Montage Gold

KONÉ GOLD PROJECT, CÔTE D'IVOIRE

Definitive Feasibility Study

National Instrument 43-101 Technical Report

3276-GREP-001

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DATE AND SIGNATURE PAGE

The independent Qualified Person (within the meaning of NI 43-101) for the purposes of this report is Sandra (Sandy) Hunter, representing Lycopodium. Further review as detailed below has been undertaken by Jonathon Abbott, Mike Hallewell, Pieter Labuschagne, Carl Nicholas, Joeline McGrath and Tim Rowles.

The author has undertaken an extensive review of Montage Gold's Koné Gold Project technical and economic data and has reviewed Montage Gold's contributions to this report.

The overall report entitled Definitive Feasibility Study, Koné Gold Project, Côte d'Ivoire was collated and signed by the following author:

Signed, this 14 March, 2022.

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1.0 SUMMARY

1.1 Introduction

This independent Technical Report comprises a Definitive Feasibility Study (DFS) for Montage Gold Corp.'s ("Montage" or "Company") Koné Gold Project ("KGP" or "Project") in Côte d'Ivoire. The DFS has been prepared by Lycopodium Minerals Pty Ltd on behalf of Montage. This Technical Report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

1.2 Property Description and Ownership

1.2.1 Property Description

The Koné Gold Project covers 300 km^2 in northwest Côte d'Ivoire 470 km northwest of Abidjan. The Koné Exploration Permit lies within the sous-prefectures of Kani and Fadiadougou within Department of Kani in the Worodougou region. The communities of Fadiadougou and Batogo lie within the permit.

The Toudian and Yarani Forest Reserves lie in part within the Koné Exploration Permit. The Toudian Reserve covers an area of approximately 5 km^2 and includes the northern portions of the planned open pits. The Company makes all efforts not to affect the forest area. The local forestry office (SODEFOR) have been kept informed as to the Company's activities and replacement planting will be undertaken as part of future programmes.

1.2.2 Ownership

The Koné Exploration Permit number 262 (PR 262) was granted to Red Back Mining (Côte d'Ivoire) SARL ("Red Back"), a wholly owned subsidiary of Kinross Gold Corporation, in 2013. In February 2017, Orca Gold Inc ("Orca") announced that it had executed a share purchase agreement with two wholly-owned subsidiaries of Kinross Gold Corporation to acquire the Koné Exploration Permit as part of a wider package of two permits and five permit applications in Côte d'Ivoire. In July 2019, Orca transferred its assets in Côte d'Ivoire to its subsidiary Montage. Montage Gold Corp listed on the Toronto Stock Exchange in October 2020.

In March 2016 and March 2019, the Koné Exploration Permit was renewed for three years. The local operating company's name Red Back Mining (Côte d'Ivoire) SARL was changed to Shark Mining CDI SARL ("Shark Mining") in August 2018.

Under the terms of the Koné Exploration Permit the company has the right to access all areas for the purpose of mineral exploration. The area is largely uninhabited outside main villages and the communities have shown significant support for the exploration activities.

1.3 Accessibility, Climate, Local Resources, infrastructure and Physiography

The Koné Gold Project is 470 km northwest of the capital Abidjan and is accessible by an established network of asphalt roads from the capital.

The communities of Fadiadougou and Batogo lie within the Koné Exploration Permit with the nearest major centre at Séguéla, 80km to the south.

Three seasons can be distinguished, namely: warm and dry (November to February), hot and dry (March to May) and hot and wet (June to October). The average annual rainfall is 1,273 mm. Average daytime maximum temperatures range from 22 to 32˚C.

There is ample space in the Permit area for the open pit, waste dumps, mineral processing plant, water catchment and tailings facilities.

The Project area is characterized by moderate relief, lying between 200 m and 420 m above sea level. The Marahoué and Yarani rivers are the main drainages in the area but the bulk of the project is cut by shallow seasonal drainages that only show significant flow in the wet season.

The Project lies within the Guinean forest-savanna ecoregion of West Africa, a band of interlaced forest, savanna and grassland running from western Senegal to eastern Nigeria and dividing the tropical moist forests near the coast from the West Ivorian savanna of the interior. Parts of the project area are covered by cashew and cotton plantations, while other areas are used for subsistence crops. There are significant areas underlain by iron rich duricrusts and are only suitable for grazing.

1.4 Geology and Mineralization

The Koné Exploration Permit lies within the Birimian Baoulé-Mossi domain, which in the Project region comprises metamorphosed sediments, volcaniclastics and volcanics flanked to the west by basement tonalite and diorites.

Much of the Project area is covered by duricrust with only very rare outcrop and deep weathering. Local stratigraphy comprises a moderately westerly dipping sequence of mafic volcanics, which are intruded by an approximately 250 m thick package of quartz diorites.

Gold mineralization generally occurs in the intrusive rocks within a wide zone of variable shearing and foliation in association with thin quartz, quartz-carbonate and sulphide veins, finely disseminated pyrite and biotite alteration. Higher gold grades are associated with greater deformation intensity and increased frequency of quartz-carbonate-sulphide veinlets.

1.5 Exploration and Resource Definition

During 2009, an 800 m by 50 m spaced soil sampling and subsequent local infill to 400 m by 50 m and 200 m by 50 m spacing identified a 2.7 km long gold in soil anomaly at Koné. The results of follow up trenching justified exploratory drilling leading to resource definition drilling.

Between 2009 and August 2021 the Koné mineralization has been tested by 102,249 m of drilling (54,703 m of core and 45,545 m of RC) on which the August 2021 Mineral Resource Estimate (MRE) are based.

The interpreted mineralization had been tested by generally 50 m spaced traverses of generally 50 m and rarely 25 m spaced holes extending to vertical depths of between 100 m and 550 m.

All sampling activities were supervised by field geologists.

All sample preparation and gold assaying of primary samples was undertaken by independent commercial laboratories. Analyses undertaken "in-house" were limited to immersion density measurements by Company personnel.

Information available to demonstrate the reliability of sample preparation and assaying include results for coarse blanks and reference standards along with inter-laboratory repeat and duplicate assaying.

Geological logging and storage of sample material along with documentation of analytical results is consistent with the author's experience of good industry standard practise.

Information available to demonstrate the representivity of the Koné RC and diamond drilling includes RC sample condition logs, recovered RC sample weights and core recovery measurements.

The author considers that the quality control measures adopted for the exploration and resource definition drilling have established that the sampling is representative and free of any biases or other factors that may materially impact the reliability of the sampling.

The author considers that the sample preparation, security and analytical procedures adopted for the 2010 to 2021 Koné drilling provide an adequate basis for the MRE and exploration activities.

1.6 Metallurgical Testing

A comprehensive comminution testwork programme has been carried out to date consisting of 65 JK Tech SMC, 67 Bond Ball Mill Work Index, 17 Abrasion Work Index and 12 Bond Low Energy Impact tests.

Table 1-1 shows the comminution testwork results from all studies. The predominant fresh mineralization zone is moderately hard in terms of resistance to SAG milling and crushing, but soft in terms of resistance to ball milling and has medium abrasivity.

Table 1-1 Comminution Testwork

		JK Tech SMC A x b			Ballmill Work Index		Abrasion Index		Crusher Work Index	
Ore Type	Deposit % Tonnes	No Samples	Relative Density	JK SMC A x b	No Samples	Bond BWi kwhrs/t	No Samples	Bond Ai g	No Samples	Bond CWi kwhrs/t
Fresh	87.4	53	2.75	31.3	54	11.4	10	0.419	11	17.0
Trans	5.5	9	2.69	76.5	9	7.8	4	0.152	1	8.5
Oxide	7.0	3	1.57	*	4	3.9	3	0.115		
Total	100.0	65	2.66	34.0	67	10.7	17	0.383	12	15.4

* Oxide Data - Off JK Tech Scale

130 leach variability samples have been tested using the optimised design conditions at anticipated site pulp temperature and dissolved oxygen levels. The metallurgical tests included oxide, transition fresh and FW fresh mineralization with results indicating that all material types are amenable to direct tank carbon in pulp (CIP) cyanide leaching.

Forecast gold recoveries were estimated based on predicted residue grades, a solution loss of 0.005 mg/L and a carbon fines loss of 0.15%. Table 1-2 estimates the gold recoveries based on the average deposit grades. Cyanide consumptions are all low to very low and lime consumptions are low for the predominant fresh zone (88%), but higher for the less dominant transition (5%) and oxide (7%) zones.

Table 1-2 Metallurgical Testwork Summary

# Samples	Domain	Processed ('000 t)	Processed Au g/t	Average Au Recovery,%	kgs/t NaCN	kgs/t CaO
53	South HW Fresh	124,107	0.69	89.10	0.26	0.55
12	North HW Fresh	469	0.56	78.13	0.37	0.43
13	South FW Fresh	17,337	0.55	87.65	0.23	0.45
17	South Transition	7,894	0.56	91.23	0.18	0.99
5	North Transition	387	0.46	88.06	0.35	0.75
21	South Oxide	9,807	0.57	93.79	0.18	2.50
9	North Oxide	917	0.47	94.17	0.13	2.79
130	**LOM**	**160,918**	**0.66**	**89.30**	**0.25**	**0.70**

The good gold recoveries, low reagent consumptions and medium-low resistance to grinding provide favourable processing economics.

1.7 Mineral Resource Estimate

MPR Geological Consultants estimated Mineral Resources for the Koné Gold Project based on the basis of RC and diamond drilling data supplied by Montage in August 2021.

Estimates with drilling spaced at around 50m by 50m are classified as Indicated, with Inferred estimates based on generally 100m spaced drilling. More broadly sampled mineralization is too poorly defined for estimation of Mineral Resources.

Recoverable resources were estimated for the Koné deposit by Multiple Indicator Kriging (MIK) of two metre down-hole composited gold grades from RC and diamond drilling. Estimated resources include a variance adjustment to give estimates of recoverable resources above gold cut-off grades for selective mining unit dimensions of five by ten by five metres (east, north, vertical) and are reported within an optimal pit shell generated at a gold price of US$1,500/oz.

The MRE have been classified and reported in accordance with NI 43-101 and classifications adopted by CIM Council in May 2014. They have an effective date of the 12th of August 2021.

Table 1-3 shows the MRE for a range of cut off grades. The figures in this tables are rounded to reflect the precision of the estimates and include rounding errors. The estimate at 0.2g/t cut-off grade represent the base case or preferred scenario. Mineral Resources that are not Mineral Reserves do not necessarily demonstrate economic viability.

Table 1-3 Indicated and Inferred Resources (August 2021)

Cut off	Indicated			Inferred		
Au g/t	Mt	Au g/t	Au moz	Mt	Au g/t	Au moz
0.1	278	0.51	4.56	32	0.35	0.36
0.2	**225**	**0.59**	**4.27**	**22**	**0.45**	**0.32**
0.3	168	0.70	3.78	14	0.56	0.25
0.4	128	0.82	3.37	9.0	0.69	0.20
0.5	99.1	0.92	2.93	5.9	0.81	0.16
0.6	76.9	1.03	2.55	3.9	0.95	0.12
0.7	59.9	1.14	2.20	3.2	1.1	0.10
0.8	46.8	1.25	1.88	1.9	1.2	0.07

1.8 Mineral Reserve Estimate

The Mineral Reserve estimate is shown in Table 1-4. This is the first Mineral Reserve estimate for the Koné project and is based on the August 2021 MRE. The figures in this table are rounded to reflect the precision of the estimates and may include rounding errors.

Table 1-4 Summary of Mineral Reserves for Koné

	Classification	Oxide			Transitional			Fresh			Total		
		Mt	Au g/t	Au M Oz	Mt	Au g/t	Au M Oz	Mt	Au g/t	Au M Oz	Mt	Au g/t	Au M Oz
South Pit	Probable	9.8	0.57	0.18	7.9	0.56	0.14	141.4	0.67	3.05	159.1	0.66	3.39
North Pit	Probable	0.9	0.47	0.01	0.4	0.46	0.01	0.6	0.57	0.01	1.9	0.5	0.03
Total	Probable	10.7	0.56	0.19	8.3	0.56	0.15	142.1	0.67	3.06	161.1	0.66	3.42

1.9 Mining

Based on the geometry of the deposit and the proximity to surface, the deposit will be mined by open pit mining using a conventional truck and shovel mining fleet.

A review of the available geotechnical information has been undertaken and a set of overall slope angles recommended by SRK Consulting. These slope angles have been used in subsequent pit optimizations and pit designs.

Pit optimizations were run using processing cost and recovery data. Mining costs were broken into base and incremental mining costs. Costs were based on West African mining contractor bids. The operating strategy assumes that mining operations will be carried out by a contractor on a cost per tonne basis, utilising a mining fleet comprised of 145t rigid body haul trucks with suitably sized loading units.

The Koné deposit will be exploited through two pits, a smaller northern pit, which reaches a depth of 130 m and a larger southern pit, which extends to a depth of 470 metres deep. The overall strip ratio for the pits is 0.90:1. Based on the assumed mining equipment, a bench height of 5 metres in the oxide, 10 metres in the transition and 15 metres in the fresh rock was designed, although geotechnical conditions allowed for up to two benches to be excavated between safety berms, within the fresh rock. There may be some opportunity to mine to higher bench heights in areas of bulk waste.

A ramp up period of 12 months was assumed at the start of the schedule the total optimized production tonnage is 35 million tonnes per year. The target for Year 1 was 9.9 Mt and 11.0 Mt of high grade crusher feed respectively, with all subsequent years targeting 11.0 Mtpa inclusive of the lower grade stockpile material after year 9. Mining dilution and recovery were not included in the schedule, as these had been included in the Resource model.

Table 1-5 shows the annualised mine production schedule with the extraction over nine year period, with a pre-strip year.

Table 1-5 Mine Production Schedule

	Description	Unit	LOM Total
Mining			
	North Pit Tonnes	Mt	1.9
	North Pit Grade	Au g/t	0.50
	South Pit Tonnes	Mt	159.1
	South Pit Grade	Au g/t	0.66
	Total Tonnes	Mt	161.1
	Total Grade	Au g/t	0.66
	North Pit Waste Tonnes	Mt	2.1
	South Pit Waste Tonnes	Mt	143.5
	Total Waste Tonnes	Mt	145.7
	Strip Ratio	W:O	0.90

Table 1-6 shows the processing schedule with the highest grade ore processed first and the remaining stockpiled lower grade ore processed after the mining operation has finished.

Table 1-6 Mine Processing Schedule

	Description	Unit	LOM Total
Processing			
	Stockpile Rehandle	Mt	71.3
	Oxide Tonnes	Mt	10.7
	Oxide Grade	Au g/t	0.56
	Transition Tonnes	Mt	8.3
	Transition Grade	Au g/t	0.56
	Fresh Tonnes	Mt	124.6
	Fresh Grade	Au g/t	0.69
	FW Fresh Tonnes	Mt	17.5
	FW Fresh Grade	Au g/t	0.56
	Total Processed Tonnes	Mt	161.1
	Total Processed Grade	Au g/t	0.66
	Total Process Recoveries	%	89.3%
	Total Recovered	000 ozs	3,059

1.10 Recovery Method

The plant design is based on a robust metallurgical flowsheet designed for optimal precious metal recovery. The flowsheet chosen is based on unit operations that are well proven in the industry. The metallurgical testwork conducted to date, has confirmed that the gold contained in the Koné mineralization is amenable to recovery via conventional cyanidation techniques and carbon adsorption.

The key criteria for equipment selection are suitability for duty, reliability, power efficiency and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements whilst maintaining a layout that will facilitate construction progress in multiple areas concurrently.

The key project design criteria for the plant are:

- Nominal throughput of 11.0 Mtpa with a grind size of 80% passing (P80) 75 µm
- Overall process plant availability of 91.3% supported by the selection of standby equipment in critical areas, reputable vendor supplied equipment and connection to an onsite LNG fired power station
- Sufficient automated plant control to minimize the need for continuous operator interface but allow manual override and control as and when required.

The treatment plant design incorporates the following unit process operations:

- Primary and closed circuit secondary crushing using a gyratory crusher and two cone crushers to produce a crushed product size P80 of approximately 38mm. Feed size preparation for a secondary crushed product is required for the grinding efficient HPGR-ball mill circuit as compared to a standard SAG mill circuit
- A crushed ore stockpile with a nominal live capacity of 22,000 wet tonnes, providing buffer storage of crushed ore with continuous reclaim feeders for the HPGR-ball mill comminution circuit
- Two parallel HPGRs in closed circuit with wet sizing screens, with undersize slurry reporting to the milling circuit via the cyclone feed hopper. Two parallel trains of ball mills in closed circuit with hydrocyclones will produce a P80 grind size of 75 µm
- Pre-leach thickening to increase the slurry density feeding the leach and carbon in pulp (CIP) circuit to minimize tankage and reduce overall reagent consumption
- Leach circuit incorporating fourteen leach tanks, arranged in two parallel trains of seven each in series, to provide 36 hours leach residence time
- A Kemix Pumpcell CIP circuit consisting of eight CIP tanks for recovery of gold onto carbon, to minimize carbon inventory, gold in circuit and operating costs. The CIP and elution circuit design is based on daily carbon harvesting

- 20 tonne split AARL elution circuit, electrowinning and gold smelting to recover gold from the loaded carbon to produce doré
- Tailings thickening to recover and recycle process water from the CIP tailings
- Tailings pumping to the tailings storage facility (TSF).

1.11 Project Infrastructure

1.11.1 Water Supply

Subject to final approval by government authorities, water will be pumped from the nearby Marahoué river, from pit dewatering and a supplementary borefield. Hydrological assessment of the river catchment indicates that the river will have flow in excess of total water demand for 7 months of the year.

The site is underlain by an overall low yielding aquifer system with an overall average groundwater piezometric level of 20 mbgl. Towards the south of the main pit, the water table is generally shallower and groundwater monitoring data indicated a fairly flat groundwater table within the pit area.

Fifteen hydrogeological exploration boreholes were drilled to determine the aquifer characteristics at the proposed Koné Gold Project mine pits. Aquifer pump tests were conducted and interpreted to derive aquifer parameters for three aquifer systems. The aquifer parameters obtained suggest overall low aquifer transmissivity with higher transmissivity associated with fracturing along geological structures.

The numerical model simulations concluded that pit de-watering will require abstraction in the order of 3,000 to 6,000 m^3/day (34 l/sec to 70 l/sec). The overall mine pit de-watering will be supplemented by perimeter de-watering boreholes that will increase the overall water-make from the mining activities slightly. It is not expected that mining will supply more than 15 to 25% of the total water balance.

Potable water for the camp and offices will be supplied from dedicated boreholes. Water quality analyses and assessment will be completed to determine any water treatment requirements.

Harvested river water, pit de-watering and supplementary borefield water will be pumped to a water storage facility (WSF) downstream of the process plant. Surface runoff from the mining area, ROM pad and stockpiles will gravity flow to this WSF. The WSF will have a capacity of approximately 7.2 Mm^3 and will enable accumulation of water during the wet season and drawdown in the dry season. In addition, water will be recycled from the tailings storage facility to the process water pond.

The processing, potable and dust suppression water requirements will be in the order of 30,000 m^3/day. The site water balance indicates that sufficient water will be available for the duration of the life of mine with the proposed WSF, river harvesting, pit de-watering and supplementary borefield.

1.11.2 Power Supply

A LNG/Solar Hybrid power plant has been assessed as the optimal power supply combination, following proposals received from West African power providers for the development of the Koné Gold power plant.

The Koné Plant is estimated to have a Maximum Demand of 44.8 MW, an average annual demand of 37 MW and an expected annual electricity energy consumption of 303 GWhr/yr. The solar farm will generate 22.7 MW of solar energy which will be coupled with an 8.8 MW of Battery Energy Storage System (BESS).

The annual power station contract payments are $20 million per annum over 5 years with an estimated transfer payment of $38M in year 6. The annual operating and maintenance cost for the power supply is estimated at $0.0998/kWhr for the hybrid gas/solar power station.

The solar PV and Battery Energy Storage Systems integration is expected to offset up to 16% of the plant electricity energy requirements, providing a fuel cost saving of $4 million per annum. The carbon footprint for the power supply will also be reduced with the solar PV offsetting up to 23,000 tonnes/year of CO_2 compared to a standalone LNG power plant. Dedicated hybrid power station control systems will be utilized to optimize the renewable energy yield whilst ensuring the security and reliability of the power supply is maintained at a high level.

1.11.3 Tailings Storage Facility

The tailings management arrangement comprises a single tailings storage facility (TSF) confined by a cross valley embankment and in-pit deposition when mining in south pit is completed. Initially the TSF will be constructed to store the tailings and will be raised annually until the mining in south pit is completed (after year 9). Tailings will be deposited in south pit for the final 6.5 years of processing.

The TSF basin will be lined with HDPE within the normal operating pond areas and a compacted soil liner elsewhere to reduce seepage. In addition, a system of underdrainage, embankment drainage and sub-liner drainage will be constructed to reduce seepage and aid consolidation of the tailings. Tailings will be deposited subaerially with the supernatant pond located away from the embankment. Water will be recovered from the supernatant pond by a suction pump with floating intake located in a channel excavated adjacent to an access causeway.

Following the completion of the mining in year nine, tailings will be deposited into the pit via four spigots located around the perimeter of the pit. The pumps will be moved progressively up the ramp as the tailings level increases. Water will be extracted from the decant pond using floating intake lines. The pond volume will be at its highest at the first year as the TSF pond will be pumped to the pit to let the TSF commence the closure process promptly. The in pit pond volume will be gradually pumped back to process plant and the pond will be reduced in the final years of operation.

The TSF will be closed and rehabilitated after deposition transferred to pit. Closure spillways will be formed to prevent water accumulating on the facilities and a waste rock cover will be placed over the tailings prior to topsoiling and revegetation.

1.12 Market Studies and Contracts

No formal market studies have been undertaken. The final product of the Koné project will be gold / silver doré bars, which will be shipped to a refinery for processing. The refined gold can either be sold by the refinery or bullion returned to the company. Preliminary quotations have been received from a refinery and transport provider.

1.13 Environmental

Environmental matters during the development phase are administered by the Ministry of Environment, Urban Sanitation and Sustainable Development and by the National Environmental Agency (Agence Nationale de L'Environnement (ANDE)). During exploitation, the Ivorian anti-pollution centre monitors environmental concerns.

The Environment Code applies to mining installations and includes the minimum environmental impact study requirements and details the relevant rules and procedures for environmental and social impact assessments for development projects. The Mining Code requires that all mining title applicants (excluding artisanal) submit an Environmental and Social Impact Study (EIES) to the DGMG and ANDE and all other institutions as required by the Mining Decree. The Mining Code also includes provisions regarding mine closure. To ensure environmental protection, mining titleholders must open an escrow account in a leading Ivoirian financial institution at the beginning of mining operations, to be used to cover costs related to the environmental management and mine closure plans. Other environmental legislation that may impact upon mining projects include the Water Code and the Forestry.

Côte d'Ivoire has been a member of the Extractive Industries Transparency Initiative (EITI) since 2008. The Mining Code also requires adhesion to good governance principles, including the Equator Principles and the EITI principles. Mining titleholders must issue ITIE reports.

There are currently no objections to the development of the Project. The Project has completed baseline data collection, to inform environment management plans. There are protected forest reserves affected and adjacent to the Project, which will be assessed during the current environmental and social impact assessment. The Project is located relatively close to the communities of Batogo, Fadiadougou and Manabri, and preliminary investigations indicate that these communities are positive towards the company.

Montage Gold is committed to managing the impacts of its operations, in conformance with recognized international best practice. The company has initiated the impact assessment process, with the development and submission of the terms of reference for the impact assessment. Results may be used to improve the design, as well as maximize the benefits without incurring excessive costs. In accordance with continual improvement processes, there are several strategies that can be used to support the Project, such as:

- Ongoing monitoring of wildlife presence in the Project area, such that management measures can be adapted to reflect changing conditions
- Assessing requirements of each of the classified forest reserves
- Ongoing community engagement, including information sharing as well as support initiatives and infrastructure development
- Optimizing the energy mix between LNG and solar
- Maintaining a grievance procedure to identify and pre-empt potential issues.

1.14 Capital and Operating Costs

1.14.1 Capital Cost

The capital estimate is summarized in Table 1-7 and Table 1-8. The initial project capital cost is estimated at US$543.9M, including a contingency allowance of US$50.5M.

Table 1-7 Captital Estimate Summary (4Q21, ±15%)

Main Area	US$M
Mine	39.6
Process Plant	320.7
TSF	50.6
Camp	3.3
Resettlement	9.5
EPCM	39.4
Owners Costs	30.3
Subtotal	**493.3**
Contingency	50.5
Grand Total	**543.9**

The duration of the detailed design and construction phase of the Project has been estimated to be 31 months commencing with the Marahoué pump station and the WSF to ensure sufficient water is available for processing. The plant is estimated to take 27 months to construct. The Mining contractor will mobilise 15 months prior to the start of processing.

The total LOM capital cost is estimated at US$835.6M, including sustaining capital costs of US$291.7M, as shown in Table 1-8. The LNG power plant and camp will be financed under a 5 year Build Own Operate Transfer (BOOT) contracts.

Table 1-8 Sustaining Capital Estimate Summary (4Q21, ±15%)

Main Area	US$M
Camp	5.7
TSF	59.4
Power	138.0
Process Plant	31.9
Closure	56.7
Grand Total	**291.7**

1.14.2 Operating Cost Mining

Contract open pit mining costs were derived from a tender process involving several West African mining contractor who were provided with a detailed mining plan. The average open pit operating cost (US$/t mined) is shown Table 1-9.

Table 1-9 Mining Costs

	Mineralized Rock (US$/t)	Waste Rock (US$/t)	Owners Costs (US$M/t)	Total Rock (US$/t))
Total	2.87	2.39	2.2	2.73

A diesel price of $0.85/L was used.

1.14.3 Operating Cost Process and Infrastructure

The process operating cost estimate has been compiled from a variety of sources, including metallurgical testwork, Montage advice, OMC comminution modelling, first principle calculations, vendor quotations and the Lycopodium database.

The process estimate comprises the following major cost centres:

- Plant and related infrastructure power
- Plant consumables, including mill media and liners, reagents and diesel for fixed plant equipment and plant mobile equipment
- Plant maintenance materials, including mobile equipment parts
- Laboratory
- Plant and administration labour
- General and administration costs.

The process operating cost includes all direct costs to produce gold/silver doré for the Project. The battery limits are the ROM feed into the primary crusher (ROM loader by Mining), production of doré in the gold room and discharge of tailings at the TSF.

Process operating costs are presented in United States Dollars (US$), to an accuracy of ±15% and are based on pricing obtained during the fourth quarter of 2021. Process operating costs have been developed for each major domain. Operating costs were developed using the plant parameters specified in the process design criteria. Table 1-10 presents the operating cost summary. In addition to the processing costs, LOM rehandle costs equate to $0.93/t processed.

Table 1-10 Process Operating Cost (4Q21, ±15%)

Cost Centre	Fixed US$'000/y	Variable Processing Costs *$/t)			LOM
		Oxide	Transition	Fresh	Fix & Var US$/t
TOTAL	17,448	5.08	5.07	6.16	7.62

G&A costs have been estimated at $10.1M/yr.

Table 1-11 shows the LOM cash cost and unit cost.

Table 1-11 Cash Cost and Unit Cost Summary (@$1,600/oz)

Description	LOM (AISC $/oz)	LOM ($/t processed)
Mining	261	4.95
Processing	424	8.04
G&A	49	0.93
Royalties	104	1.97
Total Cash Cost	838	15.89
Sustaining Capital	77	1.46
Closure	19	0.35
All-in Sustaining Costs	933	17.71

1.15 Economic Analysis

An economic analysis has been carried out for the project using a cash flow model. The model has been constructed using annual cash flows taking into account annual processed tonnages and grades for the CIP feed, process recoveries, metal prices, operating costs, refining charges, royalties and capital expenditures (both initial and sustaining). Unless otherwise stated all currencies refer to US$. The financial analysis used a base price of US$1,600/oz. The financial assessment of the project is carried out on a "100% equity" basis and the debt and equity sources of capital funds are ignored. No provision has been made for the effects of inflation. Current Côte d'Ivoire tax regulations are applied to assess the tax liabilities. Discounting has been applied mid year from the first year of operation. The results of the financial model are summarized in Table 1-12. A breakdown of the annualised operating and economic details can be found in Tables 22-4 and 22-5.

Table 1-12 Financial Model Summary @ $1,600/oz

Description	Units	LOM
Feed Tonnage	Mt	161.1
Waste Rock	Mt	145.7
Total Mined	Mt	306.7
Strip Ratio	W:O	0.90
Feed Grade Processed (average)	g/t	0.66
Gold Recovery (average)	%	89.3%
Gold Production	'000 oz	3,059
Annual Gold Production (average)	'000 oz/y	207
Pre-production Capital Cost	US$M	(544)
Sustaining Capital Cost	US$M	(292)
Total Capital Cost	US$M	(836)
Net Revenue	US$M	4,890
Selling Costs	US$M	(14)
Royalties	US$M	(318)
Total Operating Costs	US$M	(2,281)
EBITDA*	US$M	2,315
Tax	US$M	(365)
Net Cashflow after Tax	US$M	1,115
$NPV_{5\%}$ After Tax	US$M	746
IRR	%	34.8%
Cash Cost	US$/oz	838
AISC	US$/oz	933

* EBITDA is a non GAAP financial measure

Table 1-13 shows the project sensitivity of the NPV, IRR, Cash Cost and AISC with gold price.

Table 1-13 Project Sensitivity

Gold Price	1,400	1,500	**1,600**	1,650*	1,700	1,800	2,000
$NPV_{5\%}$	417	582	**746**	799	881	1,043	1,367
IRR	21.8%	28.3%	**34.8%**	37.0%	40.3%	47.0%	60.9%
Cash Cost	825	831	**838**	858	861	869	884
AISC	920	927	**933**	953	957	964	979
Payback	3.8	3.1	**2.7**	2.5	2.4	2.1	1.8

*** Three year trailing average (December 31, 2021)**

1.16 Recommendations

1.16.1 Geology

Recommendations for future work at Koné comprises additional exploratory and resource drilling which includes infill and extensional/close off drilling at Koné designed to improve confidence in the definition of mineralization extents.

1.16.2 Environmental

By initiating the impact assessment process early, results have been used to improve the design, increasing the benefits of the study without incurring excessive costs. To support the Environmental and Social Management Plan, the following activities are recommended to continue:

- Ongoing monitoring of wildlife presence in the Project area
- Monitoring of impacts on each of the classified forest reserves
- Recording of community engagement, including information sharing as well as support initiatives and infrastructure development
- Maintaining a grievance procedure to identify and pre-empt potential issues.

1.16.3 Mining

As part of the geotechnical review, SRK Consulting (UK) Ltd recommended that the development of a 3D deposit-scale structural model to assist with the spatial prediction of local/inter-ramp scale structures significant to geotechnical analysis. Given the competent nature of the rock mass, structures could have influence on overall pit slope stability depending on their orientation relative to the slope. Further investigations should consider geophysics data and field mapping to assist with characterising potential fault zones.

1.16.4 Metallurgical Testwork

The sizing and performance guarantees associated with the installation of HPGRs will require further laboratory and pilot scale vendor testing.

1.16.5 Infrastructure

Water

The numerical groundwater modelling will be advanced.

Tailings Storage Facilities and Water Management

To advance the design to the next phase of study the following activities are recommended to be included in the scope of the definitive feasibility study:

- Expanding topography to include all areas potentially impacted by a dam break
- Sterilization of infrastructure footprints
- Site inspection visit by KP project manager, COVID-19 permitting
- Update of the design based on the findings of the above investigations.

Electric Power Supply

Further options for the LNG supply chain, including gas storage are to be explored in the next phase.

2.0 INTRODUCTION

The Koné Gold Project lies within Montage Gold's Koné Exploration Permit in Côte d'Ivoire. The Project lies within the sous-prefectures of Kani and Fadiadougou, 470 km northwest of Abidjan. In February 2017, Orca Gold Inc ("Orca") announced that it had executed a share purchase agreement with two wholly-owned subsidiaries of Kinross Gold Corporation to acquire the Koné Exploration Permit as part of a wider package of two permits and five permit applications in Côte d'Ivoire. In July 2019, Orca transferred its assets in Côte d'Ivoire to its subsidiary Montage. Montage successfully listed on the Toronto Stock exchange in October 2020.

The Project comprises an open pit mining operation with the process plant, water storage facility and tailings storage facility located near the pit.

2.1 Basis of Technical Report

This Technical Report has been compiled by Lycopodium Minerals Pty Ltd (Lycopodium), Brisbane, Australia, with sections prepared and signed off by the seven Qualified Persons (QPs – identified below), to prepare a Canadian National Instrument NI 43-101 compliant Definitive Feasibility Study.

The qualified persons (QPs) responsible for Sections in this Technical Report are as follows:

- Jonathon Abbott (MPR Geological Consultants Pty. Ltd.), responsible for report Sections: 1.7, 12 and 14
- Sandy Hunter (Lycopodium Minerals Pty. Ltd.), responsible for report Sections: 1.10, 1.11.2, 1.14, 17, 18.2, 21, 22 (overview), 25.1.3 and 26.1.6
- Michael Hallewell (MPH Minerals Consultancy Ltd.), responsible for report Sections: 1.6, 1.16.4, 13, 25.1.1 and 26.1.3.
- Pieter Labuschagne (AGE Pty. Ltd.), responsible for report Sections: 1.11.1, 1.16.5 (part), 18.1, 25.1.4 and 26.1.4
- Carl Nicholas (Mineesia Ltd.), responsible for report Sections: 1.13, 1.16.2, 20, 25.1.6 and 26.1.7
- Jo McGrath (Carci Mining), responsible for report Sections: 1.8, 1.9, 1.16.3, 15, 16, 21.2.1, 21.3.1, 25.1.2 and 26.1.2
- Tim Rowles (Knight Piésold Pty. Ltd.), responsible for report Sections: 1.11.3, 1.16.5 (part), 18.1.7, 18.1.9, 18.1.10, 18.1.11, 18.3, 18.4 and 26.1.5.

2.2 Qualified Person Site Inspection

A summary of the QP site visits is detailed in Table 2-1.

Table 2-1 Summary of QP Site Visits

Qualified Person	Site Visit
Jonathon Abbott	23/08/18 – 24/08/18
Carl Nicholas	13/03/21 – 18/03/21
Jo McGrath	17/11/21 - 19/11/21

2.3 Effective Date

The Effective Date of this report is 14 February 2022. There were no material changes to the scientific and technical information of the Project between the Effective Date and signature date of this Report.

2.4 Abbreviations

µm	Micron
a	Annum
AAS	Atomic Absorption Spectrometry
ANDE	Agence Nationale de L'Environnement
Ag	Silver
Au	Gold
BOOT	Build Own Operate Transfer
CIP	Carbon-in-Pulp
EITI	Extractive Industries Transparency Initiative
ESIA	Environment and Social Impact assessment
g/l	grams per litre
g/t	grams per tonne
GPS	Global Positioning System
ha	Hectare
HPGR	High Pressure Grinding Rolls
HQ	Exploration drill size (96 mm OD / 63.5 mm ID)
IRR	Internal Rate of Return
Km	Kilometer
km^2	square kilometres
kV	Kilovolt
kWh	kilowatt hour
l	Litre
l/s	litre per second
M	Million
MIK	Multiple Indicator Kriging
Min	Minutes
Mm^3	million cubic metres
Mt	million tonnes
Mtpa	million tonne per annum
NPV	Net Present Value
NQ	Exploration drill size (75. 5mm OD / 47.6 mm ID)
oz	31.10348 grams
P_{80}	80% of a unit process product particle size is below a given size, based on particle size distribution
PEA	Preliminary Economic Assessment

ppm	parts per million
PQ	Exploration drill core size (122.6 mm OD / 85 mm ID)
QAQC	Quality Assurance Quality Control
RC	Reverse Circulation
RF	Revenue Factor
ROM	Run-of-Mine
RQD	Rock Quality Designation
SABC	Semi Autogenous Ball mill Crushing
SAG mill	Semi Autogenous Grinding mill
SD	Standard Deviation
SG	Specific Gravity
t	metric tonne (1,000 kg)
TOR	Terms of Reference
TSF	Tailings Storage Facility
TTG	Tonalite-Trondhjemite-Granodiorite
WSF	Water Storage Facility

3.0 RELIANCE ON OTHER EXPERTS

The author of this report is not qualified to provide comment on the legal issues associated with the Project, including any agreements, joint venture terms and the legal status of the exploration permits and mining tenure included in the Project. Montage retains copies of the relevant legal titles as provided by the government of Côte d'Ivoire to the Koné permit (Permis de Recherche Miniére No. 262).

Lycopodium has relied on the advice of other experts in the preparation of this report as follows:

General: the Author has relied on information provided by Montage for Sections 1.2, 1.3, 4, 5 and 6.

Geology: the Author has relied on information provided by Montage for Sections 1.4, 1.5, 1.16.1, 7, 8, 9, 10 and 11, 26.1.1.

Metallurgical Testwork: the Author has relied on information provided by MPH Minerals Consultancy Ltd for Sections 1.6, 1.16.4, 13, 25.1.1 and 26.1.3. Lycopodium has reviewed the metallurgical testwork results and concurs with their interpretation.

Resources: the Author has relied on information provided by MPR Geological Consultants Ltd for Sections 1.7, 12 and 14.

Mining: the Author has relied on information provided by Carci Mining for Sections 1.8, 1.9, 1.16.3, 15, 16, 21.2.1, 21.3.1, 25.1.2, and 26.1.2.

Hydrogeology: the Author has relied on information provided by AGE Pty. Ltd. for Sections 1.11.1, 1.16.5 (part), 18.1, 25.1.4 and 26.1.4.

Tailings Storage: the Author has relied on information provided by Knight Piésold Pty. Ltd. for Sections 1.11.3, 1.16.5 (part), 18.1.7, 18.1.9, 18.1.10, 18.1.11, 18.3, 18.4 and 26.1.5.

Environment and Social: the Author has relied on information provided by Mineesia Limited for Sections 1.13, 1.16.2, 20, 25.1.6 and 26.1.7.

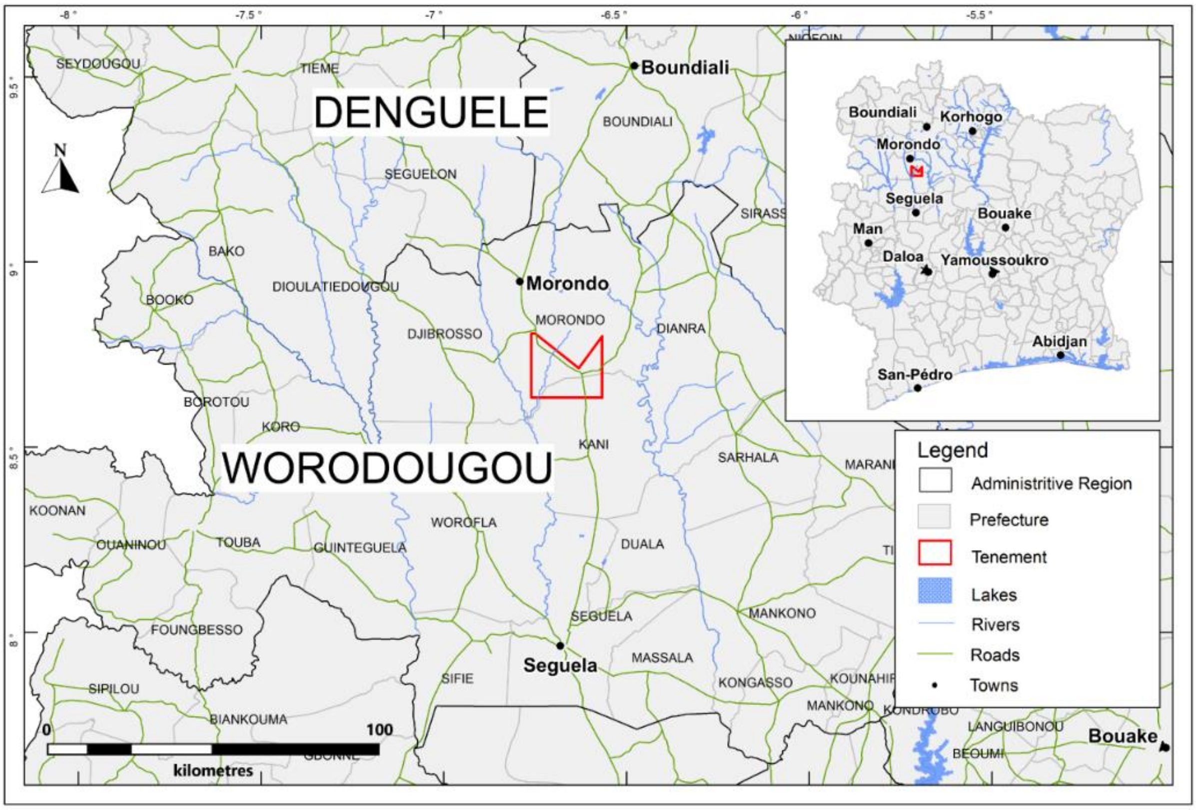
Financial: the Author has relied upon the financial analysis by Montage in Sections 1.15 and 22 of this report. Lycopodium has reviewed the inputs and basis for the financial analysis.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Koné Exploration Permit covers 300 km^2 in northwest Côte d'Ivoire around 470 km northwest of Abidjan. It straddles the sous-prefectures of Kani and Fadiadougou within the Department of Kani in the Worodougou region (Figure 4-1). The communities of Fadiadougou and Batogo lie within the licence with the nearest major centre at Séguéla, 80km to the south.

Figure 4-1 Project Location Map



Source Montage, May 2021

The Toudian and Yarani Forest Reserves lie in part within the Koné Exploration Permit. The Toudian Reserve covers an area of approximately 5 km^2 and includes the northern portions of the open pit design. The Company makes all efforts not to affect the forest area. The local forestry office (SODEFOR) is kept informed as to the Company's activities and replacement planting will be undertaken as part of future programmes.

4.2 Mineral Tenure

4.2.1 Mineral Tenure Framework

The Republic of Côte d'Ivoire reformed the Mining Code in March 2014. Exploration Permits are awarded by presidential decree after Ministerial approval from the Ministry in charge of mines and comprise five different types of titles as follows:

- Prospecting Permit - Up to 2,000 km^2, non-exclusive and granted for one year
- Exploration Permit - Up to 400 km^2, exclusive and granted for 4 years, plus 2 renewals of 3 years with the possibility of a third renewal for 2 years under extraordinary circumstances
- Mining Licence - Granted for up to 20 years with option of 10-year renewals
- Semi Industrial Mining Licence - Ivorian nationals or Ivorian majority cooperatives of companies only, up to 1 km, 4-year period, renewable
- Artisanal Mining Licence - Ivorian Nationals or Ivorian majority co-operatives only, maximum of 25 Ha. 2-year period, renewable.

Once Exploration Permit applications are submitted, coordinates of the area applied for are verified to ensure no overlap with other applications or granted licences. At this stage the applicant's technical and financial capability to undertake the work program proposed in the application is assessed. The application is then assessed by a mining commission and if approved a draft decree is presented by the Minister for Mines to a presidential cabinet for signature.

For a company to take a mining licence, the company must form a local entity and the state can take up to 10% free carry in any mining operation and up to 15% with further financial contribution. Mining royalties for gold extraction vary with gold price (Table 4-1).

Table 4-1 Summary of Royalties

Gold Price US$/ounce	<1,000	1,001-1,300	1,301-1,600	1,601-2,000	>2,000
Percent Royalty	3.0	3.5	4.0	5.0	6.0

4.2.2 Project Mineral Tenure and Ownership

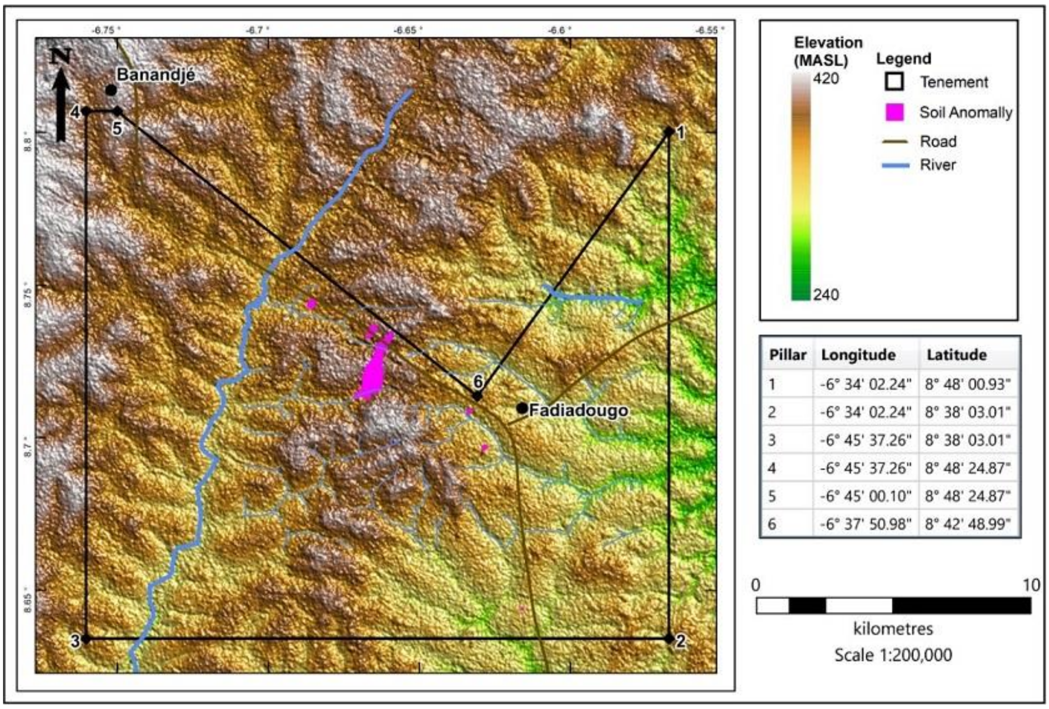
The Koné Exploration Permit number 262 (PR 262) was granted to Red Back on 22nd March 2013 under the 1995 Mining Code. It was renewed in March 2016 and March 2019 under the 2014 Mining Code for three years committing Montage to the expenditure requirements in Table 4-2.

Table 4-2 Exploration Permit Expenditure Commitments

	CFA	US$
March 2019 to March 2020	395,000,000	681,000
March 2020 to March 2021	451,000,000	778,000
March 2021 to March 2022	220,000,000	379,000

The Koné Exploration Permit will expire on 22nd March 2022. An application has been submitted to renew for a further two years. Figure 4-2 shows the lease boundary relative to the SRTM elevation along with latitude and longitude of the lease corners.

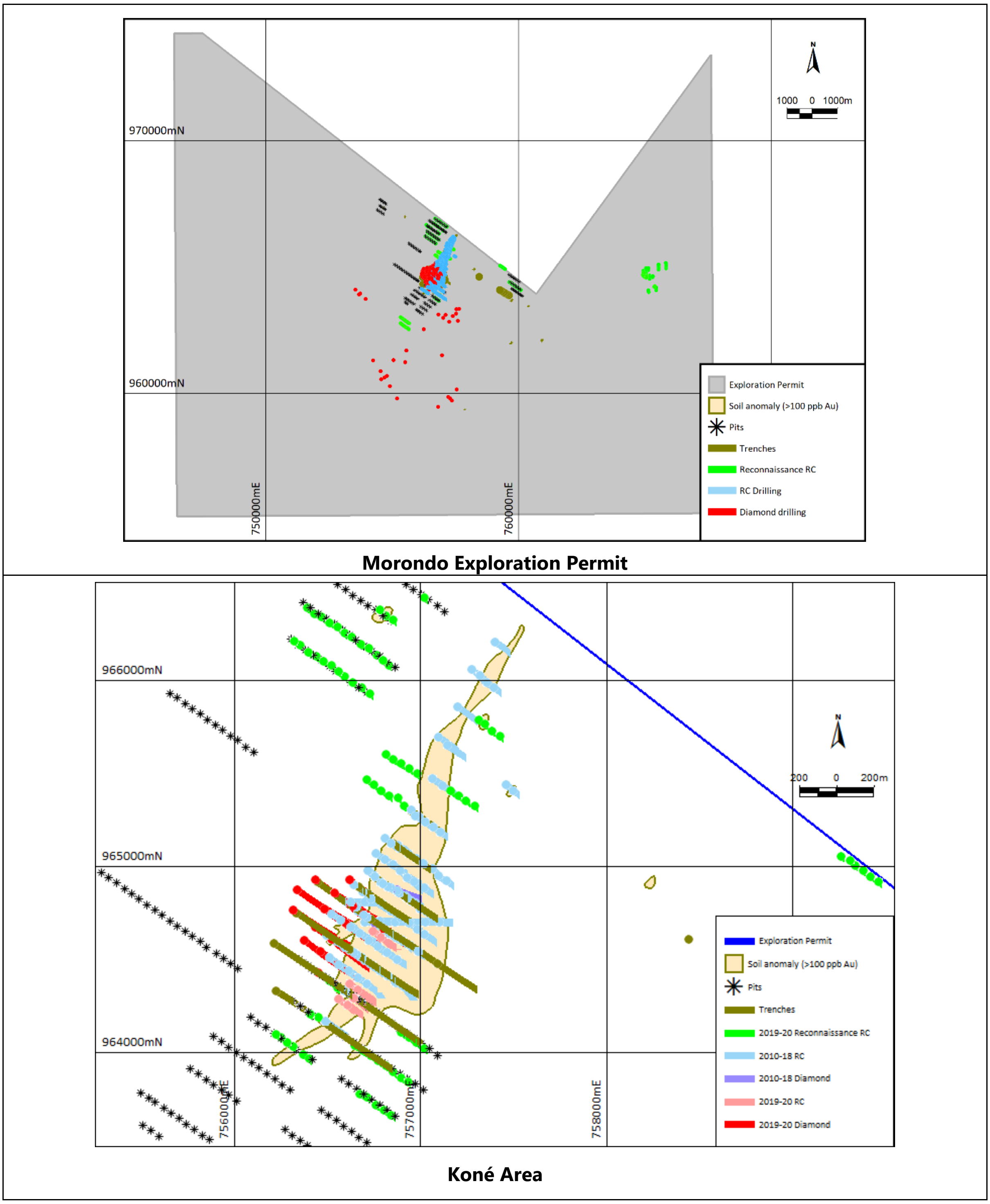
Figure 4-2 Exploration Permit Boundaries and SRTM Elevation



Source: Montage, August 2020

Figure 4-3 presents the locations of trench and drill hole sampling relative to the soil anomaly and Exploration Permit which is shown as a thick black line. The coordinate system used in this figure and throughout this report is World Geodetic System (WGS84) Zone 29 N coordinates. The Project is centred at around 757,000 mE, 963,300 mN.

Figure 4-3 Soil Anomaly, Trenching and Drilling Locations



Morondo Exploration Permit

Koné Area

Produced by MPR in February 2021 from information supplied by Montage

Under the terms of the Exploration Permit the company has the right to access all areas for the purpose of mineral exploration. The area is largely uninhabited outside main villages and the communities have shown significant support for the exploration activities.

To the extent known, the Project is not affected by any other factors that would affect access, title, or the right or ability to perform work on the properties, which would be considered as abnormal to established exploration work practices in the local and regional setting.

The Company has all the permits necessary to conduct the proposed work program on the property.

On February 1st 2017, Orca announced that it had executed a share purchase agreement with two wholly-owned subsidiaries of Kinross Gold Corporation whereby Orca would acquire from Kinross all the issued and outstanding common shares of two wholly-owned exploration companies located and operating in Côte d'Ivoire, which collectively own and have rights to the Koné Exploration Permit and one other exploration permit and five exploration permit applications in Côte d'Ivoire.

The transaction was subject to approval of the Acquisition by the Minister of Industry and Mines of Côte d'Ivoire which was received in October 2017 and the transaction closed on October 2, 2018.

On August 13, 2018, as a condition to the closing of the transaction with Kinross, the name of Red Back Mining (Côte d'Ivoire) SARL was changed to Shark Mining CDI SARL and this change has been registered with the relevant Government departments.

On July 13, 2019, Orca concluded a corporate restructuring of its assets in Côte d'Ivoire that resulted in the creation of a new subsidiary, Montage Gold Corp. Orca transferred all of its permits and permit applications in Côte d'Ivoire to Montage and subsequently entered into a share purchase agreement with Avant Minerals Inc ("Avant") pursuant to which Avant transferred its assets in Côte d'Ivoire and Burkina Faso and net cash of C$ 3.8 million to Montage. Montage subsequently raised a further C$8.2 million to fund exploration activities in Côte d'Ivoire. Orca reports Montage as a subsidiary in its financial statements.

On December 19, 2019 Maverix Metals acquired the 2% net smelter return royalty on the Koné Exploration Permit from Kinross.

Once a licence is granted by decree the company has legal right to explore for mineral commodities, the code also encompasses rights and access of the legal owners of the land and any activities undertaken by the company are undertaken with permission of the local stake holders.

The company is in continuous communication with the local communities and should any exploration activities effect farming or other activities of the local holder, clear guidelines are provided both under the mining code and by the department of agriculture and the relevant authorities. Works undertaken by the company to date has been 'low impact' from both environmental and community perspectives and there has been no direct effect on the environment or activities of local stakeholders.

Works undertaken by the Company to date has been 'low impact' from both environmental and community perspectives and there has been no direct effect on the environment or activities of local stakeholders.

In order to convert an Exploration Permit to a Mining Licence, the Company must complete an Environment and Social Impact assessment (ESIA) which both elaborates a community development plan jointly with local communities and administrative authorities and constitute a development fund for the benefit of the local villages identified as "affected localities".

There are no particular environmental stipulations for an Exploration permit though the company should operate as guided by the Equator Principles. Applications and granted licences cannot cover gazetted forest areas and access to farmland or areas held by local stakeholders must be negotiated with the stakeholders.

Under the 2014 Mining code holders of an exploration permit are required to respect and comply with the principles of good governance in particular as stipulated in the Equator Principles and Extractive Industries Transparency Initiative (EITI). This means companies holding exploration permits must, at all stages project development be responsible for respecting, protecting and promoting human rights among communities affected by extractive activities. In addition, companies holding a valid mining title must report to the national office of the EITI all mining revenues and social contributions paid to the state. The company is required to provide regular statutory filings to the state and must undertake exploration activities described in the decree for the permit held.

To the extent known, the Project is not subject to any environmental liabilities.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, PHYSIOGRAPHY

5.1 Accessibility

Côte d'Ivoire offers relatively well developed road infrastructure, the second largest port in West Africa, and a modern airport with a national airline that serves all of the major capital cities in the region.

The Project is accessible by an established network of roads from the capital Abidjan. The 230km road between Abidjan and Yamoussoukro is a four-lane motorway with access by sealed road via Boauflé, Daloa and Séguéla to Kani. The road from Kani to the Company's base in the village of Fadiadougou as far as Boundiali in the north is sealed road.

The Koné Mineral resource area lies within 1 km of the main Séguéla-Boundiali road. Bush tracks provide generally good wet and dry season access. Exploration activities can be undertaken throughout the year.

5.2 Climate

Three seasons can be distinguished, namely: warm and dry (November to March), hot and dry (March to May) and hot and wet (June to October). The average annual rainfall is 1,273 mm. Average daytime maximum temperatures range from 22 to 32˚C.

5.3 Local Resources, Infrastructure

Agriculture is crucial for the country in terms of revenues and employment, with the country being the world's largest producer and exporter of cocoa beans. Natural resources play a key role in the country's economy, especially fossil energy and ores.

Séguéla, 80km south of the Project has most modern amenities including banks, hotels and other major services. Fadiadougou, Kani and the surrounding villages provide unskilled labourers who have been trained for exploration operations. For future potential development, it envisaged that much of the professional and skilled labour would be sought from larger centres within the country.

Power is supplied to the main communities by the national power grid but the project area is not currently supplied with electricity.

5.4 Physiography

Côte d'Ivoire is a sub-Saharan nation in southern West Africa. The country is approximately square in shape. Its southern border is a 515 km (320 mi) coastline on the Gulf of Guinea on the north Atlantic Ocean. On the other three sides it borders five other African nations: Liberia to the southwest, Guinea to the northwest, Mali to the north-northwest, Burkina Faso to the north-northeast, and Ghana to the east.

The Project area is characterized by moderate relief between 200m and 420m above sea level (Figure 4-2, Figure 5-1). The Marahoué and Yarani rivers are the main drainages in the area but the bulk of the project is cut by shallow seasonal drainages that only show significant flow in the wet season.

The Project lies within the Guinean forest-savanna ecoregion of West Africa, a band of interlaced forest, savanna and grassland running from western Senegal to eastern Nigeria and dividing the tropical moist forests near the coast from the West Côte d'Ivoirian savanna of the interior. Parts of the project area are covered by cashew plantations, other areas by subsistence crops and large areas are underlain by iron rich duricrusts and are only suitable for grazing.

Figure 5-1 Photograph of Koné Resource Area (Facing North)

Source: Montage

6.0 HISTORY

Red Back applied for the Koné Exploration permit on 28th July 2008. An "Autorisation de prospection" was issued on 22nd June 2009. This allowed the start of basic exploration including soil geochemistry and geological mapping representing the first modern exploration of the area.

Table 6-1 summarizes the main field exploration activities undertaken by previous tenement owners

Table 6-1 Field exploration undertaken by previous owners

Activity	Red Back 2009-10	Sirocco 2013-14	Orca 2017-2019
Worldview imagery (km^2)	230	-	-
Ground magnetics (km^2)	4.68	-	-
Soil samples	4,877	-	-
Rock chip samples	61	2	6
Trenching (number/metres)	9/4,155 m	3/610 m	-
RC drilling (holes/metres)	8/943	43/3,431	64/13,360
Diamond drilling (holes/metres)	-	-	2/527.8

There has been no reported production from the Project. There are, however, several broad depressions within the resource area that may represent old workings of indeterminate age.

During the second half of 2009, an 800 by 50 m spaced soil sampling identified a 2.6 km long gold in soil anomaly at Koné. Infill soil sampling and trenching was completed in late 2009 and early 2010.

In July 2010, the licence application was approved by Comine (inter-ministerial committee) and an authorisation to conduct a preliminary drilling campaign was granted in September 2010.

Red Back completed eight RC holes in September 2010 but work was curtailed due to the Presidential elections and subsequent unrest.

On 22nd March 2013, the licence application was granted by Presidential decree 198-2013 under the permit number 262.

On the 22nd of May 2013 Kinross Gold signed an option agreement with Sirocco Gold Côte d'Ivoire SARL (Sirocco) covering the Koné permit. Sirocco completed several further trenches and a drill programme comprising 43 holes for 3,340m in late 2013 and early 2014.

Following the signing of an agreement to acquire the Koné Exploration permit in addition to other exploration assets in February 2017 and the receipt of Ministerial approval for the transaction in October 2017, Orca commenced work in the area drilling an RC programme in November 2017. This was followed in February 2018 by a two-hole core drilling programme and in May by the commencement of a resource definition drill programme culminating in the Mineral Resource Estimate completed in October 2018 which is described in a NI43-101 Technical Report with an effective date of the 3rd of October 2018 (Abbott, 2018). No other mineral resource estimates, including historic estimates have been produced for the Project.

Orca continued exploration in 2019 with a program of ground geophysics, pitting and soil sampling.

On July 13, 2019, Orca's assets were transferred to its subsidiary Montage and since that time Montage has been focussed on exploration in the wider Koné Exploration Permit and on diamond core drilling to test the depth extents of the Koné Deposit.

Sections 9 and 10 outline exploration activities conducted by all tenement holders.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geological Setting

The following summary of the Project's regional geological setting is derived from Goldfarb et al 2017 and Baratoux et al 2011.

Côte d'Ivoire is largely underlain by the Birimian Baoulé-Mossi domain with the west of the country underlain by the Archaean Man-Leo Shield (Figure 7-1). The Baoulé-Mossi domain contains small slivers of Archean rock, but is dominated by Lower to Middle Proterozoic Birimian rocks deformed during the Eburnean orogeny (2 to 1.8 Ga).

The domain consists of vast granitoid/gneiss Tonalite-Trondhjemite-Granodiorite ("TTG") complexes intermittently broken by narrow, elongate and generally greenschist facies metamorphosed northerly trending volcano-sedimentary belts (Goldfarb et al 2017). These greenstone belts host most of the known gold deposits of West Africa, with some exceptions such as the younger conglomerate and sandstone hosted gold found in Tarkwaian sediments that unconformably overlie the Birimian.

Three main intrusive episodes have been identified:

- Calc-alkaline biotite and amphibole bearing TTG suites, forming large generally elongate and irregularly shaped regions of granitic gneiss that were syn/post tectonically emplaced into the greenstone belts. 2,250 to 2,120 Ma

- Calc-alkaline potassic granodiorite-granite suites, biotite and K-feldspar bearing with rare amphibole and muscovite, undeformed and sub-circular or elliptical which cross cut older units, but are locally affected by shear zones. 2,120 to 2,090 Ma

- Undeformed potassic granites, occasionally metaluminous or syenitic with abundant K-feldspar often with a biotite association, amphibole is usually absent. 2,110 to 2,070 Ma.

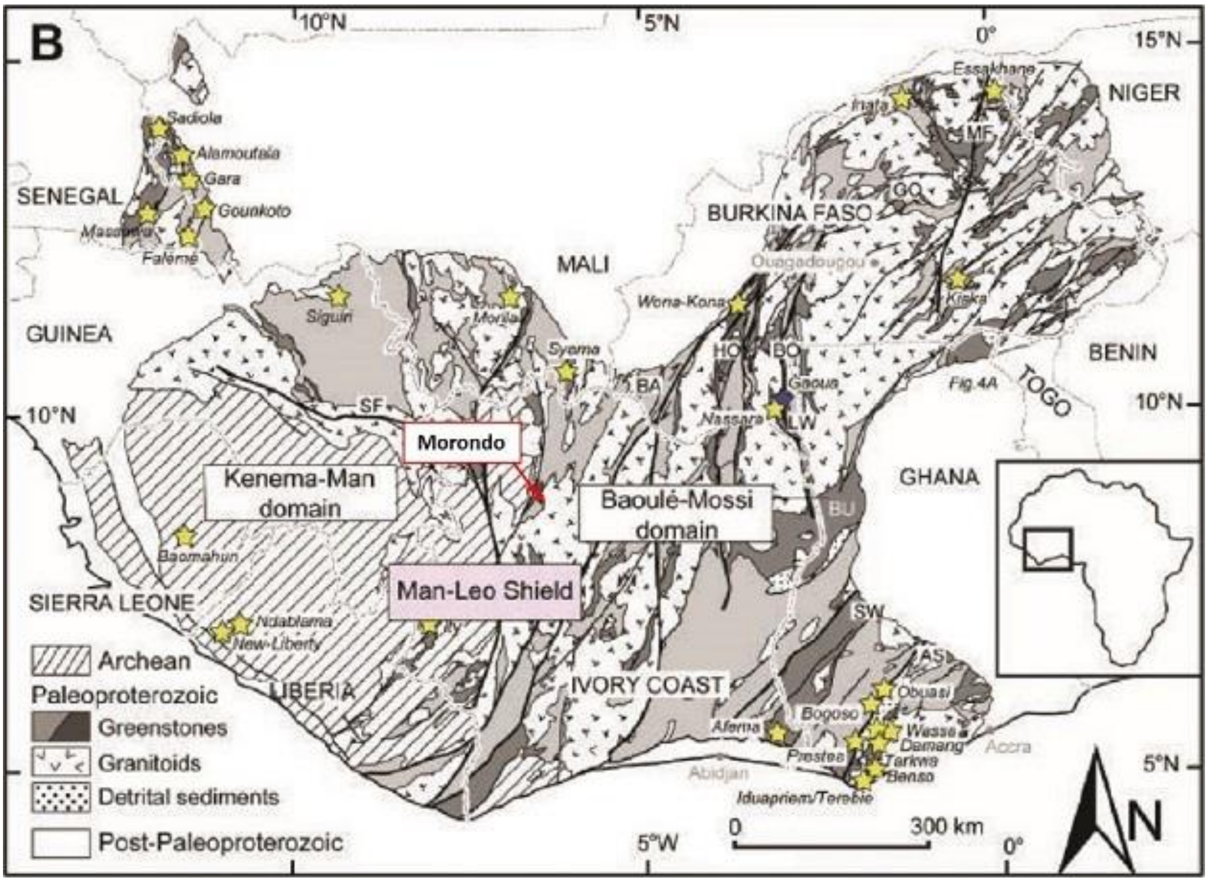
The TTG suites are commonly referred to as 'Belt Type' granites and the potassic suites are referred to as 'Basin-Type' granites reflecting the source and age of the intrusive suites. The TTG suites are derived from melting during subduction and form elongate domes or antiforms between and around the greenstone belts. The Basin Type granites are emplaced both into the sedimentary basins and the surrounding TTG suites during the later transpressional 'D2' events. They are likely the result of re-melting of the TTG suites and metasediments.

The Birimian Supergroup is formed in what is likely to have begun a rift or series of rifts and associated volcanic arcs in a Precambrian cratonic block. Basins and sub-basins formed within these arcs were filled with basal tholeiitic successions which are overlain by calc-alkaline mafic to acid volcanic rocks interstratified with clastic and chemical sediments. Subsequent orogenesis is referred to as the Eburnean Orogeny; the onset of this compressional event with accretion and amalgamation of the Paleoproterozoic arcs back on to the Archean continental margin, timing of this is now widely accepted to have been initiated ca. 2,130 Ma and continued for 25 to 30 Ma. This compressional event was followed by 100 Ma of transcurrent tectonism and exhumation. This extended tectonic period is thought to have broad implications for the formation of the orogenic gold deposits in the region.

Typically, at the district/deposit scale, mineralization is associated with secondary and tertiary structures to these primary shear zones, commonly as dilatational zones related to sinistral or oblique strike slip movement. These crustal scale structures have been reactivated throughout the history of the Birimian, initially as basin controlling extensional faults, followed by reactivation during the Eburnian as thrusts and subsequently transcurrent faults (described as D1 and D2 events during the Eburnean Orogeny).

Structurally, most mineralization is associated with the 'D2' phase of deformation where compressive stress shifted to transpression and transcurrent shearing/ strike slip faulting. Gold mineralization is typically hosted as brittle ductile quartz veins, stockworks, breccias and disseminated orebodies, usually in second order structures as dilational jogs, regional fold systems and rheology contrasts. Host rocks are highly variable as mineralization is structurally controlled and include volcanic rocks, sedimentary rocks and granites.

Figure 7-1 Geology of the Man leo Shield



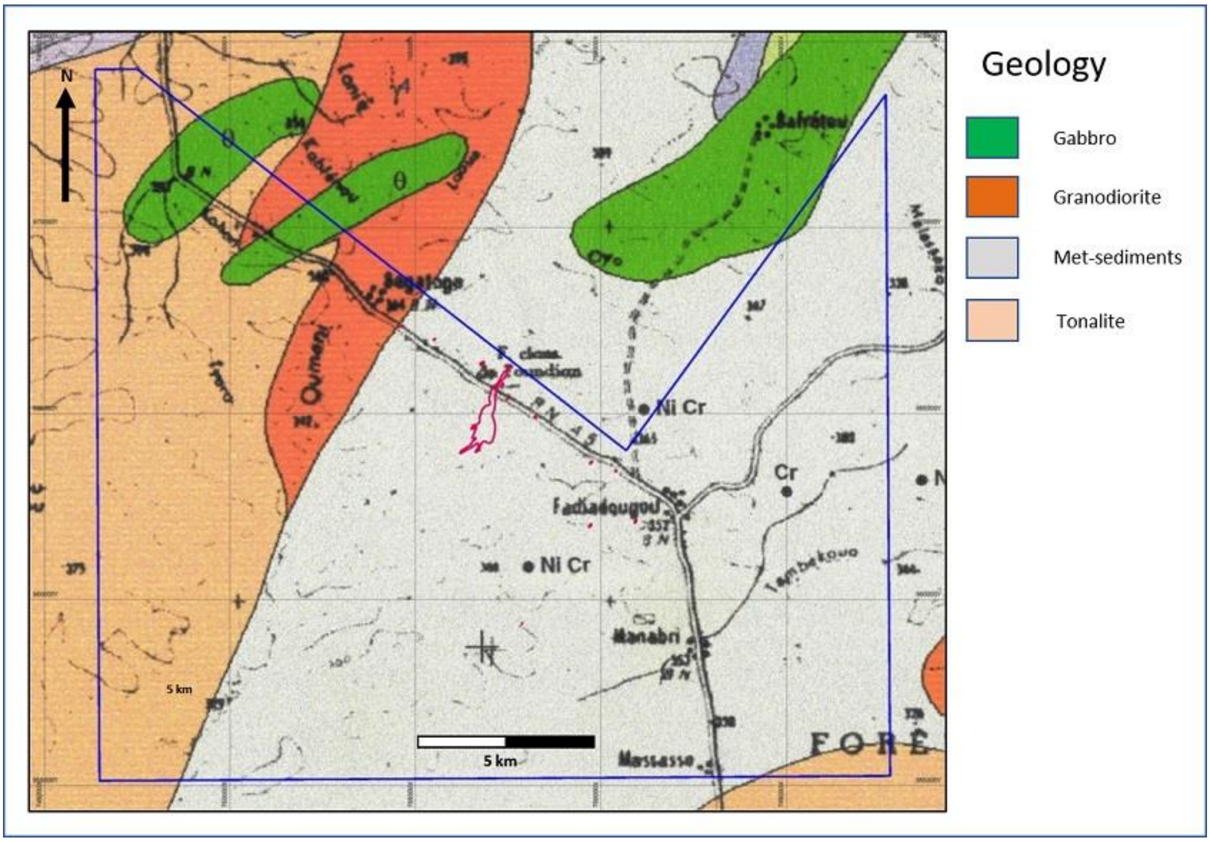
Source: Montage. Base map modified in October 2018 from Goldfarb et al, 2017.

7.2 Koné Exploration Permit Geological Setting

Much of the project area is covered by duricrust interpreted to represent remnant peneplain surfaces with only very rare outcrop and deep weathering. The local geology is not yet fully understood.

Regional mapping indicates the project overlies Birimian sediments, volcaniclastics and volcanics flanked to the west by basement tonalite and diorites (Figure 7-2). The rocks have been metamorphosed to greenschist facies. Regional aeromagnetic data shows strong north east – south west trends interpreted to reflect the distribution of underlying rock units.

Figure 7-2 Geological Map of the Koné Exploration Permit



After: 1:200,000 Geology, Mankono Sheet, 1995, Republic of Côte d'Ivoire. Source: Montage, August 2020

7.3 Koné Deposit Geological Setting and Mineralization

Koné is a mesothermal, structurally controlled gold deposit hosted within a north-south trending, westerly dipping (50･), composite package of sheeted 20-30m thick diorite intrusions which have been emplaced by multiple intrusive pulses (Figure 7-3). These diorite intrusions are of the same composition and genetically associated but display a variety of textures. The package as a whole is up to 350m in true thickness and can currently be traced along strike for 2.4km (Figure 7-4).

The diorite bodies at Koné have intruded into the contact zone between two different sequences of mafic volcaniclastic rocks which form the hangingwall and footwall of the deposit. The diorite intrusions have been age dated to 2168 ± 5 Ma indicating they were emplaced during the later stages of the Eburnian orogeny (2200 – 2100Ma).

The hangingwall volcaniclastics are characterized by a polymictic volcaniclastic sequence of proximal volcanic facies. The footwall volcaniclastics contain smaller clasts, are foliated and display strong deformation at the footwall contact of the diorite domain.

Koné is interpreted to have formed as part of a stacked thrust-shear under a compressional tectonic regime. The major thrust is located at the footwall of the diorite body where a 5-20m wide zone of shallowly plunging/horizontal tight folds can be observed within volcanic rocks immediately below the footwall contact of the diorite. This folding progressively decreases as you move outward from the contact into the footwall volcanic sequence. The upper contact of the diorite displays little to no deformation and has no mineralization related to it.

Within the diorite, higher gold grades (>1g/t) are associated with swarms of foliation parallel, 1-2mm quartz + pyrite ± chalcopyrite veinlets which form distinct corridors of mineralization. Recent observations from drill core indicate these higher grades are related to discrete zones of more intense shearing and localised slivers of highly deformed volcanic material within the diorite domain. These features are beginning to be resolved as secondary thrust shears peeling off the main footwall thrust structure. In between the high grade zones, lower grades (0.2g/t to 1.0g/t) are associated with disseminated pyrite mineralization and in general the diorite package is mineralized over most of its width, averaging >200m over the main part of the deposit, with a maximum of up to 330m (MDD015B, 330.7m grading 0.58g/t).

Regional metamorphism in the Boundiali belt is greenschist facies however the mineral assemblages observed at Koné consist of chlorite, biotite, amphibole, magnetite and pyrite with peripheral epidote-quartz-amphibole vein assemblages. These higher temperature, higher pressure mineral assemblages are interpreted to be related to the major thrust structure controlling the Koné deposit.

The deposit is intruded by multiple dykes with the majority post-dating mineralization by as much as 50 Ma and seen cutting foliation. One set of dykes, known as the early green dyke displays deformation and folding.

Figure 7-3 Section through the centre of the deposit, displaying the major units and DFS Pit Design

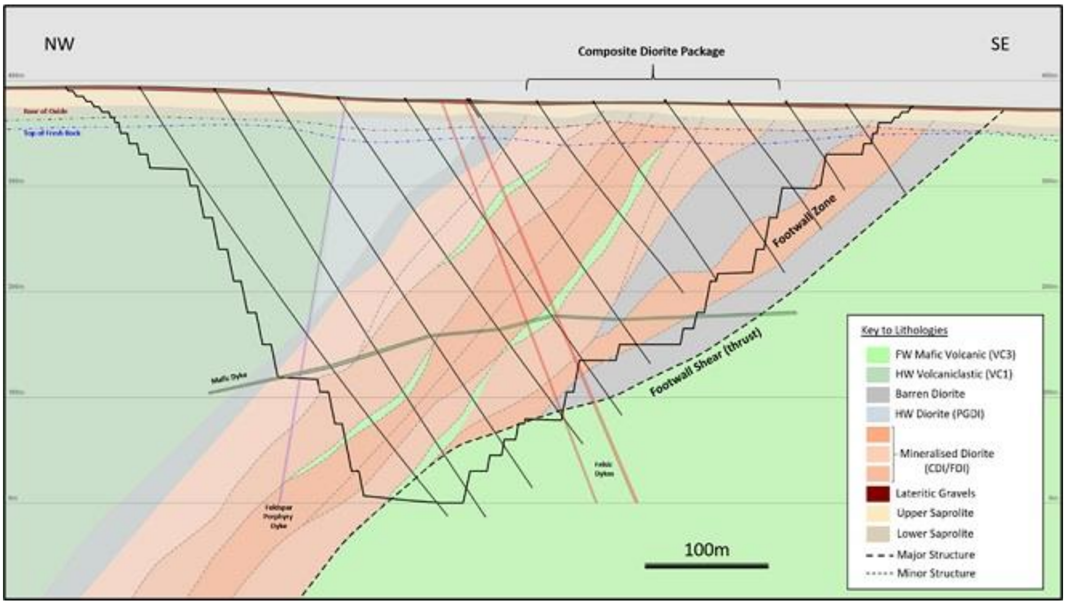
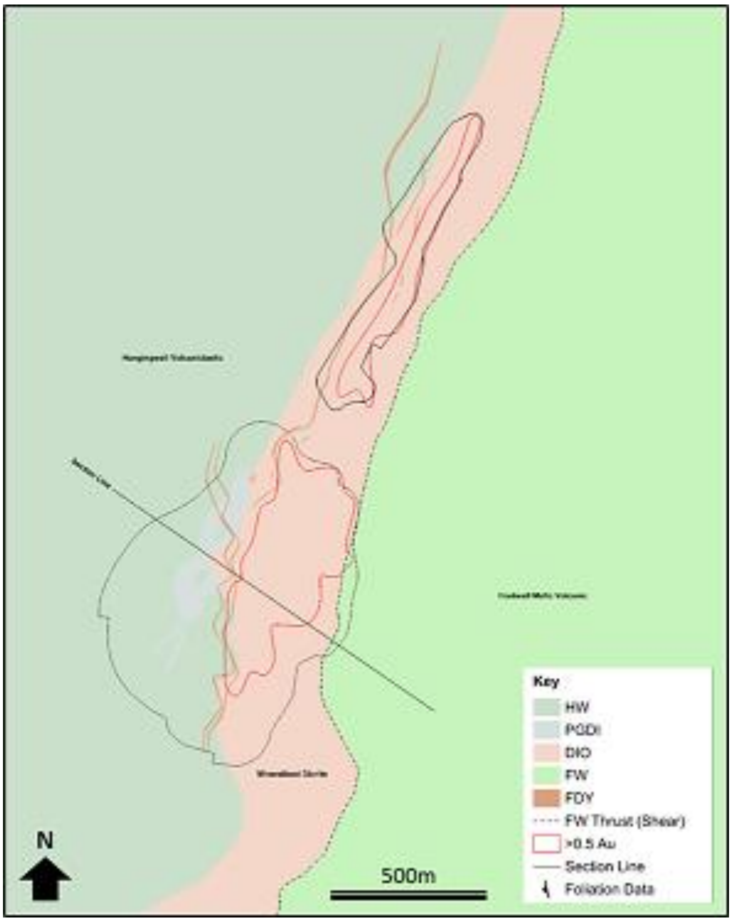


Figure 7-4 Plan of the deposit with major units defined and DFS Pit Design



Source: Montage, June 2021

7.3.1 The Diorite Sequence

A non-linear series of diorite bodies form a composite diorite domain with interstitial, localised, slivers/rafts of volcanic material that are common but not continuous. This Diorite domain is the primary host of gold mineralization at Koné.

Within the diorite domain, the individual diorites bodies are numerous but defining individual diorite intrusions is problematic. The diorite intrusions are of the same composition but are internally variable in both texture and grain size. Foliation intensity is variable and the contacts are often obscured by deformation (Figure 7-5).

At present, whilst the current logging scheme captures coarse and fine grain size variations in the diorite from a modelling perspective these units are being treated as a single diorite domain as the rock properties are consistent from a mining perspective and both are mineralized to a similar extent.

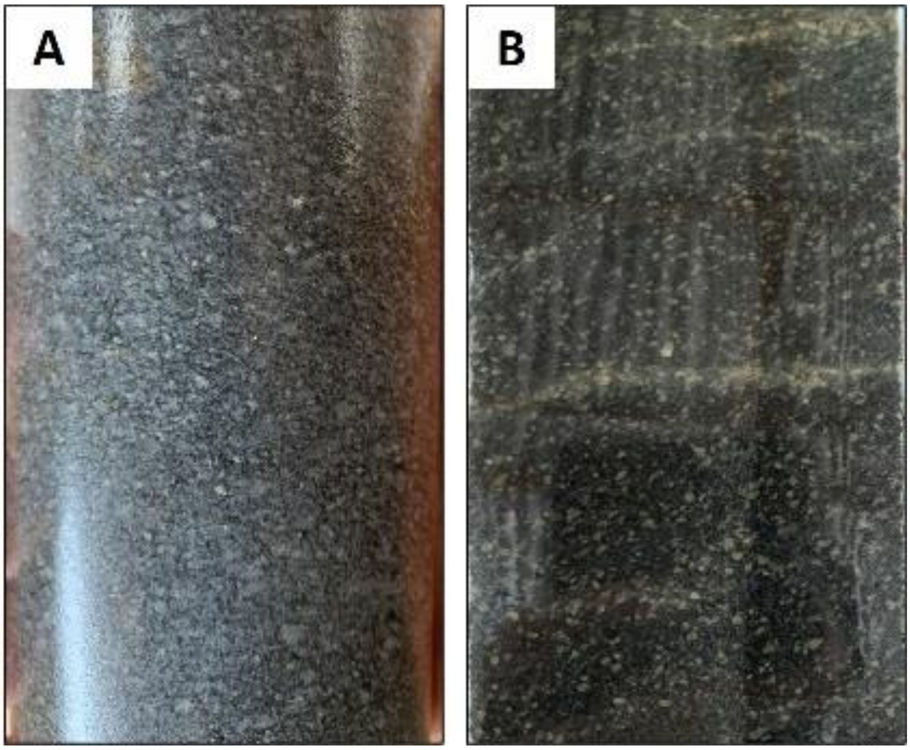
Figure 7-5 Example of sharp contact between two diorite bodies

Sharp contact between two diorite bodies highlighted by appearance of porphyritic texture in MDD017 at 214.15m depth. The contact is dipping 53° towards 279° (striking 009°). Numerous observations of this nature indicate a complex diorite package with stacked diorite bodies. Source: Montage

7.3.1.1 Coarse Grained Diorite (CDI)

Diorite with up to 2mm grain size composed of domains of fine plagioclase ± quartz and domains of mafic minerals – predominantly biotite. This lithology is moderate/strongly magnetic but in localised patches/zones in the core. The main textures observed under this code are porphyritic, caused by albitised plagioclase phenocrysts and equigranular texture highlighted by plagioclase crystals (Figure 7-6). This unit hosts gold mineralization, associated with 1-2mm, foliation parallel sulphide bearing quartz veinlets and disseminated pyrite due to the brittle nature of deformation compared to more ductile deformation seen in other units.

Figure 7-6 Coarse Grained Diorite



A – Equigranular texture composed of mainly coarse plagioclase and biotite.
B – Porphyritic texture caused by albitised plagioclase phenocrysts in a finer dark matrix
Source: Montage

7.3.1.2 Fine Grained Diorite (FDI)

The fine diorite unit is very closely related to the coarse diorite (Figure 7-7). Its composition is the same as the coarse diorite and locally it is simply a finer grained, recrystalized or altered version of the same lithology. This finer variation of the diorite is observed failing in a brittle manner comparable to the coarse unit but far more ductile deformation can also be observed in the finer intervals. The FDI commonly displays a gradational contact with coarser intervals of diorite and textural variations within a single diorite body lead to this code being somewhat subjective and difficult to log consistently. This lithology is dominated by more foliated textures highlighted by alignment of biotite and amphibole and is moderately to strongly magnetic but in localised patches. The rock is fine grained and grey in colour, and composed of domains of plagioclase and foliated biotite. Coarse grained intrusive textures are observed in localised patches throughout this lithology.

This unit hosts gold mineralization associated with 1-2mm sulphide bearing veinlets and disseminated pyrite.

Figure 7-7 Fine Grained Diorite

Fine grained foliated diorite (FDI) from MDD038 at 117.1m depth showing mafic minerals defining foliation and wrapping around plagioclase and quartz crystals. Late milky quartz vein observed in left hand side of the image. Source: Montage

7.3.1.3 Black Siliceous Diorite (BSD)

One distinctive fine grained diorite is observed. Characterized by its' very fine crystal size, dark grey/black colour and siliceous nature (Figure 7-8), this unit contains abundant magnetite and often hosts quartz-sulphide veinlets containing mineralization. The continuity of this lithology across sections and the deposit is yet to be proven, but early studies show correlation to higher grade intervals. Currently this is thought to be due to the alteration hardening of diorite caused by silicification creating a preferable unit to fail in a brittle fashion causing formation of quartz-sulphide veinlets.

Figure 7-8 Black Siliceous Diorite

BSD from MRRD001 at 227.3m depth with deformed VQS present in the left of the photo. Source: Montage

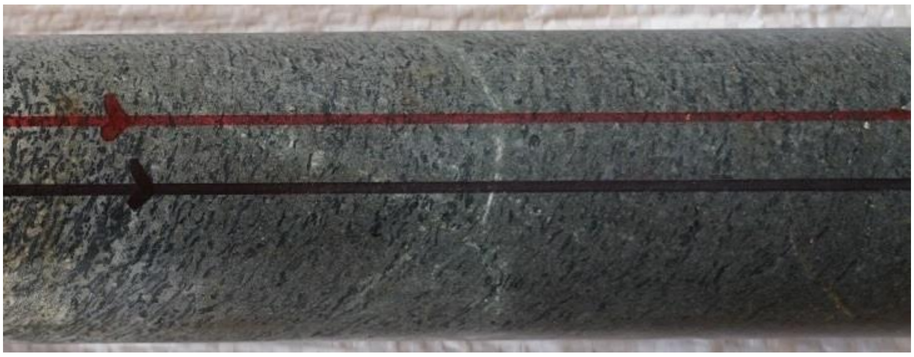
7.3.2 Hanging Wall Geology

7.3.2.1 PGDI – Pale green Diorite

The Pale green diorite is a rock of varying grainsize, characterized by its' distinctive pale green colour caused by large, foliation parallel amphibole crystals and abundant fine chlorite related to regional metamorphism (Figure 7-9). The remainder of the groundmass is composed of plagioclase and biotite. It has a moderate foliation with the same orientation as the diorite package (north-south, 50° west).

This unit is interpreted as an early intrusive into the hanging wall volcanic sequence. Although direct observation of the contact between the volcaniclastic (VC1) and this unit is rare, the drill spacing and distribution of this lithology, dictates the boundary to be sub-vertical. Main body diorites are seen intruded through this unit and barren, pre-mineralization sulphides are observed (Figure 7-9) aiding the early intrusion interpretation.

Figure 7-9 Pale green Diorite



Example of the Pale Green Diorite (PGDI) from MDD029 at 316.5m depth. Source: Montage

7.3.2.2 Mafic Volcaniclastic type 1 (VC1)

A poorly sorted volcaniclastic rock of polymictic clasts (<5cm in length) within a fine groundmass composed of ultrafine plagioclase, chlorite and biotite (phlogopite). Some sections lack clasts displaying only planar foliation (Figure 7-10). Planar foliation wrapping round the clasts is highlighted by the chlorite and phlogopite (Figure 7-11). Occasional albite, amphibole alteration of the clasts and localised zones of magnetite can be observed. Little to no deformation is present in this lithology and it is barren.

Figure 7-10 Mafic Volcaniclastic type 1

Example of VC1 showing sheared diorite clasts within a fine biotite-chlorite-amphibole matrix, from MDD071 at 102.5m depth. Source: Montage

Figure 7-11 Mafic Volcaniclastic type 1

VC1 from MRRD001 at 63m depth. Note the large clast in the center of the photograph with the foliation of the rock wrapping around it. Source: Montage.

7.3.3 Footwall Geology

7.3.3.1 Mafic Volcaniclastic type 3 (VC3)

Strongly deformed (Figure 7-12), compositionally banded mafic volcaniclastic with varying clast sizes within a groundmass of mafic chlorite ± biotite bands and fine plagioclase bands (60% and 40% respectively). Ductile deformation is observed throughout this volcaniclastic, shown by pervasive folding and deformation of clasts. Increased folding is observed proximal to the footwall contact of the Diorite. Folded quartz-sulphide veinlets are observed, still within foliation planes which have also been deformed. Grades average around 0.7g/t in this unit due to its' proximity to the footwall shear controlling mineralization.

Figure 7-12 Mafic Volcaniclastic type 3

Example of VC3 showing intense deformation of clasts and foliations from MDD038 at depth 238.2m. Source: Montage

7.3.4 Mineralization

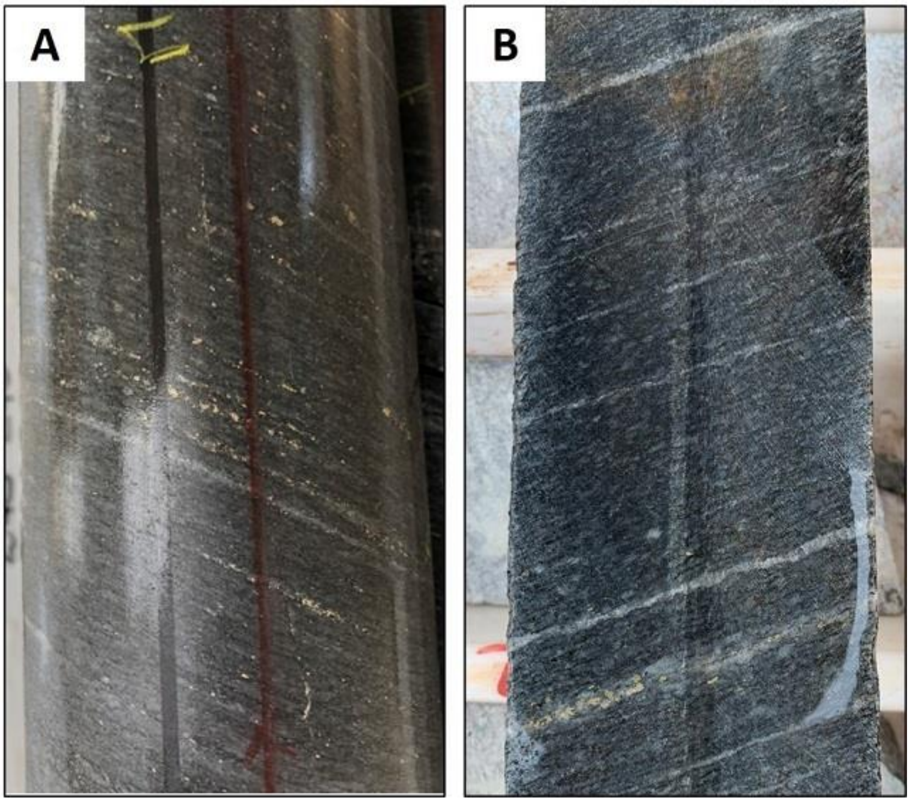
The Koné Gold deposit is hosted by the composite diorite body and is characterized by very large mineralized true widths, averaging >200m over the southern half of the deposit, with a maximum of up to 330m (MDD015B, 330.7m grading 0.58g/t) i.e. almost the entire width of the composite diorite body.

Higher grades (1-1.5 g/t) are associated with high density "swarms" of 2-5mm thick, foliation parallel (Figure 7-14), translucent white to smoky quartz veinlets containing fine grained sulphide (Figure 7-13). Lower grades are related to disseminated fine grained pyrite. Importantly, no significant silicification of the host rocks is associated with the mineralization resulting in positive comminution characteristics.

Mineralization at Koné is interpreted to be controlled by a major thrust shear at the footwall contact of the diorite contact (Figure 7-15). The volcaniclastic rocks in the footwall of the thrust have deformed plastically whereas the more rheolgically competent diorite has developed brittle/ductile shears. This is expressed in drill core as brecciated zones of diorite, associated slivers of internal volcanic rocks, and localised shears with very localised folds of foliation within the diorite and footwall volcanic (Figure 7-16 and Figure 7-17).

Geological observations from recent diamond drill core have started to resolve a spatial relationship between these shears and the VQS vein "swarms" which host higher grades. Further work is scheduled to develop this interpretation.

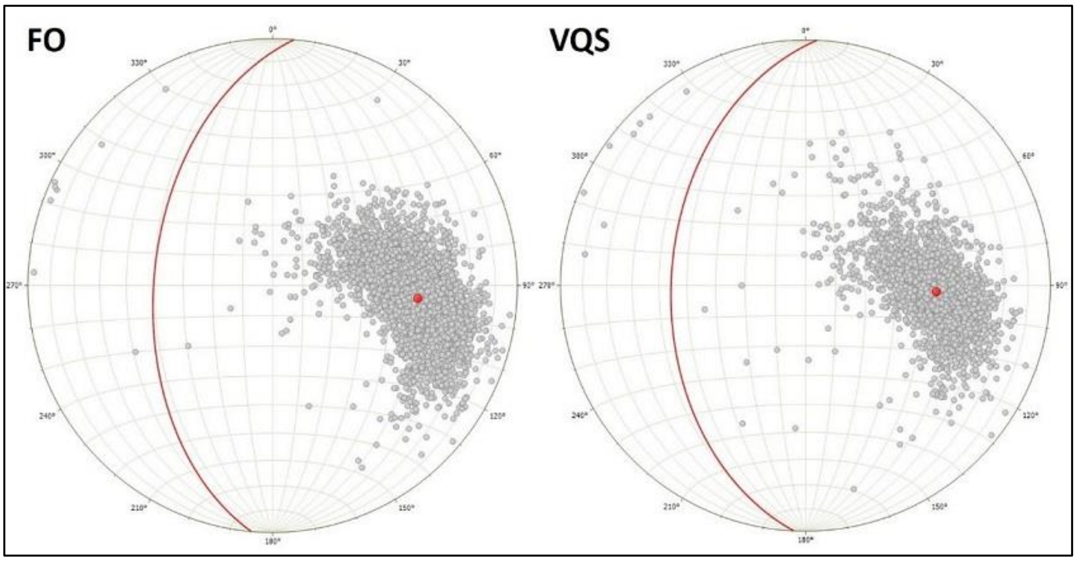
Figure 7-13 Mineralized Foliation & VQS vein swarm



A – Mineralized foliation (pyrite aligned in foliation plane) and VQS veins in fine grained diorite from MRRD001 at 245m depth (1.09g/t).
B – VQS vein swarm related to grade in MDD050 at 415.8m depth (2.85g/t).
Source: Montage

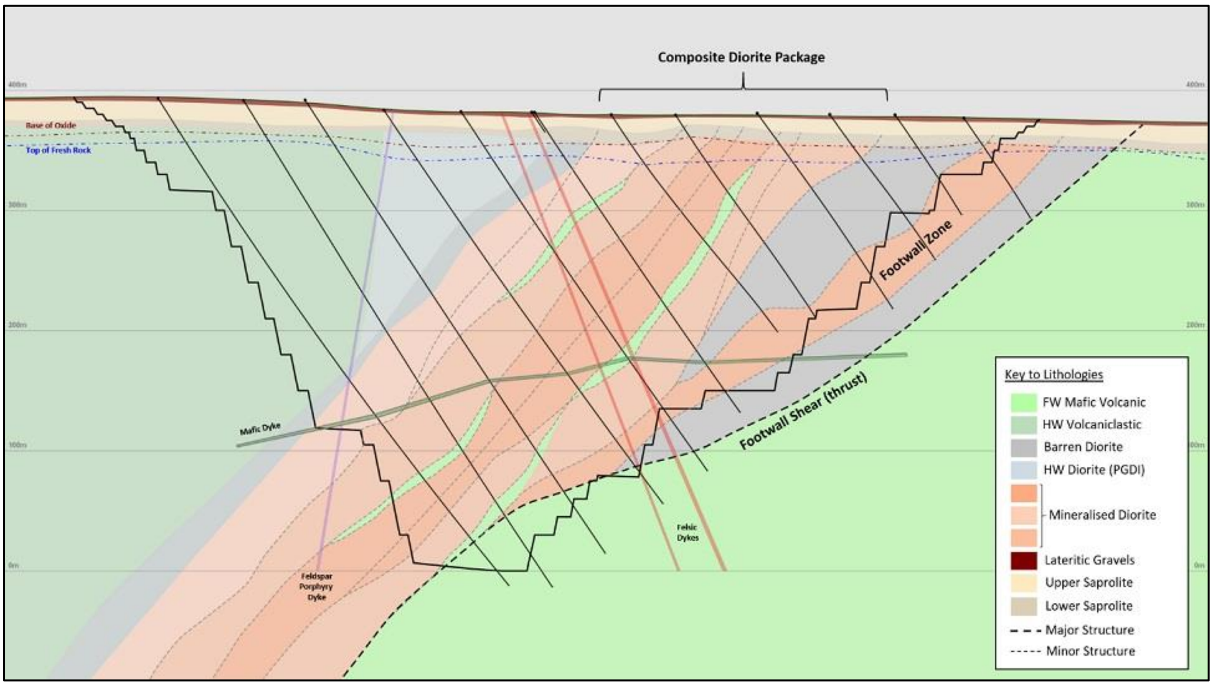
A phase of barren pyrite is present in the hanging wall that has a 1-2mm grainsize and euhedral cubic in form. Pyrite related to the mineralization is either <0.5mm disseminated globules of ultrafine pyrite that appear to be replacing magnetite locally, or internally with the VQS veins.

Figure 7-14 Foliation orientations



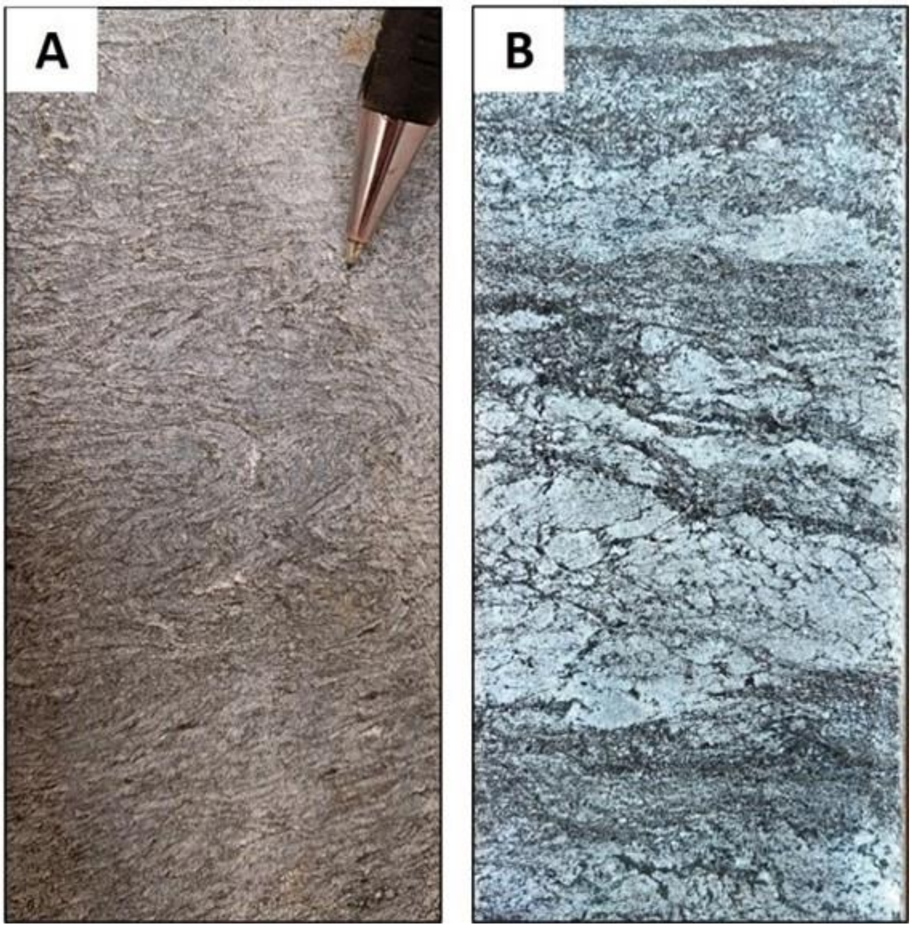
Comparison between foliation orientations (left hand stereo-net) and VQS measurements (right hand stereo-net) from diamond drill core. Both sets of features have the same spread of data and have average dips of 40° West and strike between 330° and 020°. Drill core is oriented using a reflex ACTIII digital orientation tools and surveyed with a gyroscope for accurate structural data. Source: Montage

Figure 7-15 Section through the centre of the deposit



Section through the centre of the deposit, displaying the major units with DFS pit outline. Source: Montage

Figure 7-16 Ductile shear within diorite



A - Example of ductile shear within diorite (MDD017, 379.4m, 0.32g/t).
B – Brecciated zone interpreted as a fluid pathway/structure through the diorite (MDD008, 255.8m, 0.54g/t). Source: Montage

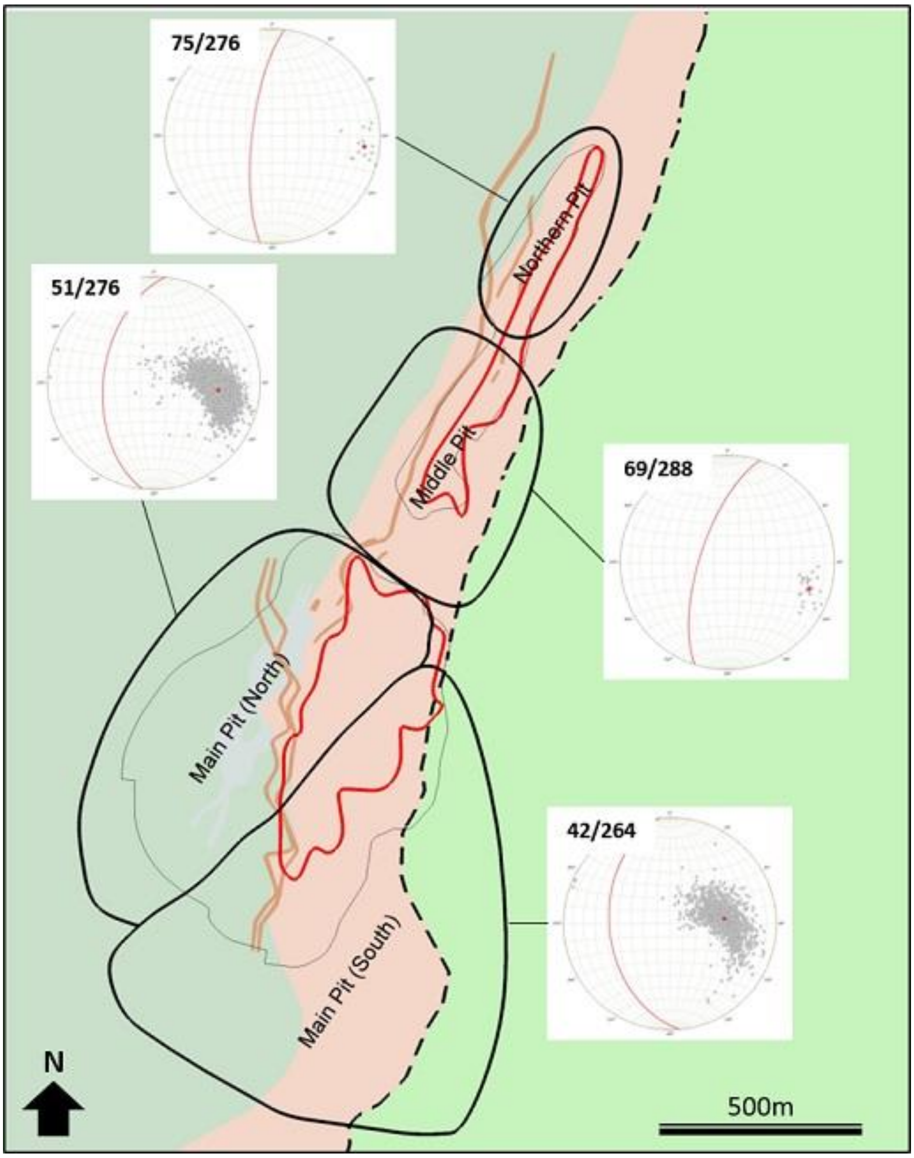
Figure 7-17 Buckle folds within the diorite

Example of buckle folds within the diorite. MRRD007, 154.06m. Source: Montage

7.3.5 Structure and Deformation

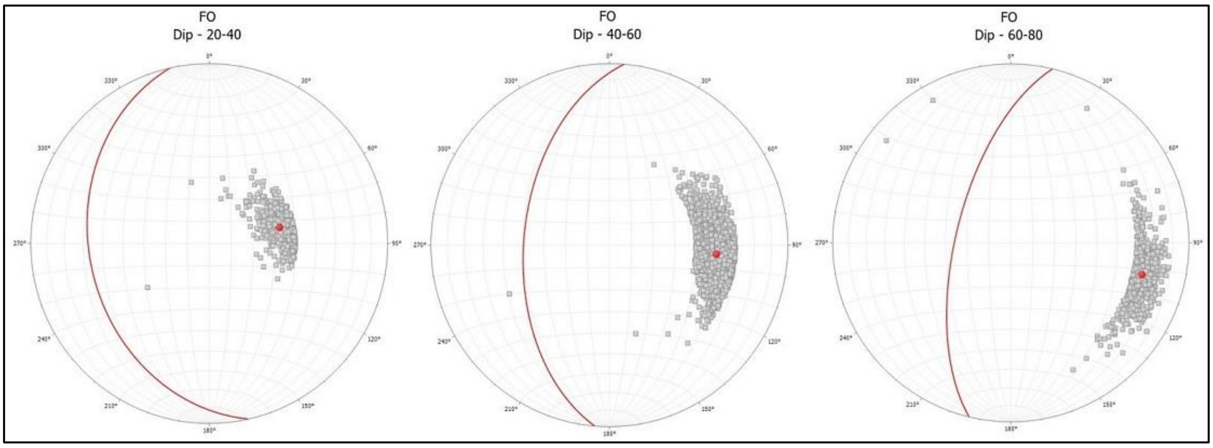
A correlation between the dip of the foliation and the strike of the host rocks can also be observed (Figure 7-18), with foliations dipping 20-50· averaging a strike of ~348· in the southern part of the deposit and dipping 60-80· striking ~014° in the northern sector. This change in strike is thought to be related to a step in the regional, mineralized structure. This step is interpreted to be a controlling factor on mineralization within this area, with the flattening and rotation of the diorite bodies by the structure allowing for fluid accommodation and subsequent mineralization. Stereo-nets of foliation data within the diorite package across the deposit are displayed below, highlighting a steepening of foliation to the north and a change in strike to the south towards south-south-east (Figure 7-19).

Figure 7-18 Correlation between the dip of the foliation and the strike of the host rocks



Plan divided into four areas displaying the steepening of the deposit to the north and rotation in the strike towards SSE to the south. Source: Montage, June 2021

Figure 7-19 Streo-nets of foliations



Stereo-nets of foliations categorized by dip angle, showing relationship between dip angle and strike direction. Source: Montage

7.3.6 Dykes

Multiple sets of dykes are observed through the deposit, displaying varying composition, orientation and deformation. Some of these can be traced across the entirety of the deposit, such as the felsic, feldspar porphyry and main late green dykes. Others are observed displaying anastomosing form and aren't continuous across drill fences and/or the deposit.

7.3.6.1 Early Green Dykes (EGD)

Intermediate to mafic foliated dykes displaying strong chlorite alteration with local biotite alteration/metamorphism related to regional greenschist/amphibolite metamorphism (Figure 7-20). The foliation is often deformed and the dykes regularly return above detection limit for gold, meaning these dykes are interpreted to be pre-/syn-mineralization. These dykes are occasionally weakly magnetic. Multiple dykes are observed, orientated sub-parallel to foliation, with occasional steeper versions observed. Continuity of these dykes is not well constrained with varying structural orientation to dyke contacts observed. Dykes are interpreted to be anastomosing and lack continuity across the deposit.

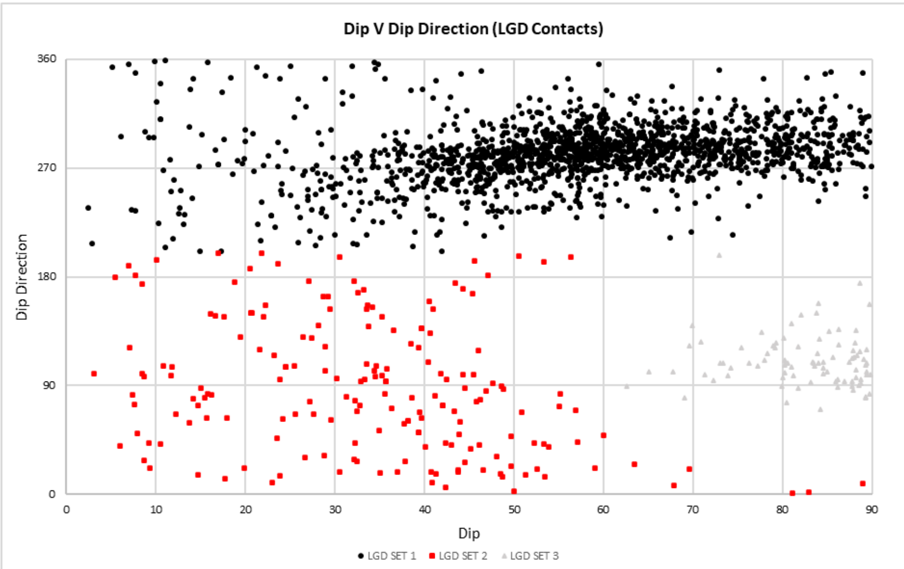
Figure 7-20 Chlorite/biotite altered foliations

Strongly chlorite/biotite altered foliated EGD intruding into CDI (contact running through the centre left of the photo at a 45° angle). Source: Montage

7.3.6.2 Late Green Dykes (LGD)

Dark green, non -magnetic, undeformed dykes, observed orientated sub-parallel and cutting foliation with three dominant orientations as shown in Figure 7-21. Sub-parallel to foliation (black), shallower than foliation with variable strike (red) and near vertical striking parallel to foliation but dipping in the opposite direction (grey). These dykes are fine grained but often have porphyritic texture with 1-2mm amphibole phenocrysts (Figure 7-22). Interpreted as late-stage, post mineral dykes shown by lack of deformation and alteration. Occasional late calcite veinlets are observed.

Figure 7-21 Late Green dykes dip and dip direction



Graph displaying the three populations of LGD dykes seen within the Koné deposit: Black circle – foliation parallel set. Red square – A less steep set of dykes displaying variable strike, some striking parallel to foliation. Grey triangle – Near vertical set of dykes, striking sub-parallel to the foliation but dipping to the east. Source: Montage

Figure 7-22 Late Green dykes

Late green dyke from MRRD007 at 185.5m. Note the dominant foliation in to Diorite (left hand side) is truncated against the LGD contact. Note the aligned amphibole running parallel to the contact.
Source: Montage

7.3.6.3 Mafic Dykes (MDY)

Two variations of mafic dykes are observed, considered to vary in time of emplacement. Both variations are characterised by abundant magnetite, black colour and late stage pyrite unrelated to mineralization (Figure 7-23).

The first type displays weak banding and deformation, with residual calcite. Further data needs to be collected to define the orientation, continuity and frequency of these early dykes.

The second variation is a late sub-horizontal version of mafic dyke, lacking any foliation or deformation and runs consistently through the deposit, observed clearly cutting foliation.

Figure 7-23 Massive Mafic dyke

Example of the sub-horizontal massive mafic dyke, MDD069 at 312.6m depth. Source: Montage

7.3.6.4 Felsic Dykes (FDY)

Felsic dykes are light grey in colour, aphanitic, massive and cross-cut the foliation at a high angle (Figure 7-24). They are intruded very late into the sequence and post-date the main deformation and mineralization event. These dykes are not magnetic and are volumetrically subordinate to the other types of dykes. They are consistent on and between sections and can be modelled easily, striking ~015° through the deposit and dipping ~75-80°. Two main felsic dykes have been logged in the main pit, with increased frequency seen to the north.

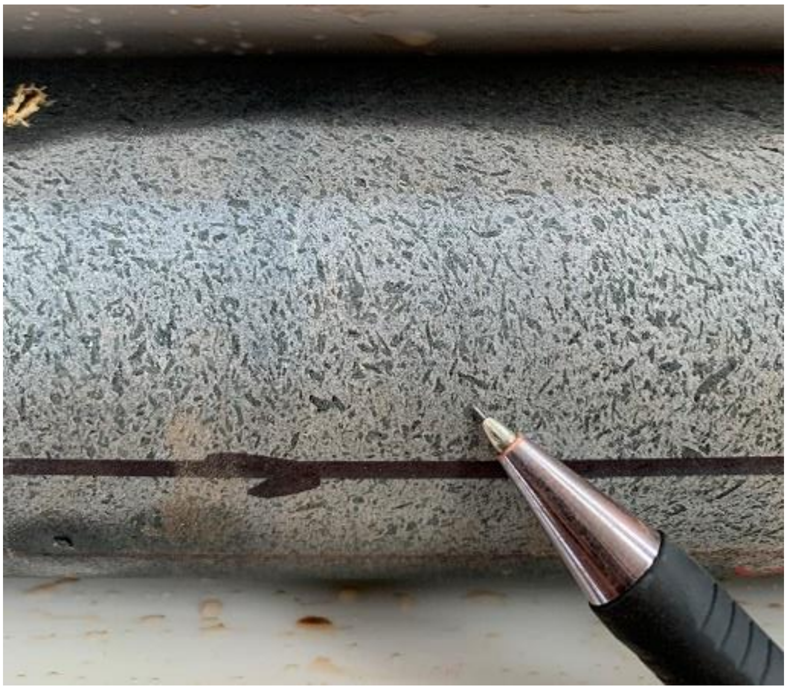
Figure 7-24 Felsic dyke

Example of FDY in MDD068 at 423.60m depth. Source: Montage

7.3.6.5 Intermediate Dyke (IDY)

Late massive, undeformed intermediate dyke. Characterized by large (<5mm) randomly orientated amphibole crystals with strong magnetite and sub-vertical, sharp contacts (Figure 7-25). Located in the west of the deposit, striking near close to north-south.

Figure 7-25 Felsic dyke

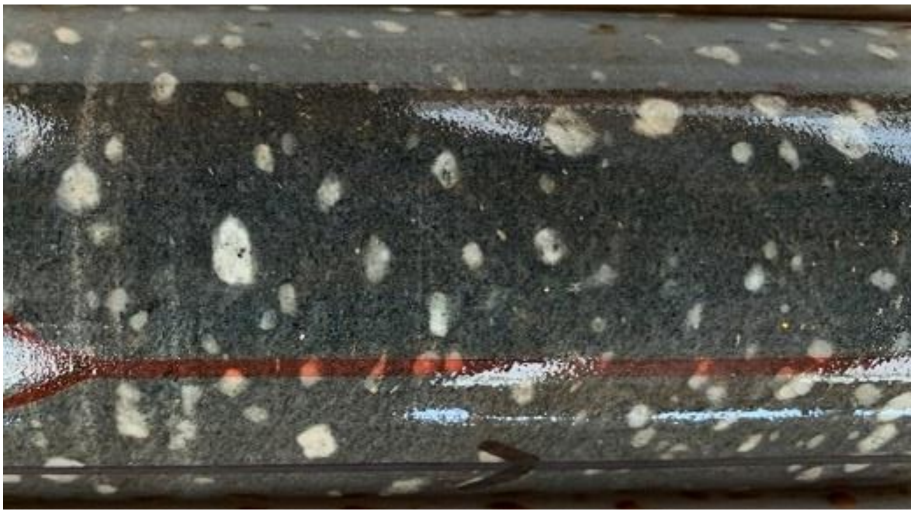


Example of IDY displaying randomly orientated amphibole in MDD010 at 184m depth. Source: Montage

7.3.6.6 Feldspar Porphyry Dyke (FPR)

Massive, unaltered, porphyritic, intermediate dyke with distinctive round feldspar phenocrysts and moderate magnetism (Figure 7-26). Sub parallel to foliation. Multiple sub-parallel dykes are seen across the deposit and all strike approximately north-south. Using Zircon U-Pb dating gave 2119 ± 4Ma, approximately 50 Ma after the dated emplacement of the diorite.

Figure 7-26 Felsic Porphyry dyke



Example of FPR dyke, with characteristic round plagioclase phenocrysts in a silvery matrix from MDD057 at 57.9m depth. Source: Montage

7.3.7 Post Mineral deformation

7.3.7.1 Faults

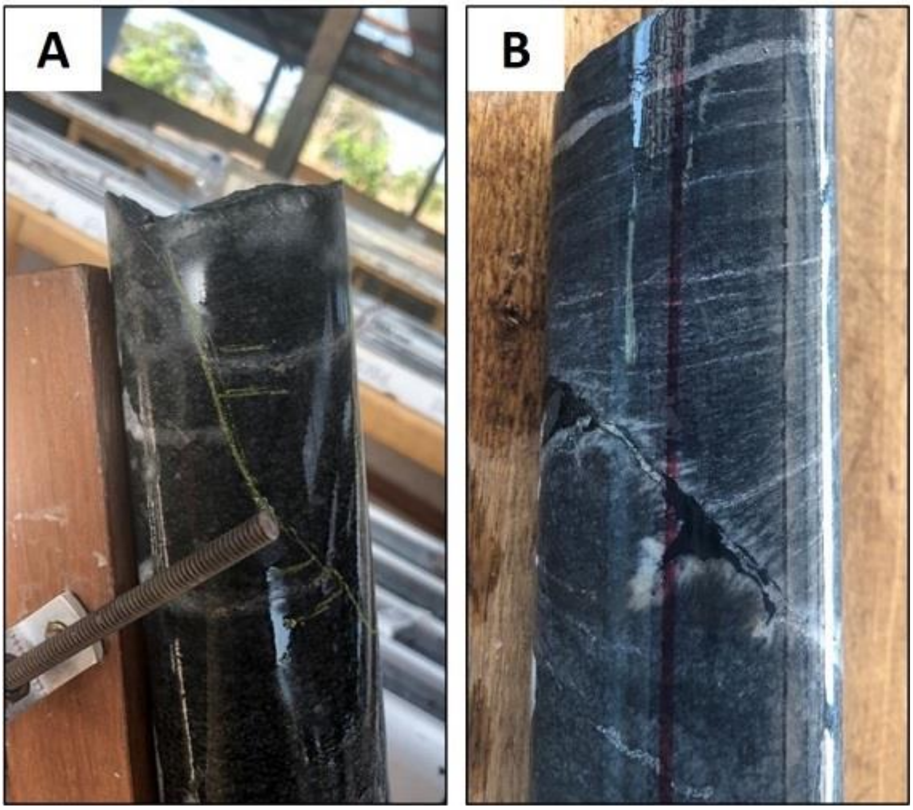
Very few larger fault, clays/breccias are observed within the deposit with the majority of faulting observed being minor faults displaying both normal and reverse movement (Figure 7-27), leading to high core recoveries (average 98% across the entire deposit). Therefore, any displacement and offset of the mineralization is most likely related a series of smaller offsets by multiple minor faults.

The minor faults display both normal and reverse movement, with some infilled by quartz ± carbonate. Varying fault orientations have been observed, with four characteristic fault sets identified. Three near vertical (>80・ dip) sets of faults are defined (Figure 7-28):

- Striking NNE-SSW, sub-parallel to the strike of the deposit
- Striking NE-SW
- A set of conjugate faults striking around E-W, most likely a strike slip pair caused by principal compression E-W.

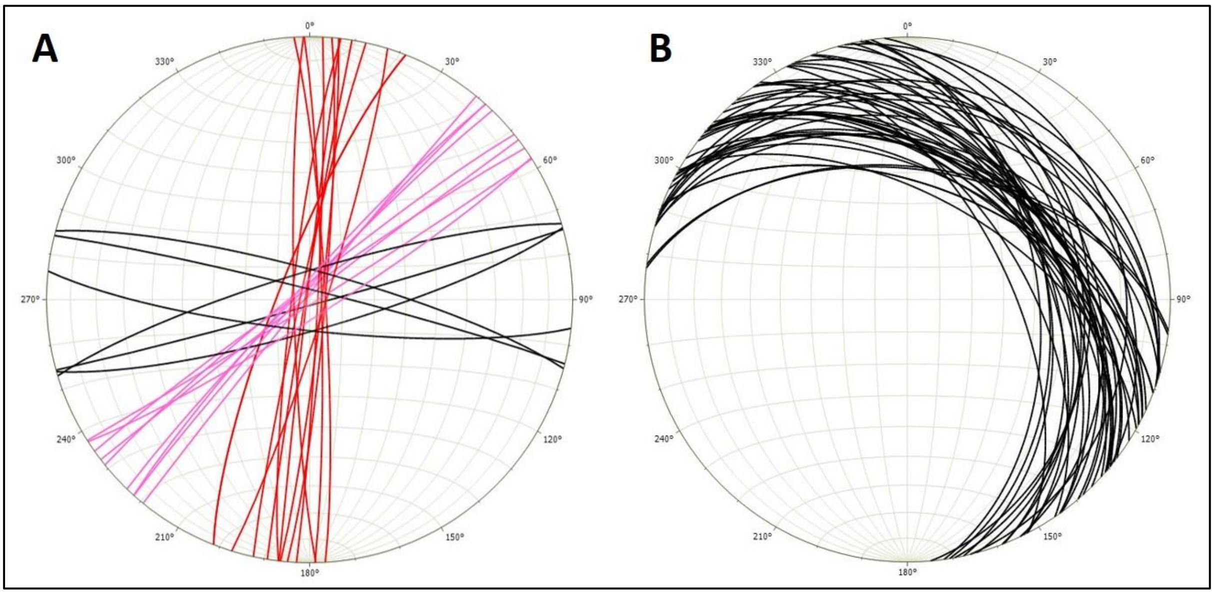
One further set of NW-SE striking faults averaging ~40・ dip, sub-parallel to drill fences is present. Further study of the fault sets needs to be conducted in order to define the movement on these faults. Minor faults can be seen offsetting mineralized veins, therefore showing that faulting occurred post mineralization and is offsetting it.

Figure 7-27 Minor healed fault



A - Minor healed Fault oriented in rocket launchers and displacing VQS vein from MRRD006 at 220m depth. The fault is dipping 69° SE and striking 045°. It has an apparent sinistral reverse fault sense of movement. B - Minor healed fault cutting across and truncating foliation and veins in MRRD007 at 149.5m depth. This fault is dipping 75°NW and striking 065°. Source: Montage

Figure 7-28 Minor faults stereo net



A – Sub-vertical faulting showing two clear orientations and a conjugate set of faults.

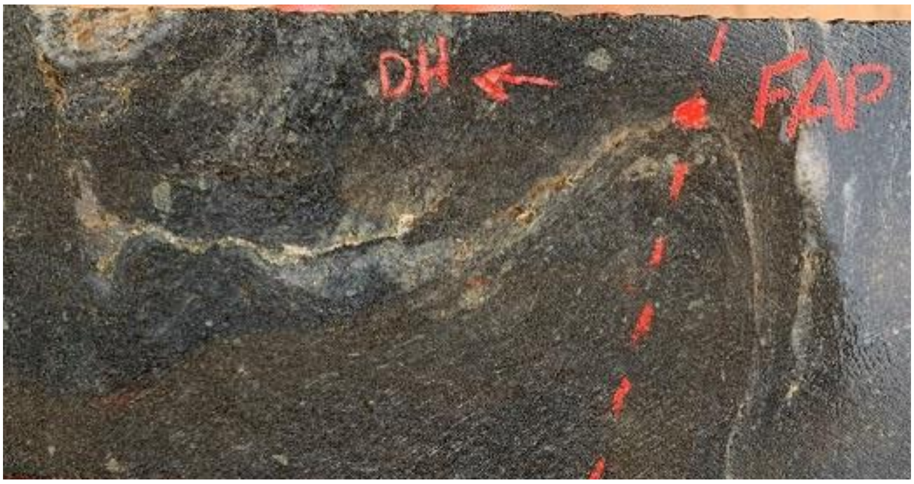
B – Shallow dipping fault set striking sub-parallel to drill fences.

7.3.7.2 Folding

In places, veins display non-linear, sinuate contacts showing post formation deformation (Figure 7-29). Although the veins now strike and dip sub-parallel to foliation, they are interpreted to have formed perpendicular to foliation and main compressive direction and rotated parallel to foliation through continued compression. Occasionally, veins are seen displaying larger open folds, with fold axial planes sub-parallel to foliation strike and dip. This suggests that the folding of the veins was also due to the continued compression of the area post formation.

Further ductile folding can be observed in the footwall volcanic units. Fold axial planes dip and strike parallel to the fold axial planes of the folded veins, foliation and major structures, showing them to all be related to the same compressional event (Figure 7-30). Due to the parallel nature of the fold axial planes with foliation and major structures, the trend and plunge of the FAPs is directly observed from the core and is sub-horizontal. This is coherent with the trend and plunge observed for the foliation and structures, therefore further supporting the folds to be caused by the same compressional event as the deposit.

Figure 7-29 Folded VQS



Example of folded VQS showing compression of mineralized event from MDD017 at 216m depth.
Source: Montage

Figure 7-30 Ductile Folding

Example of ductile folding within the mafic volcanic unit due to the footwall shear zone (MDD038, 232.8m, 0.04g/t). Source: Montage

8 DEPOSIT TYPES

The Koné deposit is considered to be an orogenic lode gold-style system, hosted by brittle ductile shearing within a quartz diorite/mafic volcaniclastic package in a Birimian Greenstone sequence of the West Africa Craton

The original targeting criteria that led to the discovery of the Koné deposit is shown in Table 8-1. Soil sampling, trenching and shallow reconnaissance drilling proved successful in the initial delineation of the mineralization.

Table 8-1 Ground Selection Criteria

1	Structure	1st order structural trend, deep seated, fertile structure with known endowment.
2	Gold Endowment	Of the structural trend.
3	Lithology	Presence of chemical and rheological host rocks, associated with a strong, wide volcanic +/- volcano-sedimentary belt, on an axis or junction site.
4	Alteration	Local evidence of extensive alteration and high fluid flow.
5	Intrusives	Area of high heat flow – presence and quantity of late intermediate to felsic intrusives.
6	Metamorphism	Unmodified by +biotite metamorphism or high strain structural reworking
7	Erosion level	High level of preservation, not deeply eroded. No local evidence of basement gneisses or migmatites.
8	Exploration	Lack of contemporary exploration over the last 20 years.

9 EXPLORATION

9.1 Introduction

During the second half of 2009, Red Back Mining completed 800m by 50m spaced soil sampling with subsequent local infill to 400m by 50m and 200m by 50m spacing identified a 2.7 km long +75 ppb gold in soil anomaly at Koné. The anomaly was tested in 2010 by 200m spaced trenches, the results of which justified exploratory drilling leading to resource definition drilling.

In 2013 Sirocco gold completed 3 trenches for a further 610m extending and infilling on the previous trench plan.

During 2019, Orca completed a program of 274 hand dug pits to follow up on weak soil geochemical anomalies in the Koné Exploration Permit. Samples from only three pits returned gold assay grades of greater than 0.5 g/t.

A small ground magnetic survey was incidental to exploration activities and did not significantly impact drill planning.

Quality control samples inserted in batches of soil, trench and pitting samples included reference standards, and coarse blanks which provide adequate confirmation of the reliability of sample preparation and analysis. The author considers that quality control measures adopted for the exploration sampling have established that the sampling is representative and free of any biases or other factors that may materially impact the reliability of the sampling and assaying.

Table 9-1 summarizes exploration work completed to date at the Project. Drilling includes tabulation of work completed by previous owners. Drilling, associated sampling and assaying procedures are described in Sections 10 and 11.

Table 9-1 Exploration Activities to Date

Activity	Red Back 2009-10	Sirocco 2013-14	Orca 2017-2019	Montage 2019-2020
Satellite Imagery Acquired				
Worldview imagery (km^2)	230	-	-	-
Ground Geophysics				
Ground magnetics (km^2)	4.68	-	-	-
Induces Polarisation (km^2)			104.7	-
Surface Sampling				
Soil samples	4,877	-	473	2,664
Rock chip samples	61	2	6	-
Trenching (number/metres)	9/4,155 m	3/610 m	-	166
Pitting (m)			1,492	

9.2 Soil Sampling

The first soil sampling program was carried out in 2009 and 2010 under contract by SEMS Exploration and was completed in two phases totalling 4,877 samples within the Koné Exploration Permit. The first phase, which covered around 11 km of strike at 800 by 50 m spacing outlined a +75 ppb gold anomaly over 2.7 km strike along the western greenstone belt margin with widths up to 500m. A second phase of in-fill sampling at 200 m by 50 m spacing confirmed and improved definition of the anomaly.

During 2019 and 2020 a further 3,137 soil samples were collected on the Koné Exploration Permit both infilling and extending previous grids. This sampling led to the delineation of the Petit Yao anomaly 8km east of the Koné deposit. Figure 9-1 shows the locations of soil samples relative to the Koné Exploration Permit, with sample locations coloured by assayed gold grade.

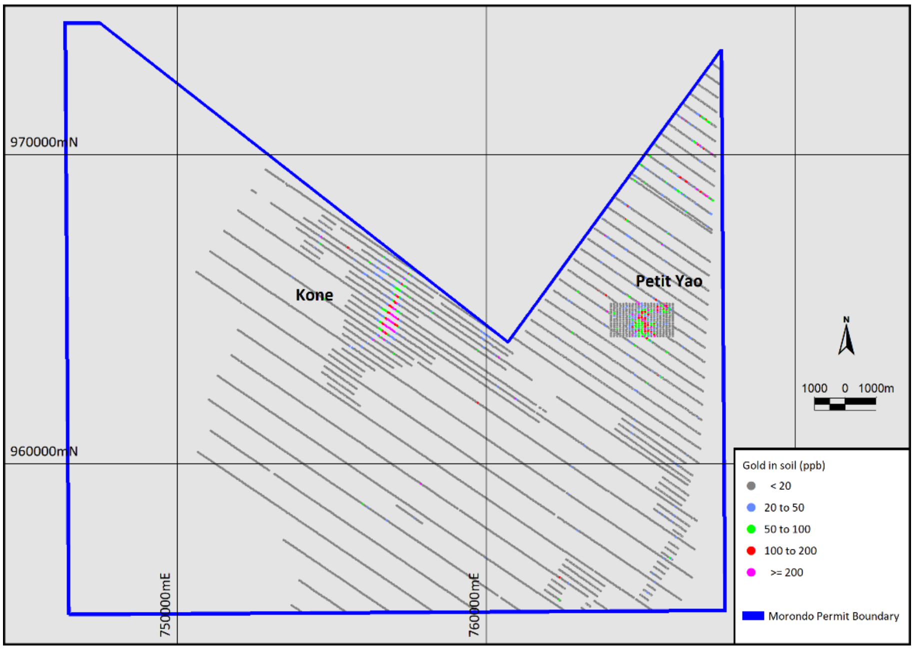
The 2009 and 2010 soil sampling phases utilized 20 to 30 cm diameter hand held augers to collect generally 2.5 to 3.0 Kg samples from depths of 50 to 60 cm, below the organic layer. Soil sampling in 2019 and 2020 was based on approximately 50 cm deep pits from which a 1 Kg sample was collected in the pisolitic horizon from below the organic layer

All samples were collected and transported to the field camp the same day under the supervision of a field geologist.

Samples from the 2009 and 2010 soil sampling were submitted to SGS for analysis. Samples from the 2019 and 2020 soil sampling programs were analysed by Bureau Veritas in Abidjan.

Quality control samples inserted at the field camp under the supervision of the Project Geologist including reference standards and coarse blanks provide adequate confirmation of the reliability of sample preparation and analysis for the 2019 and 2020 soil sampling.

Figure 9-1 Soil Sampling Distribution



Source: Montage, February 2021

9.3 Trenching

Nine trenches totalling 4,155m were completed in 2010 with a further 610m in three trenches excavated in 2013. Excavation of the trenching was contracted to the youth community of Fadiadougou village. The trenches were dug by hand to a typical width of 1 m and an average depth of 3 m, with some sections reaching 3.5 m depth. Trenching typically bottomed in the mottled clay zone, only rarely exposing saprolite material.

Field geologists employed by Red Back (2010) and Sirocco (2013) supervised the trench sampling and mapped the trenches compiling detailed trench sections (Figure 9-2).

A total of 2,201 channel samples of generally 2m, and rarely 5m length were collected at the base of the northern wall of trenches. For each sample interval the floor was first cleaned to avoid contamination and then a 2.0 to 2.5 kg sample was collected. Field duplicates were routinely collected from a second channel cut along the line of the primary sample. All samples were transported to the field camp the day of collection under the supervision of a field geologist.

Samples from the 2010 trenches were submitted to SGS for analysis, with the samples collected during 2013 submitted to Bureau Veritas. Quality control samples were inserted at the field camp under the supervision of the Project Geologist and included standards and blanks which provide adequate confirmation of the reliability of sample preparation and analysis.

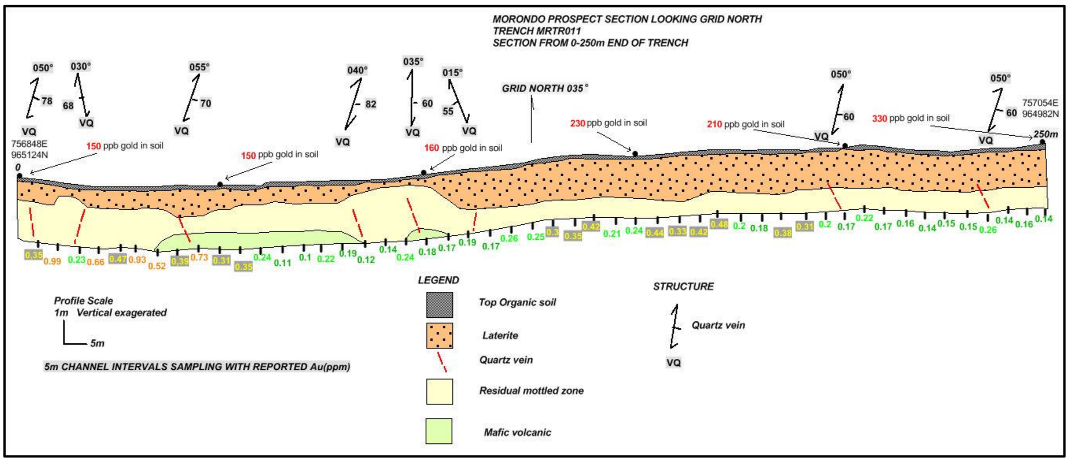
Significant intercepts from the trenching at Koné are shown in Table 9-2. True intercept thicknesses are interpreted to approximate 75% of interval lengths.

Montage's 2019 trench sampling comprised the collection of 83, two metre length samples from 14 channels excavated from road cuttings in the east of the Koné Exploration Permit area. These samples, which were submitted to Bureau Veritas for analysis returned a maximum gold grade of 0.016 g/t are not considered to be significant in terms of overall exploration of the Koné Exploration Permit.

Table 9-2 Significant intercepts for 2009 and 2010 trenching

Trench	Collar Location			Length	Azimut h	Intercept (m)			Au g/t
	mE	mN	mRl	(m)		From	To	Length	Uncut
MRTR001	756,733	964,716	382	424	125	20	222	202	1.11
MRTR002	756,620	964,555	378	444	125	92	294	202	0.67
MRTR003	756,886	964,856	388	250	125	0	212	212	0.82
MRTR004	756,666	964,889	392	352	124	174	334	160	0.75

Figure 9-2 Example annotated trench section



Trench MRTR010. Source: Montage

9.4 Pit Sampling

During 2019, Orca completed a program of 274 hand dug pits to follow up low tenor soil geochemical anomalies in the vicinity of the Koné resource and wider Koné Exploration Permit area. Pits were dug at average spacings of around 50m by 200m to an average depth of 5m and the north wall of the pit sampled. Orca geologists supervised the pit sampling and mapped the pits prior to backfilling.

A total of 628 channel samples for intervals of 0.1 to 4.5m length were submitted to Bureau Veritas in Abidjan for analysis for gold by fire assay. Field duplicates were routinely collected from a second channel cut along the line of the primary sample. All samples were transported to the field camp the day of collection under the supervision of field geologists.

Quality control samples were inserted at the field camp under the supervision of the Project Geologist and included standards and blanks providing adequate confirmation of the reliability of sample preparation and analysis.

Samples from only three pits returned gold assay grades of greater than 0.5 g/t. Due to the deep weathering and regolith encountered in the pits, they are interpreted to poorly test for bed-rock mineralization, the pitting program was discontinued. The Company considers that pit sampling does not meaningfully add to the exploration dataset and they are not detailed in this report.

9.5 Magnetic Survey

In 2010 Red Back completed a ground magnetic survey over the Koné prospect. A caesium vapour ground magnetic survey was conducted with 10m stations along 100m spaced E-W lines for 48-line km. The survey measured total magnetic intensity and targeted providing information on the local magnetism associated with discrete bodies. The surveys were diurnally corrected before being processed.

High gold grade trench samples broadly coincide with traces of magnetite. In an attempt to delineate zones of magnetite associated gold mineralization magnetic, susceptibility readings were taken at 2m intervals along trench sample intervals. The susceptibility readings were highly variable, which is considered to be mainly due to the small surface area recorded (1 cm^2).

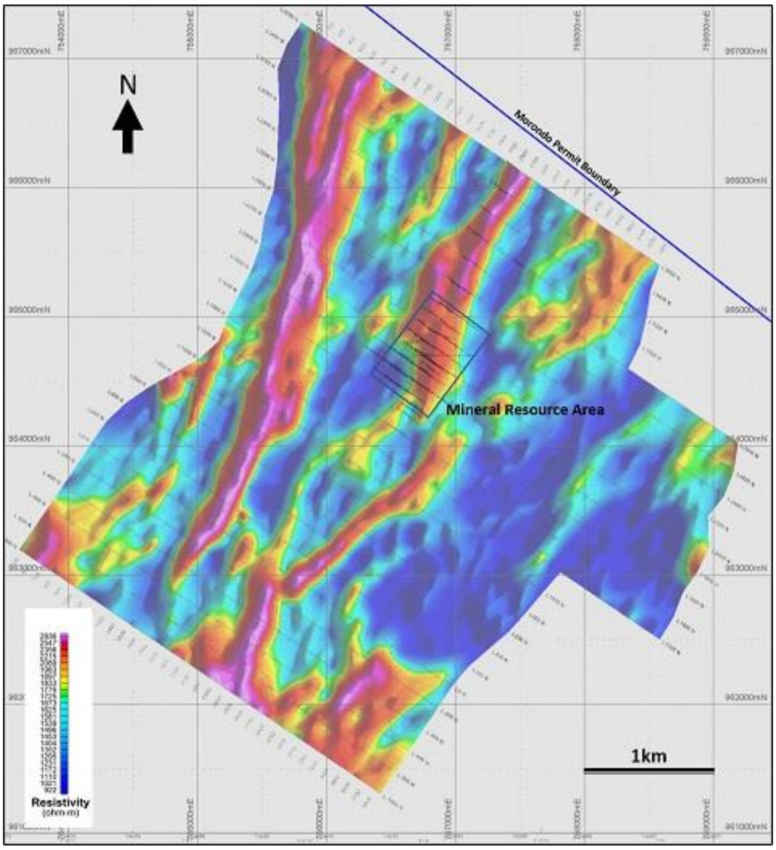
The ground magnetics are dominated by three, east-west trending magnetic highs that are considered to be mapping the extent of surficial duricrust and as a result the survey has been of limited use.

9.6 Gradient Array Induced Polarisation Survey

A Gradient Array Induced Polarisation Survey was carried out in early 2019 covering 104-line kilometres encompassing the Koné resource area.

The survey used a line spacing of 200m and an electrode spacing of 25m. As Figure 9-3 shows, the survey successfully mapped the various geological domains in the Koné resource area with the resistivity component being of particular use in mapping the intrusive mineralization host.

Figure 9-3 Induced Polarisation survey



Apparent Resistivity. Source: Montage, August 2021

10 DRILLING

10.1 Introduction and Summary

As summarized in Table 10-1, drilling information available for Koné totals 353 RC and 50 diamond holes for 50,017 m. The RC drill metres shown in Table 10-1 for 2019 to 2020 Koné area drilling include 493.3 m of pre-collared portions of seven diamond holes.

In addition to RC and diamond drilling in the Koné area, which informs Mineral Resource estimates, Montage's drilling at the Koné includes shallow reconnaissance RC drilling testing exploration targets identified by soil and rock chip sampling, and 41 diamond holes drilled for geotechnical investigations, of which only 4 holes have had samples have been submitted for gold analysis. Information from entirely un-assayed drill holes does not inform resource modelling.

Central portions of the currently interpreted Koné mineralization have been tested by generally 50 m spaced northwest southeast traverses (125° UTM) of RC and diamond holes generally inclined to the southeast at around 55 degrees. These holes are generally spaced at around 50 and rarely 25 m along the traverses with each traverse extending to vertical depths of around 60 to 490 m.

Although undertaken by different corporate entities, field procedures and key staff were consistent for all Koné drilling phases ensuring consistency in the sampling methodology. All field sampling activities were supervised by field geologists with industry standard methods employed for sampling and geological logging.

Information available to demonstrate the sample representivity for the Koné RC and diamond drilling includes RC sample condition logs, recovered RC sample weights and core recovery measurements.

The quality control measures adopted for the Koné RC and diamond drilling have established that the sampling is representative and free of any biases or other factors that may materially impact the reliability of the sampling. As assessment of the Koné Gold Project continues, and higher confidence resource estimates are targeted additional investigations of sample reliability may be warranted.

Reconnaissance RC drilling completed in 2019 focussed on the general area surrounding the Koné mineralization and returned several low tenor anomalies (<0.2 g/t Au). The 2020 reconnaissance drilling targeted the Petit Yao prospect and intersected narrow mineralized zones. The 2021 reconnaissance drilling tested potential mineralization to the south of the Koné area with no significant results and also added drilling to the petit Yao prospect in the form of further shallow RC drilling and a short programme of deeper RC holes which returned several mineralized intercepts.

Table 10-1 Koné drilling campaigns

Company	Phase	Holes			Metres		
		RC	Diamond	Total	RC	Diamond	Total
Red Back	2010 Koné area	8	-	8	943.0	-	943.0
Sirocco	2013 Koné area	43	-	43	3,341.0	-	3,341.0
Orca	2017 - 2018 Koné area	64	2	66	13,360.0	527.8	13,887.8
Montage	2019 – 2020 Reconnaissance	152	-	152	6,153.0	-	6,153.0
	2019 – 2020 Koné area	96	43	139	10,297.3	16,256.9	26,554.2
	2019 – 2020 Geotechnical	-	7	7	-	956.5	956.5
	2021 Reconnaissance	42	-	42	3,629.0	-	3,629.0
	2021 Geotechnical	-	30	30	-	984.6	984.6
	2021 Koné area – assayed	134	94	228	17,604.0	37,918.7	55,522.7
	2021 Koné area – not assayed	-	7	7	-	2,146.6	2,146.6
Subtotal resource drilling		**345**	**139**	**484**	**45,545.3**	**54,703.4**	**100,248.7**
Total		**539**	**183**	**722**	**55,327.3**	**58,791.1**	**114,118.4**

Table 10-2 presents the number and proportion of assayed mineralized domain estimation dataset composites within the volume of mineralization classified as Indicated or Inferred in the optimal pit constraining Mineral Resource Estimates by drilling type and year. This table provides an indication of the relative contribution of assays from each laboratory to Mineral Resource Estimates.

Table 10-2 Mineralized domain composite estimation dataset within resource volume by drilling group

Year	Number of composites			Proportion of composites		
	RC	Diamond	Total	RC	Diamond	Total
2010	316	-	316	1%	-	1%
2013	1,336	-	1,336	5%	-	5%
2017	1,070	-	1,070	4%	-	4%
2018	4,079	233	4,312	14%	1%	15%
2019	825	818	1,643	3%	3%	6%
2020	2,109	4,024	6,133	7%	14%	21%
2021	3,786	10,047	13,833	13%	35%	48%
Total	**13,521**	**15,122**	**28,643**	**47%**	**53%**	**100%**

10.2 Koné RC Drilling

10.2.1 Drilling and sampling procedures

The RC drill rigs (Figure 10-1) generally utilized 140mm (5.5 inch) face sampling bits. Samples were collected over 1m down-hole intervals from the base of the cyclone with a systematic procedure adopted for sample handling from collection at the cyclone to the laboratory dispatch stage as follows:

- Each metre sample was collected from the cyclone in a new 55 by 100 cm plastic sample bag labelled with the hole number and interval and weighed at the rig with the weight recorded on the drill log sheet
- The bulk sample was then passed through a three-tier riffle splitter with an approximately 3kg primary "original" sub-sample collected in a plastic bag which was then sealed
- The bulk sample was passed through riffle splitter a second time to produce an approximately 3kg archive sample with the remaining bulk sample stored in the original bag
- Duplicates were collected by passing the bulk sample through the riffle splitter a third time producing another approximately 3 kg sub-sample
- Samples tags were added to each sub-sample from numbered ticket books, with the hole number and interval clearly written on the ticket stub for reference
- The 100 cm x 55 cm plastic bags containing the bulk reject sample were left at the drill site in ordered lines
- The riffle splitter was cleaned thoroughly with compressed air between samples
- All sub-samples (original, archive and duplicate) were transported to the field office at the end of the shift, where the archive sample is stored and original and duplicates prepared for despatch to the analytical laboratory
- All assay pulps were returned to the field office from the laboratory and stored for future reference.

The 1m RC samples were submitted for analysis, with the exception of selected samples from the 2013 RC drilling which were composited over 2m intervals for assaying.

Field geologists geologically logged all RC holes over 1m intervals with information including rock types, structures, quartz veining type and percentages, sulphide occurrence and alteration type and intensity recorded on paper drill log sheets. Sieved samples were retained for reference in plastic chip trays.

Figure 10-1 Drilling at Koné in 2013

10.2.2 Collar and down-hole surveying

Drill hole locations prior to 2018 were set out using a handheld GPS and after that by Differential GPS (DGPS) and marked with wooden stake. Drill rigs were aligned with designed azimuths using compasses corrected for magnetic declination.

Upon completion of the drilling, a cement marker, inscribed with the drill hole name, was placed at the collar. After drilling all collars were surveyed using DGPS equipment, with the exception of two 2021 resource area RC holes for which only handheld GPS coordinates are available. RC Holes were down-hole surveyed as follows:

- 2010 holes were generally surveyed with a single shot Camteq Pro shot instrument at intervals of around 30m
- 2013 holes were generally surveyed at intervals of around 80m with a Reflex Ez-Trac single-shot survey tool (Reflex)
- 2017 holes were generally surveyed at intervals of around 40m with a Reflex tool.
- 2018 RC and diamond holes were generally surveyed at intervals of around 30m with a Reflex tool
- 2019 and 2020 Koné holes were generally surveyed with a Reflex tool at intervals of around 10 to 20m
- 2021 Koné holes were generally surveyed with a gyro tool at intervals of around 10 to 25m, and less commonly with Reflex tool at intervals of around 10 to 35m
- Reconnaissance holes were generally not down-hole surveyed.

It is considered that hole paths have been located with sufficient accuracy for the Mineral Resource estimates and exploration activities.

10.2.3 Sample representivity

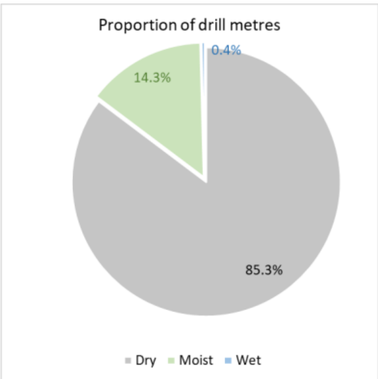
RC sample condition

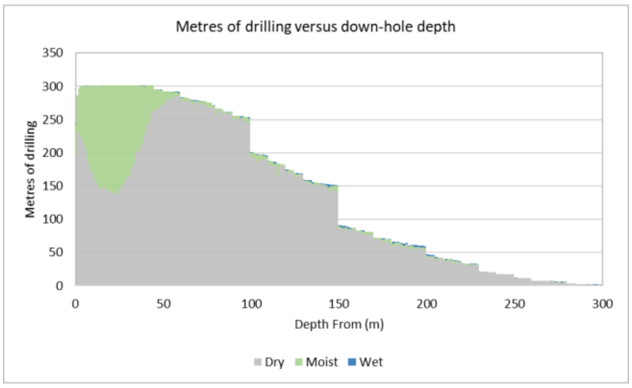
Sample condition is an important factor in the reliability of RC sampling, and wet samples can be associated with unrepresentative, potentially biased samples.

For all RC drilling field geologists recorded sample condition with samples assigned to dry, moist, or wet categories. Site visit observations suggest that samples logged as moist have little apparent moisture and, in terms of sample quality can be considered as effectively dry.

The summaries of sample condition logging in Figure 10-2 demonstrate that wet samples provide only a small proportion of the RC drilling and any uncertainty over the reliability of these samples does not significantly affect confidence in resource estimates.

Figure 10-2 Sample condition logging for Koné RC drilling





RC Sample recovery

In conjunction with bit diameters, density measurements, and moisture content estimates where available recovered sample weights provide an indication of sample recovery for RC drilling which is an important factor for assessment of the reliability of the sampling.

Sample recovery for high quality RC drilling typically averages around 80%, and estimated recoveries of consistently less than approximately 70% can be associated with unrepresentative samples and significantly biased assay grades.

Field procedures for the RC drilling programs generally included weighing recovered sample material, with weights available for most of this drilling. No sample weights are available for the 2010 and 2013 RC campaigns which represent around 6% of the data informing Mineral Resource estimates.

Sample recovery was estimated for each weighed sample from bit diameters supplied by Montage with densities assigned by oxidation domain using the values used for resource estimates. No moisture content estimates are available for Koné RC samples, and sample recovery estimates make no allowance for moisture. In the author's experience, this is likely to result in some overstatement of average recoveries for oxidized and fresh samples.

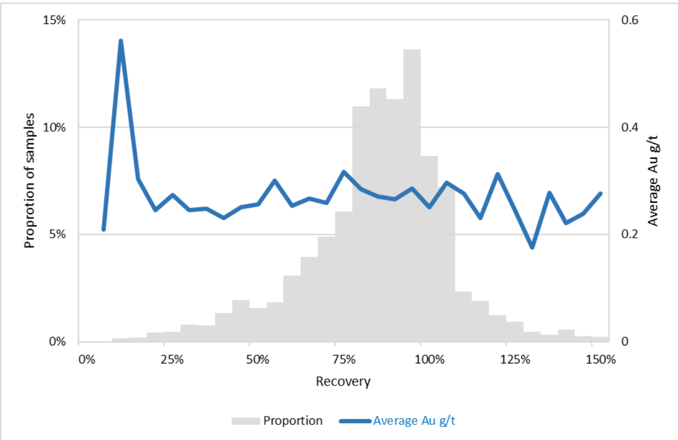
Table 10-3 summarizes RC sample recovery estimates by logged sample condition and Figure 10-3 shows average gold grade for increments of sample recovery. Notable features of this table and figure include the following:

- At 85%, average estimated RC sample recovery is consistent with the good quality RC drilling
- Samples logged as moist or wet show proportionally lower average recoveries than dry samples
- There is no notable association between estimated recovery and average gold grade.

Table 10-3 RC sample recovery estimates

Sample Condition	Number of Samples	Average Recovery
Dry	35,357	86%
Moist	5,431	81%
Wet	148	49%
Undefined	1,470	82%
Total	**42,406**	**85%**

Figure 10-3 Gold grade versus sample recovery for RC drilling



10.3 Diamond Drilling

10.3.1 Drilling and sampling procedures

Diamond drilling utilized triple tube core barrels where necessary to achieve good core recovery with generally 3m drill runs and shorter runs where necessary to maximize core recovery. The drilling was conducted at PQ diameter (122.6 mm hole diameter) to depths of around 37-75 m, and HQ diameter (96 mm) for deeper drilling. Seven holes drilled during 2019 included RC pre-collars to down-hole depths of 60 to 120 m.

All on-site core handling was supervised by a company geologist. At the drilling site, core was placed directly in core trays. Where possible core was oriented using a Reflex ACT III for 2019 and 2020 programs. Core recovery was measured at the drill site prior to delivery of the core to the camp.

Core handling and sampling procedures included the following:

- Drill core was transported to the field office at the end of every shift
- After geological logging the core was halved with a diamond saw with samples collected over generally 1m intervals (minimum 0.05m) assigned by logging geologists, respecting lithological changes
- Sampled half core was placed in plastic sample bags in sequence and securely stored before batch assignment and submission to the assay laboratory
- All core was digitally photographed prior to cutting in a wet and dry state and stored in plastic core trays at the field office.

All core was geotechnically logged at the drill site prior to transport to the field office, with core recovery, rock quality designation (RQD), rock strength and weathering recorded. After transport to the field office, core was geologically logged with rock type, stratigraphic subdivisions, alteration, oxidation and mineralization routinely recorded along with foliation, cleavage, faulting and veining, including structural measurements of these features.

10.3.2 Collar and down-hole surveying

Drill hole locations were set out using a handheld GPS and after that by Differential GPS and marked with wooden stake. Drill rigs were aligned with designed azimuths using compasses corrected for magnetic declination.

Upon completion of the drilling, a cement marker, inscribed with the drill hole name, was placed at the collar. After drilling all diamond hole collars were surveyed using DGPS equipment, with down-hole surveying as follows:

- 2018 holes were generally surveyed at intervals of around 30m with a Reflex tool
- 2019, 2020 and 2021 holes were generally surveyed with a Gyro tool at intervals of around 5m, with information available for 22 of the 2021 holes including only initial widely spaced Reflex surveys.

The author considers that hole paths have been located with sufficient accuracy for the Mineral Resource estimates and exploration activities.

10.3.3 Sample representivity

To provide a consistent basis for analysis, measured core recoveries for 0.1m to 6.0m core runs available for all resource area diamond holes were composited to 3m intervals reflecting the dominant length. The review dataset excludes information from geotechnical diamond holes outside the resource area which do not inform mineral resource modelling.

Core recoveries for these intervals average 99.0% (Table 10-4) with only approximately 5% of composites showing recoveries of less than 90%. These recoveries are consistent with the author's experience of high-quality diamond drilling.

Table 10-4 Core recovery for 3m composites from diamond drilling

Oxidation	Number	Minimum	Average	Maximum
Oxide	1,574	17.33%	89.79%	140.33%
Transitional	783	43.89%	95.97%	128.67%
Fresh	17,057	50.00%	99.98%	174.82%
Total	**19,414**	**17.33%**	**98.99%**	**174.82%**

10.4 Reconnaissance RC Drilling

Reconnaissance RC holes were inclined at 50 or 55o at orientations and hole spacings reflecting interpreted local mineralization trends and previous exploration sampling. Hole spacings vary from rarely around 20m to around 200m spaced traverses.

The 2019 and 2020 reconnaissance RC drilling targeted several exploration targets identified by soil and rock chip sampling. Drilling completed in 2019 focussed on the general area surrounding the Koné mineralization and returned several low tenor anomalies (<0.20g/t Au). The 2020 reconnaissance drilling targeted the Petit Yao prospect and intersected narrow mineralized zones.

The 2021 reconnaissance RC drilling included 18 holes for 1,823m targeting potential mineralization to the south of the Koné area with no significant results and 24 holes in the Petit Yao area. The Petit Yao drilling which comprised 1,832m of shallow drilling with average hole depths of 39m and the 1,806m of deeper RC drilling up to 114m depth returned several mineralized intercepts

The reconnaissance RC drilling generally did not include such rigorous surveying, or sampling and assaying procedures as adopted for resource drilling. The report author concurs with this approach, and considers it appropriate for this drilling.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Introduction and Summary

For discussion of field sampling, sample preparation and analysis, this sampling and analyses are subdivided as follows:

- **Exploration sampling** including soil sampling, trenching and pitting
- **RC and diamond Koné area drilling** informing the Mineral Resource estimates
- **Reconnaissance RC drilling** testing several exploration targets identified by soil and rock chip sampling in the Koné area. These did not include as rigorous surveying, or sampling and assaying procedures as adopted for resource drilling.

References to "inhouse" personnel in this report refer to personnel employed by directly Red Back, Orca or Montage respectively reflecting the changes in project ownership. Although undertaken by different corporate entities, field procedures and key staff have remained consistent for all Koné drilling phases ensuring consistency in the sampling methodology. Sample submission and sample security procedures have been consistent for all sampling phases.

All sample preparation and gold assaying samples from the Koné drilling and exploration sampling was undertaken by independent commercial laboratories. These laboratories are independent of the issuer and provided services under industry standard commercial arrangements. Analyses undertaken by inhouse personnel were limited to immersion density measurements by Orca and Montage personnel. No analyses were undertaken by Red Back personnel.

All field sampling activities were supervised by field geologists with industry standard methods employed for sampling and geological logging.

Routine sampling and assaying procedures included Quality Assurance Quality Control (QAQC) monitoring of the reproducibility and accuracy of sample preparation and assaying which are consistent with the author's experience of good industry standard practises. This included routine submission of coarse blanks and reference standards along with interlaboratory repeat assaying.

The handling, sampling, transport, analysis and storage of sample material along with documentation of analytical results is consistent with the author's experience of good, industry standard practise.

The author considers that quality control measures adopted for sampling and assaying of the Koné drilling and exploration have established that the field sub-sampling, and assaying is representative and free of any biases or other factors that may materially impact the reliability of the sampling and analytical results.

The author considers that the sample preparation, security and analytical procedures adopted for the Koné drilling and exploration sampling provide an adequate basis for the Mineral Resource estimates and exploration activities.

11.2 Sample Submission Procedures and Security

For all sample types, all sample handling and sub-sampling was supervised by inhouse geologists. Prior to collection by laboratory staff, all sample collection and transportation were undertaken or supervised by inhouse personnel. No other personnel were permitted unsupervised access to samples before collection by laboratory staff.

Routine sample handling procedures for soil, trench and RC drill samples comprised the following:

- Inhouse personnel transported soil, trench and RC drill samples directly to the sample storage facility in Fadiadougou where the samples were arranged in order (Figure 11-1) and archive samples separated and stored.
- Diamond core was delivered to the field office by inhouse personnel and after geological logging the core was sampled with samples subsequently treated consistently with other sample types
- Field duplicate samples, which were routinely collected from RC and diamond drilling were collected consistently with and assayed in the same batch as original samples providing an indication of the repeatability of field sub-sampling procedures and checking for sample-misallocation by field staff, the laboratory and during database compilation
- Coarse blanks, comprising samples of un-mineralized granite collected from well outside the mineralized area were inserted into sample sequences at pre-defined intervals. These blanks, which were blind to the assay laboratories test for contamination during sample preparation, and provide a check of sample misallocation by field staff, the laboratory and during database compilation
- Samples of certified reference standards were inserted into sample sequences at pre-defined intervals. Assay results for these standards, which were prepared by Rocklabs Ltd in Auckland New Zealand (Rocklabs), Ore Research & Exploration P/L in Perth (OREAS), Western Australia or Geostats Pty Ltd (Geostats) provide an indication of assaying accuracy
- Certified reference standards and coarse blanks were inserted into the sample sequence at pre-defined interval.
- All samples were packaged in sequence into polywoven sacks and sealed with plastic ties for transport to the analytical laboratory
- A sample submission form detailing sample number sequences and specifying analytical methods was prepared and for each batch. A hardcopy submission form was included with the submitted samples and an electronic copy emailed to the laboratory.

Samples submitted to Bureau Veritas in Abidjan or SGS in Yamoussoukro for analysis were delivered to the laboratory by inhouse personnel. Samples assayed by Intertek were collected from the Fadiadougou field office by Intertek staff.

Figure 11-1 Fadiadougou sample organisation and storage facility

11.3 Primary Assay Laboratories

Primary samples from the Koné exploration sampling and drilling were submitted to one of three commercial laboratories for gold grade analysis (Table 11-1). The accreditation status of each laboratory are outlined below. Sample preparation and analytical procedures for each sampling phase and laboratory are described in following sections.

For samples submitted to SGS, sample preparation was performed by SGS in Yamoussoukro Côte d'Ivoire with analysis at the SGS laboratory in Tarkwa, Ghana or less commonly SGS Ouagadougou, Burkina Faso. SGS preparation facilities and analytical laboratories at Yamoussoukro, Tarkwa and Ouagadougou respectively are not accredited by any recognized accreditation authority. SGS services include quality assurance protocols in line with ISO 17025.

All sample preparation and analyses of samples analyses by Bureau Veritas was undertaken by Bureau Veritas in Abidjan, Côte d'Ivoire. Bureau Veritas Abidjan is not accredited by any recognized accreditation authority. The laboratory operates under the ISO 17025 accreditation of the Bureau Veritas Vancouver as endorsed by the Standards Council of Canada.

All sample preparation and analyses of samples analyses by Intertek was undertaken by Intertek Minerals Ltd (Intertek) in Tarkwa, Ghana. In December 2017 Intertek was accredited by the South Africa National Accreditation System (SANAS) in accordance with ISO/IEC 17025:2005 (Facility Accreditation Number T0796). The accreditation demonstrates technical competency for a defined scope and the operation of a quality management system.

Table 11-1 Analytical laboratories by sampling phase

Year	Soil/Pit	Trenching	Reconnaissance RC Drilling	Koné Area Drilling	
				RC	Diamond
2009-10	SGS	SGS	-	BV	-
2013	-	BV	SGS	-	-
2017	-	-	-	BV	
2018	-	-	-	INT	BV
2019	BV	BV	BV	BV	-
2020-21	BV	-	BV/INT	BV/INT	BV/INT
Key: SGS: SGS, BV: Bureau Veritas, INT: Intertek					

11.4 Exploration Sampling

11.4.1 Soil Sampling

All soil samples were collected and transported to the field camp the same day under the supervision of a field geologist.

Samples collected from the 2009 and 2010 auger soil sampling were analysed by SGS. All sample preparation was completed by SGS Yamoussoukro. After checking and drying, samples were pulverized to nominally to 90% passing 75 microns. Pulverized samples were then transported by SGS to their Tarkwa laboratory for analysis by 50g fire assay with Aqua Regia digest and DIBK extraction with AAS determination at a 1ppb detection limit.

Sample preparation and analysis for samples from the 2019 soil sampling program was completed by Bureau Veritas in Abidjan, Côte d'Ivoire utilizing sample preparation and analyses methods consistent with those employed by SGS for the 2009 and 2010 soil sampling.

Quality control samples were inserted into sequences of soil sampling at the field camp under the supervision of the Project Geologist. Coarse blanks and Geostats certified reference standards were submitted in batches of 2019 soil samples at an average frequency of around 1 standard or blank per 77 primary samples for both types.

Assay results for coarse blanks and Rocklabs (2009-10) and Geostats (2019-20) standards included in batches of soil samples provide adequate confirmation of the reliability of sample preparation and analysis (Table 11-2).

Table 11-2 Coarse blanks and reference standards included soil samples

Coarse Blanks				
Assay Group	Number Samples	Gold assay (ppb) Minimum	Average	Maximum
2009-10 SGS	137	1	5.92	29
2019 Bureau Veritas	77	1	1.16	3

Reference Standards with five more assays				
Reference Standard	Number Samples	Gold grade (ppb) Expected	Avg. Assay	Avg. vs. Expected
GLG305-1	5	101.6	99.8	-2%
GLG305-3	5	55.5	52.4	-6%
GLG310-3	10	119.3	113.5	-5%
GLG313-5	10	83.4	66.7	-20%
GLG908-4	13	65.9	64.0	-3%
GLG910-2	13	24.7	21.6	-13%
GLG914-3	5	205.8	205.2	0%
GLG916-1	12	5.1	8.6	70%
OXA26	38	79.8	82.4	3%
OXA45	27	81.1	99.3	22%
OXD43	5	401	462	15%
OXD57	33	413	407	-1%
OXE42	8	610	605	-1%
OXE56	24	611	592	-3%

11.4.2 Trenching

Samples collected from the 2009 and 2010 trenches were submitted to SGS for analysis. Sample preparation was undertaken by SGS Yamoussoukro. After checking and drying, samples were pulverized to nominally to 90% passing 75 microns. Pulverized samples were then transported by SGS to their Tarkwa laboratory for analysis by 50g fire assay with Aqua Regia digest and DIBK extraction with AAS determination at a 1ppb detection limit.

Samples from the 2013 trenches were analysed by Bureau Veritas utilizing sample preparation and analyses methods consistent with those employed by SGS.

Assay results for coarse blanks included in batches of trench samples at an average frequency of around one blank per 18 primary samples are summarized in Table 11-3 with samples assaying at below the detection limit of 0.01 g/t assigned gold grades of half the detection limit. This table demonstrates that coarse blank assays show very low gold grades relative to typical Koné mineralization with no indication of significant contamination or sample misallocation.

Samples of Rocklabs certified reference standards were included in batches of trench samples at an average frequency of around 1 standard per 45 primary sample. Although, as expected there is some variability for individual samples, average assay results closely match expected values (Table 11-3).

Table 11-3 Coarse blanks and reference standards included with 2009-10 trench samples

Coarse Blanks					
Assay Group	**Number Samples**	**Gold assay (g/t) Minimum**	**Average**	**Maximum**	**Proportion > Detection**
2009-10 SGS	117	0.005	0.017	0.22	38%
2013 Bureau Veritas	3	0.005	0.028	0.07	67%

Reference Standards				
Reference Standard	**Number Samples**	**Gold grade (g/t) Expected**	**Avg. Assay**	**Avg. vs. Expected**
2010 SGS				
OXD27	10	0.416	0.422	1%
OXD43	4	0.401	0.418	4%
OXE56	4	0.611	0.640	5%
OXF65	10	0.805	0.835	4%
OXH37	3	1.286	1.337	4%
OXH52	13	1.291	1.347	4%
OXI7	3	2.384	2.360	-1%
Combined	48	0.956	0.983	3%
2013 Bureau Veritas				
OXD27	1	0.416	0.480	15%
OXI67	1	1.817	1.780	-2%
Combined	**2**	**1.117**	**1.130**	**1%**

11.4.3 Pit Sampling

Samples from the 2019 pitting program were submitted to Bureau Veritas in Abidjan, Côte d'Ivoire for analysis.

After checking and drying, samples were pulverized to nominally to 90% passing 75 microns and analysed for gold by 50 g fire assay with lead collection, solvent extraction and AAS determination with a lower detection limit of 0.01ppm.

Coarse blanks and OREAS certified reference standards were submitted in batches of pit samples at an average frequency of around 1 per 26 and 57 primary samples respectively. Gold assay grades reported for these samples are summarized in Table 11-4 with assays reported as below the detection limit of 0.01 g/t assigned values of half the detection limit.

Table 11-4 demonstrates that coarse blank assays show very low gold grades, and average assay results for standards closely match expected values, supporting the reliability of sample preparation and assaying for the pit samples.

Table 11-4 Coarse blanks and reference standards included with 2019 pit samples

Coarse Blanks					
Assay Group	**Number Samples**	**Gold assay (g/t) Minimum**	**Average**	**Maximum**	**Proportion > Detection**
2019 SGS	24	0.005	0.006	0.020	13%

Reference Standards				
Reference Standard	**Number Samples**	**Gold grade (g/t) Expected**	**Avg. Assay**	**Avg. vs. Expected**
OREAS-214	2	3.030	2.945	-3%
OREAS-251	9	0.504	0.502	0%
Combined	11	0.963	0.946	-2%

11.5 Koné RC and Diamond Drilling

11.5.1 Sample Preparation and Analysis

Primary analyses of samples from the RC and diamond drilling in the Koné area, which provide the basis for the current Mineral Resource estimate were undertaken by several commercial laboratories (Table 11-1). Table 11-5 presents the number and proportion of assayed mineralized domain estimation dataset composites within the volume of mineralization classified as Indicated or Inferred in the optimal pit constraining Mineral Resource Estimates by laboratory. This table provides an indication of the relative contribution of assays from each laboratory to Mineral Resource Estimates.

Sample preparation and analytical methods were consistent for all laboratories and comprised the following:

- Each batch received was laid out in sequence, weighed and checked in to the Bureau Veritas system. Inhouse geologists responsible for sample submission to the laboratory were informed of any missing samples or extra samples not listed on the submission form, and a replacement or corrected submission form prepared by inhouse personnel

- Each, nominally 3 Kg sample was jaw crushed to >80% passing 2 mm and riffle split to produce two 1.5 kg sub-samples. After every twentieth sample and at the end of each assay batch a crusher flushing sample of barren vein quartz was used to clean the crusher plates

- A 1.5 kg sample was pulverized in a ring mill to 85% passing 75 microns and a 250 g sub-sample of the pulverized material collected as the primary sample pulp.

Pulp samples were analysed for gold by 50 g fire assay with lead collection, solvent extraction and AAS determination with a lower detection limit of 0.01ppm.

Table 11-5 Estimation dataset by assay laboratory

Laboratory	Number of composites	Proportion of composites
ALS	1,942	7%
Bureau Veritas	18,559	66%
Intertek	7,669	27%
Total	**28,170**	**100%**

11.5.2 Routine Monitoring of Sampling and Assay Reliability

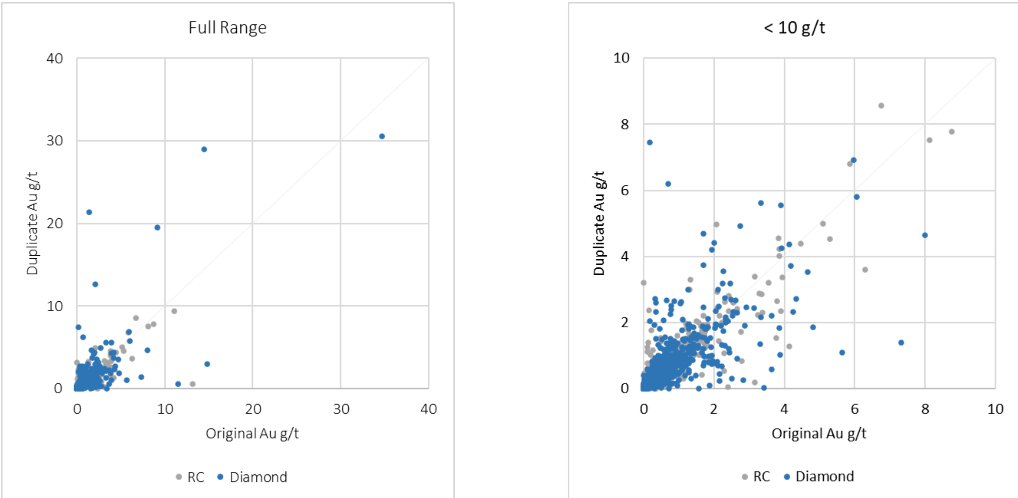
11.5.2.1 Field Duplicates

Field duplicates were collected for Koné RC and diamond drilling at average frequencies of around 1 duplicate per 25 primary samples for both drill types. Field duplicates were collected consistently with and assayed in the same batch as original samples.

The summary statistics and scatter plots in Figure 11-2 demonstrate that although there is some scatter for individual pairs duplicate assay results generally correlate reasonably well with original results demonstrating the adequacy of field sub-sampling procedures.

Figure 11-2 Field duplicates for Koné RC and diamond drilling

Au g/t	RC				Diamond			
	Full Set		0.1 to 10 g/t		Full Set		>0.1 g/t	
	Orig.	Dup.	Orig.	Dup.	Orig.	Dup	Orig.	Dup
Number	1,974		831		1,671		905	
Average	0.28	0.27	0.60	0.59	0.43	0.44	0.65	0.65
Difference		-4%		-2%		3%		0%
Variance	0.53	0.41	0.73	0.67	1.54	2.07	0.69	0.69
Coef. Variation	2.57	2.35	1.41	1.38	2.87	3.24	1.27	1.27
Minimum	0.01	0.01	0.10	0.10	0.01	0.01	0.10	0.10
1st Quartile	0.01	0.01	0.19	0.19	0.04	0.04	0.20	0.19
Median	0.07	0.07	0.32	0.34	0.14	0.14	0.37	0.37
3rd Quartile	0.27	0.27	0.65	0.67	0.41	0.42	0.74	0.74
Maximum	13.2	9.39	8.76	8.57	34.7	30.6	8.00	7.46
Correl. Coef.	0.85		0.91		0.76		0.70	



11.5.2.2 Coarse Blanks

Coarse blanks were routinely included in assay batches from all phases of Koné RC and diamond drilling at an average frequency of around one blank per 22 primary samples.

Table 11-6 summarizes gold assays for these blanks by assay laboratory with samples assaying at below the detection limit of 0.01 g/t assigned values of half the detection limit. This table excludes two anomalous samples from the 2018 drilling with gold grades of 0.56 and 1.10 g/t, and two anomalous samples from the 2018 drilling with gold grades of 0.35 and 0.68 g/t which appear to reflect misallocation.

Table 11-6 demonstrates that coarse blank assays show very low gold grades relative to typical Koné mineralization with no indication of significant contamination or sample misallocation.

Table 11-6 Coarse blanks included with Koné drill samples

Laboratory	Number Blanks	Gold assay (g/t) Minimum	Average	Maximum	Proportion > Detection
Bureau Veritas	2,517	0.005	0.01	0.05	17%
Intertek	1,330	0.005	0.01	0.12	10%
SGS	129	0.005	0.01	0.04	15%
Combined	**3,976**	**0.005**	**0.01**	**0.12**	**14%**

11.5.2.3 Reference Standards

For all phases of Koné RC and diamond drilling samples of certified reference standards prepared by commercial standards suppliers were inserted in assay batches at an average rate of around 1 standard per 24 primary samples.

For the 2010 and 2013 drilling programs, the reference standards were sourced from Rocklabs. For the 2017 and 2018 drilling, standards were sourced from OREAS. For the 2019, 2020 and 2021 drill programmes both Geostats and OREAS standards were used. Expected gold grades for the standards range from around 0.3 to 6.1 g/t covering the range of typical gold grades shown by Koné drill hole samples.

Table 11-7 summarizes assay results for standards included in batches of drill samples by assay laboratory. This table excluded a small number of standards for which fewer than five samples were analysed by each laboratory, and a small number of standards for which reported assays match expected values so poorly they are suggestive of sample misallocation. Table 11-7 demonstrates that although, as expected there is some variability for individual samples, average assay results closely match expected values.

Table 11-7 Reference standards included with Koné drill samples

Laboratory	Reference Standard	Number Samples	Gold grade (g/t) Expected	Gold grade (g/t) Avg. Assay	Avg. vs. Expected
Bureau Veritas	G308-2	229	1.11	1.07	-5%
	G314-1	320	0.75	0.77	4%
	G315-4	227	0.32	0.32	0%
	G316-6	228	1.40	1.45	3%
	G316-8	42	6.11	6.13	0%
	G319-2	224	3.96	4.00	1%
	G908-4	94	0.96	0.97	1%
	G910-10	99	0.97	0.98	1%
	G912-7	223	0.42	0.42	1%
	G913-2	42	2.40	2.44	2%
	G916-2	229	1.98	2.00	1%
	G916-4	42	0.51	0.51	0%
	OREAS-210	61	5.49	5.46	-1%
	OREAS-214	40	3.03	3.05	1%
	OREAS-219	30	0.76	0.77	1%
	OREAS-222	28	1.22	1.24	2%
	OREAS-231	56	0.54	0.53	-3%
	OREAS-232	35	0.90	0.89	-1%
	OREAS-250	8	0.31	0.38	22%
	OREAS-250b	60	0.33	0.32	-4%
	OREAS-251	63	0.50	0.51	2%
	OREAS-253	20	1.22	1.23	1%
	Combined	**2,400**	**1.47**	**1.49**	**4%**
Intertek	G314-1	21	0.75	0.77	3%
	G908-4	24	0.96	0.93	-4%
	G910-10	15	0.97	0.94	-3%
	OREAS-210	241	5.49	5.52	1%
	OREAS-214	248	3.03	3.05	1%
	OREAS-219	80	0.76	0.79	4%
	OREAS-231	87	0.54	0.55	2%
	OREAS-232	76	0.90	0.91	1%
	OREAS-250	12	0.31	0.32	4%
	OREAS-250b	76	0.33	0.33	0%
	OREAS-251	51	0.50	0.51	3%
	OREAS-253	75	1.22	1.24	2%
	OREAS-502b	50	0.50	0.49	-2%
	OREAS-504b	137	1.61	1.59	-1%
	Combined	**1,193**	**2.26**	**2.27**	**5%**
SGS	OxH52	12	1.29	1.27	-1%
	OxH66	12	1.29	1.28	0%
	Oxi67	9	1.82	1.82	0%
	SH41	10	1.34	1.31	-2%
	Combined	**43**	**1.41**	**1.40**	**-1%**

11.5.3 Cyanide Leach and Screen Fire Duplicates

In August 2018, for 59 RC sample intervals with original Intertek assays, additional field duplicates were collected and submitted to Intertek for gold analysis by 50 g fire assay consistent with the original assaying, bulk cyanide leach with AAS finish (with fire assay on tails) and screen fire assay. These duplicates were assigned new sample identifiers and were blind to Intertek.

As summarized in Table 11-8 , with the exception of the five anomalous duplicates with assay results that match original samples so poorly they are suggestive of sample misallocation and a single high grade outlier, average duplicate assays from each method reasonably match average original fire assay grades. These results provide additional support for the reliability of Intertek fire assays.

Table 11-8 Alternate method duplicate assays versus original assays for Koné drill samples

		Original Intertek FA	Fire Assay	Duplicate CN Leach	Screen Fire
Full dataset (59)	Average (Au g/t)	1.42	1.23	1.18	1.10
	vs. Original		-14%	-17%	-23%
	vs. Duplicate FA			-4%	-10%
Exclude anomalous (54)	Average (Au g/t)	1.21	1.32	1.26	1.19
	vs. Original		10%	5%	-2%
	vs. Duplicate FA			-5%	-11%
Exclude anomalous and > 10 g/t (53)	Average (Au g/t)	1.05	1.04	1.08	1.01
	vs. Original		-1%	3%	-4%
	vs. Duplicate FA			4%	-3%

11.5.3.1 Inter-laboratory repeats

Information available to demonstrate the accuracy of primary gold assaying for Koné drill samples includes several sets of inter-laboratory fire assay repeats. As outlined below these repeats help support the general accuracy of the primary analyses.

August 2018 ALS, Rosia Montana, Romania

The samples repeated by ALS comprised 239 samples originally assayed by Bureau Veritas in 2017 and 649 samples originally assayed by Intertek in 2018. These samples included 42 coarse blanks for which ALS reported very low gold grades for each of the coarse blanks. The blanks provide little information about general accuracy of the original assaying and these results were excluded from the review dataset

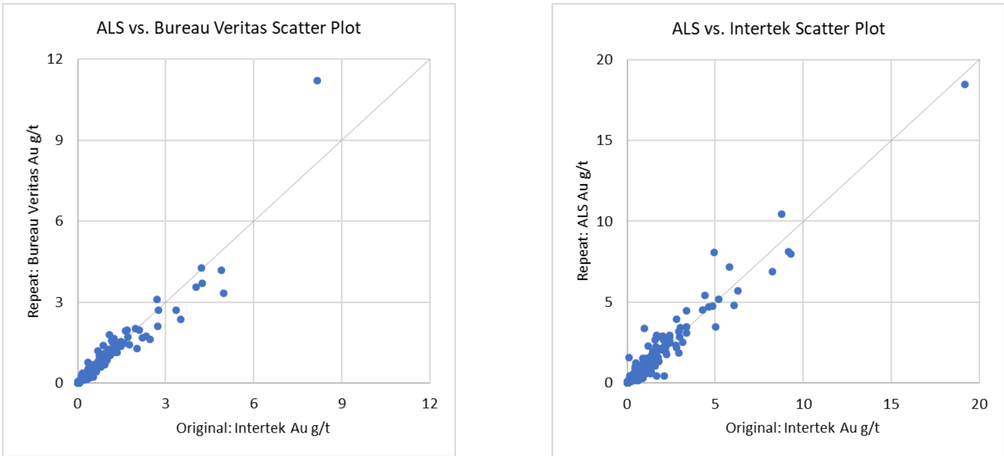
In February 2016 ALS Rosia Montana was accredited by the Standards Council of Canada in accordance with ISO/IEC 17025:2005 (Accredited Laboratory Number 742).

The summary statistics and scatter plots in Figure 11-3 demonstrate that although there is some scatter for individual pairs the ALS repeat assay results generally correlate reasonably well with original results providing additional confidence in the accuracy of the primary Bureau Veritas and Intertek assaying.

Reasons for the slight difference in average grade shown for repeats of Intertek assays are uncertain. The magnitude of this difference is not significant at the current level of project evaluation.

Figure 11-3 ALS interlaboratory repeat assays of Koné drill samples

	ALS vs. Bureau Veritas		ALS vs. Intertek	
	Original Au g/t	**Repeat Au g/t**	**Original Au g/t**	**Repeat Au g/t**
Number	228		618	
Average	**0.66**	**0.66**	**0.67**	**0.69**
Difference.		**-1%**		**4%**
Variance	0.96	1.05	1.71	1.74
Coef. Variation.	1.47	1.57	1.96	1.91
Minimum	0.01	0.01	0.01	0.01
1st Quartile	0.14	0.12	0.10	0.11
Median	0.36	0.34	0.29	0.30
3rd Quartile	0.77	0.82	0.72	0.75
Maximum	8.17	11.20	19.18	18.45
Correl. Coef.	0.96		0.97	

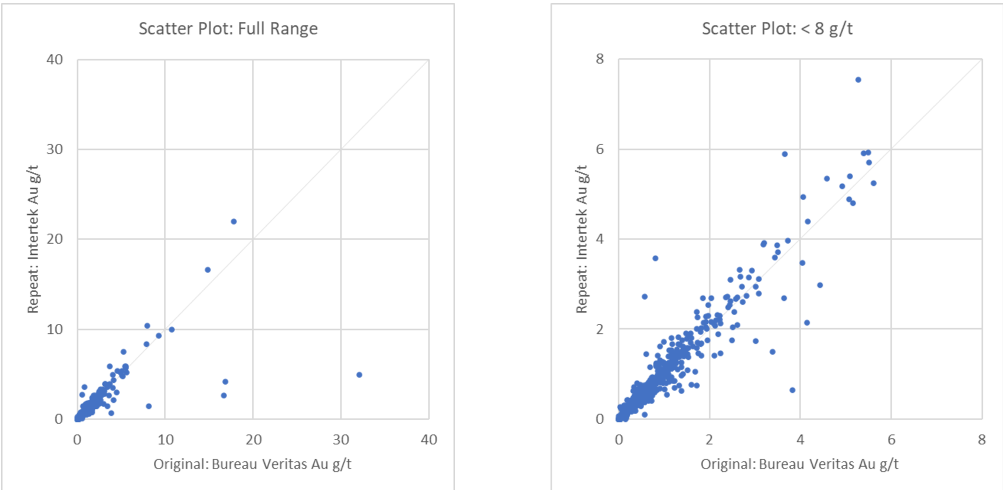


February 2021 Bureau Veritas repeats

In February 2021, 989 sample pulps with original Intertek gold assays were submitted to Bureau Veritas Bureau Veritas in Abidjan, Côte d'Ivoire for analysis. As demonstrated by the comparative statistics and scatter plots in Figure 11-4 , with the exception of a small number of poorly correlating higher grade pairs, the Bureau Veritas assays generally reasonably match the original assays confirming the general consistency of results reported by the two primary assay laboratories used for later phases of Koné drilling.

Figure 11-4 Bureau Veritas interlaboratory repeat assays of Koné drill samples

	Full dataset		0.10 to 8.0 g/t	
	Original Intertek Au /t	**Repeat BV Au g/t**	**Original Intertek Au /t**	**Repeat BV Au g/t**
Number	984		816	
Average	**0.76**	**0.75**	**0.80**	**0.83**
Difference.		**-1%**		**4%**
Variance	2.35	1.40	0.73	0.81
Coef. Variation.	2.01	1.58	1.06	1.08
Minimum	0.01	0.01	0.10	0.10
1st Quartile	0.17	0.18	0.28	0.29
Median	0.42	0.44	0.52	0.54
3rd Quartile	0.90	0.91	1.00	1.02
Maximum	32.1	22.0	5.61	7.54
Correl. Coef.	0.76		0.95	



August 2021 SGS repeats

In August 2021, a total of 956 sample pulps comprising 315 and 641 samples originally assayed by Bureau Veritas and Intertek respectively were analysed by SGS Ouagadougou, Burkina Faso.

As demonstrated by the comparative statistics and scatter plot in Figure 11-5 although there is some variability for individual samples, the SGS repeats generally reasonably match the original Bureau Veritas assays with very similar average gold grades. A key feature of these repeats is the comparatively low average gold grades relative to the estimation dataset, and lack of samples with assays of greater than 2 g/t.

Figure 11-6 compares the SGS repeats with original Intertek gold assays. This figure excludes two pairs, for which assays correlate so poorly they are suggestive of sample misallocation (MR461318 1.66 vs 0.05 and MR457828 0.72 vs 0.08 g/t). Figure 11-6 indicates that although the SGS repeats correlate comparatively well with the original assays, there is a general trend for SGS to report marginally lower average grades than Intertek. In the author's opinion this variability is not significant at the current stage of project evaluation.

Figure 11-5 SGS interlaboratory repeats of Bureau Veritas assays

	Full dataset		0.10 to 8.0 g/t	
	Original BV Au /t	**Repeat SGS Au g/t**	**Original BV Au /t**	**ReDFSt SGS Au g/t**
Number	315		175	
Average	**0.22**	**0.23**	**0.36**	**0.36**
Difference.		**2%**		**-2%**
Variance	0.07	0.07	0.08	0.08
Coef. Variation.	1.20	1.15	0.79	0.80
Minimum	0.01	0.01	0.10	0.10
1st Quartile	0.05	0.06	0.17	0.17
Median	0.13	0.14	0.26	0.26
3rd Quartile	0.32	0.30	0.48	0.42
Maximum	1.87	1.53	1.87	1.53
Correl. Coef.	0.90		0.86	

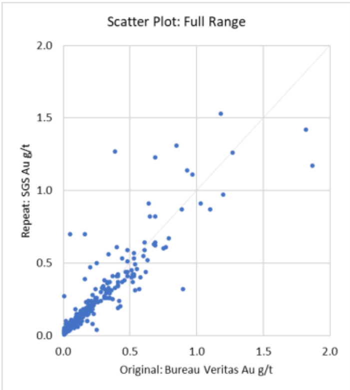
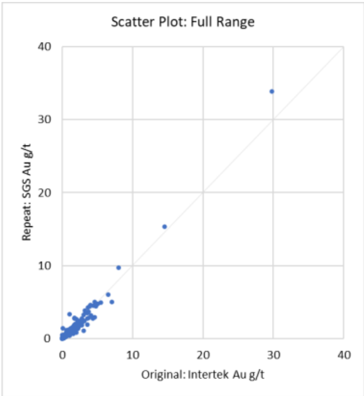
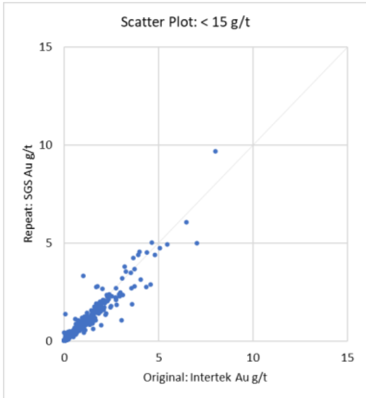


Figure 11-6 SGS interlaboratory repeats of Intertek assays

	Full dataset		0.10 to 20 g/t	
	Original Intertek Au /t	Repeat SGS Au g/t	Original Intertek Au g/t	Repeat SGS Au g/t
Number	639		482	
Average	**0.76**	**0.70**	**0.93**	**0.84**
Difference.		**-7%**		**-10%**
Variance	2.58	2.90	1.48	1.41
Coef. Variation.	2.12	2.43	1.31	1.42
Minimum	0.01	0.01	0.10	0.10
1st Quartile	0.13	0.12	0.25	0.22
Median	0.33	0.32	0.50	0.46
3rd Quartile	0.86	0.77	1.10	0.95
Maximum	29.83	33.90	14.58	15.30
Correl. Coef.	0.98		0.97	





11.6 Reconnaissance RC Drilling

11.6.1 Sample Preparation and Analysis

Samples Analysis from the 2019 reconnaissance RC program, which primarily focused on central portions of the Koné Exploration Permit including the Koné area were submitted to Bureau Veritas in Abidjan, Côte d'Ivoire for analysis consistently with earlier assaying of drill hole samples by this laboratory described above.

Samples from the 2020 reconnaissance RC drilling which targeted the Petit Yao Prospect were submitted to the Intertek laboratory in Tarkwa, Ghana for analysis. After checking and drying, samples were pulverized to nominally to 90% passing 75 microns and a 1 kg sample analysed by 12-hour Leachwell Bulk Leach Extractable Gold (BLEG) and AAS determination with a lower detection limit of 0.01 ppm.

11.6.2 Monitoring of Sampling and Assay Reliability

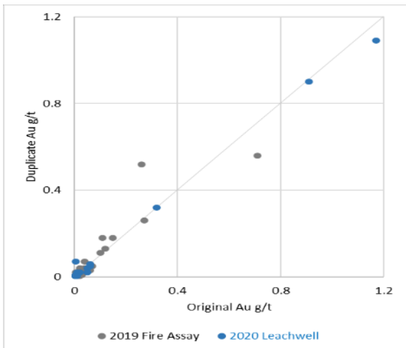
Routine Field Duplicates

Routine field duplicates were collected for the 2019 and 2020 RC reconnaissance drilling at average frequencies of around one duplicate per 29 and 39 primary samples respectively (Figure 11-7). These samples were collected consistently with and assayed in the same batch as original samples providing an indication of the repeatability of field-sub-sampling.

As expected for exploratory drilling, a large proportion of the routine field duplicate intervals from the reconnaissance RC drilling the returned very low gold grades, with only 36 out of the 125 combined set assaying at greater than detection limit of 0.01 g/t. The small numbers of duplicates with elevated gold grades provides a less reliable indication of sampling repeatability than the datasets available for other drilling groups.

Figure 11-7 Field duplicates for reconnaissance RC drilling

	Full set				Greater than detection Limit			
	2019		2020		2019		2020	
	Orig.	Dup.	Orig.	Dup.	Orig.	Dup.	Orig.	Dup.
Number	57		68		26		10	
Average	0.04	0.05	0.04	0.04	0.09	0.10	0.27	0.26
Difference.		6%		-2%		7%		-2%
Variance	0.01	0.01	0.03	0.03	0.02	0.02	0.16	0.14
Coef. Variation.	2.36	2.23	4.13	4.03	1.57	1.46	1.51	1.47
Minimum	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.02
1st Quartile	0.01	0.01	0.01	0.01	0.03	0.02	0.03	0.03
Median	0.01	0.01	0.01	0.01	0.04	0.04	0.06	0.06
3rd Quartile	0.03	0.03	0.01	0.01	0.09	0.10	0.26	0.26
Maximum	0.71	0.56	1.17	1.09	0.71	0.56	1.17	1.09
Correl. Coef.	0.92		1.00		0.93		0.998	



Coarse Blanks and Reference Standards

Coarse blanks and reference standards were included in batches of samples from the 2019 and 2020 reconnaissance RC drilling at average frequencies of around one sample per 23 and 35 primary samples respectively. Gold assays reported for these samples are summarized in Table 11-9 with samples assaying at below the detection limit of 0.01 g/t assigned values of half the detection limit.

Reference standards in Table 11-9 identified with a prefix of "G" were produced by Geostats. The "OREAS" prefixed standard was produced by ORE Research & Exploration Pty.

Table 11-9 demonstrates that, for both Bureau Veritas and SGS coarse blank assays show very low gold grades, and average assay results for standards closely match expected values, supporting the reliability of sample preparation and assaying for the reconnaissance RC samples.

Table 11-9 Coarse blanks and reference standards included with 2019-20 reconnaissance RC samples

Coarse Blanks					
Assay Group	**Number Samples**	**Gold assay (g/t)**			**Proportion > Detection**
		Minimum	**Average**	**Maximum**	
2019 Bureau Veritas (FA)	69	0.005	0.007	0.030	12%
2020 SGS (LW)	129	0.005	0.006	0.050	4%

Reference Standards				
Reference Standard	**Number Samples**	**Gold grade (g/t)**		**Avg. vs. Expected**
		Expected	**Avg. Assay**	
2019 Bureau Veritas (FA)				
G314-1	6	0.75	0.81	6%
G316-8	5	6.11	5.98	-13%
G908-4	6	0.96	0.98	2%
G910-10	5	0.97	0.97	0%
G913-2	6	2.40	2.40	0%
G916-4	6	0.51	0.51	0%
OREAS-251	22	0.50	0.51	1%
Combined	**56**	**1.33**	**1.32**	**0%**
2020 SGS (LW)				
G314-1	16	0.75	0.77	2%
G316-8	6	6.11	5.98	-13%
G908-4	24	0.96	0.93	-3%
G910-10	15	0.97	0.94	-3%
G913-2	6	2.40	2.49	9%
G916-4	6	0.51	0.56	5%
Combined	**73**	**1.42**	**1.41**	**-1%**

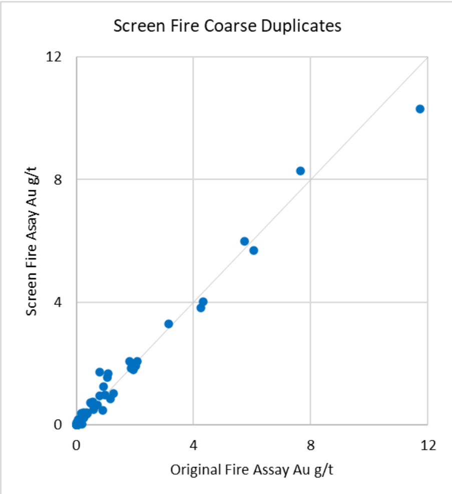
Alternative Method and Inter-laboratory Duplicate Assays

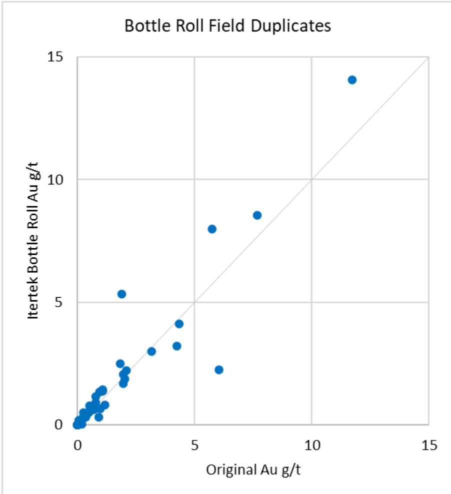
Information available to demonstrate the accuracy of primary Bureau Veritas gold fire assaying for samples from the 2019 reconnaissance RC drilling includes screen fire assays performed by Bureau Veritas on duplicate splits of coarse reject samples and field duplicate bottle roll analyses performed by Intertek, Ghana.

The summary statistics and scatter plot in Figure 11-8 demonstrate that although there is some scatter for individual pairs the screen fire and bottle roll duplicate assays correlate reasonably well with original results providing additional confidence in the accuracy of the primary Bureau Veritas fire assaying.

Figure 11-8 Alternative method and inter-laboratory duplicates for reconnaissance drilling

Au g/t	Bureau Veritas Screen Fire Coarse Reject Duplicates				Intertek Bottle Roll Field Duplicates			
	Full set		> Detection		Full set		<10 g/t	
	Orig.	Dup.	Orig.	Orig.	Orig.	Dup.	Orig.	Dup.
Number	92		57		46		45	
Average	**0.77**	**0.77**	**1.24**	**1.24**	**1.48**	**1.59**	**1.25**	**1.32**
Difference.		**0%**		**0%**		**8%**		**5%**
Variance	3.18	2.93	4.57	4.15	5.33	7.00	3.06	3.61
Coef. Variation.	2.32	2.22	1.73	1.64	1.56	1.66	1.40	1.44
Minimum	0.01	0.01	0.02	0.01	0.01	0.01	0.01	0.01
1st Quartile	0.01	0.01	0.08	0.04	0.13	0.17	0.11	0.16
Median	0.05	0.04	0.36	0.39	0.62	0.65	0.59	0.63
3rd Quartile	0.62	0.72	1.16	1.68	1.86	1.81	1.82	1.67
Maximum	11.74	10.30	11.74	10.30	11.74	14.08	7.66	8.56
Correl. Coef.	0.99		0.99		0.94		0.89	





11.7 Density Measurements

Bulk density measurements available for the Koné drilling include 4,125 immersion measurements performed by inhouse personnel on 10 to 15 cm lengths of diamond core which were oven dried for 24 hours at 1000C and wax coated to prevent water absorption. Densities were measured by the Archimedes method with allowance for the wax coating.

Table 11-10 summarizes the primary oven-dried wax coated immersion density measurements available for Koné coded by combined mineralized domain, oxidation zone and rock type wire-frames interpreted by Montage from drill hole logs. This table, which excludes three samples supplied with negative densities shows that for each oxidation zone average density measurements show little variability with rock type and mineralized domain.

Information available to demonstrate reliability of the in house density measurements includes 50 immersion measurements performed by SGS on core samples collected from around 10 cm deeper down-hole than each paired in house measurement. The SGS measurements test different material, and as expected there is some variability for individual pairs. However average SGS results closely match the inhouse measurements supporting the reliability of these data (Figure 11-9).

The author considers that the available density measurements provide an adequate basis for the current Indicated and Inferred Mineral Resources estimates.

Table 11-10 Bulk density measurements by oxidation and rock type

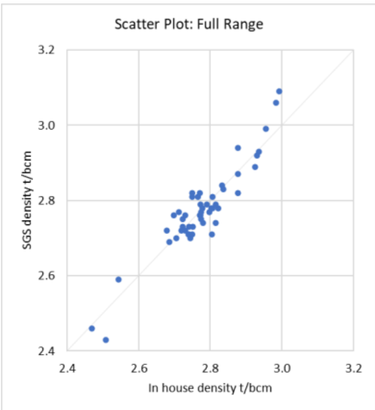
Oxidation Zone	Mineralization Zone	Rock Code	Density (t/bcm) Number	Minimum	Average	Maximum
Completely Oxidized	Background	CDI	42	1.19	1.66	2.21
		PGDI	48	1.28	1.63	2.47
		VC	275	1.15	1.67	2.62
		Subtotal	**365**	**1.15**	**1.66**	**2.62**
	Mineralized Envelope	CDI	110	1.16	1.66	2.56
		PGDI	2	1.56	1.80	2.03
		VC	6	1.56	1.76	1.99
		Subtotal	118	1.16	1.67	2.56
	Combined		**483**	**1.15**	**1.67**	**2.62**
Transition	Background	CDI	9	2.17	2.57	2.80
		PGDI	19	2.23	2.53	3.04
		VC	95	1.58	2.52	2.93
		Subtotal	**123**	**1.58**	**2.53**	**3.04**
	Mineralized Envelope	CDI	81	1.70	2.59	2.90
		PGDI	-	-	-	-
		VC	7	2.36	2.55	2.81
		Subtotal	**88**	**1.70**	**2.59**	**2.90**
	Combined		**211**	**1.58**	**2.55**	**3.04**
Fresh	Background	CDI	185	2.65	2.83	3.33
		PGDI	230	2.33	2.81	3.27
		VC	552	2.43	2.84	3.39
		Subtotal	**967**	**2.33**	**2.83**	**3.39**
	Mineralized Envelope	CDI	2,047	1.73	2.81	3.64
		PGDI	7	2.70	2.76	2.86
		VC	410	2.26	2.82	3.23
		Subtotal	**2,464**	**1.73**	**2.81**	**3.64**
	Total		**3,431**	**1.73**	**2.81**	**3.64**

Figure 11-9 SGS versus in house paired density measurements

	In house (t/bcm)	SGS (t/bcm)
Number	50	
Average	**2.77**	**2.78**
Variance	0.01	0.01
Coef. Variation.	0.04	0.04
Minimum	2.47	2.43
1st Quartile	2.73	2.72
Median	2.77	2.77
3rd Quartile	2.81	2.82
Maximum	2.99	3.09
Correl. Coef.	0.93	

By Oxidation Zone

	Number	Average (t/bcm) In house	Average (t/bcm) SGS	Difference
Oxide	1	2.47	2.46	-0.4%
Transition	2	2.53	2.51	-0.6%
Fresh	47	2.79	2.79	0.1%
Total	**50**	**2.77**	**2.78**	**0.1%**



12 DATA VERIFICATION

Verification checks undertaken by the author to confirm the validity of information for the RC and diamond drilling in the database compiled for the current study include the following:

- Checking for internal consistency between and within database tables
- Spot check comparisons between database entries and original field records
- Comparison of assay entries with laboratory source files
- Comparison of assay values between nearby holes and between different sampling phases.

These checks were undertaken using the working database compiled by the author and check both the validity of Montage's master database and potential data transfer errors in compilation of the working database.

The consistency checks showed no significant inconsistencies.

While visiting Montage's field office in Fadiadougou, the author compared original field records with database entries. These checks included 180 down hole survey table records and down hole depths and sample identifiers for 5,523 assay intervals representing approximately 25% and 33% of database entries respectively at that time. Relative to the drill holes informing the current estimates, these checks represent 1.4% and 6% of down hole survey and assay records respectively. These spot checks showed no significant inconsistencies.

For 99.5% of routine assays from RC and diamond drilling, and all 956 of the inter-laboratory repeat assays performed by SGS in 2021 the author compared database assay entries with gold grades in laboratory source files supplied by Montage (Table 12-1). These checks showed no inconsistencies.

The author considers that the resource data has been sufficiently verified to form the basis of the current Mineral Resource estimates and exploration activities, and that the database is adequate for the current estimates and exploration activities. The author considers that the data verification process included no limitations or failures.

Table 12-1 Database versus laboratory source file checks for RC and diamond samples

Period	Number of assays		Proportion
	In database	Checked	Checked
2010	925	925	100.0%
2013	1,766	1,766	100.0%
2017-18	13,878	13,878	100.0%
2019-20	22,812	22,812	100.0%
2021	49,765	49,307	99.1%
Combined	**89,146**	**88,688**	**99.5%**

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Testing 2014

The initial testwork was performed in 2014 by SGS Minerals Services UK Ltd, Cornwall and consisted of three cyanide bottle roll tests using D100 90 µm, 40% solids, 0.5 g/l NaCN concentration at pH 10.7 on three fresh composite RC drill chip samples.

This average leach extraction was 96.9% on an average 1.55 g/t Au head grade.

Reagent consumptions were low averaging 0.11 kg/t NaCN and 0.49kg/t CaO.

13.2 Metallurgical Testing 2018

In September 2018, ALS Global (ALS) in Perth Australia undertook a program of metallurgical testwork on four samples of diamond core from Koné, which were designated as the oxide, transition, fresh and FW fresh samples from the south pit.

Composites produced by ALS from the supplied core were subjected to tests including head assay determination, Bond ball mill work index (BWi) determination, grind establishment testwork, gravity-recoverable-gold (GRG) determination and cyanide leaching both with and without carbon to identify if there was any preg-robbing characteristics.

"Standard" non-optimised cyanide leach conditions were used as follows:

- 40% solids w/w
- pH 10.7
- 0.50 g/l NaCN concentration
- 48 hour residence time

The results are summarised in Table 13-1.

Table 13-1 Summary of 2018 Cyanide Leach Testwork Results

LEACHING TESTWORK: SUMMARY OF RESULTS						
Comp ID	Crush/Grind Size	Leach Duration (hrs)	Leach Type	Au Grades (g/t)		Au Extraction (%)
				Head	Tail	
Oxide	P_{100} 20mm	504	Coarse-crush IBR	1.49	0.06	96.5
	P_{100} 10mm			1.43	0.05	96.5
	P_{100} 5mm			1.10	0.05	95.5
	P_{100} 1mm			1.20	0.06	95.2
	P_{80} 75µm	48	Direct Leach	1.38	0.03	97.8
			CIL	1.31	0.04	97.3
			Gravity/Leach	1.15	0.04	97.0
Transition	P_{100} 20mm	504	Coarse-crush IBR	0.94	0.19	80.7
	P_{100} 10mm			1.28	0.31	76.1
	P_{100} 5mm			0.98	0.21	79.2
	P_{100} 1mm			0.98	0.11	88.9
	P_{80} 75µm	48	Direct Leach	1.71	0.06	96.5
			CIL	1.24	0.08	93.5
			Gravity/Leach	0.91	0.05	94.5
Fresh	P_{100} 20mm	504	Coarse-crush IBR	1.20	0.75	37.1
	P_{100} 10mm			1.06	0.52	51.2
	P_{100} 5mm			1.24	0.53	57.4
	P_{100} 1mm			0.87	0.19	78.7
	P_{80} 75µm	48	Direct Leach	1.04	0.09	91.4
			CIL	1.00	0.08	92.5
			Gravity/Leach	0.91	0.08	91.2
FW Fresh	P_{100} 10mm	504	Coarse-crush IBR	1.85	1.16	37.3
	P_{100} 5mm			1.86	0.89	51.9
	P_{80} 75µm	48	Direct Leach	1.81	0.22	87.9
			CIL	1.81	0.29	83.9

Summary highlights of this work are as follows:

- Heap leaching failed to achieve satisfactory gold dissolution on the predominant fresh ore type and so this process route was discounted
- No preg-robbing was observed and the levels of organic carbon were low, CIP was selected as the appropriate cyanide leach process
- Good gold leach extraction rates were achieved 91 - 92% from a ~1.0 g/t Au feed grade on the fresh composite. The FW fresh composite was lower at 88% and the cyanide consumption was higher due to the presence of additional copper sulphides in this ore type
- Unadjusted gravity recoveries were reasonable at 23% on the predominant fresh sample but no additional benefits were seen in terms of leach kinetics or terminal cyanide leach residue. As a consequence, gravity concentration was rejected from the future flowsheet development testwork

- The preliminary Bond Ball Mill Work indices were in line with subsequent multiple sample testing programs.

13.2 Metallurgical Testing 2020

The scope of work was designed to optimise the whole ore primary crush – SAB comminution circuit followed by CIP flowsheet, provide carbon modelling data and generate samples for third party thickener vendor testwork as well as TSF design testwork.

13.2.1 Variability Comminution & Physical Testing

A total of 43 samples were submitted for comminution testing and included SMC and Bond ball mill grindability testing on all samples, as well as Bond low-energy impact and Bond abrasion testing on eight samples.

This phase of testwork focussed on the Fresh domain which comprises 87% of total resource tonnage.

Table 13-2 Summary of 2020 Comminution testing

		Average				
Oxidation Zone	**No Samples**	**A x b**	**SCSE kWh/t**	**CWi kWh/t**	**BWi kWh/t**	**Ai g**
Fresh	39	30.0	11.5	15.8*	11.3	0.45*
Transition	4	107	6.9	8.5**	7.0	0.12**

* From seven samples
** From one sample

These results were used to support SAB comminution circuit equipment sizing.

13.2.2 Leach Conditions Optimization Testing

In total, 36 leach optimization tests were completed using the Fresh Master Composite Sample and the objective of the tests was to determine the optimal conditions.

Upon completion of the tests, the optimized leach conditions were as follows:

- Grind Size P80 target of 75 µm
- Pulp Density 50% solids (w/w)
- Pulp pH 10.5 - 10.7 (maintained with lime)
- 36 hour leach retention time
- Cyanide concentration of 0.5 g/l NaCN (maintained for 8 hours)
- Dissolved oxygen concentration of ~30 mg/l (tests sparged with oxygen)
- No pre-aeration

- No lead nitrate addition.

The leach residue assays from the Fresh composite sample, were consistently low (0.06 – 0.07 g/t Au) in all tests performed excluding test 1, 2, 3 & 4 at coarser grind sizes and test 29 at shorter (24hr) residence time.

Cyanide and lime consumptions were also consistently low at 0.18 kg/t and 0.25kg/t respectively. The introduction of oxygen sparging had a very beneficial effect on cyanide consumption rates, reducing it from circa 0.48 kg/t to 0.18 kg/t and in addition, lime consumption reduced from 0.35 kg/t to 0.25 kg/t. Oxygen is important in reducing cyanide consumption rates and to a lesser extent, lime consumption rates. Oxygen sparging was adopted for all subsequent variability testing.

Rheological testing in conjunction with comparison cyanide leach testing at different % solids confirmed that it is possible to increase pulp density on the fresh mineralization, to 55% solids. 50% solids was adopted as the standard pulp density to ensure a conservative approach.

The cyanide attenuation tests demonstrated that the dosing of cyanide will only be required in the first few tanks.

13.2.3 Leach Variability Testing

A total of 39 variability samples were processed using the optimized conditions detailed in the previous section.

The average back calculated head assay was 1.10 g/t Au which compares well with the measured head assay 1.10 g/t Au.

The leach kinetic results indicated that the samples continued to leach over the entire 36 hour leach retention time for the majority of the samples. The average cyanide and lime consumptions were all low for fresh mineralization at 0.19 kg/t NaCN and 0.22 kg/t CaO respectively. The low cyanide consumptions were consistent with all tests using oxygen sparging.

Figure 13-1 shows the relationship between g/t Au in feed and residue for each domain.

The previous PEA testwork results conducted in 2018 are consistent with this subsequent 2020 phase of work.

Figure 13-1 Relationship between g/t Au in Feed and g/t Au in Residues by Domain

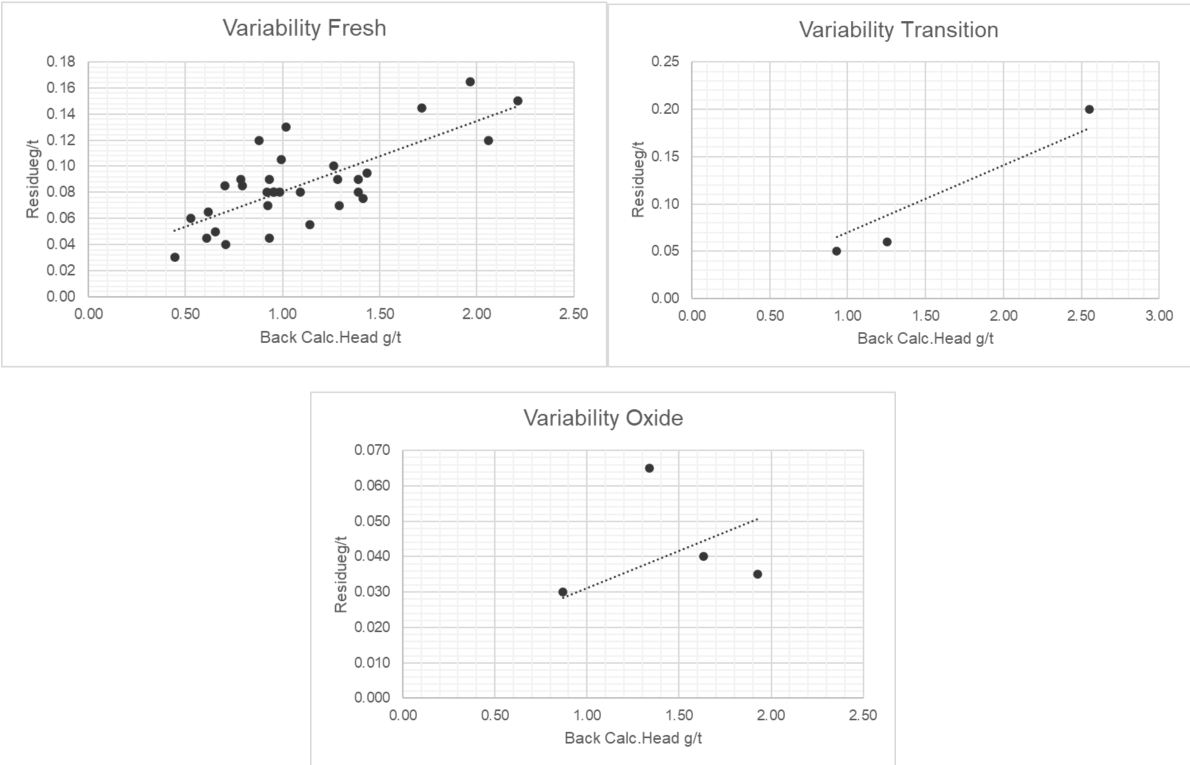


Figure 13-2 indicates that there is a linear relationship between feed sulphur and residue grade. This was consistent across all samples tested.

Figure 13-2 Fresh Samples Head Sulphur and Residue Gold Grade

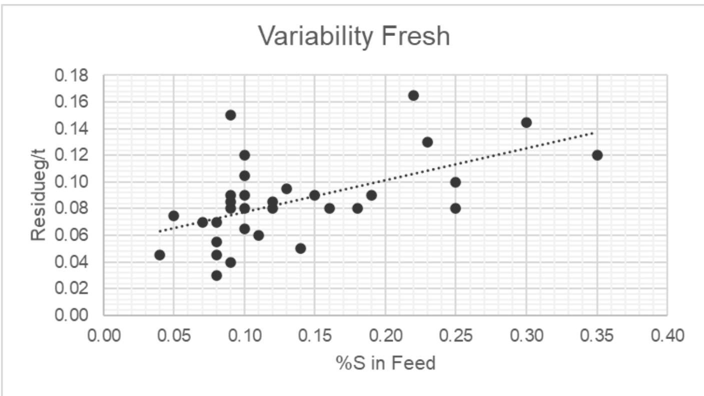
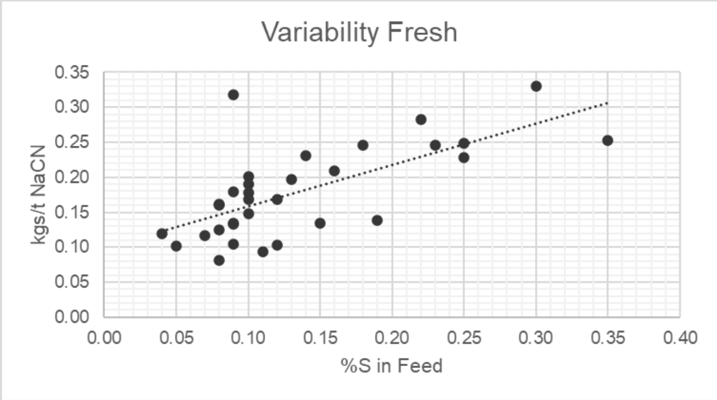


Figure 13-3 indicates that there is a linear relationship between feed sulphur and cyanide consumption. This was consistent across all samples tested.

Figure 13-3 Fresh Samples Head Sulphur and Cyanide Consumption



13.2.4 Carbon Modelling

Full Carbon Modelling testwork was performed on the Fresh Composite sample and a 60% Fresh - 40% Oxide blend sample to determine whether the oxide impeded carbon loadings. Leach & adsorption kinetics were also conducted on an 80% Fresh - 20% Oxide Blend sample.

The kK constants are summarized below in Table 13-3.

Both mineralization types show moderate adsorption kinetics.

Table 13-3 Carbon Adsorption Constants

Constant	Fresh Comp	Fresh 80% Oxide 20% Comp
Kinetic (k) h-1	0.0018	0.0093
Equilibrium (K) g/t	27,696	8196
Product (kK)	50	76

Barren solution assays of <0.010 mg/l are achievable.

13.2.5 Tailings Sample Generation

Three samples were generated for Geotechnical and Environmental testing by Knight Piésold. The samples were generated using the Fresh, Transition and Oxide Composites.

13.2.6 Thickener Tests

Twenty kilograms of the Fresh, Transition, Oxide and 20% & 30% Oxide Blend Composite samples were also delivered to Outotec for a full suite of S/L Separation tests. The samples were ground to the target grind size P80, discharged from the mills and provided as a slurry.

The results are summarized in Table 13-4

Table 13-4 Dynamic Thickening Testwork Results

<table>
<tr><th rowspan="2">Sample</th><th colspan="2">Feed</th><th colspan="2">Flocculant</th><th colspan="2">Underflow</th><th>Overflow</th></tr>
<tr><th>Flux (t/(m²h))</th><th>Liquor RR (m/h)</th><th>Type</th><th>Dose (g/t)</th><th>Meas. Solids (% (w/w))</th><th>YS (Pa)</th><th>Solids (mg/l)</th></tr>
<tr><td>Fresh Composite</td><td>0.80</td><td>3.31</td><td>910 VHM</td><td>30</td><td>65.2</td><td>11</td><td><100</td></tr>
<tr><td>Transition Composite</td><td>0.80</td><td>3.50</td><td>923 SH</td><td>40</td><td>57.7</td><td>28</td><td>280</td></tr>
<tr><td>Oxide Composite</td><td>0.70</td><td>8.32</td><td rowspan="4">945 VHM</td><td>30</td><td>36.2</td><td>3</td><td>110</td></tr>
<tr><td>60% Fresh 40% Oxide</td><td>0.80</td><td>5.11</td><td>60</td><td>50.9</td><td>31</td><td>1162</td></tr>
<tr><td>70% Fresh 30% Oxide</td><td>1.09</td><td>6.27</td><td>60</td><td>50.3</td><td>16</td><td>827</td></tr>
<tr><td>80% Fresh 20% Oxide</td><td>1.09</td><td>5.59</td><td>60</td><td>52.1</td><td>15</td><td>410</td></tr>
</table>

The testwork shows that the fresh & transition composite sample can be thickened to well in excess of the planned leach circuit feed % solids (50%). However, the 30% and 40% oxide blend samples produced underflow densities of around 50% solids. The 20% oxide blend sample produced underflow density of 52% solids and on this oxide ore blend, the testwork concluded that flocculant would to be supplemented by circa 35g/t coagulant to control thickener overflow clarity.

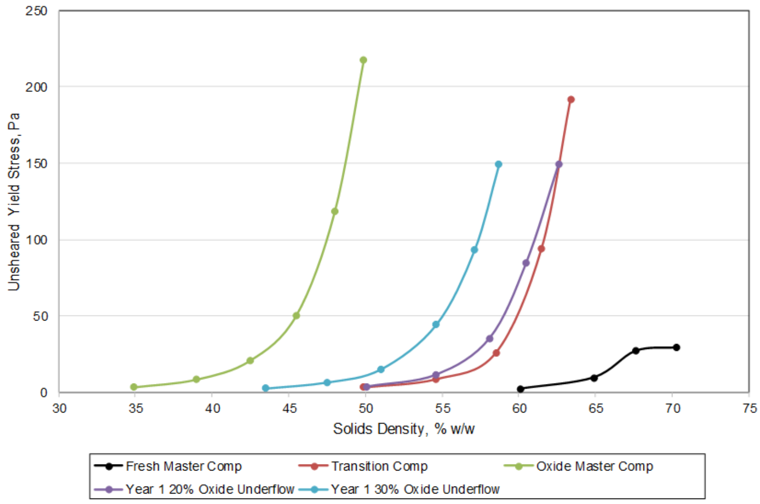
13.2.7 Rheological Tests

Five samples were submitted for rheology testwork. The three mineralization zone master composites (Fresh, Transition, and Oxide Comps) were initially tested. The rheology tests were completed on "pre-leach" samples that were ground to the target grind size P80 and processed as-is. The samples were not thickener underflow samples.

Two thickener underflow samples were also submitted for rheology testwork. These samples were referred to as "80% Fresh 20% Oxide Underflow" and "70% Fresh 30% Oxide Underflow". These samples were products from dynamic thickener tests conducted.

Figure 13-4 summarizes the relationship between unsheared yield stress and pulp density for the different samples tested. The fresh composite showed the most favourable flow behaviour, followed by transition and the worst behaviour was observed in the oxide blends.

Figure 13-4 Thickener Test Yield Stress Curve



The thixotropic nature of the oxide mineralization requires that the oxides are blended to <20% to maintain 50% solids feeding the leach circuit and avoid any adverse affects associated with carbon adsorption kinetics.

13.3 Metallurgical Testing 2021

The scope of work was designed to advance the flowsheet development and optimization to DFS level and was designed to deal with the 2020 Metallurgical testing recommendations as follows:

- Additional comminution testing to increase the variability sampling particularly on oxide mineralization
- Extend variability testing, specifically on low grade samples to verify metallurgical response at the lower grades at site ambient temperature and design DO conditions
- Further metallurgical testing to re-evaluate whether introduction of a gravity stage has a beneficial effect on metallurgy.

13.3.1 Variability Comminution & Physical Testing

The sample selection for this phase of work focused on increasing the variability study data base for the predominant fresh ore type whilst specifically generating more data for oxide (Saprolite and Saprock) mineralisations. Testwork focused on increasing the variability study data base for the predominant fresh ore type whilst specifically generating more data for oxide (Saprolite and Saprock) mineralisations.

A total of 69 SMC tests were conducted over all studies (2018, 2020 and 2021) and the total data base was used to support the DFS comminution circuit design.

The SMC results of all studies are summarized in Table 13-5.

Table 13-5 Summary of All SMC Test Results

Domain	# Sample	A	b	A x b	Hardness Percentile	t_a[1]	DWI (kWh/m^3)	DWi Percentile	M_{ia} (kWh/t)	M_{ih} (kWh/t)	M_{ic} (kWh/t)	SCSE (kWh/t)	Relative Density
Fresh	50	88.3	0.36	31	82	0.3	8.98	77	24.2	19.0	9.8	11.32	2.74
FW Fresh	3	79.5	0.39	31	82	0.3	9.01	78	24.0	18.8	9.7	11.35	2.77
Trans	9	71.4	1.10	77	29	0.7	4.24	24	13.3	9.1	4.7	8.04	2.69
Saprolite	3	99.8	21.34	2131	0	10.0	0.08		1.0	0.3	0.2		1.57
Saprock	4	89.2	5.40	489	0	5.0	0.57	1	2.9	1.3	0.7	5.14	2.54

On average, the fresh and FW fresh results were characterized as moderately hard with respect to resistance to impact breakage. The three oxide saprolite samples tested resulted in extremely soft A x b's, and were not used in the comminution design. The four oxide saprock samples tested were classified as very soft, with A x b's ranging from 308 to 690, and all fell within the 1st percentile of hardness of JKTech's database. These values should also be used with caution.

The Bond Low Energy Impact test results from all studies are presented in Table 13-6. The samples fell in the hard to very hard range of hardness of the SGS database.

Table 13-6 Summary of All Bond Low Energy Impact Test Results

Domain	# Sample	Average (kWh/t)	Min (kWh/t)	Max (kWh/t)	Std Dev (kWh/t)	Relative Density	Hardness Percentile
Fresh	11	17.0	7.7	31.8	6.2	2.75	87
Trans	1	8.5	3.3	15.2	2.6	2.67	37

A total of 67 Bond Ball Mill Work Index tests were performed using a closed screen size of 106 µm and these results for all studies are summarized in Table 13-7.

On average, the fresh samples were characterised as being soft range of hardness, whilst the transition and oxides were very soft.

Due to the naturally occurring very high fines content of the four oxide (saprock) samples, a modified test procedure was applied to those samples. The calculated overall work index takes into account the amount of fine material that was removed prior to the test. The test results averaged 3.9 kWh/t, so oxide ores can be characterised as being very soft with a hardness percentile of <1.

Table 13-7 Summary of All Bond Ball Mill Work Index Test Results

Domain	# Sample	Mesh of Grind	F80 (mm)	P80 (mm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile	Feed passing (%)	Bulk Density	Closing Screen Size (mm)	Mib (kWh/t)
Fresh	50	150	2,374	86	1.9	11.6	23.4	13.8	1,860	106	14.6
FW Fresh	4	150	2,324	86	2.3	9.7	10.2	16.3	1,896	106	11.8
Trans	9	150	2,271	87	3.1	7.8	3.9	18.2	1,806	106	9.0
Oxide	4	150	2,036	93	4.4	3.9					

A total of 17 samples were subjected to the Bond Abrasion Index during all studies. The test results are summarized in Table 13-8. The Ai values ranged from 0.120 g to 1.040 g, meaning that the samples ranged all the way from very mild to very abrasive when placed in the SGS database.

Table 13-8 Summary of All Abrasion Work Index

Domain	# Sample	AI (g)	Percentile of Abrasivity
Fresh	10	0.442	69
FW Fresh	0		
Trans	4	0.152	34
Oxide	3	0.115	21

13.3.2 XRD Clay Speciation

In total, 11 oxide samples (6 saprolite and 5 saprock samples) were submitted for XRD analysis. The main clay minerals include montmorillonite and kaolinite.

13.3.3 Gravity Concentration

Gravity separation was investigated in more detail than the 2018 work, to determine if it would have a positive impact on the overall gold recovery and project economics

The test program was focused upon fresh composite samples and comprised of the following:

- 4 E-GRG tests
- 4 Bulk gravity separation and cyanidation tests (concentrate and tail leaches)
- variability gravity separation and cyanidation tests.

Table 13-9 summarises the E-GRG testwork results. The GRG grain size considering the typical lower gold grades in the fresh ore exhibit fine to very fine gold grain size on the AMIRA GRG scale and is not efficiently recovered by gravity separation. FLSmidth Knelson forecast gold recoveries of ~15% using four KC – QS48.

The FLSmidth Knelson test results do not support installation of a gravity circuit unless the ore had significant preg-robbing properties, which it does not.

Table 13-9 Summary of E-GRG Testwork

Summary of E-GRG Testwork				
Sample No	2	3	4	5
g/t Au	0.89	1.76	0.64	1.00
E-GRG	50	61	42	49
Av GRG Size, ųms	52	70	50	46
FLS GRG	17	25	14	16

13.3.4 Cyanide leach tailings diagnostic testwork

Following the completion of the 2020 PEA stage testwork, six cyanide leach residues were selected for QEMScan bulk mineralogy investigation.

The Gravity Amenability Test "GAT" was conducted on a 3.7 kg composite made up of all six samples. 35% of the gold was recovered into a 6% weight yield low grade concentrate (~1 g/t Au) and this concentrate was upgraded using a Mozley panner to produce a 0.07% weight yield gravity concentrate assaying 21.4 g/t Au and containing 9% (~1% overall) of the gold in tailings.

The diagnostic leach results showed that the majority of the gold is locked in sulphides or in silicates and/or associated with fine sulphide particles which are locked within larger silicate particles.

The QEMScan mineralogy showed that the major sulphide mineral is pyrite with minor Chalcopyrite. The grain size of the pyrite ranges between 18-46 microns, whereas the chalcopyrite was more consistently finer ~20 microns.

The Pyrite grains ranged between 89-98% free and liberated, the remaining particles occur as complex particles associated with host rock minerals.

The Chalcopyrite grains ranged between 66-99% free and liberated, the remaining particles occur as complex particles associated with host rock minerals.

13.3.5 Cyanide Leach Variability Testwork

The objective of this testwork was to further define gold leach extraction and reagent consumptions using new lower gold grade samples that were in line with the lower DFS resource grades.

A total of 130 samples (31 PEA & 99 DFS) were tested using the optimised conditions as follows:

- 1 kg standard bottle roll tests
- 35-40oC pulp temperature
- Grind Size P80 target of 75 µm
- Pulp Density 50% solids (w/w)
- Pulp pH 10.5-10.7 (maintained with lime)
- 36 hour leach retention time
- Initial cyanide concentration dosage of 0.5 g/l NaCN and then 0.4 g/l NaCN maintained for the first 8 hours of the test. After 8 hours, the cyanide concentration decayed until the end of the test
- Dissolved oxygen concentration of ~20 mg/l (tests sparged with oxygen)
- No pre-aeration
- No lead nitrate addition.

Upon completion of each test, the pulp was filtered and the pregnant leach solution (PLS) was submitted for analysis (Au, CNS and a multi-element ICP Scan). The solids were washed and submitted for Au assay (duplicate 30 grams cuts – fire assay to extinction). A subsample of the final residue was also submitted for sieve analysis. Solution (kinetic) subsamples were taken for gold assays on all tests (2, 4, 8, 12, 24 and 32 hours).

Table 13-10 summarise the 130 cyanide leach variability test results conducted to support the DFS study.

Table 13-10 Summary of all 130 DFS Sample Results

Sample	CN Test No.	Feed Size P_{80}, µm	Average DO ppm	Reagent Cons. kg/t of CN Feed		Au Extraction, %							Avg. g/t Au Residue	*ppm Cu Act Meas*	*%S Head Meas*	Back Calc Au , g/t	Direct g/t Au
				NaCN	CaO	2 h	4 h	8 h	12 h	24 h	32 h	36 h					
SOUTH HW FRESH	min	62	17	0.12	0.34	30.9	43.3	60.2	71.6	81.8	85.1	86.7	0.05	39	0.04	0.61	0.63
SAMPLES	max	78	27	0.39	0.81	62.4	77.2	87.8	91.0	93.9	95.9	95.4	0.19	516	0.30	2.14	1.33
53	avg.	70	20	0.26	0.55	43.2	58.0	71.8	79.1	89.4	90.3	91.5	0.09	230	0.14	1.09	0.97
NORTH HW FRESH	min	65	18	0.17	0.31	50.7	56.4	62.5	65.3	69.4	70.7	70.8	0.06	75	0.38	0.25	0.23
SAMPLES	max	76	26	0.61	0.52	75.0	78.6	81.4	82.5	87.2	87.8	87.8	0.23	2100	1.96	1.84	1.76
12	avg.	70	20	0.37	0.43	59.6	66.2	71.8	74.2	77.1	78.3	79.2	0.13	509	0.90	0.71	0.64
SOUTH FW FRESH	min	68	16	0.12	0.26	26.9	26.7	54.9	63.3	74.8	79.4	81.4	0.04	132	0.03	0.40	0.41
SAMPLES	max	78	23	0.59	1.10	64.7	72.3	79.8	85.6	94.2	97.7	95.6	0.30	949	0.44	1.62	2.08
13	avg.	72	19	0.23	0.45	44.4	54.6	69.1	75.8	85.2	87.7	88.7	0.10	393	0.12	0.88	0.83
SOUTH TRANS	min	54	19	0.06	0.43	11.8	25.1	45.2	57.7	77.5	81.9	82.6	0.02	16	0.01	0.26	0.25
SAMPLES	max	78	24	0.92	2.18	86.5	92.6	95.9	94.7	98.4	98.3	97.9	0.29	1050	0.17	2.73	1.40
17	avg.	69	21	0.18	0.99	40.7	54.4	71.2	79.4	89.2	91.5	92.4	0.06	249	0.03	0.90	0.76
NORTH TRANS	min	64	19	0.11	0.38	23.8	35.9	49.0	57.6	84.4	86.5	85.0	0.04	180	0.02	0.30	0.33
SAMPLES	max	69	25	0.76	1.34	64.0	73.1	84.1	87.3	90.5	91.5	91.7	0.14	1070	0.43	1.25	1.10
5	avg.	68	22	0.35	0.75	47.6	57.8	68.7	76.1	87.0	88.8	89.1	0.08	604	0.14	0.77	0.68
SOUTH OXIDE	min	59	16	0.08	1.61	19.8	44.8	79.7	79.7	88.4	89.5	90.7	0.02	45	0.01	0.25	0.25
SAMPLES	max	87	22	0.63	4.48	93.4	97.2	99.0	99.0	99.2	99.8	98.2	0.07	1360	0.01	1.99	1.18
21	avg.	69	19	0.18	2.50	69.4	81.9	90.8	92.2	95.0	95.1	95.6	0.03	327	0.01	0.88	0.65
NORTH OXIDE	min	61	18	0.09	1.85	78.9	84.5	81.6	87.5	90.3	91.8	91.7	0.02	191	0.01	0.32	0.38
SAMPLES	max	71	22	0.18	6.16	94.3	96.1	97.0	99.0	99.0	99.9	97.5	0.06	591	0.02	0.80	0.72
9	avg.	66	20	0.13	2.79	87.8	92.4	93.2	95.2	96.1	95.5	95.1	0.03	367	0.01	0.59	0.54

The average gold concentration in leach residues achieved was good averaging 0.08 g/t Au on the predominant fresh ore domain which represents 63.1% of gold ounces and 0.10 g/t Au on the FW Fresh domain which represents 25.2% of gold ounces. The smaller transition and oxide ore domains representing 4.7% and 7.1% of gold ounces respectively have even better leach extraction efficiencies and the average leach residue g/t Au were 0.06 g/t Au and 0.03 g/t Au respectively.

Figure 13-5, Figure 13-6 and Figure 13-7 show the relationships between gold concentration in residue and feed that was used to adjust final residue assays based upon resource feed gold grade in the financial modelling for each domain.

Figure 13-5 Relationship between g/t Au in Residues and g/t Au in Feed for Fresh Domains

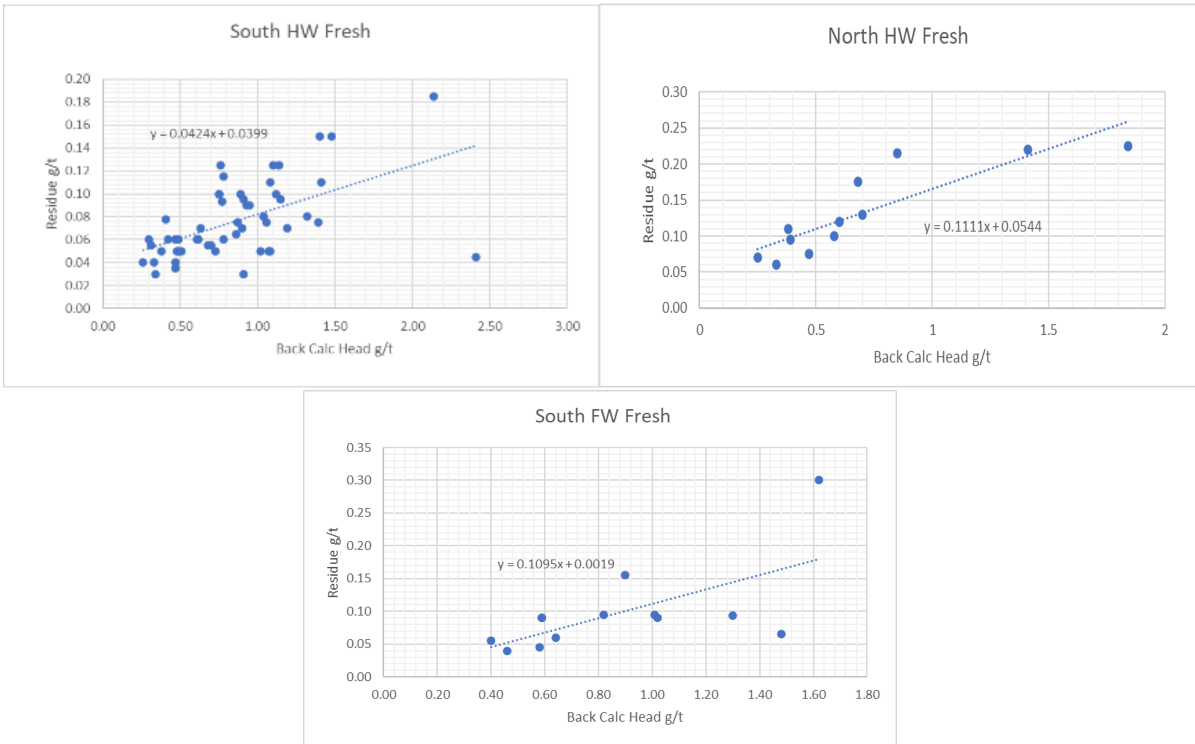


Figure 13-6 Relationship between g/t Au in Residues and g/t Au in Feed for Transition Domains

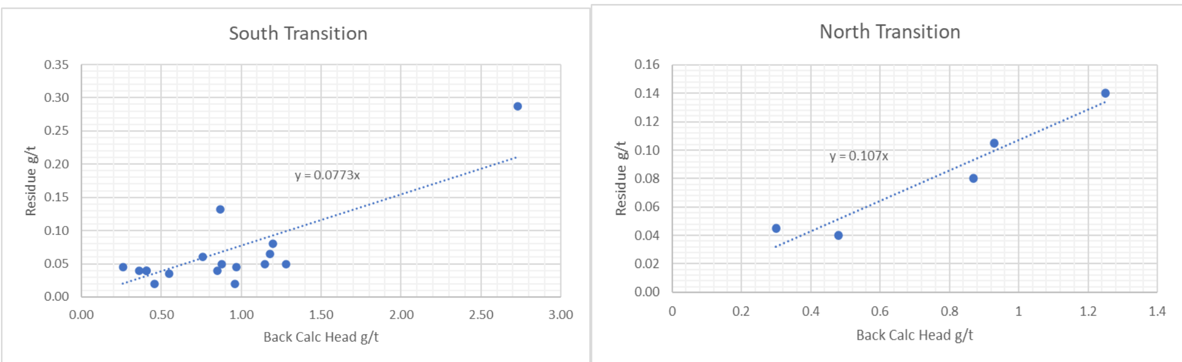


Figure 13-7 Relationship between g/t Au in Residues and g/t Au in Feed for Oxide Domains

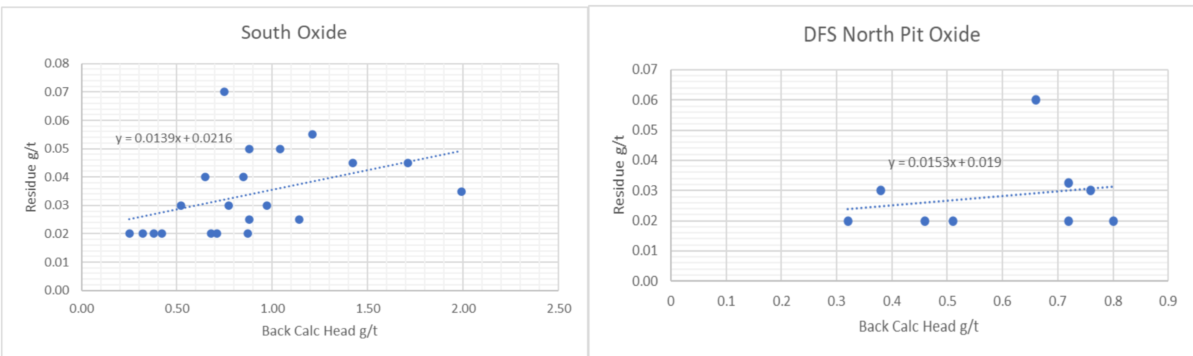


Figure 13-8 shows the gold leach extraction kinetics for each of the ore domains. The results indicated that oxide samples had the fastest leach kinetics and leaching was essentially complete after ~24 hours. The Fresh, Fresh FW and Transition test results indicated that the samples continued to leach over the entire 36 hour leach retention time. North HW Fresh is the worst performer due to the elevated sulphides which were found to encapsulate the gold.

Figure 13-8 Gold Leach Kinetics

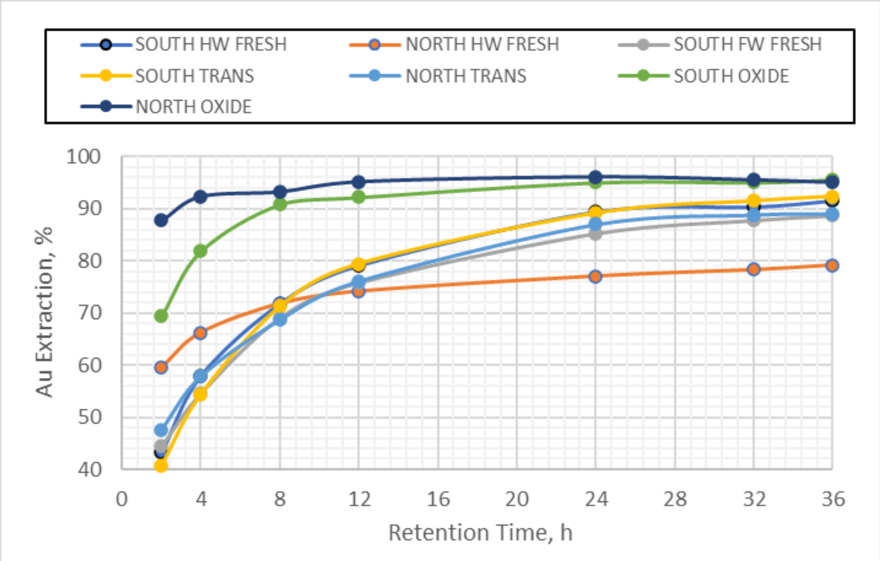


Figure 13-9 show cyanide consumption rates were generally all low averaging 0.25 kg/t based upon domain tonnage weighted average for all samples tested.

North Fresh & Transition domains were found to consume more cyanide due to the localised higher copper levels in the North zone.

Figure 13-9 Summary of Sodium Cyanide Consumptions

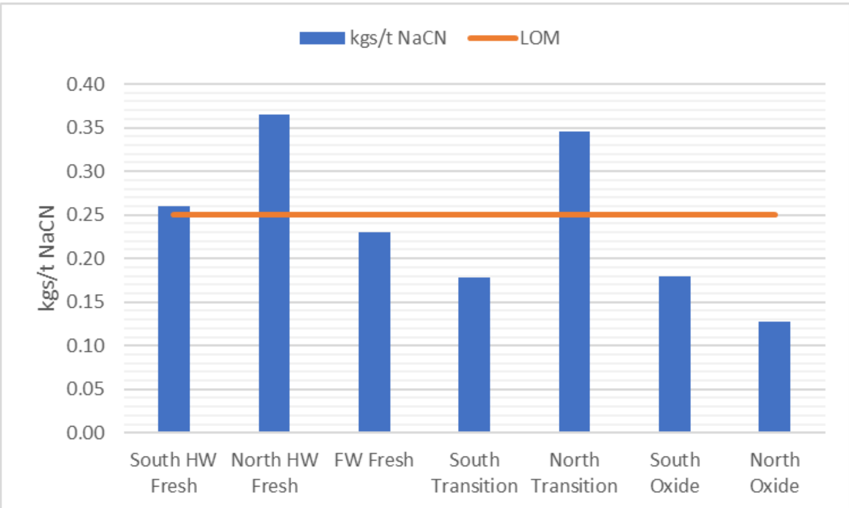
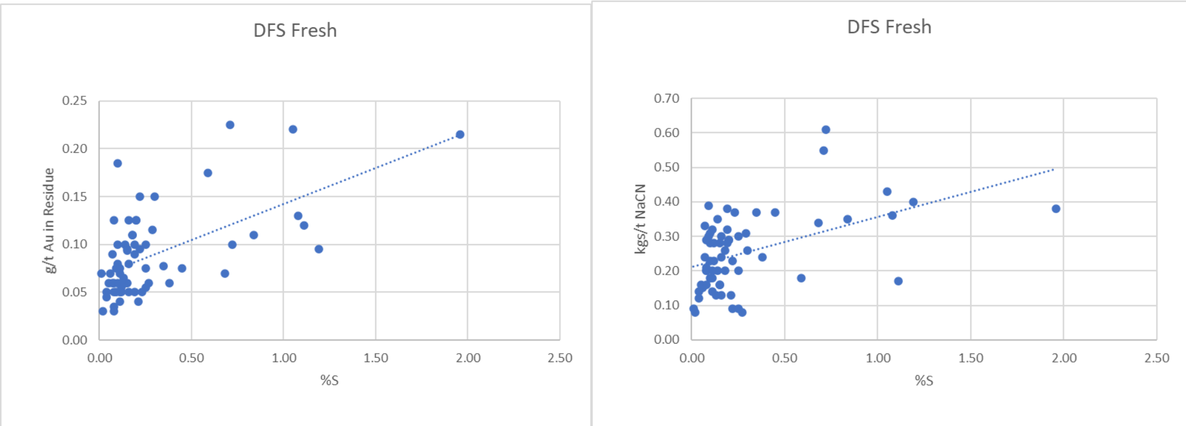


Figure 13-10 shows the relationship between %Sulphur and cyanide leach residue and sodium cyanide consumption for all fresh samples tested.

Figure 13-10 Effect of %S on Cyanide leach Residue and kg/t NaCN consumption



As highlighted above in the diagnostic testwork conducted on PEA cyanide leach residues, a small portion of the gold is encapsulated by sulphides.

Figure 13-11 provides a breakdown of the reported sodium cyanide consumption by species.

For the predominant fresh ore domains, the cyanide consumption is 30 to 40% due to Cu in solution, 20 to 30% due to Fe in solution, 10% due to thiosulphate and the remainder is not accounted for in solution and is deemed to have been lost as cyanates breakdown to form ammonia gas.

Sodium cyanide consumption for the transition ore domains appear to be much more associated with copper in solution whereas for oxides a lot of the consumption is due to hydrolysis of cyanate products since it is unaccounted for in pregnant solutions. The higher pulp temperature is likely to increase gaseous product losses.

Localised minor quantities of copper in the ore clearly result in elevated sodium cyanide consumptions.

Figure 13-11 Cyanide Consumption Breakdown by Species

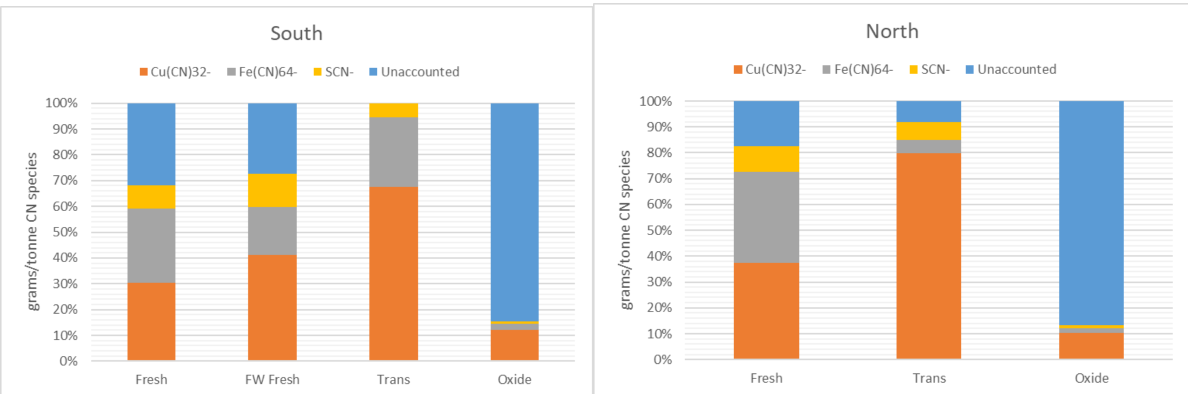
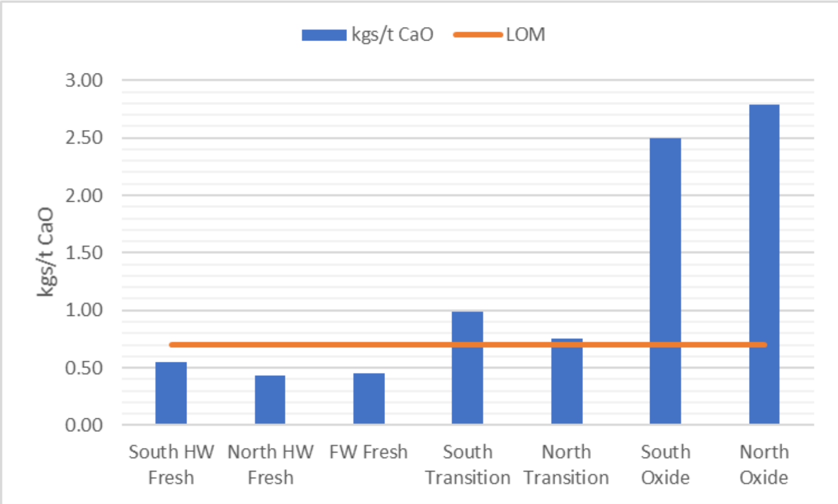


Figure 13-12 summarises lime consumption by domain. The two oxide domains were found to consume significantly more lime than the transition and fresh domains, but their impact on overall LOM consumptions is low because of the relatively tonnages in these domains.

Figure 13-12 Summary of Lime Consumptions



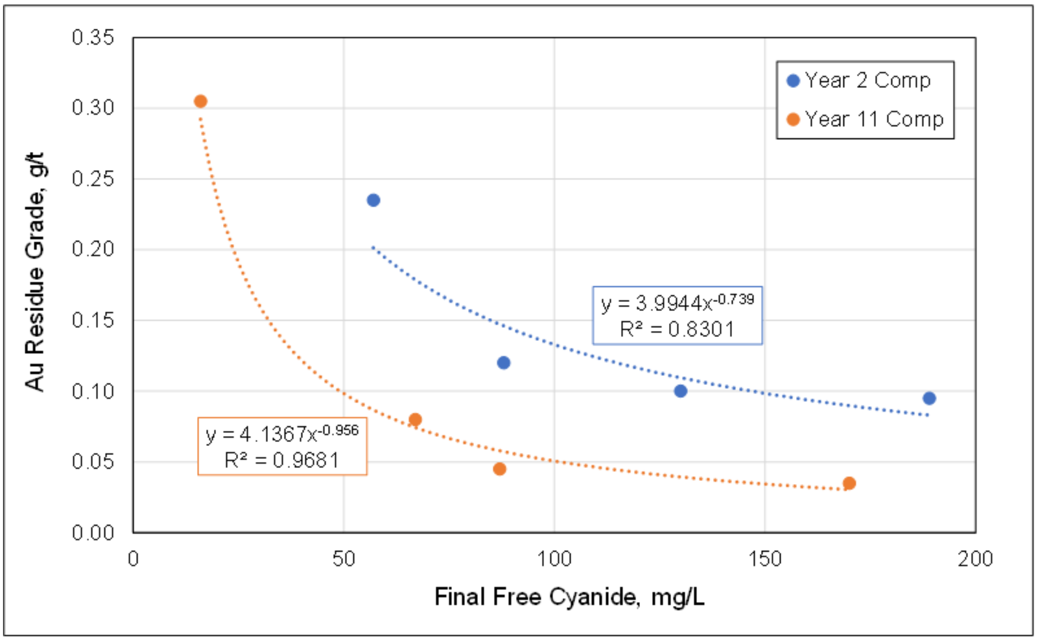
13.3.6 Sodium Cyanide Decay Testing

Two composite samples representing Year 2 (High Fresh blend) and Year 11 (High Transition blend) were produced to conduct sodium cyanide decay testing at 0.50, 0.42, 0.33 and 0.25 g/l free cyanide concentrations. The tests were set up with varying cyanide additions for the first 8 hours (0.25-0.5 g/l NaCN) and then no additional cyanide was added for the remainder of the test periods. The objective was to determine the free cyanide decay during the 36 hour tests and evaluate the impact on cyanide consumption as well as the final gold extractions and solution analyses. The remaining test conditions were identical to the variability tests.

Figure 13-13 shows that ~130ppm NaCN free cyanide in the final leach tank is the optimal free cyanide concentration for both the Year 2 consisting of predominantly fresh ore and the Year 11 containing a higher proportion of transition ore.

Detailed chemical analyses were performed and used for environmental modelling.

Figure 13-13 Relationship between Free Cyanide and g/t Au in residues



13.3.7 Carbon Modelling

Additional modelling (leach kinetic and adsorption kinetic) tests were completed using the Year 2 and Year 11 Composite samples and the results were compared to the Fresh Composite (from the previous modelling test results, 17236-01). The objective of the testwork was to see how the plant might respond to changes in the feed to the leach and CIP plants as the mine progresses through the different ore zones.

The results provided re-assurance that if the number of stages of adsorption is increased from six to eight, then solution assays in tailings of <0.005 mg/l are achievable.

kK constants of 84 and 90 were obtained for the Year 2 ad Year 11 composite respectively.

13.3.8 Thickener Tests

Two new composite samples were blended together to produce 20 kg solids as a pulp for further third party vendor thickener testing. These new blends consisted of 90% fresh and 10 % oxide, in one sample the 10% oxide was saprolite and in the other sample was saprock.

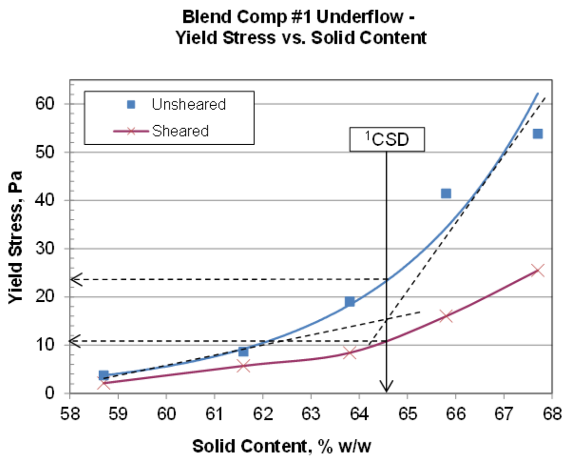
The objective was to confirm if coagulant is required in the design with the lower quantities (10% max) of oxide in the typical plant feed blend.

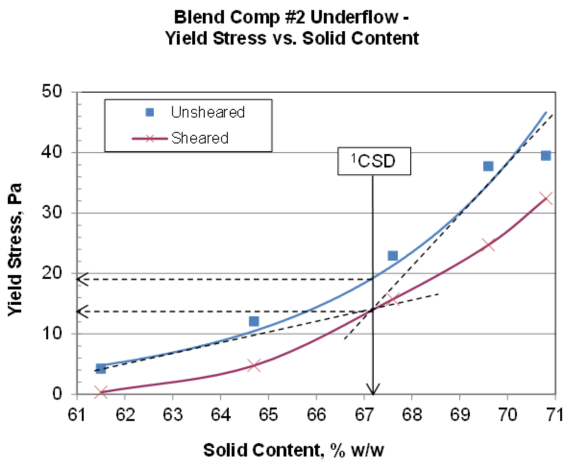
The results show that adding 5g/t coagulant produced less than 200ppm supernatant overflow clarity. At the existing thickener design flux rate of 1.09 tph/m^2, thickener underflow densities of 59% and 62.5% were achieved with the saprolite and saprock blended samples respectively.

13.3.9 Rheology

Figure 13-14 shows the relationship between Yield Stress (Pa) and Solids Content for Blended Composite No 1 (10% Saprolite : 90% Fresh) and Blended Composite No 2 (10% Saprock : 90% Fresh). The critical solids density was found to be much higher than the operating %solids for leaching (50% solids w/w) and the stress yield values at this operating % solids were low which provides some assurance that carbon adsorption will not be adversely affected by the planned design % solids range.

Figure 13-14 Yield Stress Vs Solids Content 10% Oxides





13.4 Metallurgical Results Summary

13.4.1 Metallurgical Sample Locations

The location of the comminution samples from all testwork campaigns are shown in Figure 13-15 and Figure 13-16.

Figure 13-15 Comminution Sample Locations – Plan

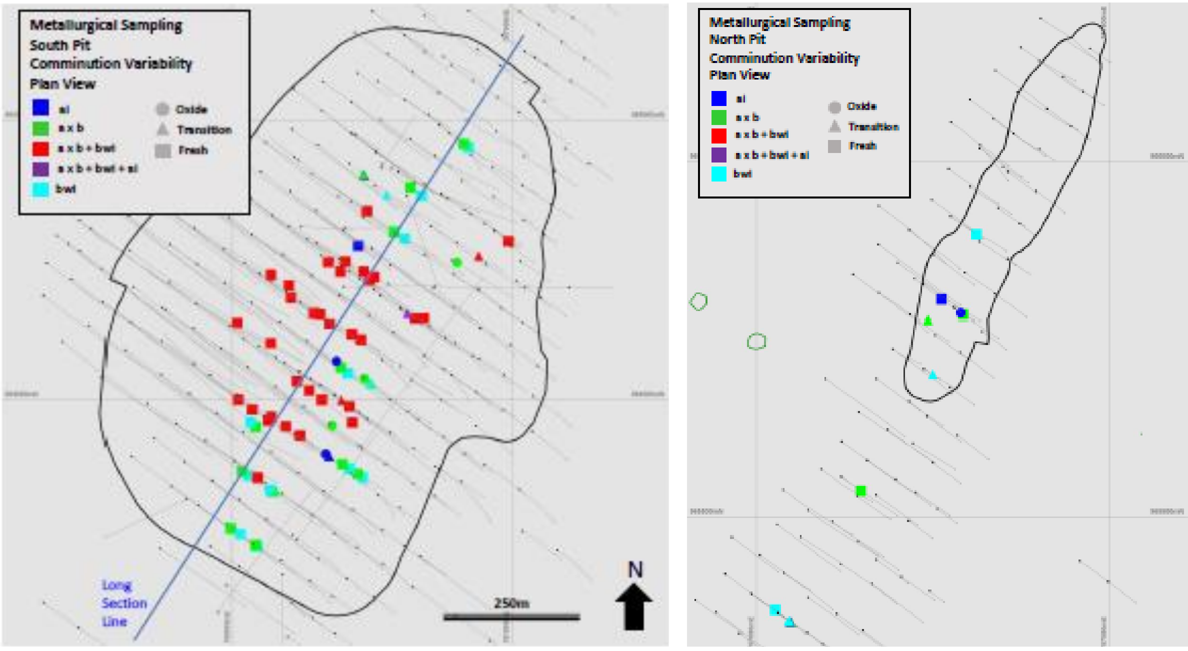
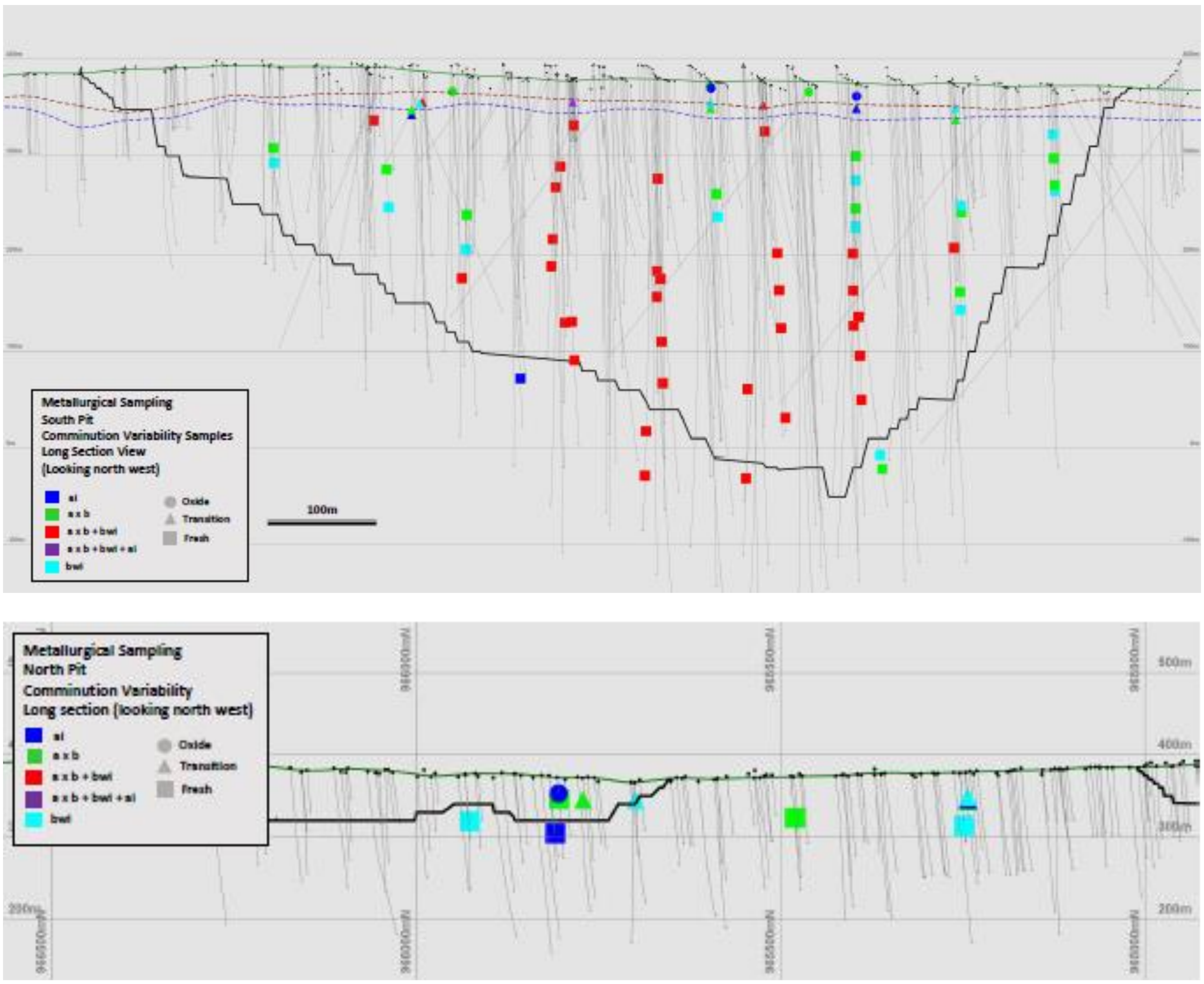


Figure 13-16 Comminution Sample Locations – Section



The location of the leach variability samples from all testwork campaigns are shown in Figure 13-17 and Figure 13-18.

Figure 13-17 Leach Variability Sample Locations – Plan

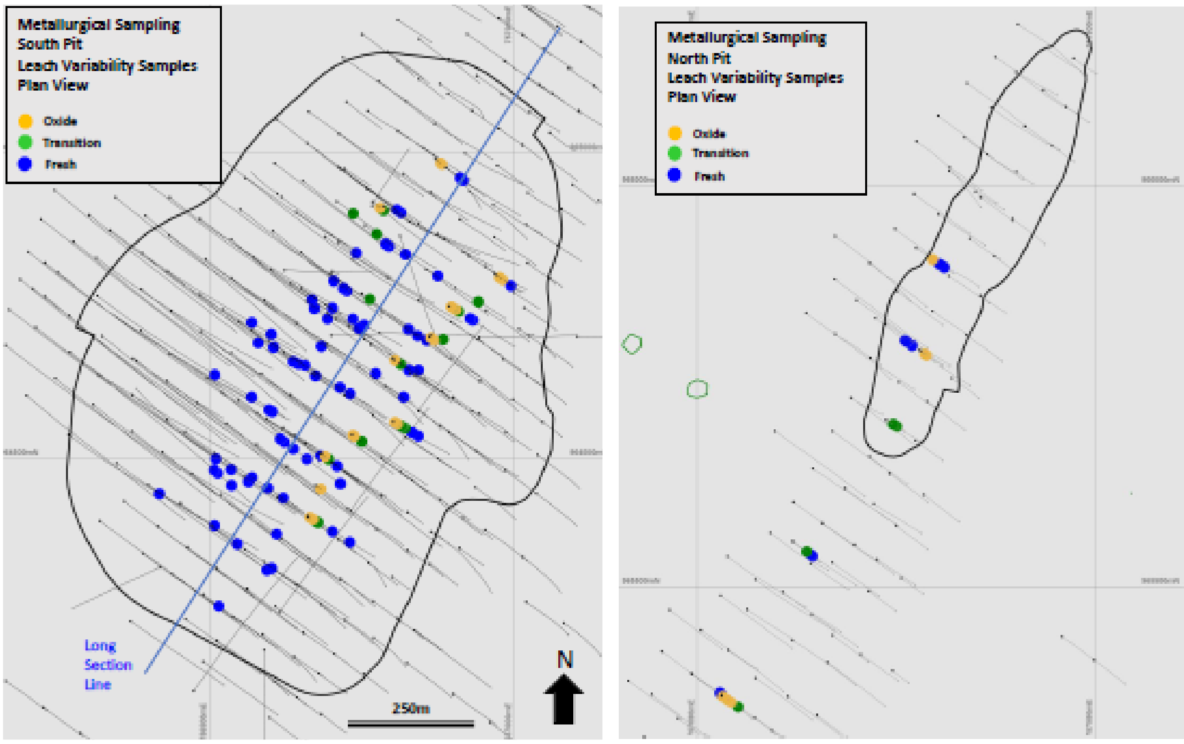
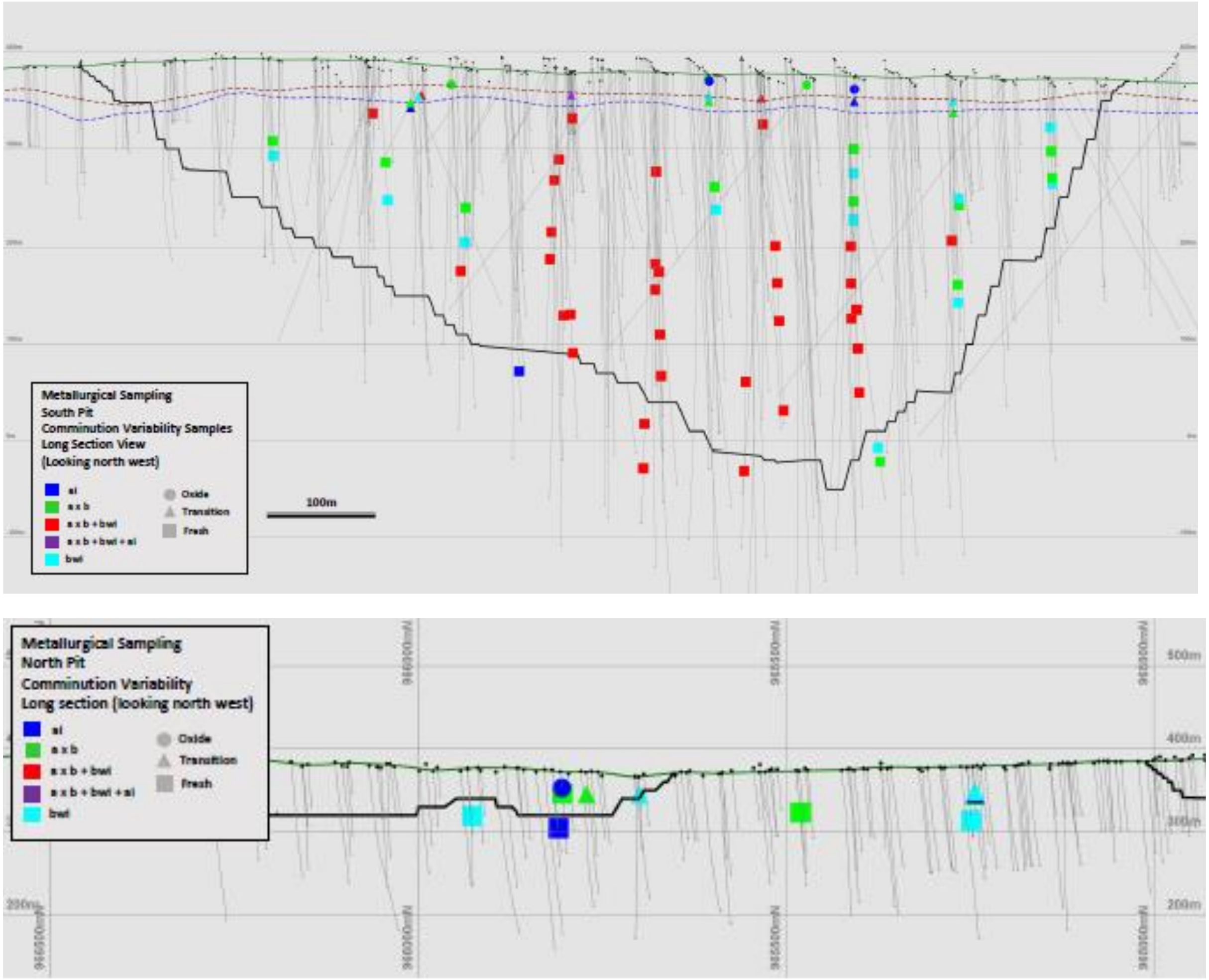


Figure 13-18 Leach Variability Sample Locations – Section



13.4.2 Comminution

Table 13-11 summarises the comminution testwork that has been conducted in all (2018, 2020 and 2021) studies.

A total of 65 JK Tech SMC, 67 Bond Ball Mill Work Index, 17 Abrasion Work Index and 12 Low Energy Impact Tests (i.e. Crusher Work Index) have been performed and were used to support the grinding circuit design. The four extremely soft SMC test results on oxide samples were not included due to them being below the JK Tech range of data used.

The predominant fresh mineralization zone is moderately hard in terms of resistance to SAG milling and crushing but soft in terms of resistance to ball milling and has medium abrasivity.

Table 13-11 Comminution Testwork

		JK Tech SMC A x b			Ballmill Work Index		Abrasion Index		Crusher Work Index	
Ore Type	**Deposit %**	No Samples	Relative Density	JK SMC A x b	No Samples	Bond BWi kwhrs/t	No Samples	Bond Ai g	No Samples	Bond CWi kwhrs/t
Fresh	**87.4**	53	2.75	31.3	54	11.4	10	0.419	11	17.0
Trans	**5.5**	9	2.69	76.5	9	7.8	4	0.152	1	8.5
Oxide	**7.0**	3	1.57	*	4	3.9	3	0.115		
Total	**100.0**	**65**	**2.66**	**34.0**	**67**	**10.7**	**17**	**0.383**	**12**	**15.4**
	* Oxide Data - Off JK Tech Scale									

Figure 13-19 shows the variability in the JK Tech SMC A x b value for 53 fresh and 9 transition samples tested during all (2018, 2020 and 2021) studies.

The predominant fresh domain accounting for 87.4% of planned processing tonnage was found to be moderately hard with variability from very hard to soft. Variation with respect to pit depth was studied but was found to be insignificant.

The transition domain accounts for 5.5% of planned processing tonnage and was found to be soft, and ranged from medium to very soft.

The oxide domain accounts for 7% of planned processing tonnage and was so soft it was off the JK Tech scale in terms of softness and so these results should be treated with caution. Figure 13-20 shows the variability of the hardness indices tested and the total number of samples tested is shown (in brackets) on the y axis.

Figure 13-19 A x b and Bond Ballmill Work Index Variability by Domain

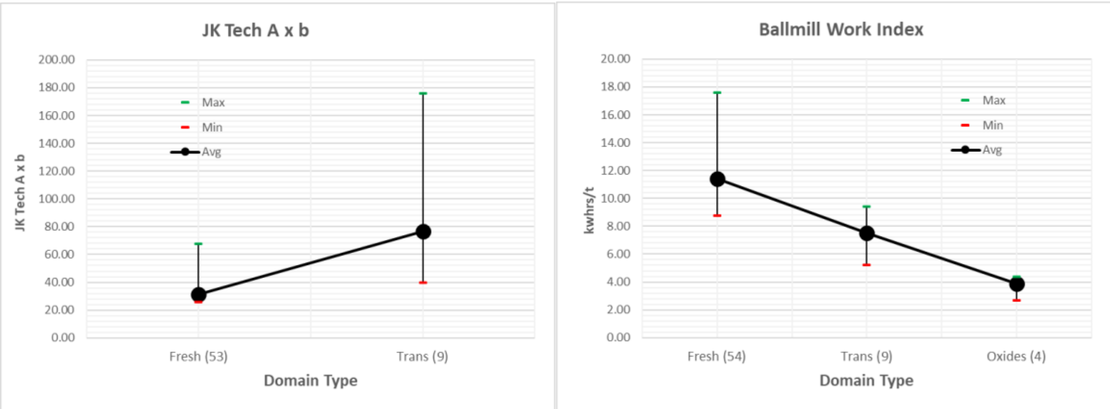
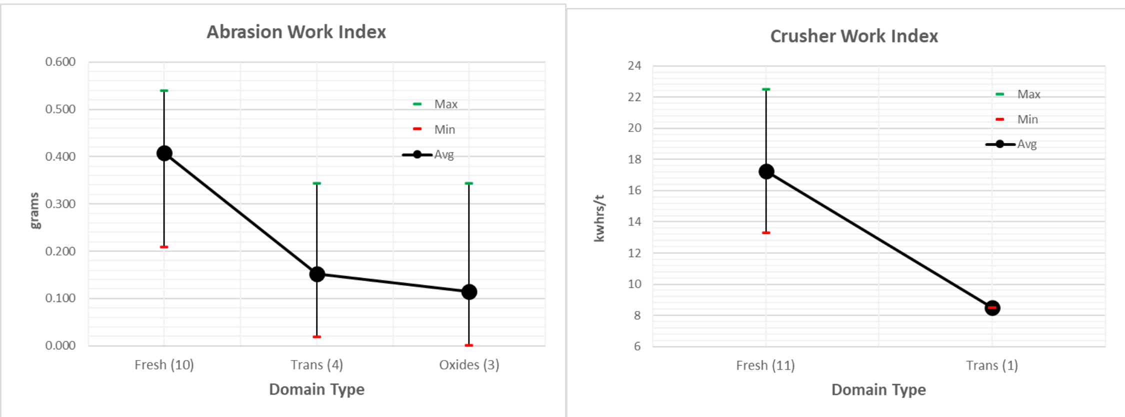


Figure 13-20 Bond Abrasion and Low Energy Impact (Crusher Work Index) Variability by Domain



The Fresh domains resistance to SAG milling was relatively consistently across all samples tested whereas the transition varies considerably, but is much softer. The oxide samples tested gave results that were all below JK Tech scale in terms of softness, this ore is extremely soft.

The resistance to ball milling in also highest and varies the most in the fresh domain but becomes softer and less varying in the transition and oxide domains.

The abrasivity of the fresh domain is the highest in the fresh domain and is lower and approximately the same in the transition and oxide. The variability of abrasivity was seen to vary by the same amount in all three ore domains.

The resistance to crushing was greatest in the fresh domain and was variable, the one transition sample tested was as expected less resistant to crushing. The oxide samples were all found to contain insufficient competent rocks to be tested.

Due to the high resistance to SAG milling observed, the high capacity required and the operational strategy to stockpile clay bearing oxides and blend them in slowly over LOM, HPGR has subsequently been considered as a better alternative to SAG milling. The HPGR sizing has been obtained using a Lycopodium in-house relationship between HPGR and SMC data. Actual HPGR testing will be required to firm up on equipment sizing and provide performance guarantees.

13.4.3 Metallurgical Data

Table 13-12 shows the summary of metallurgical data based upon actual Koné resource gold grades.

The gold leach extraction efficiencies are based upon arithmetic averages of all data tested in each ore domain and the relationship between gold head grades and leach residues has been used to normalise all leach extraction data to the actual resource gold concentration for each ore domain.

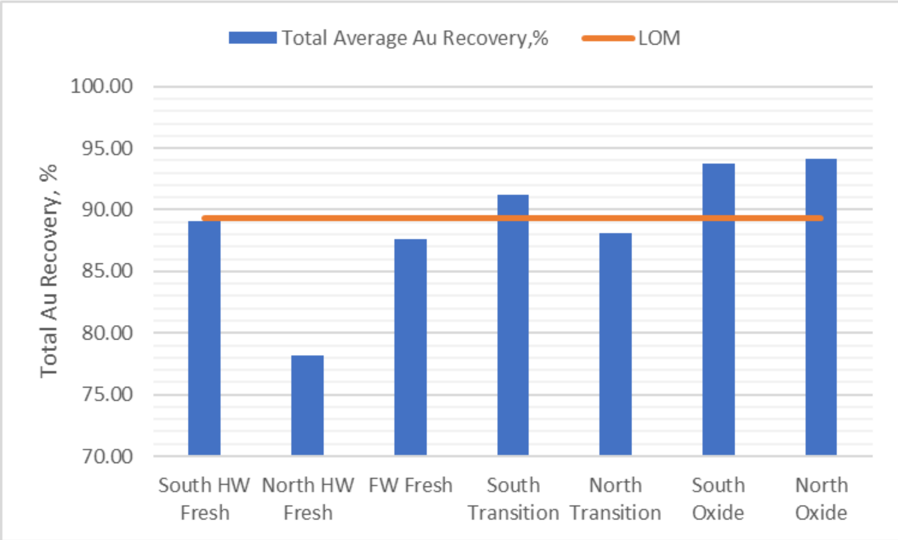
A solution assay of 0.005 mg/l has been used to calculate soluble loss of gold, the actual modelled mg /l Au were lower. 0.15% gold loss to carbon fines has been assumed.

Table 13-12 Summary of Metallurgical Data Produced

# Samples	Domain	Processed ('000 t)	Processed Au g/t	Calc Residue	Au Leach Extraction %	Soluble Au Losses		Carbon Fines Loss, %	Au Recovery %	kgs/t NaCN	kgs/t CaO
53	South HW Fresh	124,107	0.69	0.07	89.98	0.005	0.72	0.15	89.10	0.26	0.55
12	North HW Fresh	469	0.56	0.12	79.18	0.005	0.89	0.15	78.13	0.37	0.43
13	South FW Fresh	17,337	0.55	0.06	88.70	0.005	0.91	0.15	87.65	0.23	0.45
17	South Transition	7,894	0.56	0.04	92.27	0.005	0.89	0.15	91.23	0.18	0.99
5	North Transition	387	0.46	0.05	89.30	0.005	1.09	0.15	88.06	0.35	0.75
21	South Oxide	9,807	0.57	0.03	94.82	0.005	0.88	0.15	93.79	0.18	2.50
9	North Oxide	917	0.47	0.03	94.43	0.005	1.06	0.15	93.21	0.13	2.79
130	**LOM**	**160,918**	**0.66**	**0.06**	**90.20**	**0.005**	**0.76**	**0.15**	**89.30**	**0.25**	**0.70**

Figure 13-21 shows the average gold recovery by domain and the weighted LOM gold recovery. The LOM gold recovery is driven mainly by South HW Fresh as this is the biggest single contributing domain to gold ounces.

Figure 13-21 Summary of Total Gold Recovery



13.4.4 Clay Speciation

The testwork confirmed that Kaoilinite (sticky clay) and Montmorrilonite and Nontronite (swelling clays) are present in major amounts (>30%) in the oxide domain. Kaolinite appears to dominate more in the Saprolite oxide, and Montmorillonite and Nontronite dominates more in the Saprock.

13.4.5 Gravity Concentration

The testwork conducted clearly shows that there are no significant benefits of installing a gravity recovery stage due to the inherently very fine grain size of the gold.

13.4.6 Leach Residue Diagnostic Testwork

The testwork confirmed that the level of free recoverable gold in tailings is low (~1%). The majority of the gold is encapsulated within sulphides and the remainder encapsulated within host rocks.

The use of higher weight yield multi gravity equipment as a scavenging stage is not justified.

13.4.7 Cyanide Leach Variability Testwork

Figure 13-22 show the maximum, average and minimum values for the g/t Au, %S and ppm Cu in all samples tested.

The DFS average g/t Au samples selected contained lower (0.71g/t Au) as planned and more in line with resource g/t Au. The range g/t Au studies contained sufficient data to provide relationships between g/t Au in feed and g/t Au in residues as shown above. The %S & ppm Cu in feed was found to have a significant greater range of values in the North HW Fresh domain, this domain only contains 2.9% of total process tonnage and 2.5% of processed contained ounces but is characterised by elevated sulphides and specifically copper sulphides. Some copper spikes were seen in North Transition domain that results in higher localised cyanide consumption.

Figure 13-22 g/t Au, %S and ppm Cu in Feed Variability by Domain

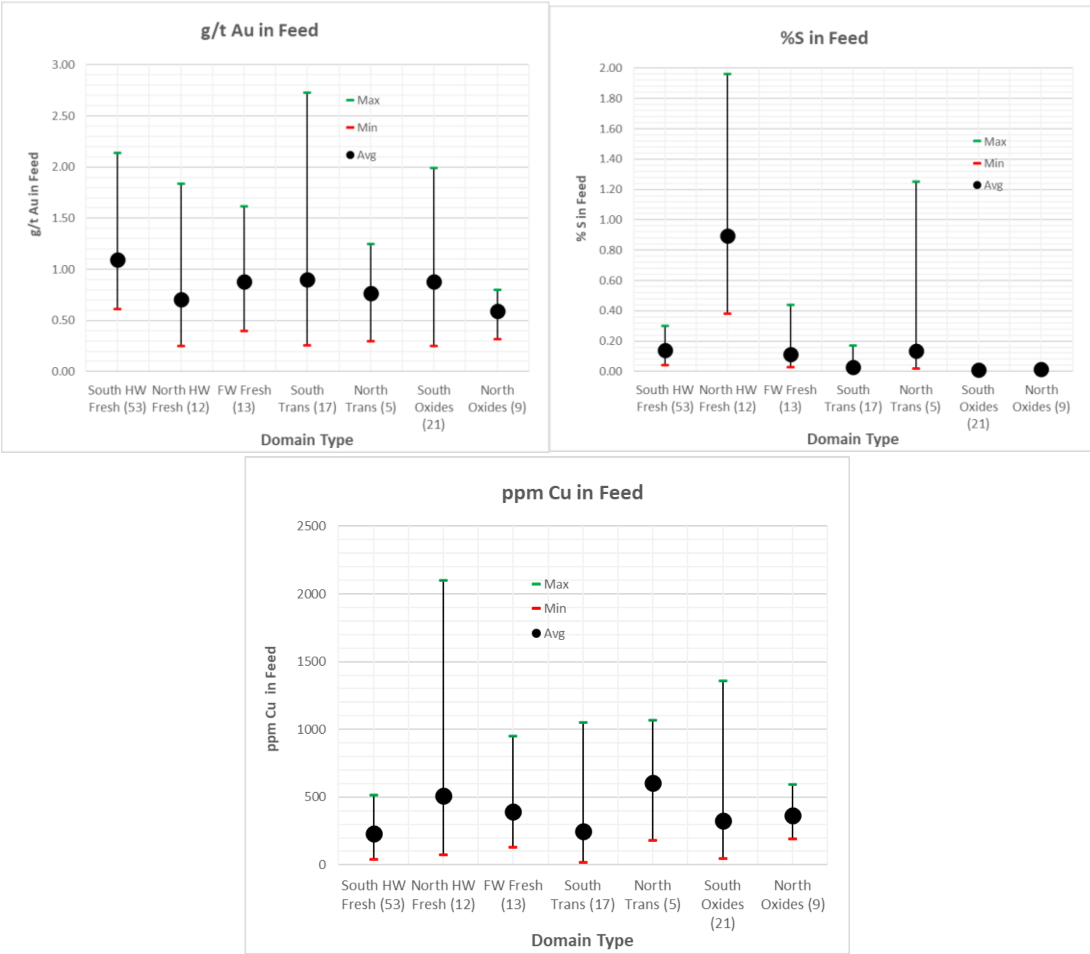


Figure 13-23 shows the variability in leach extraction efficiency for each ore domain. The low proportion North HW fresh ore type was seen to vary the most, between 70.8% and 87.8% averaging 79.2% and is due to the increased levels of sulphides and specifically copper sulphides. The South HW & FW Fresh, Transition and Oxide ore domains have higher gold leach extractions mainly due to these ores containing less sulphides.

Figure 13-23 % Au Leach Extraction Variability by Domain

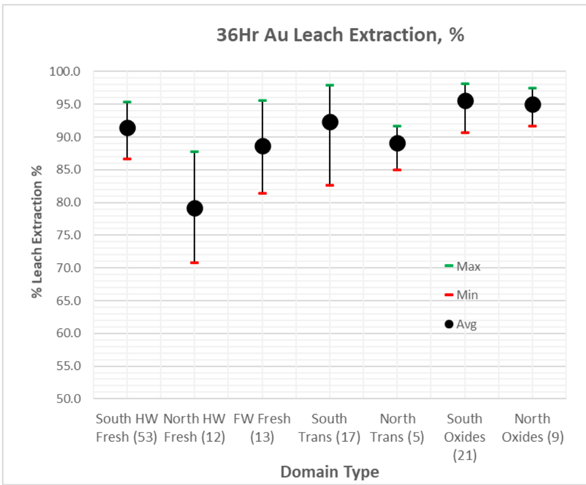


Figure 13-24 shows the variability observed with sodium cyanide consumption by ore domain. The variability in the sodium cyanide consumptions is highest in the transition ore domain due to localised higher copper bearing zones. Small localised higher copper zones in the small North HW fresh and Trans ore domains also results in elevated cyanide consumptions.

Figure 13-24 kg/t NaCN consumption by Domain

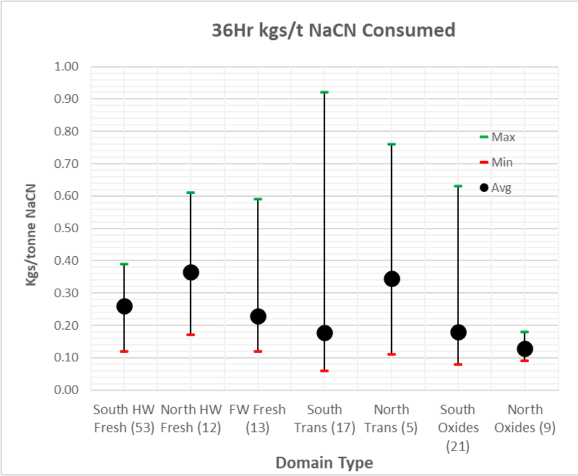
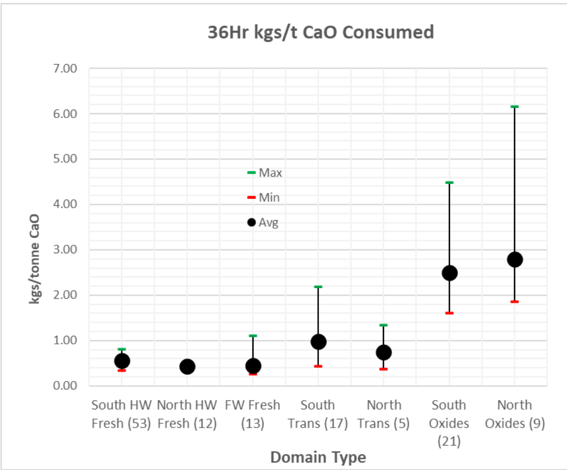


Figure 13-25 shows the variability in hydrated lime consumptions by domain. The consumption is consistently low for the HW fresh, FW fresh and Transitional ore domains but is much higher and varies more in the oxide ore domains.

Figure 13-25 kg/t CaO consumption by Domain



13.4.8 Cyanide Leach Decay Testwork

Sodium cyanide decay testing has shown that typical anticipated free sodium cyanide levels in final leach tailings will be ~130ppm for the predominant fresh ore type to ensure adequate free cyanide is present to leach the gold within 36 hours. This provides some room for reducing the free cyanide concentration in the plant which will reduce cyanide consumption rates slightly.

The sodium cyanide need only be added to the first two or worse case, three tanks. Stage addition along the tank train does not appear to be required.

13.4.9 Carbon Modelling

The latest DFS carbon modelling results using 10% Oxide as Saprolite and/or Saprock with 90% Fresh provides assurance that the carbon adsorption efficiency is not adversely affected by this blend of oxide with fresh ore.

Furthermore, the lower grade pregnant solutions due to the current lower resource gold head grade can produce <0.005 mg/l gold in solution using the DFS modified design of 8 stages of Kemix adsorption.

13.4.10 Silver Carbon Modelling

Bottle rolls were conducted on four composite samples representing HW Fresh, FW Fresh, Transition and Oxide ores to determine the adsorption stage recoveries for silver. The same conditions were employed as in the previously mentioned carbon modelling testwork.

Table 13-13 shows the silver stage adsorption efficiencies obtained from these tests.

Table 13-13 Silver Stage Adsorption Efficiencies

ADR Ag Fresh Efficiency, %	45.5
ADR Ag FW Fresh Efficiency, %	37.5
ADR Ag Trans Efficiency, %	58.3
ADR Ag Oxide Efficiency, %	46.2

The Ag Adsorption stage recoveries obtained from this phase of testing were used in conjunction with the mgs/l Ag results derived from the 36hr pregnant solution assays from the 130 variability bottle roll tests. This methodology was used to provide a ratio between gold and silver that would theoretically load onto carbon and this ratio was then used to calculate the %Au in dorè.

The %Au in dorè has been estimated for two distinct phases of the project as follows:

Years 1 to 9 inclusive treating high grade ore only

Years 10-15 inclusive treating low grade stockpiled ore

The estimated dorè fineness are shown in Table 13.14.

Table 13-14 Silver Stage Adsorption Efficiencies

	Distribution	g/t Au	Au Rec, %	%Au in Dorè
HG Ore	60%	0.82	90.3	88.1
LG Ore	40%	0.42	86.3	76.5
LOM	**100%**	**0.66**	**89.3**	**86.5**

13.4.11 Thickener Testing

The additional DFS thickener testing at the fixed flux rate of 1.09 tph/m^2 has been shown that small (5g/t) doses of coagulant are required to ensure the thickener supernatant clarity is <200 ppm solids. The additional benefit of adding this small amount of coagulant is that the flocculant dosage can be reduced. The thickener underflow densities using the anticipated 10% oxide blends whether using Saprolite or Saprock oxides, all confirm that thickener underflow densities can be higher than previously anticipated which allows more recycle of free cyanide in the tailings thickener overflow water to the leach feed in order to dilute pre-leach thickener underflow down to achieve target 50% solids for leaching. This will ultimately provide potential to reduce the cyanide consumption rates in practice, this effect is not factored into the financial modelling and so provides some potential to reduce sodium cyanide consumption costs as the project goes into production.

13.4.12 Rheological Properties

The rheological properties of the 10% oxide blends proposed do not show any signs of increasing the stress yield values to levels that would adversely affect adsorption of gold onto carbon and this has been supported by the carbon modelling work described above.

The critical solids densities calculated by plotting stress yield versus % solids in pulp are supported and verified by the additional thickener testing described above.

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

Recoverable resources were estimated for the Koné deposit by 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 mineralization styles.

The estimates are based on RC and diamond drilling data supplied by Montage in August 2021. Details of this sampling and assay are described in previous sections of this report.

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 resource reporting.

The Mineral Resource estimates have been classified and reported in accordance with NI 43 101 and the classifications adopted by CIM Council in May 2014. Estimates tested by drilling spaced at around 50 by 50m are classified as Indicated, with Inferred estimates based on generally 100m spaced drilling.

The estimates are constrained within an optimal pit generated at a gold price of $US 1,500/oz below a topographic wire frame produced by Montage from DGPS surveys.

Resource modelling was undertaken in a local grid defined by Montage, which comprises a rotation of 35° and plan view offset from WGS84 and an elevation increase of 1,000 m (Table 14-1) This transformation aligns the RC and diamond drilling traverses with local grid east-west section lines. All figures, coordinate and direction references in this chapter reflect local grid.

Table 14-1 WGS84 to local grid transformation

	WGS84	Local Grid
Easting Northing	756,452.21 mE 964,427.14 mN	5,000.00 mE 24,600.00 mN
Rotation Elevation change	-35° +1,000 m	

14.2 Mineralization Interpretation and Domaining

Drilling at Koné has delineated a northerly trending mineralized zone which dips to the west at around 50°. The transition from gold mineralization to barren host rock is generally characterized by diffuse grade boundaries.

The current resource modelling incorporates a mineralized envelope interpreted by MPR on the basis of composited gold grades capturing continuous intervals of greater than 0.1 g/t, and a background domain. Drill samples within the background domain, which extends to generally around 130 m from the mineralized envelope show generally low gold grades and rare, generally discontinuous zones of elevated gold grades. The background domain does not contribute to Mineral Resource estimates. Domain boundaries were digitized on cross sections, snapped to drill hole traces where appropriate, then wire framed into a three dimensional solid.

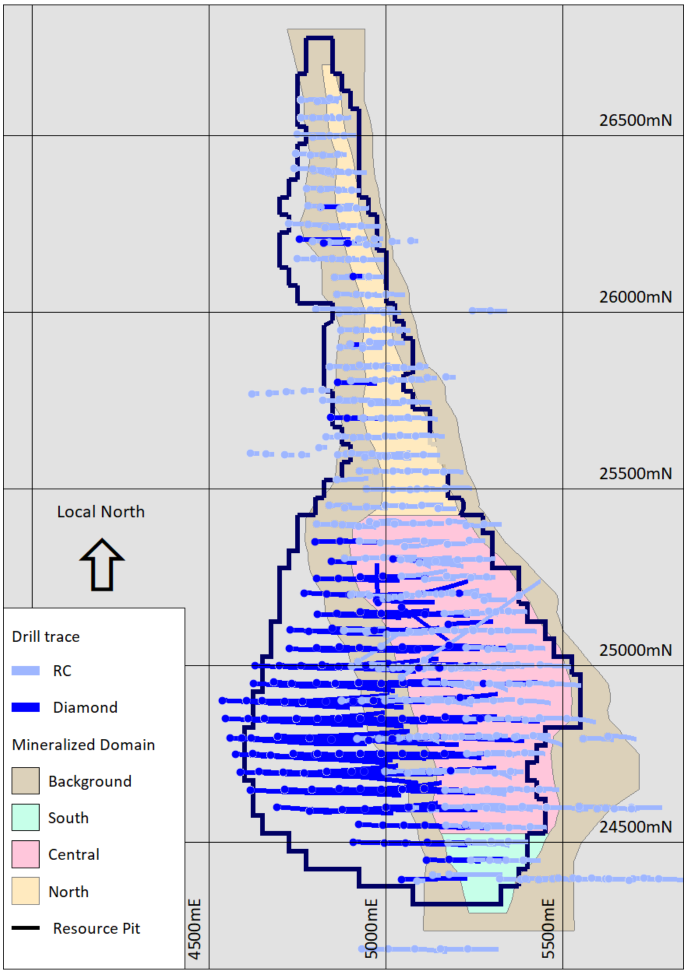
The mineralized envelope around 130 which strikes north-north east (350) and dips to the west an average of around 50° is interpreted over 2.4 km of strike with horizontal widths ranging from around 35 to 450 m and averaging around 215 m. True widths are up to 350 m.

The mineralized envelope is subdivided into three mineralized domains comprising lower grade southern and northern domains, and a main higher grade central zone. For each mineralized domain, average drill hole composite gold grades are higher in the western portion than in the east.

Montage supplied surfaces representing the base of complete oxidation and the top of fresh rock interpreted from drill hole geological logging. These surfaces were used for flagging of estimation dataset composites into oxide, transition and fresh subdomains, density assignment and partitioning resources by oxidation type. Within the mineralized envelope area, the depth to the base of complete oxidation ranges from around 8 to 45 m and averages 24 m with fresh rock occurring at of around 15 to 56 m averaging 35 m.

Figure 14-1 shows the surface expression of the mineralized domain relative to traces of the RC and diamond drilling utilized for Resource estimation. Figure 14-3 shows example cross sections of the estimation domains relative to drill hole traces coloured by composited gold grades and block model estimates. These plots demonstrate that for some cross sections, the resource drill holes do not all penetrate the full width of the mineralized envelope. For these sections drilling preferentially tests the western, generally higher average gold grade portions of the mineralized envelope.

Figure 14-1 Mineralized Domain and RC and Diamond Drill Traces



Produced by MPR in August 2021 from information supplied by Montage. Local Grid.

14.3 Estimation Dataset

The estimates are based on two metre down hole composited gold grades from RC and diamond drilling comprising 41,555 composites with gold grades ranging from 0.000 to 51.16 g/t and averaging 0.35 g/t. Samples from RC and diamond drilling provide approximately equal proportions of the combined mineralized domain composites.

Table 14-2 presents univariate statistics of composite gold grades for the estimation dataset subdivided by mineralized domain and oxidation zone. Notable features of these statistics include the following:

- At 0.04 g/t, the mean gold grade for the background domain composites is notably lower than for the mineralized domains demonstrating that the domaining has been effective in assigning most mineralized composites into the mineralized domains

- The central mineralized domain contains notably more composites than the south and north domains with this domain providing around 81% of the combined mineralized domain dataset

- At 0.50 g/t mean gold grade for composites from the central mineralized domain is notably higher than for the south and north mineralized domains

- With the exception of the South Mineralized Domain which represents around 4% of the combined mineralized domain dataset, and contains comparatively few completely oxidized and transition zone composites average composite grades for each mineralized domain show relatively little variability between oxidation zones

- Gold grades show strong positive skewness with a coefficient of variation of around two indicating that MIK is an appropriate estimation technique.

Table 14-2 Estimation dataset statistics

Au g/t	Background Domain				South Mineralized Domain			
	Comp. Ox	Trans.	Fresh	Total	Comp. Ox.	Trans.	Fresh	Total
Number	2,310	675	7,263	10,248	119	42	1,190	1,351
Mean	**0.04**	**0.02**	**0.04**	**0.04**	**0.06**	**0.06**	**0.25**	**0.23**
Variance	0.02	0.00	0.02	0.02	0.01	0.01	0.11	0.10
Coef. Var.	3.18	2.91	3.08	3.12	1.55	1.34	1.32	1.40
Minimum	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
1st Quartile	0.01	0.01	0.01	0.01	0.01	0.01	0.05	0.04
Median	0.01	0.01	0.01	0.01	0.02	0.03	0.15	0.13
3rd Quartile	0.04	0.02	0.04	0.04	0.07	0.06	0.34	0.31
Maximum	2.75	1.39	5.70	5.70	0.46	0.31	4.59	4.59
Au g/t	**Central Mineralized Domain**				**North Mineralized Domain**			
	Comp. Ox	**Trans.**	**Fresh**	**Total**	**Comp. Ox.**	**Trans.**	**Fresh**	**Total**
Number	2,348	1,261	21,801	25,410	994	381	3,171	4,546
Mean	**0.53**	**0.50**	**0.49**	**0.50**	**0.33**	**0.28**	**0.30**	**0.31**
Variance	0.83	0.75	1.07	1.04	0.51	0.18	0.51	0.48
Coef. Var.	1.73	1.75	2.10	2.05	2.14	1.53	2.35	2.25
Minimum	0.000	0.000	0.000	0.000	0.005	0.005	0.000	0.000
1st Quartile	0.11	0.09	0.09	0.10	0.07	0.05	0.05	0.06
Median	0.27	0.23	0.23	0.24	0.16	0.16	0.15	0.15
3rd Quartile	0.63	0.57	0.55	0.55	0.34	0.33	0.34	0.34
Maximum	24.03	13.43	51.16	51.16	12.42	4.47	25.18	25.18

14.4 Estimation Parameters

The block model frame work used for MIK modelling covers the full extents of the informing composites and mineralized domains. It comprises panels with dimensions of 25 m east-west by 50 m north-south and 10 m vertical defined in local grid coordinates.

For each domain, assayed composites from all three oxidation subdomains were combined for determination of indicator thresholds and class mean gold grades. This approach reflects the generally limited variability in average composite gold grades with oxidation zone and provides sufficient composites to generate robust conditional statistics.

Indicator grade thresholds were defined using a consistent set of percentiles for data in each domain. All class grades were determined from bin mean grades with the exception of the upper bins, which were reviewed on a case by case basis and an appropriate grade selected to reduce the impact of small numbers of outlier composites. For the Central Mineralized Domain, the upper bin grade was estimated from the bin mean grade with 15 outlier gold grade composites cut to 15 g/t. In the author's experience the approach adopted for determination of upper bin grades is appropriate for MIK modelling of highly variable mineralization such as Koné.

Table 14-3 presents the indicator thresholds and bin mean grades with the value and source of the upper bin grades used for estimation shown below the upper bin mean grade.

Indicator variograms were modelled for each indicator threshold from the combined mineralized domain composites. For determination of variance adjustment factors a variogram was modelled from composite gold grades. The modelled variograms are consistent with geological interpretation and trends shown by composited gold grades, showing an average westerly dip of around 50°.

As an example of the variogram models, Figure 14-2 presents a three dimensional variogram surface map of the median indicator variogram model at variogram value of 0.85.

The four progressively more relaxed search criteria (Table 14-4 used for MIK estimation were aligned with dominant domain mineralization orientation and inclined towards the west at 50°. Search pass 4 informs a small number of panels in broadly sampled areas. Panels informed by this search pass represent around 0.3% of estimated mineral resources and reliability of these estimates does not significantly impact confidence in estimated resources.

The model estimates include a variance adjustment to give estimates of recoverable resources above gold cut off grades for selective mining (SMU) dimensions of five by ten by five metres (east, north, vertical). The variance adjustments were applied using the direct lognormal method and the adjustment factors listed in Table 14-5.

Figure 14-2 Three dimensional variogram plot

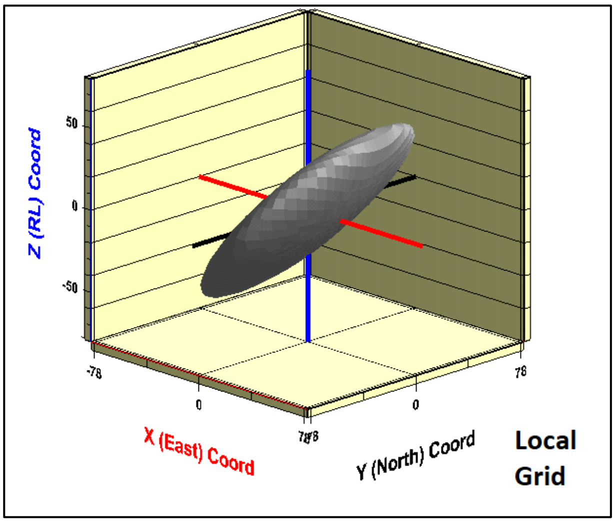


Table 14-3 Indicator thresholds and bin mean grades

Percentile	Background Domain		South Mineralized Domain		Central Mineralized Domain		North Mineralized Domain	
	T'hold (Au g/t)	Mean (Au g/t)	T'hold (Au g/t)	Mean (Au g/t)	T'hold (Au g/t)	Mean (Au g/t)	T'hold (Au g/t)	Mean (Au g/t)
10%	0.005	0.005	0.007	0.005	0.037	0.019	0.013	0.007
20%	0.005	0.005	0.030	0.019	0.075	0.056	0.040	0.026
30%	0.005	0.005	0.060	0.044	0.117	0.096	0.070	0.054
40%	0.007	0.006	0.100	0.078	0.170	0.143	0.108	0.088
50%	0.014	0.011	0.135	0.120	0.240	0.202	0.150	0.128
60%	0.024	0.019	0.190	0.163	0.330	0.281	0.203	0.174
70%	0.037	0.030	0.265	0.225	0.461	0.390	0.285	0.242
75%	0.048	0.043	0.324	0.292	0.555	0.505	0.340	0.311
80%	0.060	0.054	0.380	0.348	0.680	0.616	0.410	0.370
85%	0.080	0.070	0.455	0.414	0.854	0.763	0.520	0.461
90%	0.110	0.093	0.575	0.510	1.140	0.983	0.685	0.598
95%	0.205	0.149	0.770	0.664	1.752	1.396	1.035	0.819
97%	0.295	0.242	0.915	0.838	2.255	1.976	1.345	1.182
99%	0.570	0.400	1.296	1.046	3.766	2.839	2.395	1.721
100%	5.700	0.920	4.590	1.810	51.160	6.418	25.175	3.524
		Median		Median		Avg. Cut 15 g/t		Median

Table 14-4 Search criteria

Ellipsoid Rotation: Z+10,Y-50 (Local Grid)				
Search	Radii (m)	Minimum Data	Minimum Octants	Maximum Data
1	60,60,15	16	4	48
2	78,78,19.5	16	4	48
3	78,78,19.5	8	2	48
4	120,120,30	8	2	48

Table 14-5 Variance adjustment factors

Domain	Block/ Panel	Information Effect	Total Adjustment
All domains	0.165	0.842	0.139

14.5 Bulk density assignment

Bulk densities of 1.65, 2.55 and 2.80 t/bcm assigned to completely oxidized, transitional and fresh material respectively. These values reflect the average of the available measurements (Table 11-10).

14.6 Classification of the Estimates

In the author's opinion, the available sampling does not define the Koné mineralization with sufficient confidence for estimation of the Measured resources. Estimates tested by drilling spaced at around 50 by 50 m are classified as Indicated, with Inferred estimates based on generally 100 m spaced drilling.

The Indicated and Inferred estimates are restricted to model panels within the mineralized envelope tested by drilling generally spaced at closer than 100 by 100 m. More broadly sampled and peripheral mineralization is too poorly defined for estimation of Mineral Resources and is not included in estimated resources.

Model estimates were classified as Indicated and Inferred by estimation search pass and two sets of sectional polygons defining areas of consistently spaced drilling for each model row.

Panels informed by search pass 1 within polygons defining the outer limits of consistently 50 by 50 m spaced drilling are classified as Indicated. Remaining panels within polygons defining the limits of generally 100 by 100 m and closer spaced drilling, including all search pass 2, 3 and 4 panels were initially classified as Inferred. Comparatively rare isolated search pass 2 and 3 panels within areas of generally 50 by 50 m drilling were re-classified as Indicated ensuring all panels within reasonably closely drilled areas are classified as Indicated, giving a consistent distribution of resource categories. These re-classified panels are commonly near surface and due to the search 1 octant requirements are not informed by this search pass.

The plots in Figure 14-3 show the polygons used for assignment of confidence categories in red and green respectively.

14.7 Model Reviews

Model reviews included comparison of estimated block grades with informing composites. These checks comprised inspection of sectional plots of the model and drill data and review of swath plots and showed no significant issues.

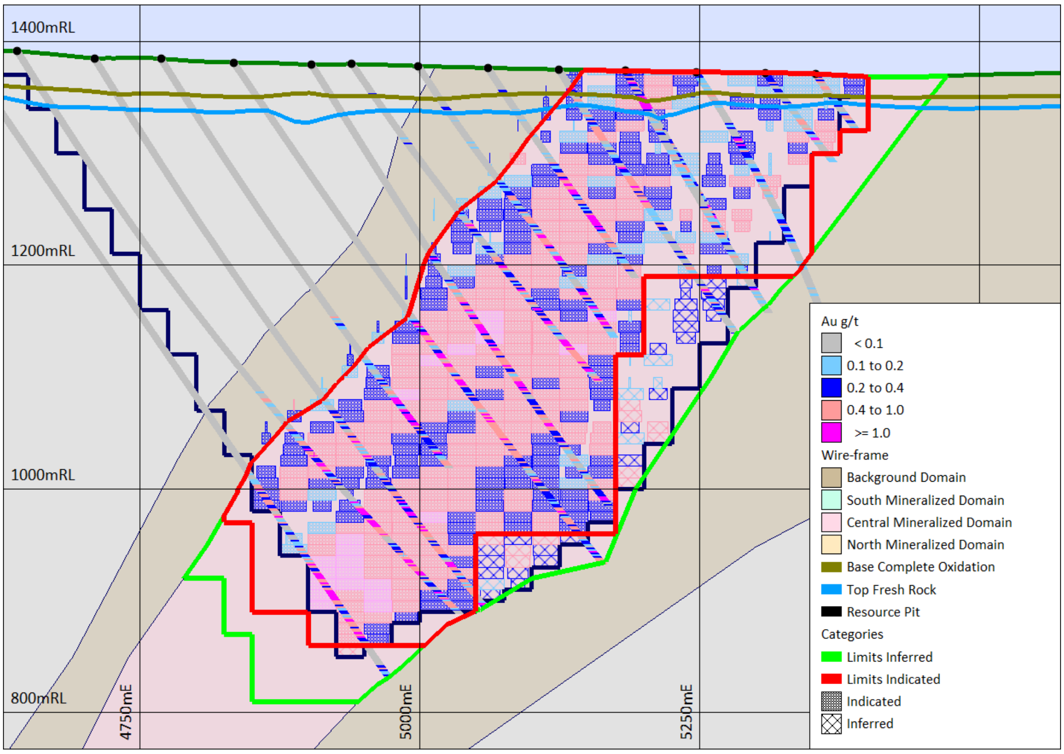
Figure 14-3 shows representative cross sections of the Koné block model. These plots show model panels scaled by the estimated proportion above 0.2 g/t cut off and coloured by the estimated gold grade above this cut off relative to the estimation domains and drill hole traces coloured by two metre composited gold grades. Model panels shown in this figure are restricted to those within the optimal pit used for constraining Mineral Resource estimates.

The plots in Figure 14-3 include instances where model blocks appear poorly correlated with mineralized intercepts in nearby drill holes. This reflects the way the resource models have been presented. Only model blocks that contain an estimated resource above 0.2 g/t gold cut off are plotted and the proportion above cut off has been used to scale the east dimension of the blocks for presentation purposes. The scaling occurs about the model block centroid coordinate and introduces the apparent miss match between data and the resource model blocks.

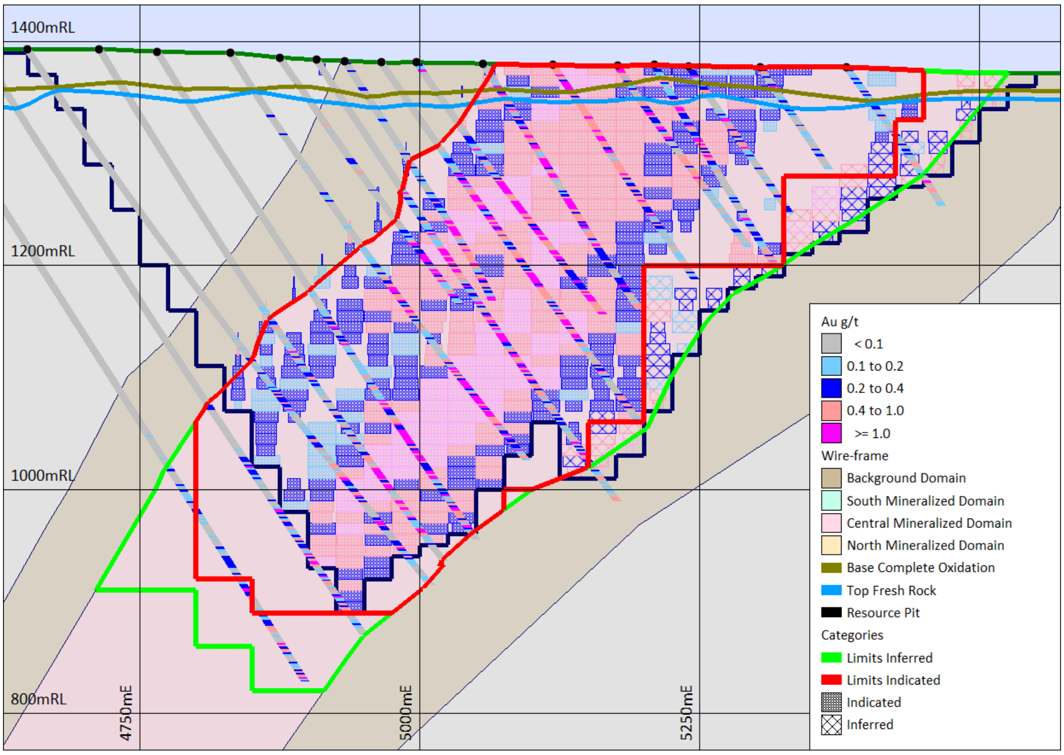
The swath plot in Figure 14-4 compares average mineralized domain composite grades and average MIK panel grades within the volume of model blocks classified as Indicted. For preparation of this plot average composite gold grades from the south, central and north mineralized domains include uppercuts of 2.7, 7.1 and 4.9 g/t respectively representing the 99.75th percentile of each dataset reducing the impact of a small number of outlier composite grades.

Figure 14-4 shows that although, as expected average MIK panel grades are smoothed relative to average composite grades they generally closely follow the trends shown by composite mean grades with the exception of peripheral areas of limited and variably spaced sampling at the extremities of the deposit. The figure shows local apparent deviations between model and composite trends which are influenced by the variability in drill hole spacing, such as clustering of drilling in areas of higher average grade mineralization in the north of the deposit. These features reflect the distribution of drilling and do not represent biases in the model estimates.

Figure 14-3 Model Blocks at 0.2 g/t cut off



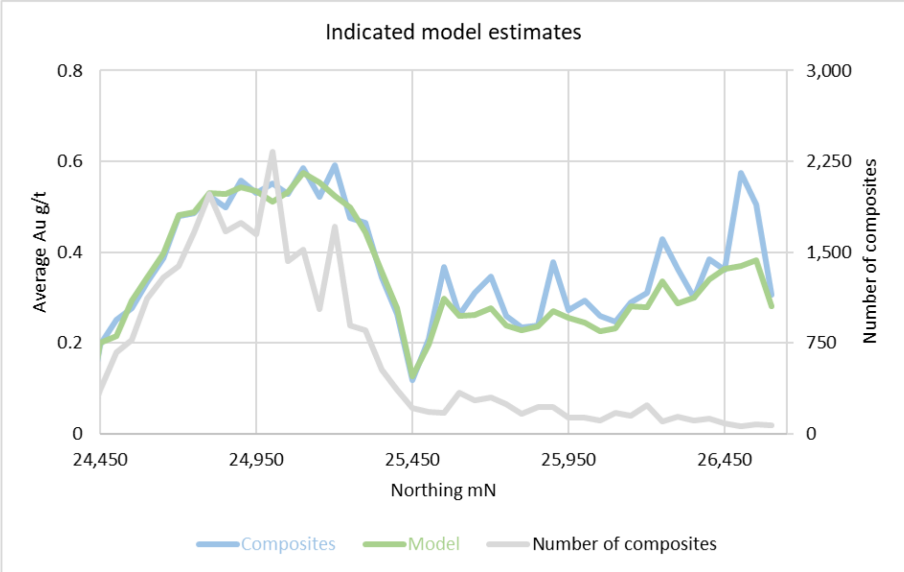
24,700m N



24,900m N

Source MPR, August 2021, Local Grid

Figure 14-4 Estimated Panel Grades Versus Composite Grades



14.8 Mineral Resource Estimates

To provide estimates with reasonable prospects for eventual economic extraction, Mineral Resources are reported within an optimized pit shell generated from parameters supplied by Montage. The optimization parameters reflect a large scale conventional open pit operation with the cost and revenue parameters detailed in Table 14-6.

The pit shell constraining the estimates extends over 2.5 km of strike to a maximum depth of around 550 m.

Table 14-7 shows the Koné Indicated and Inferred Mineral Resource Estimates for a range of cut off grades. The author considers the estimates at 0.2 g/t represent the base case or preferred scenario. Table 14-8 shows the estimates at 0.2 g/t cut off subdivided by oxidation type. The figures in these tables are rounded to reflect the precision of the estimates and include rounding errors.

The Mineral Resource estimates have an effective date of the 12th of August 2021.

There are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that may materially affect the Mineral Resource estimates.

Table 14-6 Resource pit shell optimization parameters

Gold Price		US$ 1,500/oz			
		Oxide	**Transition**	**Fresh**	**Total**
Wall angle		35º	40º	60º	
Average mining cost		US$ 2.27/t	US$ 2.37/t	US$ 3.11/t	US$ 2.99/t
Mill processing cost		US$ 6.66/t	US$ 6.24/t	US$ 7.82/t	US$ 7.82/t
Mill recovery	South/Central	94.34%	91.07%	88.97%	
	North	93.03%	90.61%	87.82%	
Government royalty		4.0%	4.0%	4.0%	4.0%
Maverix royalty		2.0%	2.0%	2.0%	2.0%
Community development		0.5%	0.5%	0.5%	0.5%
Selling costs		US$ 4.71/oz	US$ 4.71/oz	US$ 4.71/oz	US$ 4.71/oz

Table 14-7 Indicated and Inferred Mineral Resource Estimates by cut off grade

Cut off	Indicated			Inferred		
Au g/t	**Mt**	**Au g/t**	**Au moz**	**Mt**	**Au g/t**	**Au moz**
0.1	278	0.51	4.56	32	0.35	0.36
0.2	**225**	**0.59**	**4.27**	**22**	**0.45**	**0.32**
0.3	168	0.70	3.78	14	0.56	0.25
0.4	128	0.82	3.37	9.0	0.69	0.20
0.5	99.1	0.92	2.93	5.9	0.81	0.16
0.6	76.9	1.03	2.55	3.9	0.95	0.12
0.7	59.9	1.14	2.20	3.2	1.1	0.10
0.8	46.8	1.25	1.88	1.9	1.2	0.07

Table 14-8 Indicated and Inferred Mineral Resource Estimates at 0.2 g/t cut off by oxidation type

Oxidation	Indicated			Inferred		
	Mt	**Au g/t**	**Au moz**	**Mt**	**Au g/t**	**Au moz**
Comp. Ox.	14	0.54	0.24	0.3	0.43	0.20
Transition	9	0.56	0.15	0.1	0.54	0.15
Fresh	202	0.59	3.86	21	0.45	3.25
Total	**225**	**0.59**	**4.27**	**22**	**0.45**	**0.32**

15 MINERAL RESERVE ESTIMATE

15.1 Statement of Reserves

The definition of a mineral reserve, as defined by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), is:

> *"A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include the application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.*
>
> *Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.*
>
> *The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situation where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported."*[1]

The mineral reserve classification is also based on the degree of certainty that can be attached to the estimate. This classification is broken into two categories; Probable and Proven. These are defined by the CIM as:

> *A "Probable Mineral Reserve" is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource/ The confidence in the Modifying Factors applying to a Probably Mineral Reserve is lower than that applying to a Proven Mineral Reserve.*
>
> *A "Proven Mineral Reserve" is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.*

This section states the estimated Mineral Reserves for the Koné deposit. The deposit is divided into the South and North Pits. There are no existing stockpiles on the property.

Table 15-1 Summary of Mineral Reserves for the Koné deposit

	Classification	Oxide			Transitional			Fresh			Total		
		Mt	Au g/t	Au M Oz	Mt	Au g/t	Au M Oz	Mt	Au g/t	Au M Oz	Mt	Au g/t	Au M Oz
South Pit	Probable	9.8	0.57	0.18	7.9	0.56	0.14	141.4	0.67	3.05	159.1	0.66	3.39
North Pit	Probable	0.9	0.47	0.01	0.4	0.46	0.01	0.6	0.57	0.01	1.9	0.5	0.03
Total	Probable	10.7	0.56	0.19	8.3	0.56	0.15	142.1	0.67	3.06	161.1	0.66	3.42

[1] https://mrmr.com.org/media/1068/cim_definition_standards_2014.pdf

15.2 Basis of Estimate

The Mineral Reserve estimate was undertaken by Carci Mining Consultants Ltd ("Carci") using Deswik mine planning software (Version 2021.2) and demonstrated that mining of the deposit is practical and economically viable. The Mining Study and the resultant Mineral Reserve estimate relied on information from the following sources:

- Joeline McGrath visited the site in November 2021. During this visit, she visited the pit areas, waste dump areas and the TSF and processing plant locations and viewed a variety of ore and waste core and met with the geology team

- The geological model was provided by Montage as an MIK model in Datamine format.

- The geotechnical review was undertaken by SRK Consulting (UK) Ltd and the report was provided to Carci by Montage. Slope parameters for the optimisation and design were obtained from this report

- Hydrogeological investigation results including ground water modelling and pit dewatering estimates were developed by Australian Groundwater and Environmental Consultants Pty Ltd and provided to Carci by Montage

- Environmental and Social impact assessment for the project was undertaken by Minesia Ltd

- General administration, processing costs and process recovery estimates were provided by Montage in consultation with Lycopodium

- The mining productivity and cost estimates are based on information received from major equipment suppliers and a mining contract tender budget exercise conducted in 2021 involving eight international mining contractors. This is complemented with data available from similar operations

- Mining parameters and mining costs were prepared and estimated by Carci in consultation with Montage utilising the information received from the sources above.

The major tasks completed in the mining study for the reserve estimation include the definition and review of the study parameters, pit limit optimisations, cut-off grade analysis and mine design. This is detailed in the "Koné Gold Project Feasibility Study Mining Report (UK21-0050)" produced by Carci as part of this work.

15.3 Pit Optimisation Key Assumptions

15.3.1 Resource Model

The pit optimisations and reserves are based on the August 2021 resource model, as discussed in Section 14. The resource model was provided by Montage and was classified by Indicated and Inferred categories. Only the Indicated category resources were used in the mining study and resultant reserve estimates.

The August 2021 geological block model was provided as a Multiple Indicator Kriged model (or Partial Percentage model). It contained resource classification, oxidation zone and density estimates for regular blocks with dimensions of 12.5 m x 12.5 m x 10 m. Grade bins were provided at 0.05 g/t increments from 0.1 g/t to 0.7 g/t and then in 0.1 g/t increments from 0.7 g/t to 1.5 g/t. All material above 1.5 g/t was contained within a single increment. The model contained a grade field and percentage field for each cut-off grade in each block.

Topographic and geological surfaces were provided by Montage and reflected the latest information available.

15.3.2 Dilution and Ore Recovery

The MIK Resource model incorporated dilution and mine recovery. After review, it was agreed with Montage that no further application or recovery or dilution factors would be necessary.

15.3.3 Geotechnical Considerations

SRK Consulting (UK) Ltd ("SRK") was commissioned to assess the geotechnical conditions of the Koné project. The following information in this section has been extracted from the SRK report entitled "Geotechnical Study on the Koné Project, Côte d'Ivoire (UK31101)", released in December 2021.

SRK initially provided a geotechnical drilling programme comprising six cored boreholes. SRK visited the site in January 2021 as part of the PEA and carried out QA/QC validation of the core and logging of the geotechnical boreholes. Samples were also selected from the most relevant lithologies and weathering degrees within the study area for intact rock strength and discontinuity shear strength characterisation.

Two additional boreholes were recommended during the PEA and these were added to the original six boreholes. The pit shell as of August 2021 and the eight cored geotechnical boreholes shown in blue are shown in Figure 15-1. The image also contains a selection of cored resource boreholes, shown in black that SRK included in the geotechnical characterisation of the open pit rock. The eight geotechnical boreholes generated 2,405 m of core.

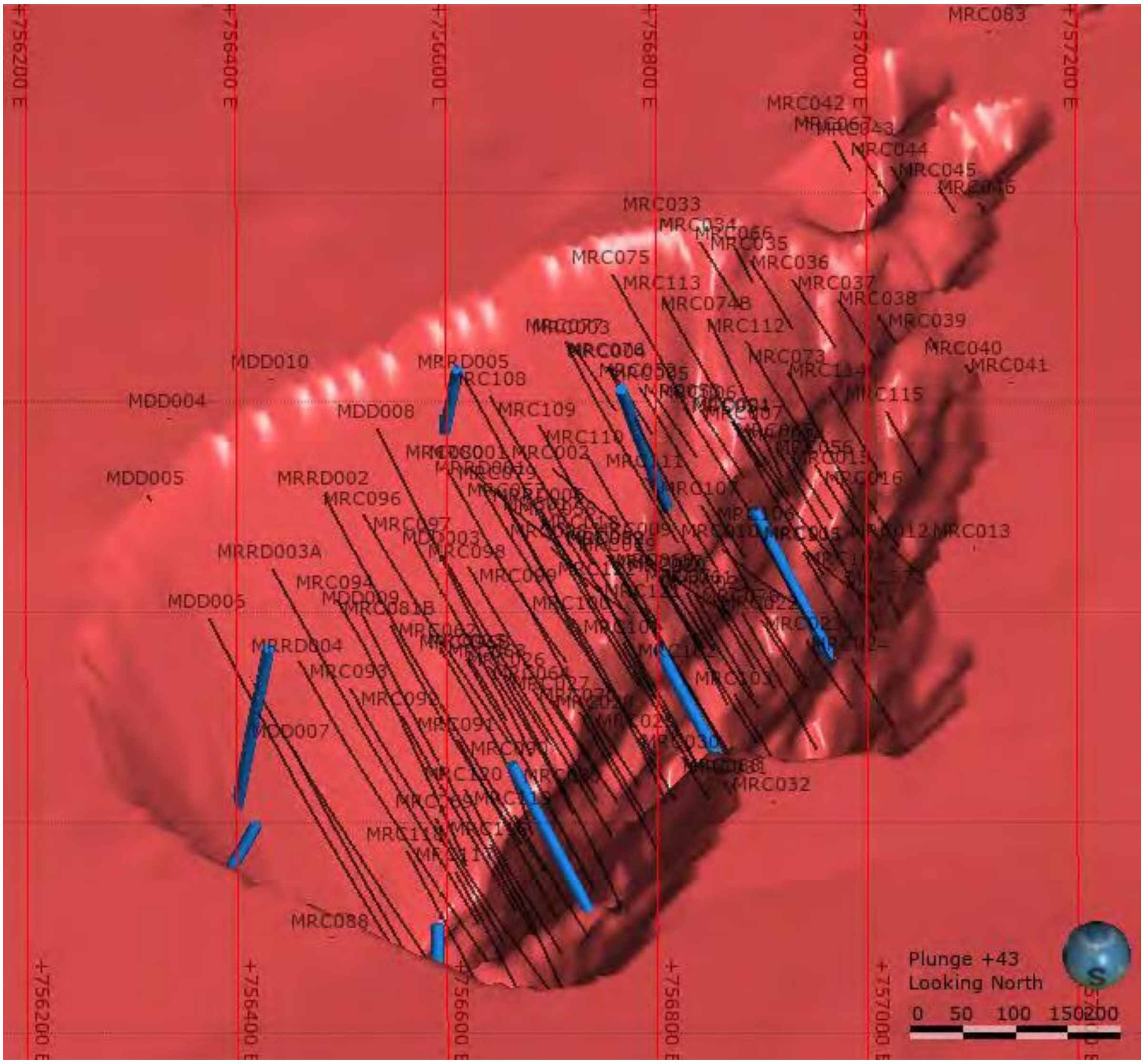
Logging analysis yielded three main geotechnical units: surficial soil and highly weathered rock ("Soil"), slightly to moderately weathered rock ("MW/SW") and unweathered rock ("UW"), with thicknesses of 15 to 25 m, 14 to 25 m and >350 m respectively. By far, UW is the predominant unit, making up 84% of the geotechnical metres drilled. For clarity and comparison, these units broadly correspond to the terms oxide, transitional and fresh rock respectively.

UW is divided into two lithological sub-units: Diorite ("DIO") and Volcanoclastic ("VC"), which exhibit uniaxial compressive strengths ("UCS") greater than 174 MPa and 140 Mpa respectively. The rock mass quality of UW, characterised through the Rock Mass Rating ("RMR"89), lies for the most part between 75 and 85. The UCS of the MW/SW unit could not be defined exclusively from laboratory tests, however it has been assigned a range between 32 Mpa and 46 Mpa. RMR89 for MW/SW lies mostly between 60 and 75.

Water has been detected 2m from the surface at its shallowest. As a consequence, for geotechnical design purposes, all bench analyses have assumed water-saturated joints, all inter-ramp or overall slope analyses have considered the rock units UW and MW/SW to be below the water table, and a lowered water table has been considered for the Soil unit.

Owing to the high intact rock strength ("IRS"), especially in the UW unit, structure will play a major role in slope design and performance at all scales. Because of this, both major structures and minor joints were analysed in detail. Joint data consisted of 2,240 poles from the eight oriented geotechnical boreholes. This was supplemented by 6,105 poles from 17 geological boreholes. Overall, seven major joint sets were identified, of which the foremost strikes N-S, dipping W at 20°-70° and the second is sub-horizontal.

Figure 15-1 Koné pit shell as of August 2021 and eight cored geotechnical boreholes



Source: Geotechnical study on the Koné Project, Côte d'Ivoire: SRK Consulting (UK) Ltd

Major structures in general exhibit very high RMR values (70 - 80 for the most part). Only certain mafic dykes (MDY) exhibit slightly lower RMR values, albeit with high IRS and high Joint Condition rating. Felsic dykes ("FDY"), although consistent and traceable through multiple drill sections, exhibit very high IRS and reasonably good Joint Condition. Therefore, with the information available, it is concluded that major structures do not pose a risk to inter-ramp or overall slopes. As for water flow along such structures, site geologists have indicated that the FDY are not porous nor very fractured, so they are unlikely to contain water. EGD, while being deformed, chlorite-altered and strongly foliated, have a high RMR (greater than 70). Thus, it does not seem probable that major structures will carry water.

In UW rock, four structural domains, North, East, South and West, were defined to carry out kinematic and wedge formation analyses. These were based solely on slope dip direction as no difference in jointing was found between DIO/VC sub-units. Benches in UW rock are 30m high with an 80° bench face angle and a berm of 8m, yielding a 66° IRA. With this configuration and using pre-split blasting, stacks of up to 7 benches can be achieved for a maximum UW slope height of 210m. Geotechnical berms have been removed although ramps are in place in the final pit design such that there is a maximum of 7 benches per stack.

Benches in slightly to moderately weathered rock are 20m high with an 80° bench face angle and 9m berms, yielding a 58° IRA. This design is valid for a maximum unit thickness of 40 metres. For soil slopes with a maximum thickness of 36 metres, a 39° IRA is acceptable under dry conditions, or if the water table is kept at a minimum of 15 metres from the surface.

Table 15-2 Geotechnical Report Bench Design Summary

Unit	Wall	BH (m)	BFA (°)	BW (m)	IRA (°)
Soil	All	6	60	4	**39**
MW/SW	All	20	80	9	**58**
UW	All	30	80	8	**66**

Table 15-3 Geotechnical Report IR and OS Design Summary

Unit	Wall	Benches per stack	IR height (m)	Bench Stack Angle (°)	Geotechnical berms (m)
Soil	All	2	12	**48**	6
MW/SW	All	2	40	**68**	9
UW	All	7	210	**68**	

SRK has stated in its report that it considers the geotechnical investigations and subsequent analyses to date to have been developed to a level appropriate for a Feasibility Study level open pit slope design. SRK does make one recommendation which should be included in any future geotechnical study and that is:

"The development of a 3D deposit-scale structural model is recommended to assist with the spatial prediction of local/inter-ramp scale structures significant to geotechnical analysis. Given the competent nature of the rock mass, structures could have influence on overall pit slope stability depending on their orientation relative to the slope. Further investigations should consider geophysics data and field mapping to assist with characterising potential fault zones."

15.3.4 Optimisation Constraints

There are several surface features that require consideration for the optimisation. The Toudian Forest Reserve covers part of the northern concession area, although this is not considered to be a hard constraint for mining activities. A major road passes through the lease area over the southern part of the north deposit. A power line is also adjacent to this road.

A trade-off study was conducted to evaluate the impact of diverting the road around the north of the north pit. Due to the small tonnage impacted by the existing road it was determined that there was no economic benefit to diverting the road.

A buffer of 200 m on either side of the existing road was applied to the optimisation and any material within this buffer was flagged as un-mineable.

15.3.5 Base Mining Cost Estimate

Unlike previous studies, which developed mining costs derived from first principles, a budget tender process was conducted to ascertain market rates for contract mining operations. Eight contractors provided a tender response.

The tender evaluation was completed as though the exercise was a formal tender. An average cost per tonne of $2.67/t was developed based on the chosen bid, excluding rehandle costs.

The preferred contractor did not provide costs broken into the different material types, so it was not possible to simply transpose the tender rates into the optimisation process. In order to honour the contractor pricing a series of optimisations were conducted, with fixed mining costs being flexed such that the overall unit mining cost matched the cost calculated from the contractor selection process.

Table 15-4 shows the fixed mining costs derived from the contractor pricing study.

Table 15-4 Summary of Fixed Mining Costs

Material	Fixed Unit Mining Cost ($/t)
Oxide Ore	2.36
Transitional Ore	2.38
Fresh Ore	2.52
Oxide Waste	1.97
Transitional Waste	1.99
Fresh Waste	2.21

Incremental haulage costs were also determined for the fleet and applied during the optimization process to account for vertical haulage. An incremental hauling cost of $0.027/t per 10 metres of vertical haul was calculated and applied to all blocks assuming a reference RL of 375m UTM (1,375m RL local grid).

15.3.6 Processing Costs

Processing costs were developed by Lycopodium in consultation with Montage. Processing costs vary based on the oxidation category for the North or South Pit. Table 15-5 shows the fixed and variable processing costs. Fixed costs were applied to the cost per tonne based on an 11 Mtpa processing rate.

Table 15-5 Processing costs ($/t processed)

Material	Fixed Costs (annual)	Variable Cost ($/t)	Total Unit Cost ($/t)
South Oxide	$21,261k	$5.02	$6.95
South Transitional	$21,261k	$4.42	$6.35
South Fresh (Hanging Wall)	$21,261k	$5.85	$7.78
South Fresh (Footwall)	$21,261k	$5.77	$7.70
North Oxide	$21,261k	$5.01	$6.94
North Transitional	$21,261k	$4.74	$6.67
North Fresh	$21,261k	$6.19	$8.12

15.3.7 Gold Price, Royalties and Selling Costs

A gold price of $1,250/oz was used in the pit optimisations and the calculations of the break-even cut-off grades. Three different royalties are applicable to the project: the government royalty of 3.5%, the Maverix Royalty of 2.0% and the Community Royalty of 0.5%.

Table 15-6 Revenue and selling parameters

Item	Value
Gold Price	$1,250/oz
Refining and Selling Cost	$5/oz
Government Royalty	3.5%
Maverix Royalty	2.0%
Community Royalty	0.5%

15.3.8 Processing Recovery

Recovery formulae were developed from metallurgical testwork dependent on head grade. The south deposit was divided into four domains, namely: oxide, transition, fresh hanging wall and fresh footwall. The north deposit was divided into three domains namely: oxide, transition, fresh hanging wall.

The processing recoveries shown in Table 15-7 have been calculated for the different metallurgical zones at the predicted break-even cut-off grades.

Table 15-7 Processing recoveries at breakeven cut-off grade and average pit inventory grade

	Processing Recovery @ cut-off grade	Processing Recovery @ average LOM grade
South Oxide	85.23%	93.36%
South Transitional	88.17%	90.78%
South Fresh (Hanging Wall)	78.34%	89.10%
South Fresh (Footwall)	86.03%	87.65%
North Oxide	86.27%	92.68%
North Transitional	85.58%	87.52%
North Fresh	69.58%	92.68%

15.3.9 Cut-off Grade Determination

Cut-off grades were calculated for oxide, transitional and fresh material within the deposit, with the fresh material for the South Pit being further divided into footwall and hanging wall material. Table 15-8 shows the calculated cut-off grades at various gold prices.

Table 15-8 Cut-off grade calculations

	$1,200	**$1,250**	$1,300	$1,350	$1,500	$1,700
South Oxide	0.23	**0.22**	0.21	0.20	0.18	0.16
South Transitional	0.20	**0.19**	0.18	0.18	0.16	0.14
South Fresh (Hanging Wall)	0.28	**0.26**	0.25	0.25	0.22	0.19
South Fresh (Footwall)	0.25	**0.24**	0.23	0.22	0.20	0.17
North Oxide	0.22	**0.22**	0.21	0.20	0.18	0.16
North Transitional	0.22	**0.21**	0.20	0.19	0.17	0.15
North Fresh	0.32	**0.31**	0.30	0.29	0.26	0.22

Given that the geological model is an MIK model with partial percentages assigned to grade bins, it is necessary to identify the corresponding grade bin for each cut-off grade. Table 15-9 shows the calculated cut-off grades at $1,250/oz and the associated grade bins.

Table 15-9 Cut-off grade bins

	Cut-off Grade	Model Grade Bin
South Oxide	0.22	0.20
South Transitional	0.19	0.15
South Fresh (Hanging Wall)	0.26	0.25
South Fresh (Footwall)	0.24	0.20
North Oxide	0.22	0.20
North Transitional	0.21	0.20
North Fresh	0.31	0.30

15.4 Pit Optimisation Results

15.4.1 Methodology and Software

Pit optimisations were undertaken in the Deswik software (V2021.1) using the Pseudoflow command. The Pseudoflow command follows the same general principle as the Lerchs Grossman algorithm and determines a series of incremental pit shells representing the break-even points based on the given block model, slope, cost and recovery data at varying revenue factors.

15.4.2 Optimisation Results and Pit Shell Selection

The optimisation process results in a series of nested shells, representing the economic material at each revenue factor. Given the reasonably standard results (shown in Figure 15-2), the Revenue Factor 1 shell was chosen for the basis of further work.

The chosen pit shell contained 157.9 million tonnes of ore at an average gold grade of 0.67 g/t, with 135.7 million tonnes of waste, producing a strip ratio of 0.86:1 (waste:ore). The shell is clearly divided into two pits with the South Pit being the larger of the two pits. The South Pit shell contained 156.0 million tonnes of ore at a grade of 0.67 g/t while the North Pit shell contained 1.9 million tonnes of ore at a grade of 0.51 g/t.

Figure 15-3 shows the chosen optimisation shell (RF=1) along with the solid identifying the footwall of the deposit. It can be seen that the east wall follows this contact and is considerably shallower than the west wall. This is due to the shell following the dip of the orebody and recovering ore in the east wall, while the west wall is contained entirely within waste rock.

Figure 15-2 Results of Koné Optimisation Study

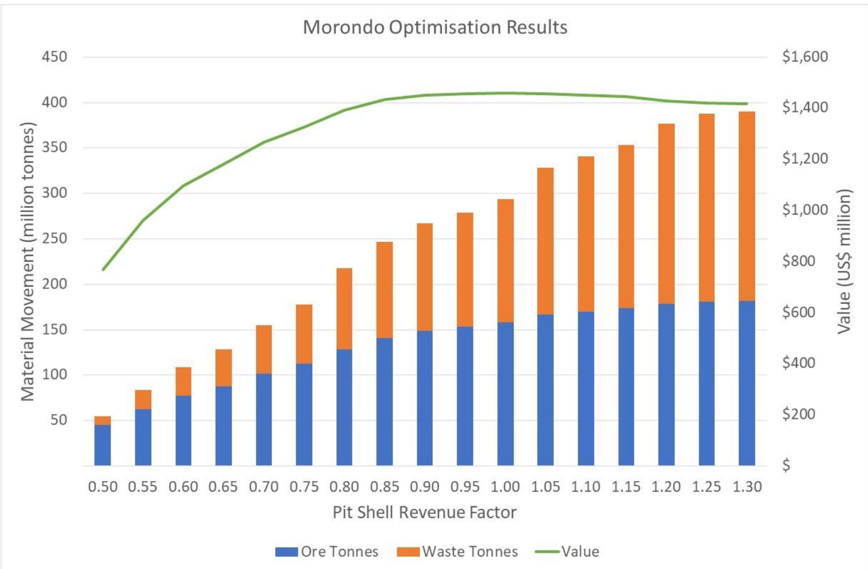
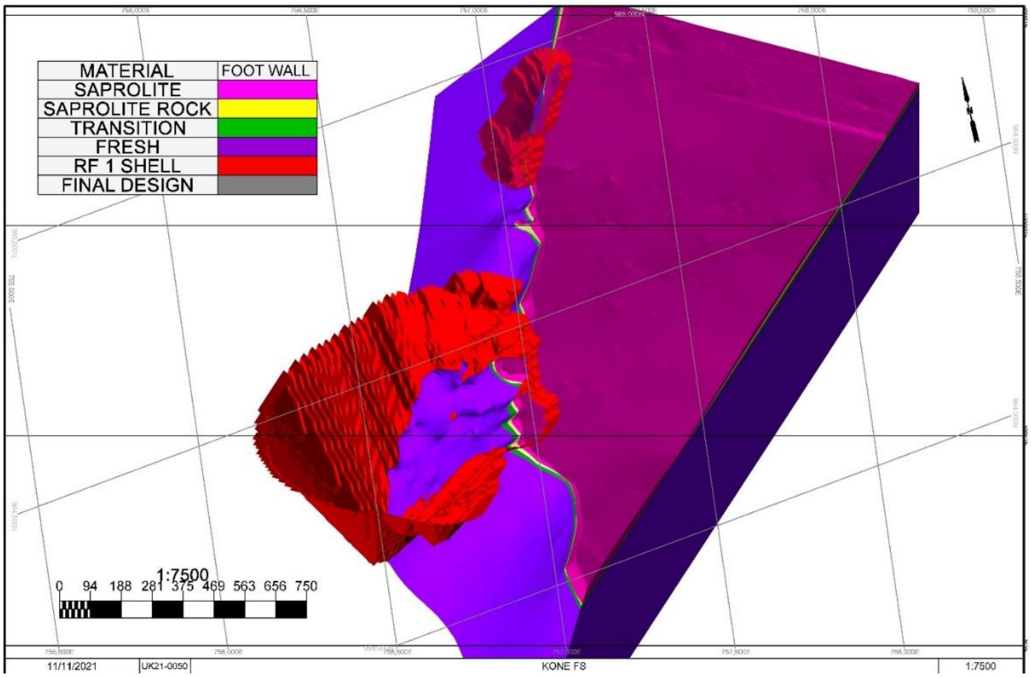


Figure 15-3 RF 1 Optimisation Shell and Footwall Oxidation State Solid



15.5 Mine Design

It should be noted that references to specific make or model of equipment are not recommendations, but were simply used to aid in design, scheduling and calculation of equipment requirements. The mining contractor may choose equipment from a different manufacturer or able to efficiently operate equipment of a different capacity.

15.5.1 Pit Development Strategy

The production strategy was developed in previous studies and refined during the Feasibility Study. The general approach is one utilising an elevated cut-off grade by mining at a higher rate than required to feed ore to the processing plant. The low-grade material is stockpiled while preferentially treating high-grade material and is processed once mining is completed. A cut-off grade of 0.5 g/t was determined to provide the best NPV and was chosen as the boundary between low-grade and high-grade.

In order to achieve this strategy, a mining rate of 35 Mtpa was used while the processing rate was capped at 11 Mtpa. This is discussed further in Section 16.

The Koné deposit can be divided into two pits. The South Pit is the larger of the two and contains the vast majority of the ore and total material (98.7% of total material and 98.8% of the ore). The North Pit can be mined in one stage while the South Pit is mined as three separate but overlapping cutbacks. At the end of mining in the North Pit, the waste from that pit is rehandled back into the pit and then supplemented with waste from the South Pit so that the North Pit is completely rehabilitated by the end of the mine life.

15.5.2 Pit Design Criteria

The geotechnical constraints for the design were based upon the main oxidation state of the material. Table 15-10 shows the geotechnical inputs for the final pit design.

Table 15-10 Modified Geotechnical Inputs to Pit Design

Geotechnical Zone	***RL's***	***Inter-ramp Slope Angle***	***Berm Width***	***Batter Angle***	***Bench Height-Designed***	***Benches per Stack***	***Geotechnical Berm***	***Stack Height***	***Overall Angle with Ramps & Geotechnical Berms***
		(°)	**(m)**	**(°)**	**(m)**		**(m)**	**(m)**	**(°)**
Oxide	Above 360	39°	3.5	60	5	1	6	10	24.6
Transition	between 340-360	58°	9	80	10	2	9	40	33.3
Fresh Rock FW	between -90 + 340	69°	6	80	10	3	*Modified Berm Width	210	40.2
Fresh Rock HW	between -90 + 340	69°	8	80	10	3	*Modified Berm Width	210	40.2

A minimum mining width of 40 metres was maintained in all cutbacks, except for the "goodbye cuts" at the base of the cutbacks, which were reduced to 30 metres.

15.5.3 Haul Road Design

The haul road design parameters used in the pit designs are summarised in Table 15-11 and are based upon the manufacture's standards and best practice operating procedures. The equipment selection process nominated CAT 785 trucks, with an operating width of 6.7m, as the preferred unit and this was used for all the productivity and haulage calculation.

Table 15-11 Haul Road Design

Parameter	Units	*Value*
Ramp Gradient	%	10%
Gradient style		Constant
Ramp Width (double lane)	m	32
Ramp Width (single lane)	m	18
Turning Radius	m	16

It is assumed that the safety windrow will be at least half of the height of the largest tyre in use on the road, therefore the safety windrow should be no less than 1.45m in height.

A distance of 0.5 metres has been left between the outside edge of the safety windrow and the edge of the ramp. This provides some protection from unravelling and bench scale failures. Road shoulders and a highwall drain have also been included in the road width calculations.

15.5.4 Waste Rock Dump Design Criteria

Waste dumps, stockpiles and a ROM pad have been designed with sufficient capacity to hold the material planned to come from the Koné open pits. The design parameters used for these designs can be seen in Table 15-12.

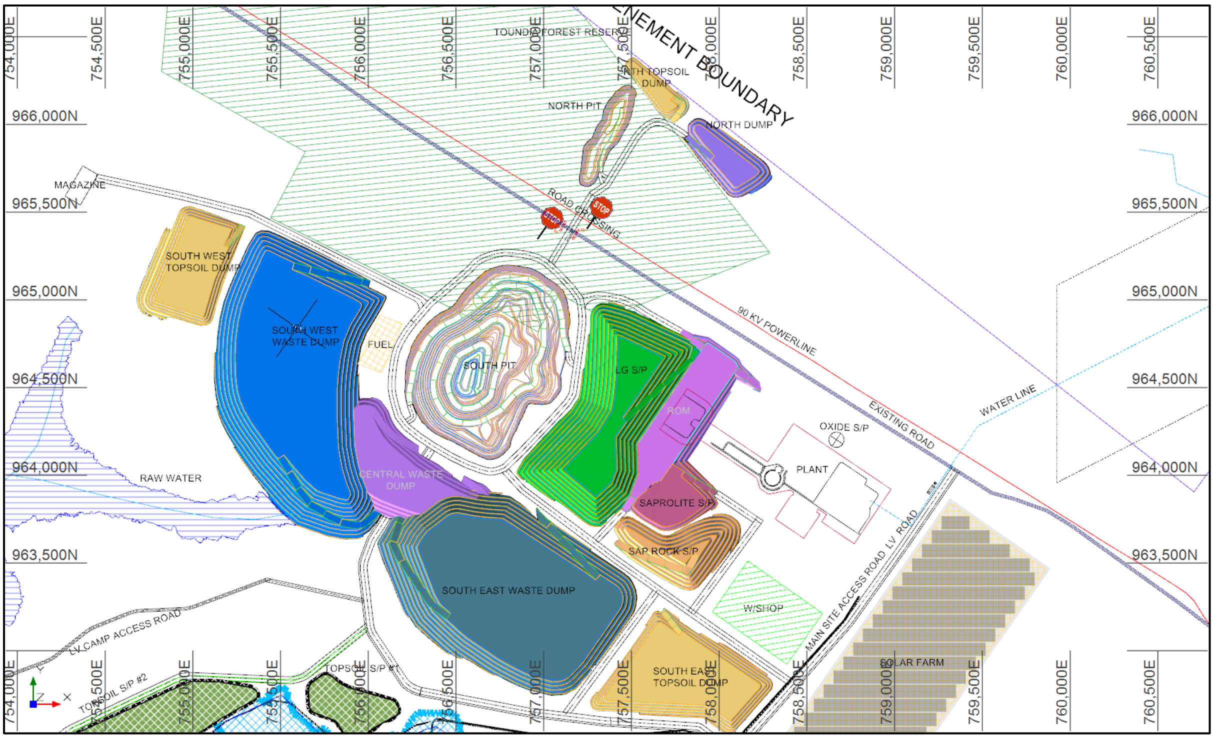
Table 15-12 Waste dump and stockpile design parameters

Parameter	Value
Lift height (m)	10
Lift angle (°)	37
Berm width (m)	10
Ramp width (m)	40
Ramp gradient (%)	10

There are six areas that are to be used to stockpile waste material: the construction of the ROM Pad, the Southwest dump, Southeast dump, Central dump, North dump and backfilling of the of the North pit in the latter years. These are shown in Figure 15 4.

The North and South Pits contain a total waste tonnage of 145.7 million tonnes, The Central Waste Dump will be the last dump to be constructed and will take waste from the lower levels of the South Pit. This will assist in balancing truck requirements towards the end of the mine life. Once mining of the South Pit is completed, tailings will be discharged into the pit void and the Central Waste Dump will be reclaimed for TSF capping and rehabilitation. Similarly, the North Dump will be reclaimed to backfill the North Pit once mining is completed in that pit.

Figure 15-4 Waste Dump Locations



15.6 Pit Stage Designs

The Koné deposits will be mined by two open pits, with the bulk of the mill feed contained within the South Pit and the smaller North Pit contributing mainly oxide and transitional ore. The pit inventory contains a total of 161.1 million tonnes of mill feed at a grade of 0.66 g/t Au. This is associated with 145.7 million tonnes of waste rock, providing a strip ratio of 0.90:1 (waste:ore).

The pit inventory by deposit is summarised in Table 15-13. This shows that 10.8 million tonnes of ore is oxide, 8.3 million tonnes is transitional and the remaining 142.1 million tonnes of material is fresh rock. Nearly 90% of the overall material is fresh rock. To best accommodate this, the excavators and trucks will be purchased and configured for fresh rock mining. A simple bucket change for the excavators will allow for shorter loading times within the oxide material.

The final engineered pit designs for the North and South Pits are shown in Figure 15-5. Overall, the design converted 102 % of the ore tonnes with 101 % of the total ounces, at the cost of an additional 7% of waste tonnes.

Table 15-13 Koné Pit Inventory

Material	*Units*	South Pit	North Pit	Total Pit Inventory
Saprolite Oxide Millfeed Tonnes	*Mt*	6.8	0.8	7.6
Saprolite Oxide Millfeed Grade	*g/t*	0.58	0.48	0.57
Saprolite Rock Millfeed Tonnes	*Mt*	3.0	0.1	3.2
Saprolite Rock Millfeed Grade	*g/t*	0.55	0.40	0.55
Transitional Millfeed Tonnes	*Mt*	7.9	0.4	8.3
Transitional Millfeed Grade	*g/t*	0.56	0.46	0.56
Fresh Footwall Millfeed Tonnes	*Mt*	17.3	0.2	17.5
Fresh Footwall Millfeed Grade	*g/t*	0.55	0.61	0.56
Fresh Hanging Wall Millfeed Tonnes	*Mt*	124.1	0.5	124.6
Fresh Hanging Wall Millfeed Grade	*g/t*	0.69	0.56	0.69
Total Fresh Millfeed Tonnes	*Mt*	141.4	0.6	142.1
Total Fresh Millfeed Grade	*g/t*	0.68	0.57	0.67
Total Millfeed Tonnes	***Mt***	**159.1**	**1.9**	**161.1**
Total Millfeed Grade	***g/t***	**0.66**	**0.50**	**0.66**
Oxide Waste Tonnes	*Mt*	17.6	1.4	19.0
Transitional Waste Tonnes	*Mt*	9.3	0.3	9.6
Fresh Footwall Waste Tonnes	*Mt*	4.6	0.2	4.8
Fresh Hanging Wall Waste Tonnes	*Mt*	112.1	0.3	112.3
Total Fresh Waste Tonnes	*Mt*	116.6	2.1	118.8
Total Waste Tonnes	***Mt***	**143.5**	**2.1**	**145.7**
Total Tonnes	***Mt***	**302.7**	**4.1**	**306.7**
Strip Ratio	*t waste : t ore*	0.90:1	1.09:1	0.90:1

The South Pit has been split into three cutbacks based upon the outputs from the NPV optimisation scheduling software Deswik.GO. The minimum mining width of 40 metres (excluding the final goodbye cuts at 30m), the double width ramp width of 32m, geotechnical berms at the required stack height or widening of catch berms, and the exit ramps designed to reduce the overall haulage have all contributed to the design of the cutbacks.

As part of the elevated cut-off grade methodology, the requirement of the initial two cutbacks is to provide sufficient feed of greater than 0.5 g/t gold to the processing plant within the early years of the project. The designs for the southern cutbacks are presented in Figure 15-6 for Cutback 1 and Figure 15-7 for Cutback 2. The third cutback is represented by the final pit design. The strip ratio for Cutback 1 is 0.32:1 with an average grade of 0.76 g/t Au. Cutback 2 has a higher strip ratio with 1.05:1 but still has an above average grade of 0.69 g/t Au. The final southern cutback has a strip ratio of 1.25:1 and produces a feed grade of 0.60 g/t Au. The cutbacks align with the elevated cut-off grade strategy by allowing access to the higher-grade ore in the first 2-3 years of the project.

Figure 15-5 North and South Pit Final Designs

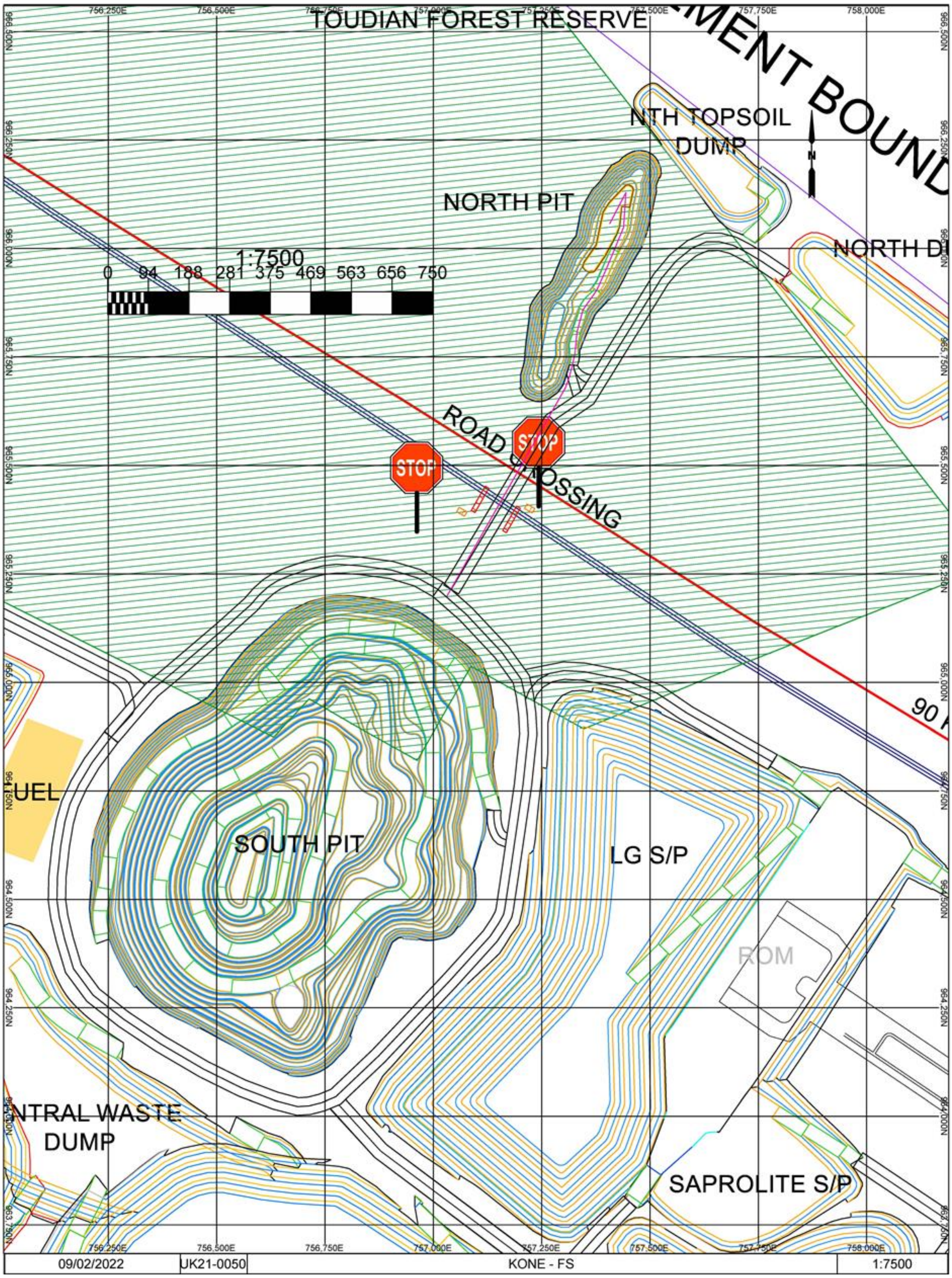


Figure 15-6 South Pit Cutback 1 Engineered Design

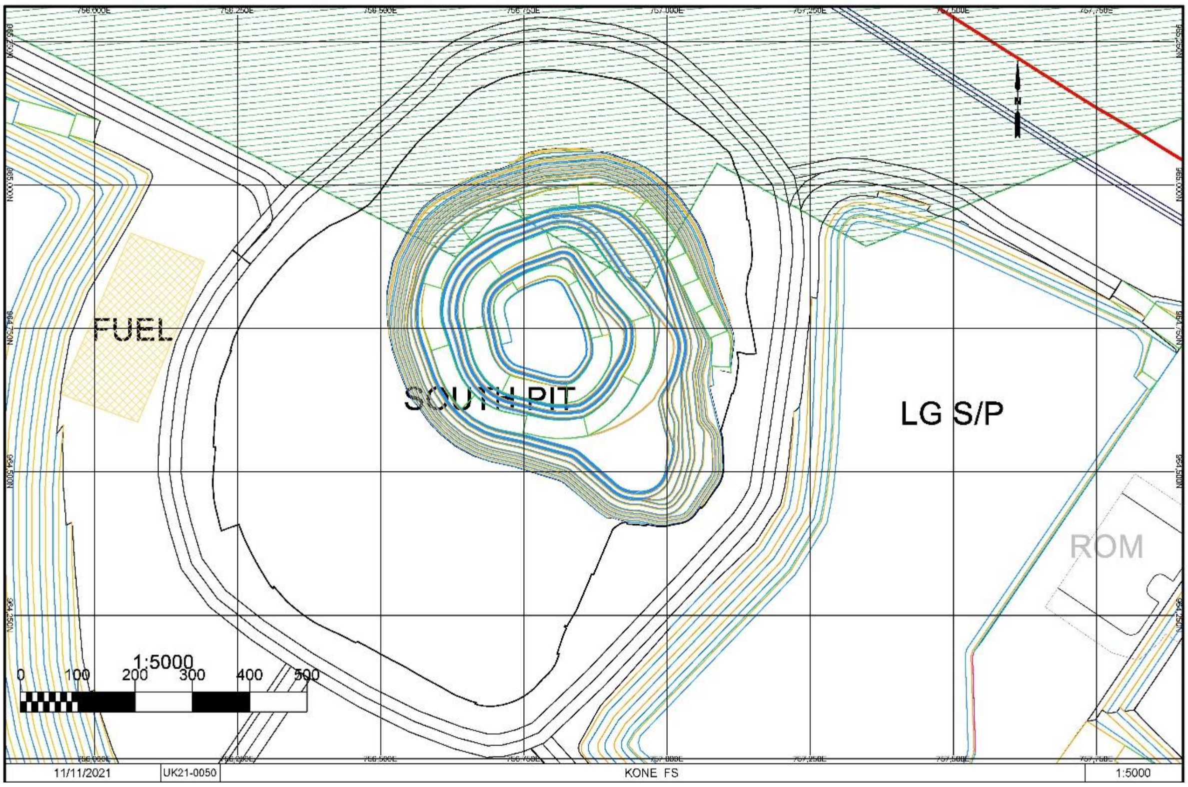
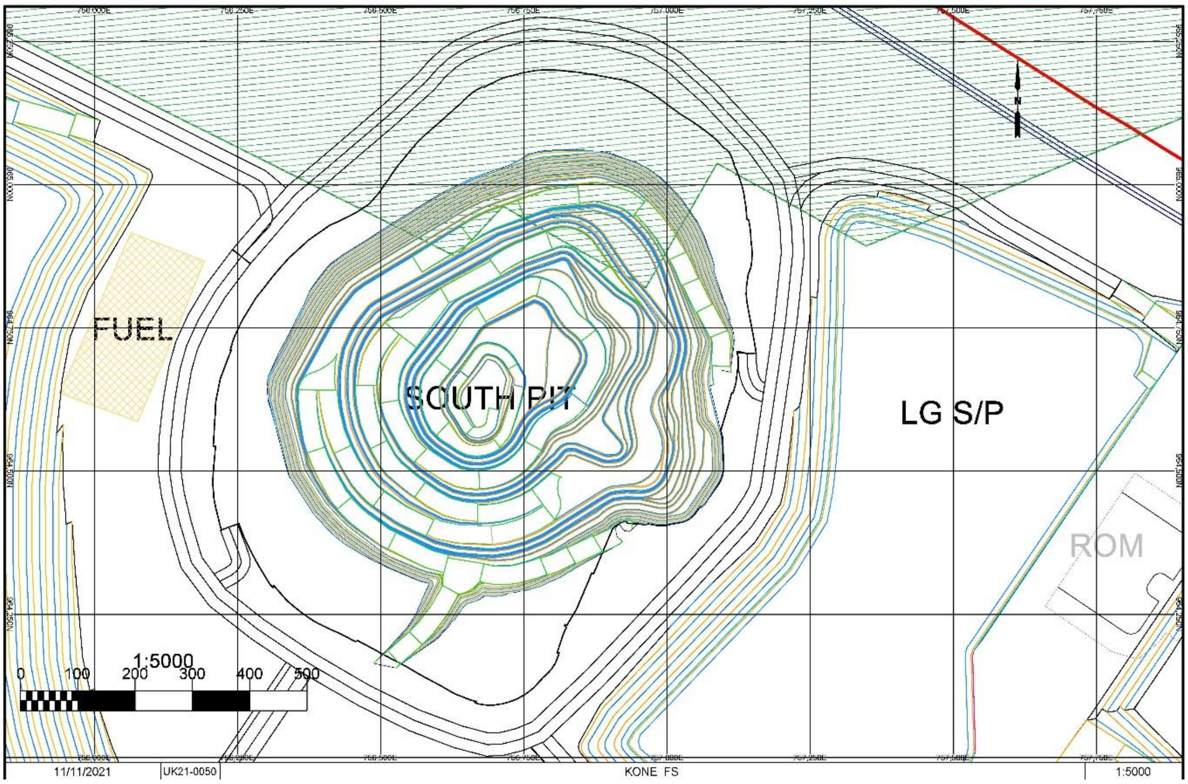


Figure 15-7 South Pit Cutback 2 Engineered Design



15.7 Risk Assessment

A risk assessment for the development stage of the Project identified risks related to the investigations, data quality, estimations, planning and decision making.

Table 15-14 Project Development Risk Assessment

Risk	Likelihood of Occurrence	Potential Severity of Impact	Risk Ranking
Over-estimation of mineral resources	Possible	Minor	Medium
Under-estimation of mineral resources	Possible	Minor	Medium
Under-estimation of mining costs for reserve estimate	Possible	Minor	Medium
Under-estimation of processing & administration costs	Possible	Minor	Medium
Over-estimation of metallurgical performance for reserve estimate	Unlikely	Minor	Low
Over-estimation of gold price for reserve estimate	Unlikely	Moderate	Medium
Over-estimate of ore reserves	Unlikely	Moderate	Medium
Pit slopes are too steep	Possible	Minor	Medium
Feasibility Design is not fully achievable	Unlikely	Minor	Low
Unsuitable mining method	Unlikely	Minor	Low
Production schedule is not achievable	Unlikely	Minor	Low

The pit limits are not significantly sensitive to change in study parameters, however the design has allowed for a buffer between the pit limits and any permanent infrastructure or waste rock dumps.

The gold price used for the mineral reserve calculations is conservative given current pricing and the optimisation parameters are robust enough to withstand a considerable decrease in gold price from 2021 levels while still remaining economic.

Geotechnical studies have indicated that, as long as groundwater is managed, the pit slopes developed are reasonable for this project. Dewatering plans have been developed consisting of both pre-operational and operational dewatering bores. Sumps have also been planned within the pit during operations to further manage surface and groundwater.

16 MINING METHODS

16.1 Mining Method

16.1.1 General Approach

The report "Koné Gold Project Feasibility Study Mining Report (UK21-0050)" produced by Carci contains a detailed discussion on the mining methods anticipated for this project.

Based on the geometry and physical properties of the orebody, the proximity of the orebody to surface and the gold grades contained within the orebody, the most appropriate mining method is conventional open pit mining. Montage has identified that its preference is to utilise a mining contractor for the mining operations, primarily due to the short life of the mining activity and capital management strategy.

The open pit mine life is 9.5 years, including one year of preproduction mining. The mining strategy utilises an elevated cut-off grade approach, where ore with a gold grade over 0.5 g/t is preferentially processed while ore with a gold grade over economic cut-off but under 0.5 g/t is considered low-grade material and stockpiled or used as supplementary feed. This strategy is in place for the first 9 years of operation. Once the pits are exhausted and all high-grade ore has been processed, the low-grade ore is reclaimed from stockpiles and processed for an additional 6 years.

Previous studies identified the most optimal mining and processing combination as a 35 Mtpa mining rate coupled with an 11 Mtpa processing rate. The blending algorithm in the Deswik scheduling software was used to set specific targets in terms of total feed tonnes and set material limits, incorporating all costs and revenues used for the determination of the mineral reserve estimate. The algorithm also provides the optimal NPV for the material available at a point in time. The software can also look forward, ensuring that blending of material in later years is not compromised by the immediate period targets.

16.1.2 Shifts and personnel

Mining will occur at the South Pit 24 hours a day utilising several shifts. The number of shifts per 24-hour period is yet to be determined but will either be two or three. Mining in the North Pit will occur for three months of the year during the dry season and on night shift only, minimises the interaction of haul trucks crossing the public road. Rehandle at the North Pit will be minimised as much as possible.

Montage will employ its own mining technical services team to oversee the contract mining operations and to provide technical guidance to the contractor. The owner's team will consist of mine management and technical staff including:

- A Mine Manager and Alternate Mine Manager
- Geologists and field assistants (resource and grade control)
- Mining Engineers (scheduling will be undertaken by the principal to a monthly level)
- Geotechnical and Hydrogeological staff

- Surveyors
- Contract management personnel (including supervisors)
- Health and Safety personnel.

The contractor will provide its own health and safety advisors, but these will be complimented by the Montage Health and Safety team, of which there will be at least one member dedicated to the mining activities. A Mines Rescue capability will be established utilising personnel from all departments and contractors.

Both Montage and contractor labour will be sourced from the local communities in the first instance, followed by the region, country and then internationally, depending on job requirements and skills available.

16.1.3 Mining Equipment

It should be noted that references to specific make or model of equipment are not recommendations, but were simply used to aid in design, scheduling and calculation of equipment requirements. The mining contractor may choose equipment from a different manufacturer or able to operate equipment of a different capacity.

The equipment selection was determined for the Feasibility Study based upon the work completed during the PEA and updated for the revised optimised pits as discussed in the Mine Optimisation Section. The project location also influenced the equipment selection as it is recognised that support for larger mining equipment (200t+ capacity trucks etc.) is limited in West Africa.

A Caterpillar 785C Widebody truck was selected based upon the optimal density of the fresh rock material matched to a body dimension that also allowed for a full load of oxide and transition material. A suitable tray can be selected to optimise the payload for all materials within the deposit.

The Caterpillar 6040 excavator was matched to the CAT 785 haul trucks for the purposes of this study.

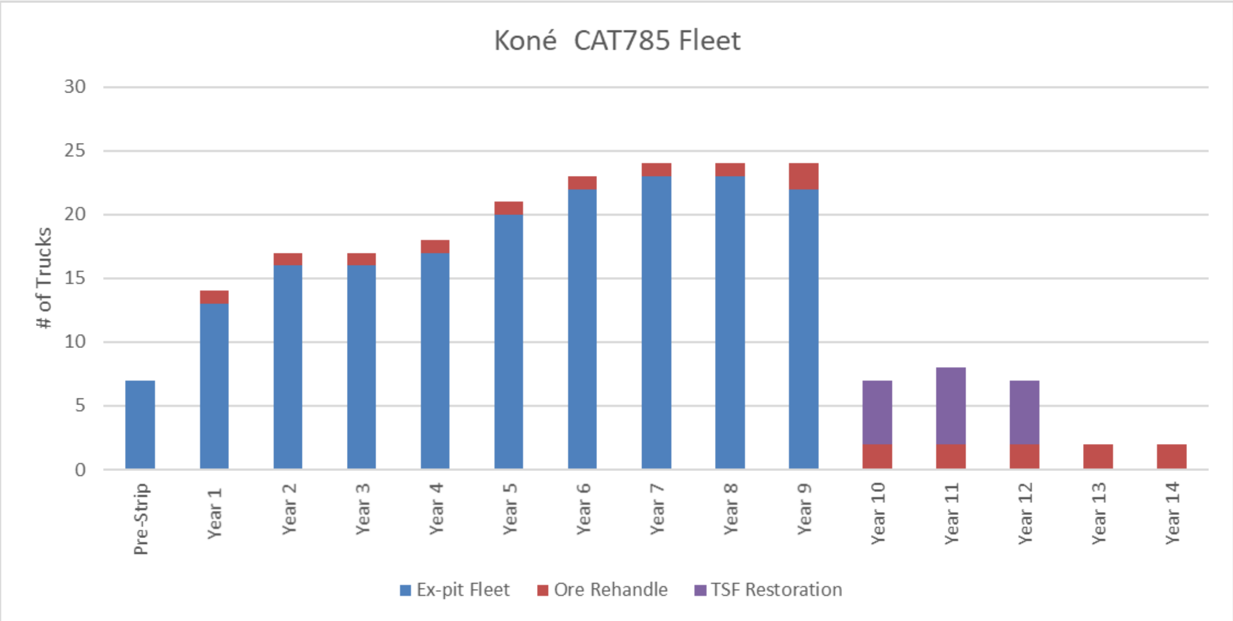
Two CAT 6040 excavators are required in the pre-production year, with the third coming online in Year 1. This gives an overall capacity of 40.8 million tonnes per year in hard rock material (which accounts for 85% of the material within the schedule). This means on average that the number of excavators utilised during a year is 2.56. This availability of equipment allows the North Pit to be campaigned on night shift only. This pit is set to mine 1 Mtpa over four years, with the active mining time set to a 3-month window during the dry season.

The total truck requirement comprises the ex-pit trucks and ore rehandle trucks (with the waste rehandle truck requirement being the repurposed ex-pit fleet). The fleet requirement increases annually from the start of the project to a peak of 24 trucks in Year 8 (Table 16-1). The open pit mining finishes in July of Year 9, however a full fleet is required up until that point. The waste rehandle for the tailings closure and the North Pit backfill utilises the retired mining fleet from August Year 9 onwards (Figure 16-1). Following the completion of mining in year 9, tailings will be deposited into the South Pit.

Table 16-1 Total CAT 785 fleet requirements

	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14
Ex-pit Fleet	7	13	16	16	17	20	22	23	23	23	0	0	0	0	0
Ore Rehandle	0	1	1	1	1	1	1	1	1	1	2	2	2	2	2
Waste Rehandle (North Pit and TSF)	0	0	0	0	0	0	0	0	0	4	5	4	5	0	0
Total CAT785	**5**	**12**	**16**	**17**	**17**	**19**	**22**	**24**	**24**	**24**	**7**	**7**	**7**	**2**	**2**

Figure 16-1 Total CAT 785 Fleet Requirements



Based on the mining schedule and Contractor tender responses, it is expected that the following ancillary equipment will be required during the primary mining operations (Table 16-2).

Table 16-2 Ancillary Equipment Requirements

	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
CAT 980 Wheel Loader	1	1	1	1	1	1	1	1	1	1
CAT D10 Track Dozer	3	3	3	3	3	3	3	3	3	3
CAT D9 Track Dozer	5	5	5	5	5	5	5	5	5	5
CAT 777E Water Cart	3	3	3	3	3	3	3	3	3	3
CAT 16M Grader	3	3	3	3	3	3	3	3	3	3
Sandvik DI650i Blast Hole Rig	5	5	5	5	5	5	5	5	5	5
Sandvik DP1500i Presplit Rig	1	1	1	1	1	1	1	1	1	1
CAT 336 Ex Rock Breaker	2	2	2	2	2	2	2	2	2	2
CAT CS78B Compactor	1	1	1	1	1	1	1	1	1	1
CAT 432F Backhoe	1	1	1	1	1	1	1	1	1	1
CAT IT38 Tool Carrier	1	1	1	1	1	1	1	1	1	1
CAT 980 Tyre Handler	1	1	1	1	1	1	1	1	1	1
CAT 745 ADT – Service Truck	2	2	2	2	2	2	2	2	2	2
CAT 745 ADT – Fuel Truck	2	2	2	2	2	2	2	2	2	2
CAT 745 ADT – Wash Truck	2	2	2	2	2	2	2	2	2	2
Lighting Plants	13	13	13	13	13	13	13	13	13	13
Dewatering Pump	1	1	2	2	2	2	2	2	2	1
Welding Truck	1	1	1	1	1	1	1	1	1	1
Rough Terrain Mobile Crane	1	1	1	1	1	1	1	1	1	1
Explosives Truck – MMU	2	2	2	2	2	2	2	2	2	2
Shotfirer's Truck – Accessories	1	1	1	1	1	1	1	1	1	1
Light Duty Vehicles	37	37	37	37	37	37	37	37	37	37
Shift Change Vehicles	6	6	6	6	6	6	6	6	6	6

16.1.4 Grade Control

It is expected that grade control of the mining benches will be conducted by RC drilling ahead of the mining front and covering a minimum of three benches, so as to assist with short to medium term planning activities.

Grade control activities will be conducted by a contractor but overseen by the Owner. Assaying of samples will be performed by the Owner in the onsite laboratory attached to the processing plant.

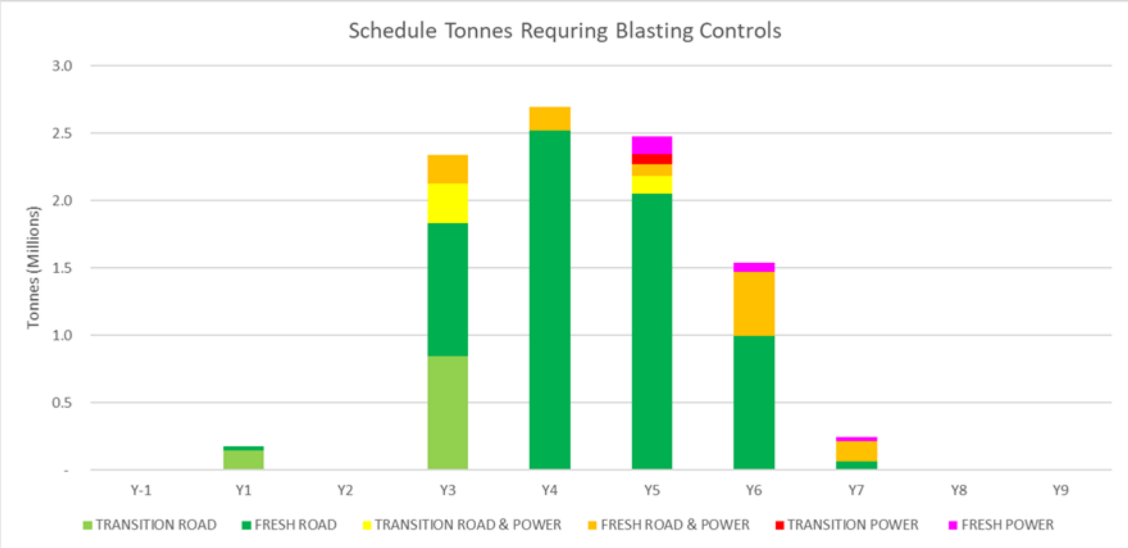
16.2 Drilling and Blasting

The production drilling and blasting will be carried out by the mining contractor on 10m benches in most situations. It is expected that most drill holes will be 165mm in diameter, although buffer rows may be at a smaller diameter. Presplit will also be undertaken on all final walls.

Blasting may have an impact on the local community which has had no exposure to any mining operations within the vicinity. The most significant impacts would be dust, vibration, noise and closure of the national highway. A program of education as well as mine site controls will need to be in place to mitigate the impact on the local community.

Both the location of the National Highway and the powerline will have an impact on blasting practices, however there are limited tonnes of material within the 500 m buffer zone for each of these pieces of infrastructure. Figure 16-2 shows the mined tonnes by year contained within the buffer zone. This represents less then 3% of the total tonnes within the project.

Figure 16-2 Scheduled tonnes within highway/powerline caution zone



Controls will be put in place for blasting within a 500 m radius of powerlines to ensure that the blasting vibrations won't affect the powerlines. The blasting vibrations can be reduced to within acceptable levels by using programmable electronic detonators to control the output vibrations and, when combined with a process of monitoring, the resulting blasts will achieve a good level of control.

A study will be undertaken prior to blasting within the 500m zone to assess how the vibrations act within the local rock. The use of adequate and correctly size stemming and suitable Quality Control practices will also be required to reduce any potential for fly rock that could damage a powerline or highway.

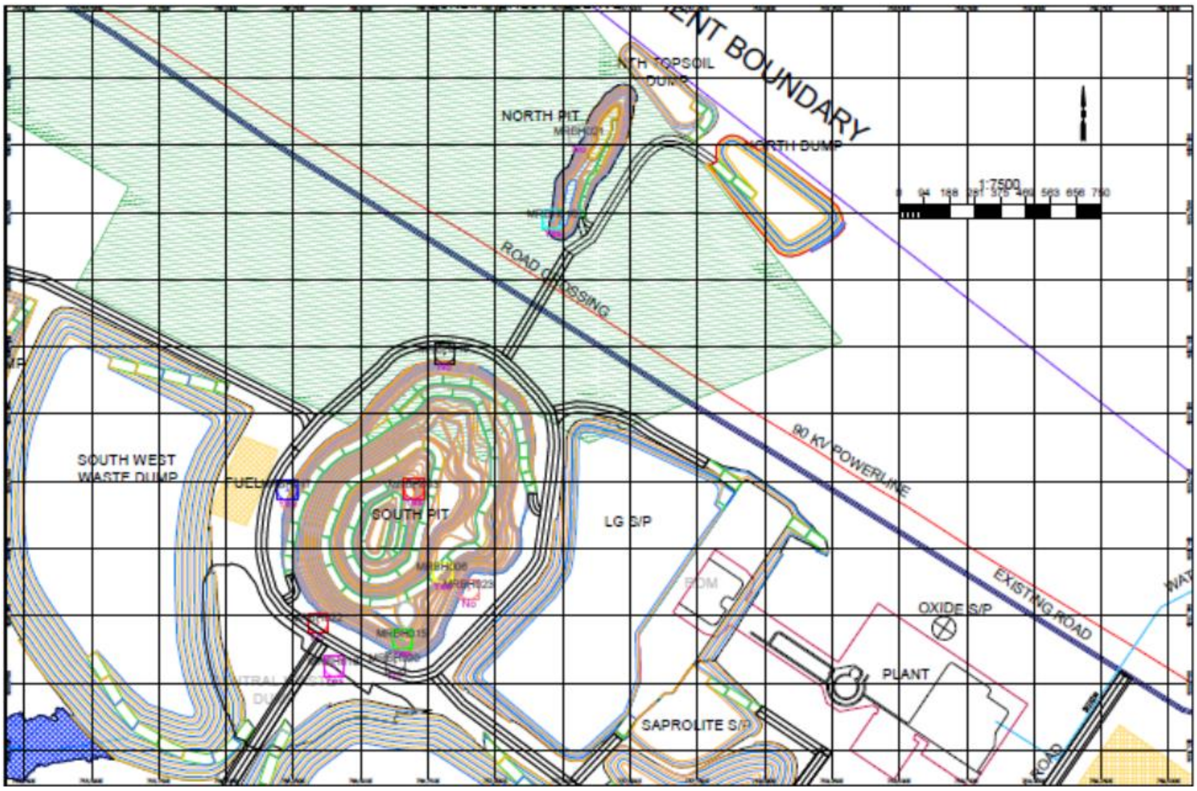
16.3 Pit Dewatering

Australian Groundwater and Environmental Consultants ("AGE") conducted a Hydrogeological and dewatering study on the Koné deposits. As part of this study, AGE proposed a pit dewatering concept that consisted of two stages. The first was a pre-mining stage, using in-pit and out of pit dewatering boreholes. These boreholes would target the high yielding areas identified by the resource definition drilling. This would provide additional time for drainage of the less permeable geology units.

The second stage involves operational dewatering through conventional dewatering methods. These methods include strategically positioned collection sumps. Water collected in these sumps will be pumped to the water storage facility for later use. AGE has recommended a minimum in-pit pump capacity of approximately 8,000 m^3/day.

Dewatering boreholes will also be established around the pit limits to be used during operations (Figure 16-3). Seven dewatering bores will be operated and will extract approximately 1,700 m^3/day. While some of these holes will be destroyed as the pit expands, the remainder will continue to operate, in conjunction with the sumps, and be sufficient to meet pit dewatering requirements.

Figure 16-3 Mine site dewatering borehole plan



16.4 Mine Production Schedule

Earlier studies had identified that an elevated cut-off grade utilising an accelerated mining schedule with stockpiling and subsequent rehandling and processing of low-grade material provided the best value for the project. The highest value was obtained through a combination of a 35 Mtpa mining rate option with an 11 Mtpa mill throughput. This combination was used for this study.

The production schedule considers both the mining schedule and the blending, stockpiling and rehandle schedule required to produce the optimal blend of ore to the processing plant to maximise the value of the project. The mining schedule, blending schedule and haulage/dumping schedule were all produced in the Deswik software. Table 16-3 shows the mining schedule at an annualised rate of 35 Mtpa while Table 16-4 shows the processing schedule at an annualised rate of 11 Mtpa.

Table 16-3 Koné Mining Schedule

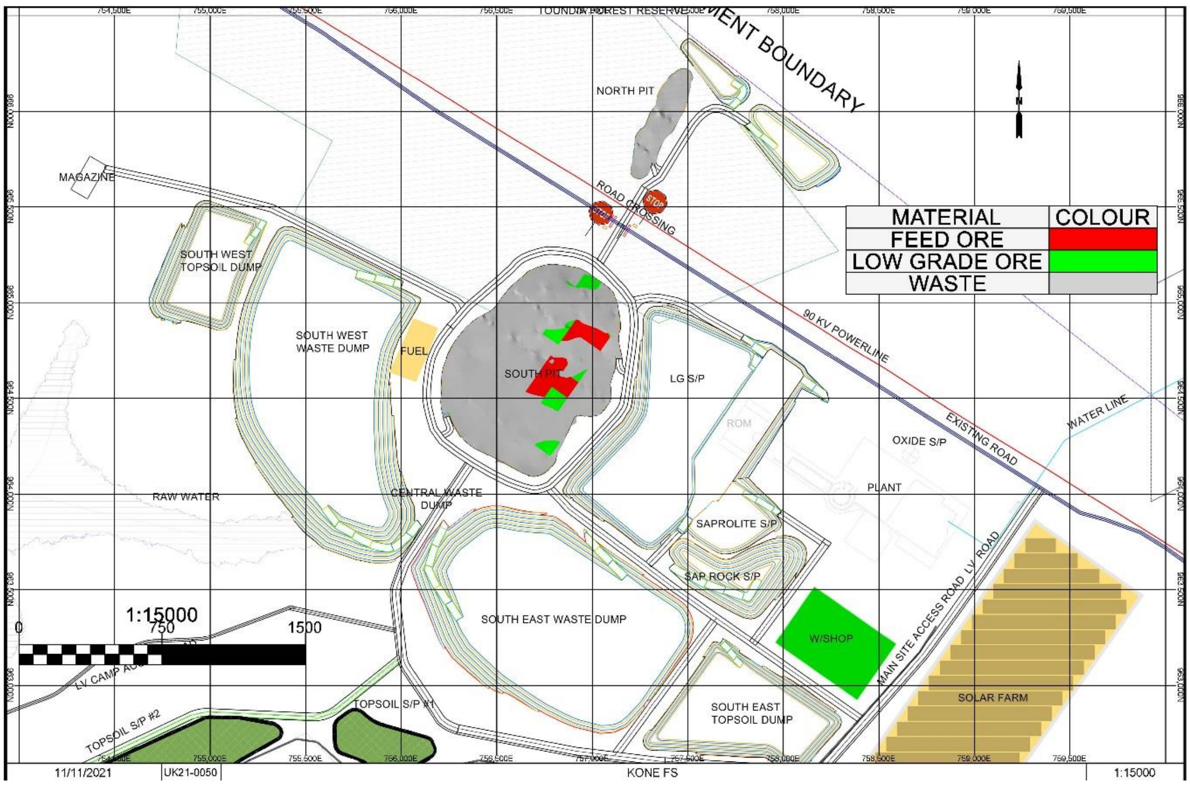
Description	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Total
Sth Oxide Tonnes	'000 t	6,453	2,217		1,137							9,807
Sth Oxide Grade	Au g/t	0.62	0.57		0.35							0.573
Sth Trans Tonnes	'000 t	228	5,859	18	981	808						7,894
Sth Trans Grade	Au g/t	0.46	0.62	0.30	0.42	0.34						0.562
Sth Fresh FW Tonnes	'000 t		1,144	2,616	1,985	2,032	2,491	2,203	1,808	2,372	684	17,337
Sth Fresh FW Grade	Au g/t		0.58	0.55	0.64	0.58	0.55	0.53	0.56	0.50	0.49	0.555
Sth Fresh HW Tonnes	'000 t		9,589	20,319	13,394	10,523	16,284	12,893	11,154	18,966	10,986	124,107
Sth Fresh HW Grade	Au g/t		0.71	0.75	0.73	0.65	0.66	0.67	0.55	0.66	0.83	0.692
Total Sth Tonnes	'000 t	6,681	18,809	22,953	17,497	13,364	18,775	15,096	12,692	21,339	11,670	159,145
Total Sth Grade	Au g/t	0.61	0.66	0.73	0.68	0.62	0.64	0.65	0.55	0.64	0.81	0.663
Nth Oxide Tonnes	'000 t				245	407	265					917
Nth Oxide Grade	Au g/t				0.49	0.46	0.45					0.47
Nth Trans Tonnes	'000 t						212	175				387
Nth Trans Grade	Au g/t						0.5	0.42				0.46
Nth Fresh Tonnes	'000 t						65	431	148			643
Nth Fresh Grade	Au g/t						0.66	0.56	0.56			0.57
Total Nth Tonnes	'000 t				245	407	541	606	178			1,947
Total Nth Grade	Au g/t				0.49	0.46	0.50	0.52	0.56			0.50
Total Ore Tonnes	'000 t	6,681	18,809	22,953	17,743	13,771	19,316	15,702	13,110	21,339	11,670	161,092
Total Ore Grade	Au g/t	0.61	0.66	0.73	0.68	0.62	0.64	0.65	0.55	0.64	0.81	0.66
Total Waste Tonnes	'000 t	11,073	16,187	12,047	17,053	21,221	15,684	19,298	21,890	9,201	1,996	145,650
Total Mined Tonnes	'000 t	17,755	34,995	35,000	34,796	34,991	35,000	35,000	35,000	30,540	13,666	306,742
Global Strip Ratio	W:O	1.66:1	0.86:1	0.52:1	0.96:1	1.54:1	0.81:1	1.23:1	1.67:1	0.43:1	0.17:1	0.90:1

Table 16-4 Koné Processing Schedule

Description	Units	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Total
Sth Oxide Tonnes	'000 t	630	733	732	692	438	216	679	733	733	733	733	733	733	733	560	9,807
Sth Oxide Grade	Au g/t	1.04	0.96	0.94	0.95	0.94	0.35	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.57
Sth Trans Tonnes	'000 t	2699	-	220	97	-	-	-	-	-	-	-	-	-	-	4879	7,894
Sth Trans Grade	Au g/t	0.91	-	0.82	0.75	-	-	-	-	-	-	-	-	-	-	0.35	0.56
Sth Fresh FW Tonnes	'000 t	414	273	1150	1710	1194	990	1647	641	454	-	-	-	7157	1708	-	17,337
Sth Fresh FW Grade	Au g/t	0.87	0.95	0.78	0.72	0.71	0.69	0.58	0.66	0.56	-	-	-	0.42	0.41	-	0.55
Sth Fresh HW Tonnes	'000 t	5,717	9,994	8,898	8,461	8,950	8,973	8,497	9,627	9,813	10,227	10,267	10,267	2,672	8,560	3,183	124,107
Sth Fresh HW Grade	Au g/t	0.89	1.00	0.89	0.77	0.84	0.79	0.60	0.86	0.88	0.44	0.44	0.44	0.44	0.43	0.43	0.69
Total Sth Tonnes	'000 t	9,460	11,000	11,000	10,959	10,582	10,178	10,823	11,000	11,000	10,960	11,000	11,000	10,562	11,000	8,621	159,145
Total Sth Grade	Au g/t	0.91	0.99	0.88	0.78	0.83	0.77	0.59	0.82	0.83	0.44	0.44	0.43	0.42	0.42	0.38	0.66
Nth Oxide Tonnes	'000 t	-	-	0.34	40.84	294.99	517.00	53.20	-	-	-	-	-	-	-	10.92	917
Nth Oxide Grade	Au g/t	-	-	0.80	0.68	0.46	0.45	0.45	-	-	-	-	-	-	-	0.32	0.47
Nth Trans Tonnes	'000 t	-	-	-	-	62	62	-	-	-	-	-	-	263	-	-	387
Nth Trans Grade	Au g/t	-	-	-	-	0.67	0.52	-	-	-	-	-	-	0.40	-	-	0.46
Nth Fresh Tonnes	'000 t	-	-	-	-	61.40	242.98	124.01	-	-	40.20	-	-	174.71	-	-	643
Nth Fresh Grade	Au g/t	-	-	-	-	0.67	0.61	0.58	-	-	0.51	-	-	0.48	-	-	0.57
Total Nth Tonnes	'000 t	-	-	0.34	41	418	822	177	-	-	40	-	-	438	-	11	1,947
Total Nth Grade	Au g/t	-	-	0.80	0.68	0.52	0.50	0.54	-	-	0.51	-	-	0.43	-	0.32	0.50
Total Tonnes	'000 t	9,460	11,000	11,000	11,000	11,000	11,000	11,000	11,000	11,000	11,000	11,000	11,000	11,000	11,000	8,632	161,092
Total Grade	Au g/t	0.91	0.99	0.88	0.78	0.82	0.75	0.58	0.82	0.83	0.44	0.44	0.43	0.42	0.42	0.38	0.66
Recovered Ounces	'000 oz	251	320	283	247	260	238	182	261	267	133	133	132	130	128	93	3,058
Global Recovery	%	91.2	91.3	90.7	90.0	90.3	89.7	88.2	90.3	90.5	85.8	85.8	85.7	86.9	85.8	88.4	89.3

Figure 16-4 shows the pit surfaces prior to the commencement of mining in Year -1. The shaded areas indicate the materials to be mined whereas the outlines indicate the designs of dumps, roads, stockpiles and ROM to be built. The mining blocks within the pits are coded by the material; red for high grade or Feed ore, green represents the low-grade ore, referred to as Stockpile ore, while the remaining grey colour is waste.

Figure 16-4 Starting Pit surfaces prior to mining pre-strip (Year -1)



At the end of the pre-strip year, the ROM pad has been constructed from waste material and the saprolite ore and sap-rock ore is stockpiled on individual stockpiles adjacent to the ROM. Roads and infrastructure pads for the workshop, plant and fuel farm are also constructed during this period. The pre-strip year starts 12 months prior to processing and waste is removed from all cutbacks. This allows for sufficient waste to be mined for the initial tailing storage facility civil requirements and for the construction of the ROM pad and skyway. The skyway is built around the crusher to provide the most optimal delivery of feed material to the ROM via direct tip. The main method of feeding the ore during the initial 9 years of mining is via direct tip from the mine haul trucks. As a majority of the sap-rock and saprolite material is mined within the first few years, this accumulates on the stockpiles. Due to the soft nature of this material, it is gradually fed into the processing plant over the life of the plant. This equates to 4.70% of total saprolite feed and 1.96% of sap-rock feed. These stockpiles are built within the first few years and reclaimed from over the next 14 years.

Year 1 is the first year at the full production rate of 35 million tonnes and requires three excavators and a fleet of 13 trucks. The processing plant is also commissioned at the start of the year with an overall annual feed rate of 9.9 million tonnes. The materials mined during this year are a combination of oxide, transitional and fresh rock as the cutbacks get deeper, which will be processed or stockpiled according to material type and grade. The total ore mined during this period is 18.8 million tonnes, with 8.1 million tonnes of high-grade ore at average grade of 0.96 g/t with the remaining 10.7 million tonnes being low-grade ore with an average grade of 0.43 g/t.

During Year 2, material is excavated from the southern cutbacks 1 and 2 only. The ore is mined from Cutback 1 and Cutback 2, producing a total of 23 million tonnes with a average grade of 0.73 g/t Au. The high-grade component of this is 12 million tonnes at 0.96 g/t Au. The remaining 10.9 million ore tonnes is low-grade with an average grade of 0.47 g/t Au and is sent to the low-grade stockpile. The total amount of material mined for the year is 35 million tonnes. The Southwest and Southeast waste rock dumps are partially active with approximately 50% of the surface area being dumped on.

The mining of the North Pit commences in Year 3 on a 3-month night shift campaign during the dry season with a total of 1 million tonnes moved. The topsoil will also need to be stripped from the North Pit and dump locations. The waste strip in the final South cutback also commences this year although some waste was previously extracted in the pre-strip year for civil construction purposes. A majority of the ore is mined from Cutback 1 and Cutback 2 with 7.7 million tonnes of high-grade ore at an average grade of 0.99 g/t, supplemented with 10 million tonnes of low-grade ore with an average grade of 0.44 g/t being delivered to the processing plant and stockpiles. The southwest, southeast and northern waste dumps are active.

During Year 4, material is mined from all cutbacks. The northern pit is active for a 3-month nightshift campaign with a total of 1 million tonnes moved. The ore within this pit is mostly low grade with 0.4 million tonnes at an average head grade of 0.46 g/t being mined. The South Pit has high grade tonnes being mined from Cutback 1 and Cutback 2, generating 5.2 million ore tonnes at an average grade of 0.93 g/t, which will be supplemented with some of the 8.6 million tonnes of low-grade feed with an average grade of 0.44 g/t. The remaining low-grade material is stockpiled (Figure 16- 5).

All three cutbacks are active during Year 5, with the high-grade plant feed coming predominantly from Cutback 1 and Cutback 2. The final pit cutback is mostly lower grade material and waste. 19.3 million ore tonnes are mined during this period with an average grade of 0.64 g/t Au. Waste material is placed on both the southwest and southeast waste dumps depending upon the closest available waste dump space. There are 1 million ore tonnes mined from the northern pit during a 3-month night shift campaign. This material is predominantly lower grade with an average gold grade of 0.5 g/t.

The northern pit is still being actively mined during Year 6 and is again mined over a 3-month period during the dry season with hauling occurring on night shift only. This will reduce the impact on the community for the road crossing of the haul trucks. The South Pit is being mined from both the Cutback 2 and the final pit with ramp accesses to the south for waste haul and to the northeast for ore. 15.1 million ore tonnes are mined during Year 6 with an average grade of 0.65 g/t Au. 11 million ore tonnes will become plant feed with the remaining 4.1 million tonnes going to the low-grade stockpile.

By Year 7, mining in the North Pit is complete however the road crossing remains active to allow backfilling to commence in the North Pit in accordance with the Mining Licence requirements. The waste rock material is taken from the South Pit and trucked to the North Pit where it is dumped and manoeuvred by dozer if required. The ore is mined from the final southern cutback but is a majority of low grade. With only 3.5 million tonnes of high-grade ore, supplemented by 9.6 million tonnes of low grade ore, the overall grade is 0.55 g/t Au. During this period, the plant will have to receive both low-grade and high-grade ore to reach the total feed rate of 11 Mtpa. The low-grade stockpile will contain the remaining material (Figure 16- 6).

Figure 16-5 Mine Progression Year 4

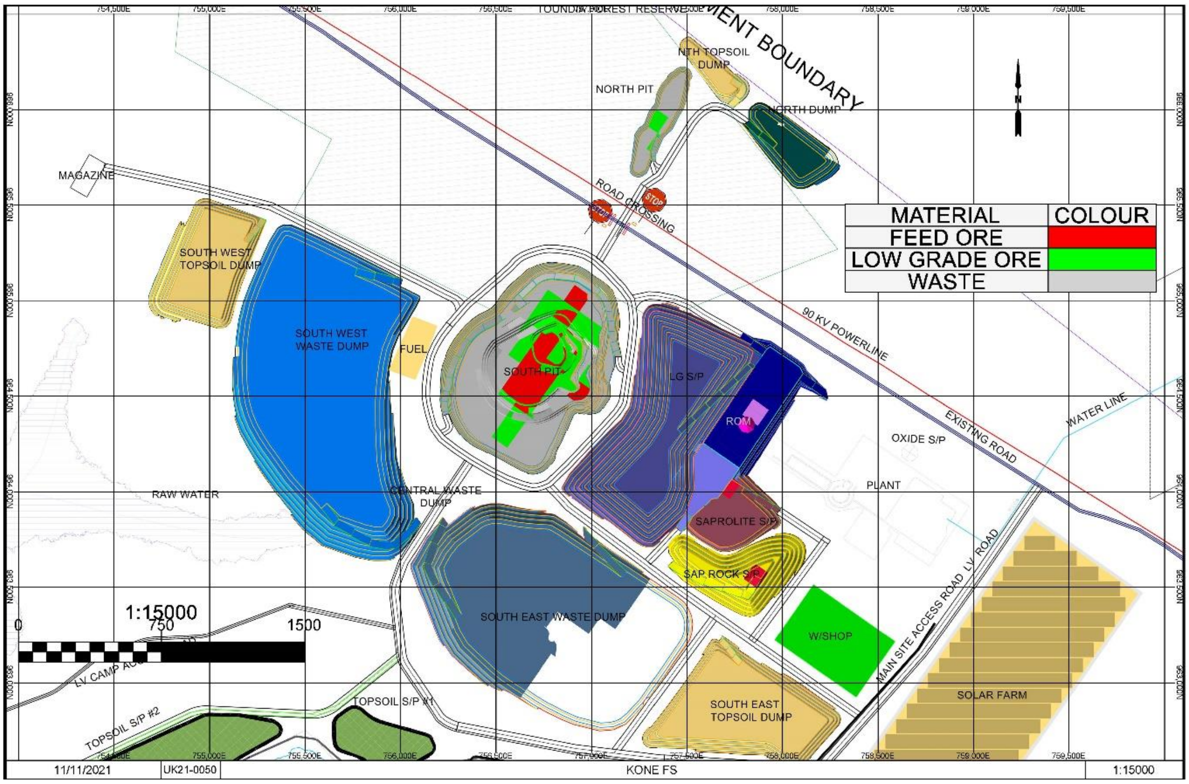
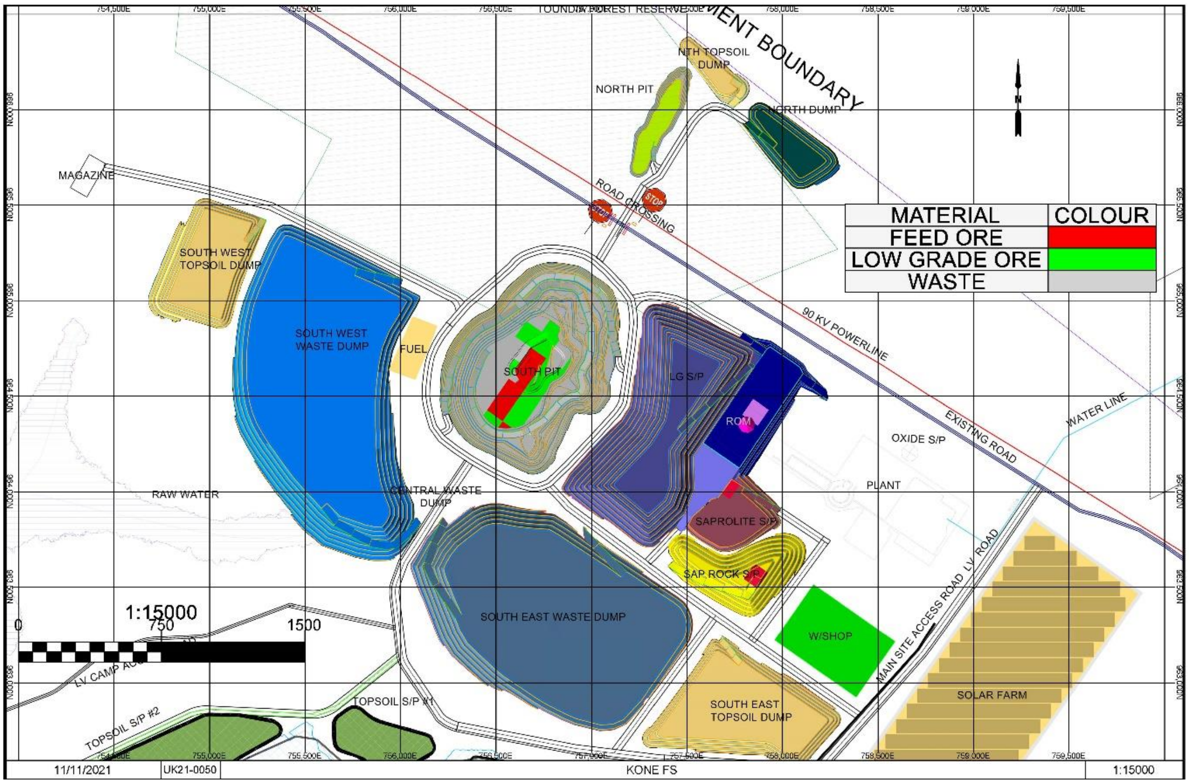


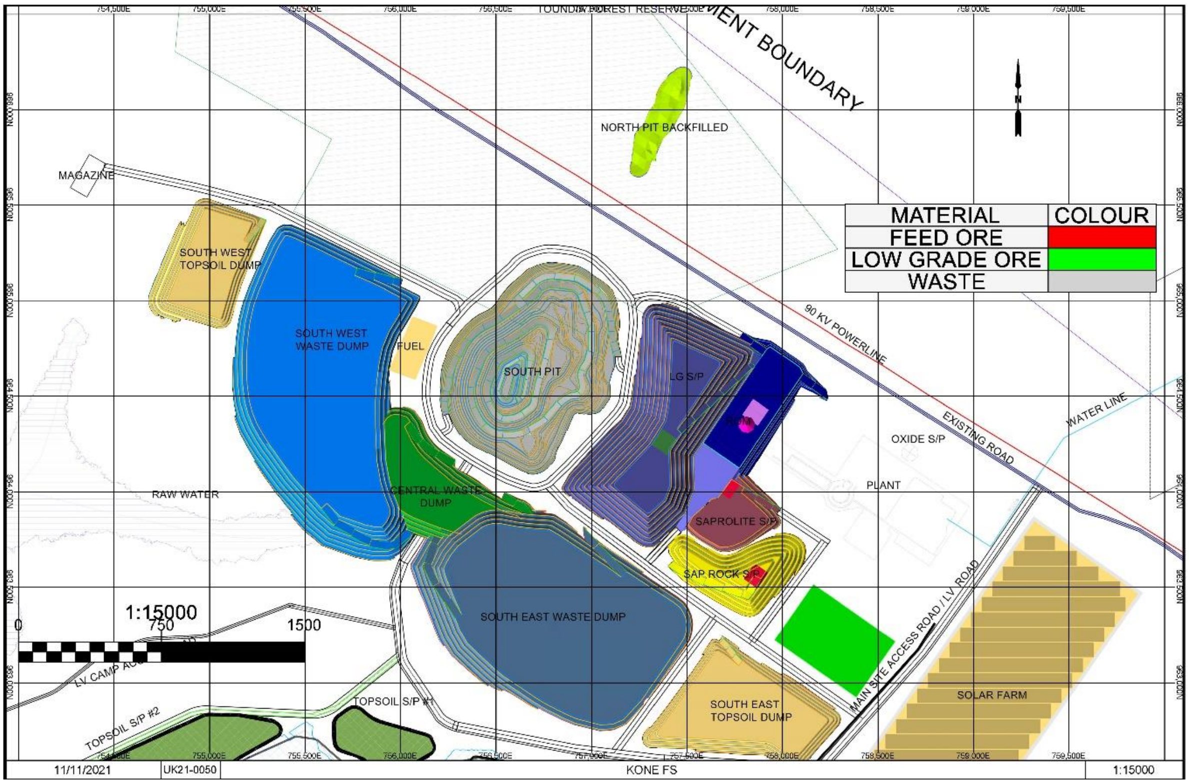
Figure 16-6 Mine Progression Year 7



The Central Waste Dump is opened at the beginning of Year 8 to allow for short haul from the deepest parts of the South Pit to the closest dump area. The impact of the Central Waste Dump is to even out the haulage fleet over the last few years and not incur additional costs for more fleet for a short period. The North Pit is partially backfilled with waste material from the South Pit so as to reduce the amount of rehandle required later in the project. Therefore, the road crossing is active during this year for rehandle. The material mined from the pit in Year 8 is all fresh hard rock material with an average grade of 0.64g/t Au. The total ore tonnes are 21 million tonnes with 13.1 million tonnes of that classified as low-grade ore and the remaining 8.3 million tonnes as high-grade ore.

Mining operations are completed in July of Year 9 (Figure 16-7). After the mining has completed, the fleet is repurposed to backfill the North Pit with material from the Northern Waste Dump and for rehandle of the low-grade material to feed the crusher. The North Waste Dump is completely reclaimed, and the topsoil dump is placed back over the pit and dump area. This area will then be seeded and will return to its original condition, at a minimum. All roads and infrastructure are also removed from this area. The ore mined from the South Pit in Year 9 is all fresh hard rock material with an average grade of 0.81 g/t Au.

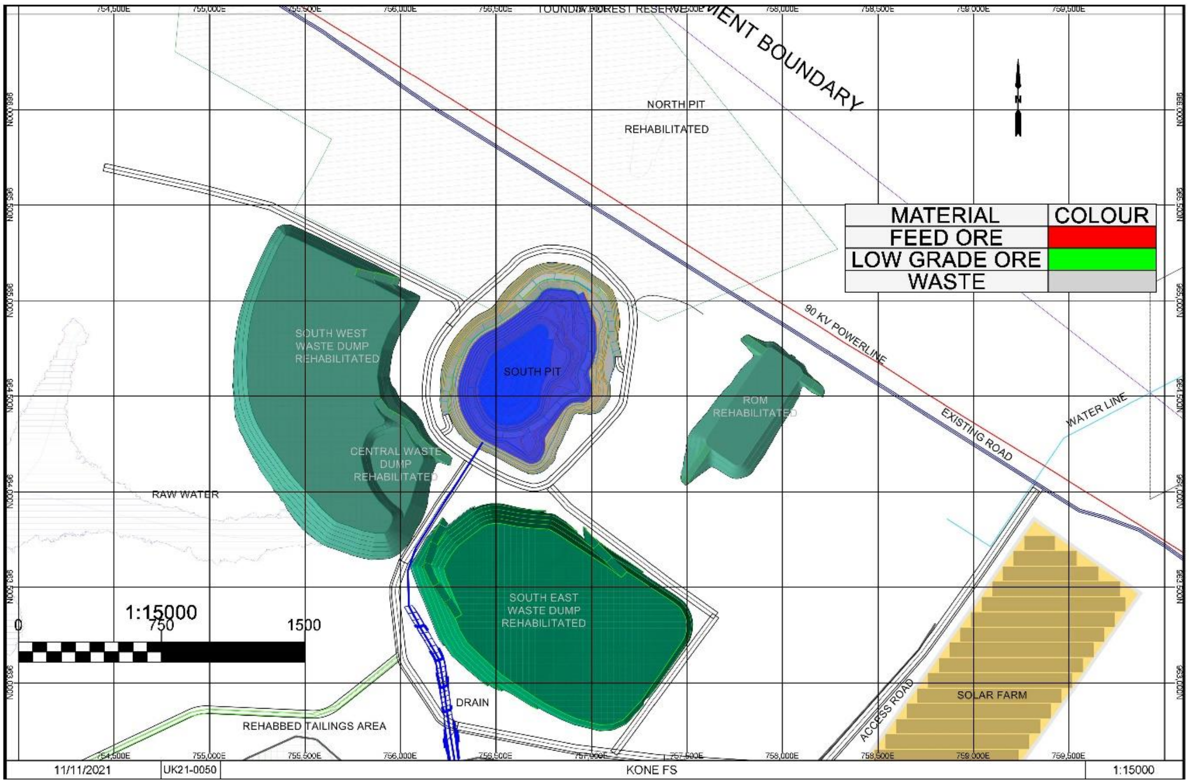
Figure 16-7 Mine Progression Year 9



By Year 10, the mining from the pits has been completed however the processing of the low-grade dumps is continuing. The Central Waste Dump is reclaimed and shaped to allow for a drainage way from the now completed South Pit to the TSF area. The material from the Central Waste Dump has been hauled from mid-year in Year 10 to commence covering the TSF for rehabilitation. The covering of the TSF will take 24 months. Once mining is completed in the previous year, tailings deposition will occur in the South Pit, allowing the TSF to dry and be rehabilitated while the processing plant is still in operation.

Figure 16-8 shows the end of project surface with all of the low-grade stockpiles reclaimed and processed, showing the southwest waste dump and the southeast waste dump as reshaped and rehabilitated. This process will begin in Year 10 after the mining has been completed and any activities to move waste from the stockpiles to cover the TSF has been completed. However, where progressive rehabilitation can take place in earlier years, this should be undertaken. The topsoil dumps will also be used to rehabilitate the waste dumps and TSF. The ROM pad will also be dozed to give a wall angle of 20 degrees. It is then covered in topsoil and seeded. The plant and other mining infrastructure will be dismantled and removed.

Figure 16-8 End of Processing Life



16.5 Mining Risk Assessment

Table 16-5 outlines the risk assessment for the operations stage of the project.

Table 16-5 Mining Operations Risk Assessment

Risk	Likelihood of Occurrence	Potential Severity of Impact	Risk Ranking
Final pit design is not achievable	Unlikely	Moderate	Medium
Mining equipment (contractor) productivities are slower than expected	Unlikely	Moderate	Medium
Grade control or resource model underperforms	Unlikely	Minor	Low
Climate stopping mining	Unlikely	Minor	Low
Water drainage management issues	Possible	Moderate	Medium
Under-estimation of pit groundwater inflows	Unlikely	Moderate	Medium
Minor pit wall failure	Possible	Minor	Medium
Major pit wall failure	Unlikely	Major	Medium
Waste dump stability issues	Unlikely	Minor	Low
Public relations issues relating to blasting	Possible	Minor	Medium

The cutbacks are large and provide opportunity to adjust mining schedules while any remedial action is undertaken and the elevate cut-off grade strategy means that there will be several years of stockpiled material, albeit at a lower grade, available after the first few years. The final pit limits are not reached for several years, providing opportunity for detailed monitoring and assessment of the behaviour of the rock mass, the control for surface water and the development of a groundwater and geotechnical model.

The highway and powerline are not affected by blasting activities for several years, providing sufficient time for community awareness campaigns to sensitise the populations, and blasting studies to optimise blasting techniques.

This study has not identified any risks that would affect the mine plans and open pit operations, provided that monitoring, analysis and reporting practices are put in place from the start of the operation.

17.0 RECOVERY METHODS

17.1 Overview

The metallurgical testwork conducted to date has confirmed that the Koné gold is amenable to recovery via conventional cyanidation techniques and carbon adsorption.

The process plant design is based on a robust metallurgical flowsheet designed for optimal precious metal recovery. The flowsheet chosen is based on unit operations that are well proven in the industry.

The key criteria for equipment selection are suitability for duty, reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements whilst maintaining a layout that will facilitate construction progress in multiple areas concurrently.

The key project design criteria for the plant are:

- Nominal throughput of 11.0 Mtpa with a grind size of 80% passing (P80) 75 µm
- Process plant availability of 91.3% supported by the selection of standby equipment in critical areas, reputable western vendor supplied equipment and connection to an onsite LNG fired power station.
- Sufficient automated plant control to minimize the need for continuous operator interface but allow manual override and control if and when required.

17.1.1 Process Flowsheet

The treatment plant design incorporates the following unit process operations:

- Primary and closed circuit secondary crushing using a gyratory crusher and two cone crushers to produce a crushed product size P80 of approximately 38mm. Feed size preparation for a secondary crushed product is required for a grinding efficient HPGR-ball mill circuit as compared to a standard SAG mill circuit.
- A crushed ore stockpile with a nominal live capacity of 22,000 wet tonnes, providing buffer storage of crushed ore with continuous reclaim with feeders for the HPGR-ball mill comminution circuit
- Two parallel HPGRs in closed circuit with wet sizing screens, with undersize slurry reporting to the milling circuit via the cyclone feed hopper. Two parallel trains of ball mills in closed circuit with hydrocyclones will produce a P80 grind size of 75 µm
- Pre-leach thickening to increase the slurry density feeding the leach and carbon in pulp (CIP) circuit to minimize tankage and reduce overall reagent consumption

- Leach circuit incorporating fourteen leach tanks, arranged in two parallel trains of seven in series, to provide 36 hours leach residence time.

- A Kemix Pumpcell CIP circuit consisting of eight CIP tanks for recovery of gold onto carbon, to minimize carbon inventory, gold in circuit and operating costs. The CIP and elution circuit design is based on daily carbon harvesting.

- 20 tonne split AARL elution circuit, electrowinning and gold smelting to recover gold from the loaded carbon to produce doré

- Tailings thickening to recover and recycle process water from the CIP tailings

- Tailings pumping to the tailings storage facility (TSF).

A simplified overall flow diagram depicting the sequence of the unit operations incorporated in the selected process flowsheet is shown in Figure 17-1. The Site Plan and Plant Layout are included in Figure 17-2 and 17-3.

Figure 17.0-1 Overall Schematic Flow Diagram

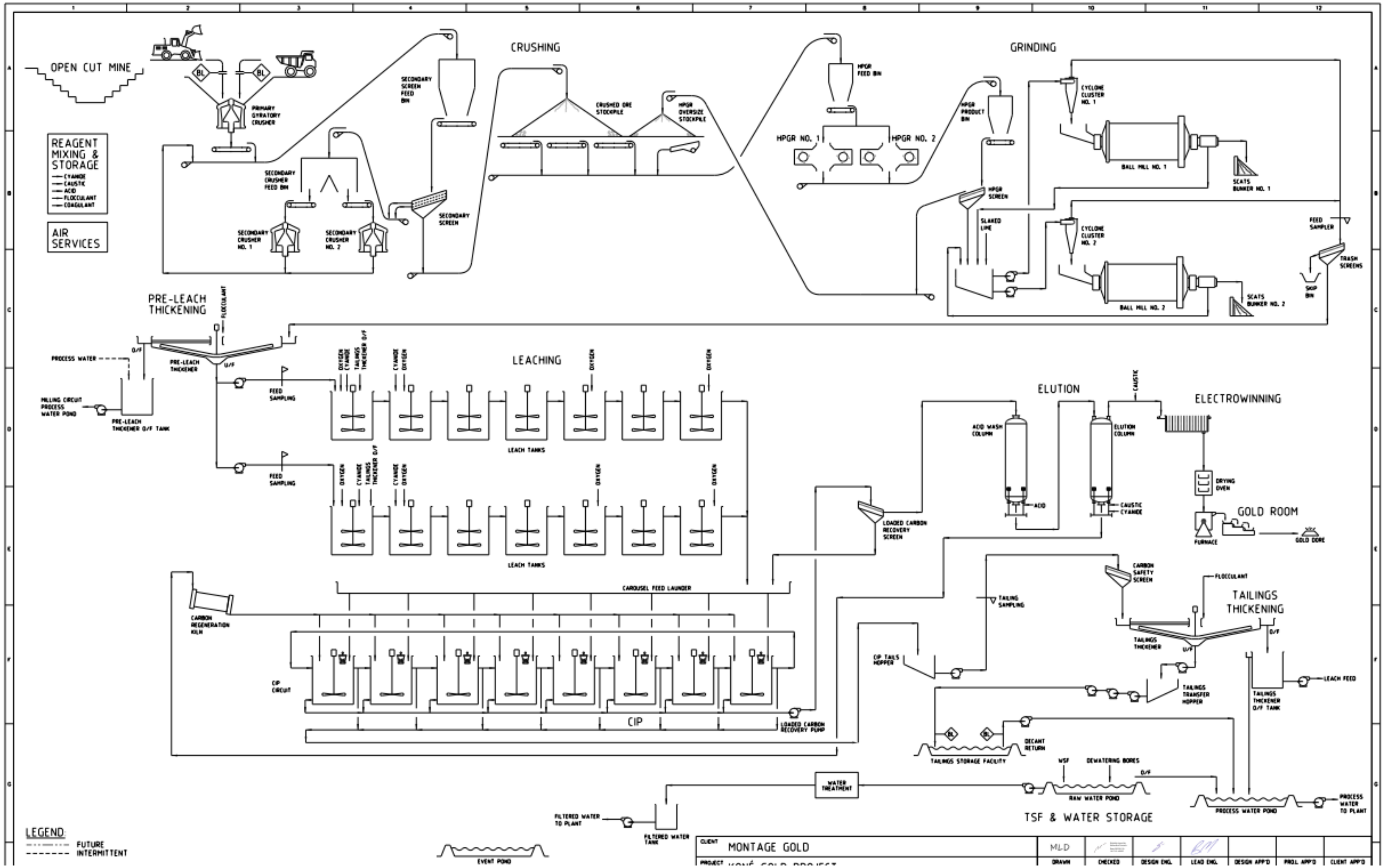


Figure 17-2 Site Plan

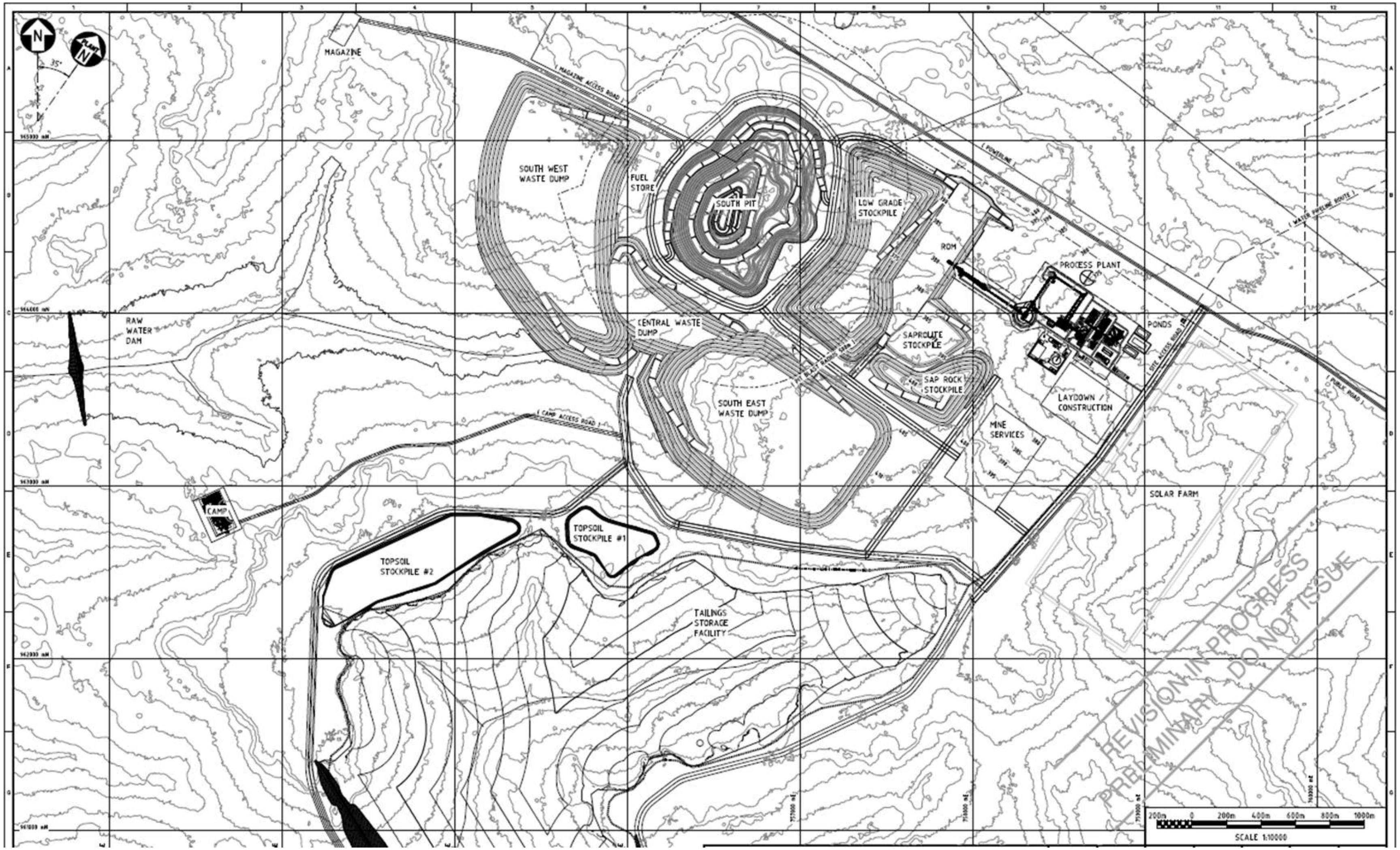
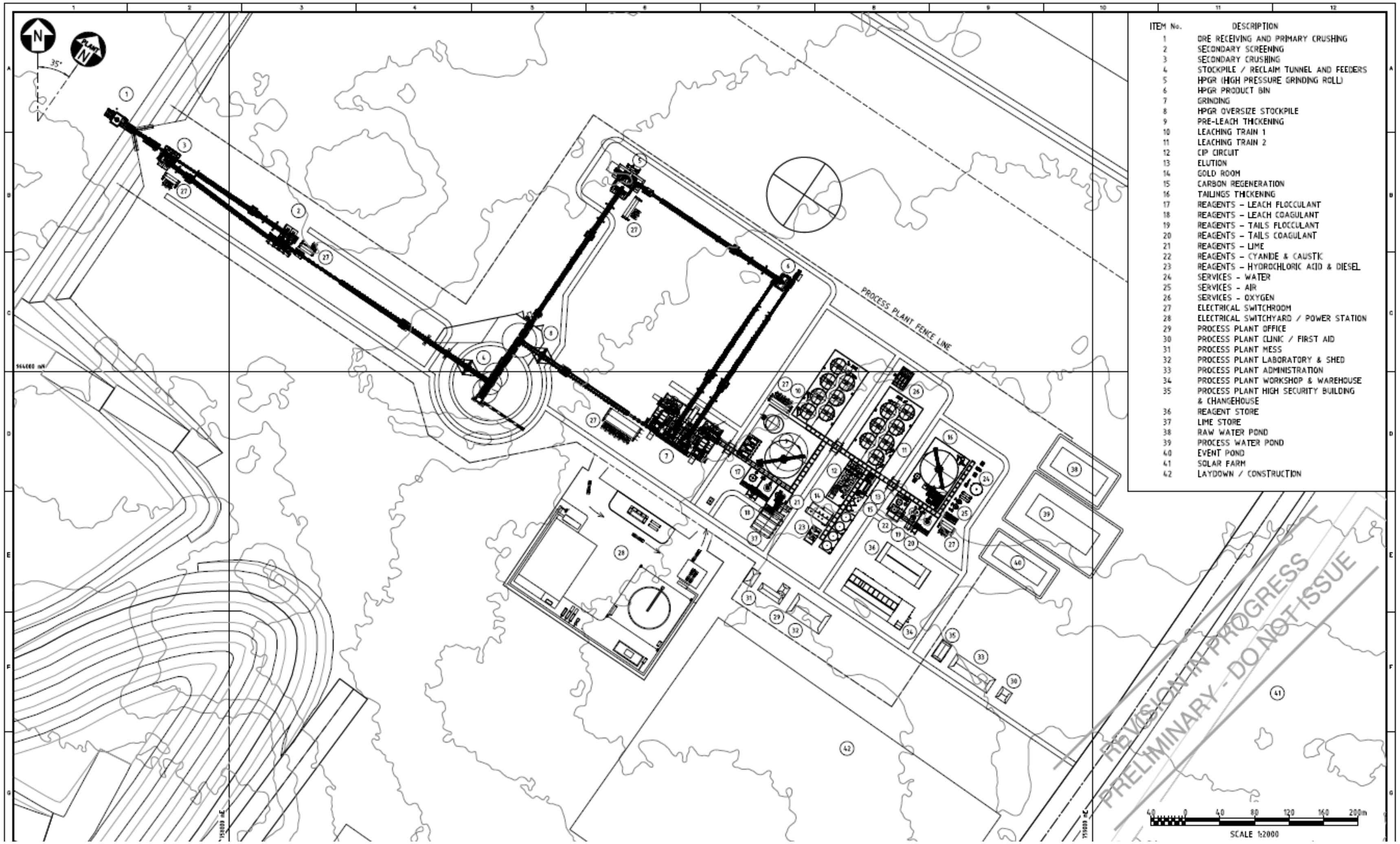


Figure 17-3 Plant Layout



17.2 Process Design Basis

The key factors considered in the process plant design and selection of equipment are outlined as follows:

17.2.1 Process Plant

The plant design has been based on a nominal capacity of 11.0 Mtpa of ore and maximum head grade of 1.29 g/t Au with maximum gold recovery 91.6%. This reflects the highest annual mine schedule grade, with design margin to cater for short term high grade batches. The plant will process relatively low proportions of saprolite and saprock oxide ore of 6.7% and transition ore of 5.1% over the life of the mine. The saprolite will be stockpiled and treated in dry season only, with saprock being treated during the wet season.

17.2.2 Comminution Circuit Selection

Following review of various comminution circuit alternatives by Orway Mineral Consultants (OMC) for equipment sizing, Montage selected a HPGR-ball mill circuit as being the most energy efficient option for the very competent fresh ore. The higher energy efficiency and HPGR technology also resulted in the lowest life cycle cost of the options.

The comminution data was also benchmarked against other ores where HPGR testwork results were available to support the recirculating load estimates and modelled product size distributions.

17.2.3 Circuit Availabilities

The open circuit primary crushing and closed circuit secondary crushing and screening equipment have been sized based on 70% availability on fresh ore. This was driven by the lower availability of the secondary crushers due to the expected higher relining / maintenance requirements.

The crushing circuit will be decoupled from the downstream plant by the crushed ore stockpile, which provides surge capacity between the lower availability crushing circuit and the higher availability downstream plant.

The closed circuit HPGR sizing is based on 80% availability on fresh ore comminution properties. The HPGR circuit is likely to achieve higher operating availabilities, similar to a milling circuit, however the HPGR and mill downtime frequencies and durations are generally not aligned. The HPGR will require more frequent, shorter downtime for inspection and preventative maintenance, along with annual rolls changeouts. The ball milling circuit will typically require less frequent, but longer downtime for relining activities. The HPGR will be decoupled from the milling circuit by the HPGR product bin which provides storage capacity between the HPGR and the downstream plant.

The milling circuit and downstream plant is sized on 91.3% availability (8,000 operating hours at 1,570 dry t/h) on fresh ore.

17.2.4 ROM Pad and Crushing Circuit

The mine will be operated at an enhanced rate to enable the higher grade material to be processed early and lower grade material processed once the mining has been completed. The ROM pad will be used to provide a buffer between the mine and the plant. Importantly, saprolite oxide ore processing will be limited to dry season only and saprock oxide ore will be limited to the wet season, each up to a maximum of 10% of the ore blend to avoid material handling issues with crushing, screening and thickening operation.

Open circuit primary crushing and closed circuit secondary crushing will be required to ensure a maximum particle size suitable for feeding the HPGR-ball mill circuit. HPGRs are sensitive to feed size and it is important that the top feed size is significantly less than the HPGR operating gap to minimise stud wear and potential for breakage. A consistently finer feed size to the HPGR will improve the HPGR tyre wear life.

Modelling of the closed circuit crushing based on primary crushed product distribution and ore hardness was used to determine the likely secondary feed recirculating load for design. Metal tramp detection and removal will be required to protect the secondary crushers and HPGR downstream.

17.2.5 Crushed Ore Stockpile

A 22,000 t live capacity stockpile will be provided for surge capacity between the crushing circuit and the milling circuit. Crushed ore will be reclaimed by three variable speed apron feeders and will report to the HPGR feed conveyor along with the HPGR screen oversize reclaimed from the HPGR oversize stockpile via vibrating feeders.

17.2.6 Milling

Comminution circuit design has been based on achieving the required 11.0 Mtpa throughput rate and grind size for the competent fresh ore, which comprises the majority of the ore over the life of mine. An HPGR-ball mill circuit has been selected to achieve the nominal circuit grind size of P_{80} 75 µm. The HPGR-ball mill circuit provides the lowest power utilisation with two parallel trains consisting of HPGRs, wet screens and ball mills.

The HPGR product will contain considerable oversize due to the pressure profile across the roll width, with little crushing occurring near the roll edges. Wet screening is necessary to achieve efficient screening down to fine sizes. A finer closing size increases the work done by the HPGR, making the overall comminution process more energy efficient by reducing the ball milling specific energy requirement. The size selected is a practical trade-off given increasing equipment size and moisture carryover (recycled to the HPGR feed) with decreasing screen open area.

The optimum HPGR feed should contain some graded fines to increase close packing in the ore bed and some moisture to improve binding and integrity of the autogenous wear protection layer. Too many fines or too much moisture can wash out the autogenous layer or result in relative slippage between the material bed and the roll increasing stud and tyre surface wear.

The HPGR product will be mixed with water in the re-pulping screen feed box to maximise the de-agglomeration of the HPGR flake product ahead of the screen. Wet screening will be conducted at less than 50% feed solids w/w to ensure high screening efficiency and minimum undersize recycle.

17.2.7 Classification

650mm diameter cyclones have been selected for the classification duty to reduce the potential for spigot blockages occurring from coarse mill discharge material and minimize the overall size of the cyclone cluster. Design has been based on up to 294% recirculating load to the ball mills.

The inclusion of a pre-leach thickener will allow operation of the grinding and classification circuit to be optimized by operating cyclones at feed densities that maximize classification efficiency, thereby reducing circulating load and overall circuit power consumption.

17.2.8 Trash Screening

Linear trash screens have been selected to prevent oversize particles and rubbish from entering the downstream leach and adsorption circuit. Although minimal trash is expected, acceptable trash screen performance will be essential for good carbon management.

17.2.9 Pre-Leach Thickening

A high rate pre-leach thickener ahead of the leach circuit has been included to thicken trash screen undersize to between 61% and 63% solids w/w. Dilution with tailings thickener overflow water to the design leach circuit feed density of 50% solids w/w will take advantage of the residual free cyanide in this water, thereby reducing overall cyanide consumption rates.

17.2.10 Leach and CIP Circuit

Head assays have indicated that the ore does not show preg-robbing characteristics and therefore a leach and CIP circuit can be considered for design.

Based on testwork, a leach time of 36 hours was selected requiring fourteen 5,000 m^3 leach tanks at 50% w/w solids density.

A Kemix Pumpcell CIP circuit consisting of eight 400 m^3 tanks was selected for recovery of gold onto carbon, to minimize carbon inventory, gold in circuit and operating costs.

17.2.11 Elution, Electrowinning and Gold Recovery

A split AARL elution circuit has been selected to remove absorbed gold and silver from carbon. The split AARL circuit can accommodate additional elution cycles if required due to the relatively short stripping time of approximately eight hours. The circuit will include an acid wash column to remove inorganic foulants from the carbon prior to elution, and a cold cyanide wash cycle has been included in the design in the event that ores with elevated copper are processed.

A 20 tonne batch size is required for the 11.0 Mtpa plant capacity at the design head grade and gold recovery.

Two parallel 33 cathode electrowinning cells are proposed for the gold room to provide a high pass efficiency and ensure a low gold tenor in the spent electrolyte returning to the strip solution tank.

A sludging cell design with in-tank washing of the cathodes has been adopted to simplify the cathode handling process. Electrowinning cell sludge will be filtered in a pressure filter prior to transfer to a drying oven before smelting to produce doré.

Barren carbon will be regenerated in a horizontal rotating kiln to remove any organic foulants.

17.2.12 Tailings Thickening and Pumping

A high rate thickener has been selected to thicken the CIP tails to maximize process water recovery and reduce the volume of tailings. The tailings thickener also allows the residual free cyanide in the process water that would normally be discharged to the tailings storage facility to be recovered for re-use in the leaching circuit.

The tailings thickener underflow will be pumped to the tailings storage facility for the first part of the mine life. For the second part of the mine life, tailings will be discharged into the pit following mining completion.

17.3 Key Process Design Criteria

The key process design criteria listed in Table 17-1 form the basis of the process design criteria and mechanical equipment list. Inputs into the design criteria include metallurgical testwork, Montage advice, comminution modelling by Orway Mineral Consultants (OMC), and Lycopodium calculations and modelling and vendor advice.

Table 17.0-1 Key Process Design Criteria

Parameter	Units	Oxide Blend*	Fresh	Source
Plant Capacity	tpa	11,000,000	11,000,000	Montage
Maximum Gold Head Grade	g Au/t	1.13	1.29	Montage
Maximum Design Gold Recovery (Leach)	%	96.3	91.6	Testwork
Crushing Circuit Utilization	%	70.0	70.0	OMC
HPGR Circuit Availability	%	80.0	80.0	OMC
Grinding Circuit and Plant Availability	%	91.3	91.3	Lyco
Crushing Work Index (CWi)	kWh/t	N.A.	19.7	Testwork
A*b (SMC Test)		1698	27.9	OMC / Testwork
Bond Ball Mill Work Index (BWi)	kWh/t	5.2	13.3	OMC / Testwork
Abrasion Index (Ai)		0.115	0.419	OMC / Testwork
Grind Size (P_{80})	µm	56	75	OMC / Montage
Leach Circuit Residence Time	hrs	36	36	Montage
Leach Slurry Density	% w/w	50	50	Montage
Number of Leach Tanks		14	14	Lyco
Number of Adsorption Tanks		8	8	Vendor
Cyanide Addition – Plant	kg/t	0.36	0.45	Testwork / Lyco
Quicklime Addition – Plant	kg/t	2.87	0.51	Testwork / Lyco
Elution Circuit Type		Split AARL	Split AARL	Lyco
Elution Circuit Size	t	20	20	Lyco
Frequency of Elution	strips / wk	7	7	Lyco

* Up to 10% oxide treated in plant feed; Comminution parameters quoted reflect 100% oxide ore, benchmarked by OMC.

Note that reagent addition in the plant includes an allowance for residual cyanide and quicklime supply at 90% active CaO.

17.4 Process Description

17.4.1 Run-of-Mine (ROM) Pad

The mine will be operated at an enhanced rate to enable the higher grade material to be processed early and lower grade material processed once the mining has been completed. Haul trucks will deliver run-of-mine (ROM) ore from the pits to the ROM pad where it will either tip directly into the ROM pocket with the excess stockpiled by ore head grade and ore type. Reclaim of ore from the various stockpiles to the ROM bin will be handled by the mine fleet.

Ore will be blended under the guidance of mine geologists and metallurgists to maintain a relatively constant feed grade to the process plant. Feed blending will also take care of the potential clayey nature of the oxide ore, with total oxide ore content limited to 10% to minimize chute blockages and rheology issues in the crushing, milling, leach and CIP circuits.

17.4.2 Crushing Circuit

ROM ore will be directly tipped from the haul trucks into the crusher ROM pocket. The ROM pocket will be designed for haul truck tipping from both sides. The gyratory crusher will process ore with a maximum lump size of approximately 1,000 mm. Crushed ore will be withdrawn from the ROM discharge pocket by a variable speed apron feeder and conveyed to the secondary screen feed bin. A belt magnet on this conveyor belt will be provided to allow material to be diverted to a metal reject bunker.

Ore will be extracted from the secondary screens feed bin by belt feeders, which will feed the secondary screens. The secondary screens will be fitted with 80mm and 40mm aperture screen decks. Oversize from these decks will report to the secondary screen oversize conveyor which will deliver coarse ore to the secondary crusher feed bin. This conveyor will be fitted with a weightometer and a metal detector to remove any tramp steel ahead of the secondary crushers feed bin to protect the secondary crushers.

Ore will be withdrawn from the secondary crusher feed bin by two secondary crusher belt feeders operating in a duty / duty configuration and report to the secondary crushers, which will also operate in a duty / duty configuration. The secondary crushers will be MP1000 or equivalent units operating with a closed side setting of 30mm. Product from the secondary crushers, along with the primary crushed ore will report to the primary crusher discharge conveyor, which will be equipped with two weightometers to separately measure the primary and secondary crushers discharge.

Secondary screen undersize will report to the stockpile feed conveyor, which will be fitted with a weightometer. This conveyor will deliver ore, with a P80 of 30mm, to the crushed ore stockpile.

Both the secondary screen feed bin and the secondary crusher feed bin will be fitted with bin blasters to assist with removing material hung up in the bins during periods of wet or sticky ore. Two dust collectors will also be installed. One unit at the primary crushing and secondary crushing area, and a second unit at the secondary screening area, to ensure that dust generated at transfer points will be adequately captured. The dust collected will be discharged onto the nearest conveyor and then finely sprayed with water for dust suppression.

The crushing circuit will be controlled from a dedicated crusher control room. Communication with the crusher control room will be achieved with a two-way radio.

17.4.3 Crushed Ore Stockpile and HPGR Screen Oversize Stockpile

The stockpile feed conveyor will discharge ore onto the crushed ore stockpile. The stockpile will have a live capacity of approximately 22,000 t, which is 15 hr of mill feed at 11 Mtpa. Ore will be withdrawn from the stockpile by up to three variable speed apron feeders. These feeders will discharge ore onto the HPGR feed conveyor.

The HPGR closed circuit screen oversize material will be stockpiled adjacent to the crushed ore stockpile and will be reclaimed via vibrating feeders onto the HPGR feed conveyor. A weightometer will be located after the apron feeders to indicate the new feed reclaimed ore tonnage to the HPGR circuit with a second weightometer after the oversize feeders to indicate the total reclaimed ore tonnage.

A ventilation fan will force air into the concrete stockpile reclaim chamber to ensure fresh air ventilates the upper part of the chamber which would otherwise have limited natural ventilation.

17.4.4 HPGR Crushing and Wet Screening

HPGR feed conveyor will discharge to the HPGR feed bin. A belt magnet on this conveyor belt will be provided to remove any metal in circuit and discharge to a metal reject bunker. HPGR feed will be drawn from the feed bin by belt feeders and feed two HPGRs operating in parallel at a controlled rate. A metal detector located above each HPGR belt feeder will detect any tramp metal remaining in the HPGR feed and will activate a diverter gate to further protect the HPGR from any metal ingress by diverting ore to the HPGR discharge conveyor below.

For optimum HPGR operation, the priority will be to maintain a level in the HPGR feed chute to minimise feed variation and associated tyre wear. The HPGR will be provided with variable speed drives on the rolls and variable pressure to optimise size reduction at the nominated throughput rate. Installed power will cater for spikes in operating power draws.

A dust collector will be used to recover dust generated in the HPGR and product discharge chutes. The dust collected will be discharged onto the HPGR discharge conveyor and will be finely sprayed with water for dust suppression.

The HPGR product will discharge onto the HPGR discharge conveyor fitted with two weightometers to measure the throughput from each HPGR. The HPGR discharge conveyor will fill the HPGR product bin, which is sized with surge capacity to ensure mill feed is available during HPGR downtime periods.

HPGR product will be drawn from the product bin via the HPGR product bin apron feeders onto the HPGR screening feed conveyors and will discharge into the HPGR screening feed bins. A dedicated weightometer on each HPGR screening feed conveyor will measure the HPGR screen feed reporting to each HPGR screening feed bin. An oxide bypass feed bin installed on one of the HPGR screen feed conveyors will provide alternate facility to feed wet sticky clay ore directly into the HPGR screening feed bin, and bypass the HPGR and crushing circuits if required. This facility is not expected to be used during normal operations, but is provided as a risk mitigation facility should issues be experienced with oxide blend processing in the crushing and HPGR circuits.

HPGR screen feed will be drawn from the HPGR screening feed bins via belt feeders and will discharge to HPGR screens feed pulping boxes. Water will be added to the repulping box to de-agglomerate the HPGR product for presentation to the milling screen for efficient sizing. The water will also assist with spreading the screen feed across the full width of the screen in the feed box.

The repulped slurry will feed the HPGR screens. A double deck screen will be utilised for this duty with the upper deck serving to:

- protect the lower deck from wear by the larger oversize particles.

- further break up agglomerates for better presentation to the lower deck.
- reduce the bed depth on the lower deck for improved screening efficiency.
- increase the overall deck area for dewatering and minimum moisture return to the HPGR.

HPGR screen undersize slurry will report to the cyclone feed hopper, providing the new feed to the milling and classification circuit.

HPGR screen oversize will report to the HPGR oversize stockpile via the HPGR oversize conveyor fitted with a weightometer for measurement of the HPGR screen oversize rate and calculation of the undersize stream reporting to the milling circuit. Under normal operating conditions, the oversize will be reclaimed at the rate it is produced.

If a HPGR is offline for an extended shutdown, the HPGR screen lower deck oversize can be diverted via the screen middlings diverter gate. This material will be less than 8mm which is still suitable for ball mill feed. This will slightly reduce the ball mill capacity at the same product grind, further extending the product bin reclaim rate during a HPGR shutdown period.

17.4.5 Grinding and Classification Circuit

The HPGR crushed ore will be milled to achieve the nominated grind size for effective gold leaching. The grinding circuit will consist of two ball mills operating in parallel and in closed circuit with a cluster of classification hydrocyclones.

Two ball mills will be installed, each with its own dedicated cyclone pack. The ball mills will be 7.0m x 10.7m EGL fixed speed mills, each fitted with two 5.1 MW twin pinion drives. Slurry from the cyclone underflow launder will be returned to the respective ball mill where it will be diluted with process water to achieve the desired milling density. Slurry exiting each ball mill will pass through the ball mill trommel and report to the appropriate cyclone feed hopper. Reject oversize material from the ball mill trommel screens will be collected in the ball mill scats bunkers. Media will be added to each ball mill to maintain the power draw as required using dedicated ball mill ball charging kibbles and hoists.

Combined undersize product from the HPGR screens and ball mill trommels from each train will flow by gravity to the respective cyclone feed hopper where it will be diluted with process water. Slaked lime slurry, used for pH control in the leach circuit, will also be added directly into each cyclone feed hopper. Slurry will be pumped to each hydrocyclone clusters for classification. Duty / standby pumps will be provided to each cyclone feed hopper to maximise operating availability with variable speed pump drives to manage the cyclone feed flow and inlet pressure.

Each cyclone cluster will be fitted with a number of spare cyclones to allow wear inspection and maintenance on-line. The cyclone underflow slurry will report to the relevant ball mill feed box while the cyclone overflow from each cluster (fine material) will flow by gravity to a common cyclone overflow boil box. There will be an in-line particle size analyser (PSA) sampler on each cyclone overflow stream to provide feed for the PSA to provide real time sizing information and feedback for control of each ball mill circuit.

The combined cyclone overflow stream will flow by gravity through a metallurgical sampler which will be used as the plant feed sampler. Both solids and solution assays will be undertaken on this sample

Three linear trash screens will be installed in a parallel configuration prior to the pre-leach thickener. Three screens have been selected to provide sufficient area for the required volumetric throughput rate. The trash screen undersize will be directed to the pre-leach thickener while trash screen oversize will be dewatered before being collected in trash bins.

The grinding area will be serviced by dedicated vertical spindle sump pumps which will allow spillage and clean up to be returned to the circuit via the cyclone feed hopper. A drive in sump will be provided to allow coarse material to be removed via FEL

17.4.6 Pre-Leach Thickening

Trash screen undersize will be thickened in a 44 m diameter high rate thickener. The feed slurry will be de-aerated in the thickener feed box prior to entry into the thickener. Flocculant will be added in the feed launder and feed well. Flocculant will be diluted with water in a static mixer to ensure adequate dispersion throughout the feed stream. Coagulant will be added to the thickener feed box.

Thickener underflow at 61% solids w/w or higher will be pumped to the leach circuit. Three thickener underflow pumps will be installed in a duty / duty / standby configuration. Thickener overflow will flow by gravity to the pre-leach thickener overflow tank from where it will be pumped directly to the milling circuit for dilution, with any excess diverted to the process water pond for re-use around the plant.

The pre-leach thickener area will be serviced by the pre-leach area sump pump which will allow spillage to be directed to the pre-leach thickener feed or directly to the leach feed distribution box.

17.4.7 Leach Circuit

Due to the number of leach tanks required to achieve a 36 hour residence time, two trains of seven mechanically agitated tanks in series will be installed. Each tank will have a live volume of approximately 5,000 m^3.

For each train, pre-leach thickener underflow will be pumped to the leach feed distribution box via a pre-leach sampler. Tailings thickener overflow water, containing residual cyanide, will be added as required to achieve the design 50% solids w/w leach feed density. The slurry from the leach feed distribution box will flow by gravity to the first tank. If the first tank is offline, the slurry will be diverted to the second tank, via a launder gate system.

The tanks will be interconnected by launders, with slurry flowing by gravity from tank to tank. Each tank will be fitted with a dual impeller mechanical agitator to ensure uniform mixing and particle suspension. All tanks will be fitted with launder bypass facilities to allow any tank to be removed from service for agitator maintenance.

On-site generated oxygen will be added via oxygen addition devices and dedicated oxygen contactor slurry pumps, to four leach tanks in each train. In the first tank of each train, all new leach feed slurry along with recycled slurry from the tank, will be introduced to oxygen gas in the contactor, and sodium cyanide solution added to the suction side of the oxygen contactor slurry pump. In the remaining tanks, recycled slurry will be introduced to oxygen gas in the contactor and sodium cyanide solution added to the suction side of the oxygen contactor slurry pump. In tanks 1 and 2, three oxygen contactors will be used for each tank and in tanks 5 and 7, two oxygen contactors will be used for each tank. It is anticipated that 16 tonnes per day of oxygen will be required, most of which will be added at the head of the trains.

Slurry will flow by gravity through the tanks and report to the CIP circuit feed launder.

The leach circuit will be serviced by eight floor sump pumps. Sump pumps will return spillage to a nearby tank.

A cyanide analyser for on-line monitoring of the free cyanide concentration will allow the sodium cyanide dose rate to be optimized. A hydrogen cyanide (HCN) gas monitor will also be installed in the leach area.

17.4.8 CIP Circuit

The CIP circuit will be a carousel system in which the feed and discharge points of each tank will be changed, and carbon will only be moved when transferring loaded carbon to the elution circuit, or returning barren carbon from regeneration. This has a number of advantages over conventional CIP including no back mixing of carbon, smaller tanks and lower gold in circuit. A total of eight tanks with a volume of 400 m^3 each will be installed.

The feed launder, discharge manifold and internal launder arrangements are integral to the carousel mode of operation. The individual tanks will be connected by an external launder. The feed launder and discharge manifold allow any tank to be either the head or tail tank in the carousel sequence.

Feed slurry will be directed to the head tank while residue slurry will be directed out of the circuit via the residue manifold. Once the desired gold on carbon loadings have been achieved in the head tank, this tank will be isolated and feed slurry will be directed to the next tank in the carousel sequence. The entire contents of the head tank will be pumped to the loaded carbon recovery screen, by one of two loaded carbon recovery pumps, to separate the loaded carbon from the slurry. The screened slurry will be returned to the feed launder. Regenerated carbon will be added to the isolated tank which will be brought back on line as the new tail tank in the carousel sequence.

Each CIP tank will be fitted with two pumping interstage screens, in order to achieve the required flowrate and one Pumpcell agitator. The pumping interstage screens consists of a pumping impellor, rotating cage, wedge wire screen, pitch blade turbine and agitator in one. The pumping impellor will be used to transfer pulp from one tank to the next. The rotating cage and stainless steel wedge wire screen will retain the carbon within the tank while allowing slurry to be pumped to the next tank in series. The pitch blade turbine will ensure that the slurry remains suspended even when there is no flow through the mechanism, which allows ease of start-up after a prolonged shutdown. The agitator will ensure even slurry and carbon suspension and mixing with the tank.

Slurry from the discharge manifold of the CIP circuit will gravitate to the CIP tails pumps. The CIP tails pumps will transfer slurry to the two duty carbon safety screens to recover any carbon escaping from worn screens or overflowing tanks. Screen undersize will gravitate to the tailings thickener. Screen oversize containing carbon will be collected in the fine carbon bin for potential return to the circuit. HCN monitors will be provided on the CIP deck and near the carbon safety screens.

A CIP tails sampler will be installed prior to the carbon safety screens for metallurgical accounting purposes.

Two vertical spindle sump pumps will be provided in the CIP areas to return spillage and clean up to the CIP feed launder. A CIP area gantry crane will also be provided to facilitate removal of Pumpcell mechanisms for maintenance.

17.4.9 Tailings Thickening and Disposal

Carbon safety screens undersize will flow by gravity to the tailings thickener feed box. Other streams such as acid waste streams will also report to the feed box. Flocculant will be diluted using a static mixer prior to being added to the tailings thickener to enhance solids settling rates. Coagulant will be added to the thickener feed box. Tailings thickener underflow will be pumped to the tailings transfer hopper. The cold cyanide wash waste will report to the tailings transfer hopper to avoid copper cyanide complexes reporting directly to the process water system via the thickener overflow. Slurry from the tailings transfer hopper will be pumped to the TSF using three stages of centrifugal pumping.

Two HCN monitors will be installed in the tailings area; one adjacent to the carbon safety screen and the other near the tailings transfer hopper.

Tailings thickener overflow will flow by gravity to the tailings thickener overflow tank. A portion of the process water from this tank will be returned to the leach circuit by the tailings thickener overflow pump for recycle of residual cyanide. The remainder of the process water will gravitate to the process water pond.

Water from the surface of the TSF will be recovered from the decant system and pumped directly to the process water pond. Underdrainage and seepage from around the TSF drainage system will be pumped back into the TSF.

The tailings thickener area will be serviced by one vertical spindle sump pump. Any spillage collected within this area will be directed to the tailings thickener feed box.

17.4.10 Elution, Carbon Regeneration and Gold Room Operations

The following operations will be carried out in the elution and gold room areas:

- Acid washing of carbon.
- Optional cold cyanide wash to remove copper from loaded carbon.
- Stripping of gold from loaded carbon using the split AARL method.

- Electrowinning of gold from pregnant solution.
- Filtration of electrowinning sludge.
- Drying of the filter cake.
- Smelting of filter cake to produce a gold doré.

The elution and gold room areas will operate seven days per week, with the majority of loaded carbon preparation and stripping occurring during day shift. The AARL stripping circuit will be automated and will contain separate acid wash and elution wash columns. The stripping circuit will be sized for a 20 tonne batch of carbon.

17.4.10.1 Acid Wash

Loaded carbon will be recovered on the loaded carbon recovery screen and directed to the rubber lined acid wash column. The acid wash column fill operation will be controlled manually. All other aspects of the acid wash and the carbon transfer sequence to elution will be automated. Acid washing of the carbon will commence after carbon transfer is complete.

Dilute hydrochloric acid, 3% w/w HCl, will be prepared prior to use and stored in the dilute acid make-up tank. During acid washing, the dilute solution of hydrochloric acid will be pumped into the column in an up-flow direction to remove contaminants, predominantly carbonates, from the loaded carbon. This process improves the elution efficiency and has the beneficial effect of reducing the risk of calcium-magnesium 'slagging' within the carbon during the regeneration process.

After the soak period has elapsed, the loaded carbon will be rinsed with treated water. This rinse water will displace any residual acid from the loaded carbon. Dilute acid and rinse water will be disposed of in the tailings thickener. Acid-washed carbon will be hydraulically transferred to the elution column for stripping. Calcium assays on loaded and barren carbon will be conducted to determine the efficiency of the acid wash step.

A vertical spindle sump pump will be provided in the acid wash area to direct spillage to the tailings thickener.

17.4.10.2 Elution

The elution sequence will be fully automated, with actuated valves used to direct solution to and from the appropriate destinations once certain set-points or time periods are met.

The split AARL elution sequence will begin with the fill of the elution column and pre-heat of lean eluate solution with simultaneous injection of caustic and cyanide into the lean eluate pump suction. The solution will be recirculated through the heat recovery and primary heat exchangers, through the elution column, through the hot side of the heat recovery heat exchangers and back into the lean eluate tank until a temperature of 95°C is achieved. The sequence will then automatically shift to the elution phase, with the temperature set point raised to 130°C, and five bed volumes (BV) of solution pumped from the lean eluate tank, through the heat exchangers and elution column to the pregnant solution tank. Caustic will be added to the pregnant solution tank during this step to ensure that a high enough solution pH is attained for electrowinning.

Following this step, five BV of treated water or barren electrowinning solution will be pumped from the stripping water tank, through the heat exchangers and elution column and into the lean eluate tank to provide lean solution for the next stripping cycle. The temperature set point will be maintained at 130°C for this step.

The final step of the sequence will be a cool down of carbon where treated water will be used to cool the carbon down to approximately 80°C. Treated water exiting the column will be directed to the leach feed distribution box.

A vertical spindle sump pump will be provided in the elution column area to direct spillage to the leach feed distribution box.

17.4.10.3 Electrowinning

Soluble gold recovery from pregnant solution will be carried out by electrowinning onto stainless steel cathodes. The electrowinning circuit will consist of two electrowinning cells in parallel, each containing 33 cathodes. A dedicated rectifier, per electrowinning cell, will supply the necessary current to electroplate the gold onto the cathode.

Once sufficient pregnant solution is available within one of the two pregnant solution tanks, electrowinning will be initiated by starting the duty pregnant solution pump. The flow of pregnant solution to the cells will be evenly split across the electrowinning distribution box and manual control valves will assist the desired linear velocity to be achieved. During the electrowinning cycle the electrowinning cell discharge will be continuously returned to the pregnant solution tank via gravity.

Once the target barren solution grades have been achieved, the electrowinning cycle is complete. Barren solution from the duty pregnant solution tank will be re-used as strip solution to conserve water and reagents, or directed to the leach circuit when required via the barren eluate pump.

Fume extraction will be provided to remove noxious gases from the cells. In addition to this, a number of gold room vent fans will be provided to ensure there is adequate ventilation inside the gold room.

17.4.10.4 Gold Room

Upon completion of electrowinning, precious metal sludge will be washed off the cathodes with a high pressure cathode washer. The gold bearing sludge will flow by gravity to a sludge hopper, from where it will be pumped to a pressure filter.

The filter cake will be thermally dried in a drying oven to remove moisture prior to smelting. Dried solids will be mixed with a prescribed flux mixture (silica, nitre and borax), prior to being charged into the diesel fired gold furnace. The fluxes added will react with base metal oxides to form a slag, whilst the gold remains as a molten metal. The molten metal will be poured into moulds to form doré ingots, which will be cleaned, assayed, stamped and stored in a secure vault ready for dispatch. A conical shaped vessel will be used for slag collection so that any precious metals prills form in the bottom and can be easily recovered and put directly back in the furnace. Low grade slag will periodically be returned to the grinding circuit, via the ball mill.

The gold room and electrowinning area will be serviced by a gold trap and dedicated gold room area sump pump. Any spillage within this area will be pumped back to the leach circuit.

17.4.10.5 Carbon Regeneration

After completion of the elution process, the barren carbon will be transferred from the elution column to the carbon dewatering screen to dewater the carbon prior to entering the feed hopper of the horizontal carbon regeneration kiln. In the kiln feed hopper any residual and interstitial water will be drained from the carbon before it enters the kiln. Kiln off-gases will also be used to dry the carbon prior to entering the kiln.

The carbon will be heated to 650 - 750°C and held at this temperature for 15 minutes to allow regeneration to occur. Regenerated carbon from the kiln will be quenched in the carbon quench vessel and pumped to the carbon sizing screen using the regen carbon transfer pump. New carbon will be added to the carbon quench vessel to ensure that the carbon is sized over the carbon sizing screen prior to entering the CIP circuit.

The screen oversize (regenerated, sized carbon) will report to the carbon transfer hopper and be returned to the CIP circuit using the barren carbon transfer pump. The quench water and fine carbon (carbon sizing screen undersize) will report to the carbon safety screen via the fine carbon hopper and pump.

A vertical spindle sump pump will be provided in the carbon regeneration area to direct any spillage back to the CIP circuit via the carbon sizing screen.

17.4.11 Reagents

17.4.11.1 Quicklime

Quicklime will be delivered to site in bulk tanker, with the option to handle bulk bags deliveries if supply is interrupted. Bulk bags will be added to the lime slaking area transfer hopper via the bag breaker. The lime slaking area transfer blower will transfer the quicklime to the lime silo. For bulk deliveries the lime unloader will transfer quicklime to the lime silo. A dust collector will be fitted to the silo to minimize particulate emissions when transferring lime into the silo.

Quicklime will be withdrawn from the lime silo, by a rotary valve and screw feeder and discharged directly to the top of the lime slaking vertimill, along with filtered water for wet grinding and control of the mill slurry temperature as the lime slaking reaction is exothermic. The slaked lime slurry density target is 20% solids (w/w) and will be transferred via dedicated pumps to the lime storage tank. Slaked lime slurry will then be circulated around the processing plant, via a ringmain, for dosing to the cyclone feed hopper for leach circuit pH control.

A vertical spindle sump pump will be provided to service the lime slaking area. This pump will report to the tailings thickener feed box.

17.4.11.2 Sodium Cyanide

Cyanide will be delivered as dry briquettes in one tonne bulk bags in boxes. Cyanide bulk bags will be added to the mixing tank via a bag breaker and be dissolved in filtered water to achieve the required 20% w/v reagent strength. The facility to dose caustic into the cyanide mixing tank to maintain a suitable solution pH has also been provided. Cyanide can also be received in bulk isotainers, which will require a filtered water supply for dissolution of the cyanide briquettes to achieve the required reagent strength. The cyanide solution will be transferred to the cyanide storage tank on completion of a mix. Cyanide solution will be dosed to the leach circuit via individual dosing pumps to the oxygen contactor pumps suction. Cyanide solution will also be pumped to the elution circuit as required.

A vertical spindle sump pump will be provided to service the cyanide and caustic mixing areas. This pump will report to the train 1 leach feed distribution box.

17.4.11.3 Caustic

Caustic (sodium hydroxide) will be delivered to site in 1.2 tonne bulk bags of 'pearl' pellets. Caustic will be added to the mixing tank via a bag breaker and be dissolved in filtered water to achieve the required 20% w/v concentration. Caustic solution will be pumped to elution and electrowinning as required. The facility to dose caustic into the cyanide mixing tank will also be provided.

17.4.11.4 Hydrochloric Acid

Concentrated hydrochloric acid (32% w/w) will be delivered to site in 1,000 L bulk boxes. The concentrated hydrochloric acid will be transferred into the dilute acid make-up tank by a positive displacement, hose type pump. Filtered water will be added to the dilute acid tank to achieve a solution concentration of 3% w/w. The solution will be mixed by using the acid wash pumps. Following completion of the mixing cycle, the dilute acid will be pumped to the acid wash column during the acid wash sequence.

The hydrochloric acid storage area will be serviced by an air operated dedicated floor sump pump which will discharge to the tailings thickener.

17.4.11.5 Activated Carbon

Activated carbon will be delivered in bulk bags. Carbon will be added to the carbon quench tank as required for carbon make-up to the CIP inventory. This addition point will allow attritioning of any friable carbon particles with subsequent fines removal on the sizing screen prior to entering the CIP tanks.

17.4.11.6 Grinding Media

Grinding media will be delivered to site in steel drums. The balls will be charged to the ball mill using a kibble and ball loading hopper. A fork lift with a hydraulic drum tipper attachment will unload balls from the drums into the ball loading hopper. The ball loading hopper will then be used to load the ball kibbles which in turn feed media into the ball mills. Media will be added as required to achieve the target ball mill power draw settings.

17.4.11.7 Flocculant

Flocculant for use in the pre-leach and tailings thickeners will be delivered to site in 750 kg bulk bags. Each thickener will be provided with a dedicated flocculant mixing and storage system due to the distances between each thickener.

Flocculant bags will be lifted by hoist to a bag breaker on the flocculant feed hopper. The vendor supplied flocculant mixing plant will automatically mix batches of flocculant with filtered water and transfer the mixed flocculant to the flocculant storage tank after each mixing cycle is complete.

Flocculant will be distributed to the pre-leach thickener using the variable speed pre-leach thickener flocculant dosing pumps which will be installed in a duty / standby configuration. A vertical spindle sump pump will be provided to service the flocculant and coagulant mixing areas. This pump will transfer any spillage to the pre-leach thickener.

Flocculant will be distributed to the tailings thickener using the variable speed pre-leach thickener flocculant dosing pumps which will be installed in a duty / standby configuration. A vertical spindle sump pump will be provided to service the flocculant and coagulant mixing areas. This pump will transfer any spillage to the tailings thickener.

17.4.11.8 Coagulant

Coagulant will be used to reduce the flocculant consumption and achieve acceptable thickener overflow clarity. Coagulant for use in the pre-leach and tailings thickeners will be delivered to site in 750 kg bulk bags. Each thickener will be provided with a dedicated coagulant mixing and storage system due to the distances between each thickener.

Coagulant bags will be lifted by hoist to a bag breaker on the coagulant feed hopper. The vendor supplied coagulant mixing plant will automatically mix batches of coagulant with filtered water and transfer the mixed coagulant to the coagulant storage tank after each mixing cycle is complete.

Coagulant will be distributed to the pre-leach thickener using the variable speed pre-leach thickener coagulant dosing pumps, which will be installed in a duty / standby configuration.

Coagulant will be distributed to the tailings thickener using the variable speed tailings thickener coagulant dosing pumps, which will be installed in a duty / standby configuration.

17.4.11.9 Plant Diesel

Diesel will be delivered to the 15 m^3 plant diesel day tank by the mine diesel tanker. Diesel in the day tank will be reticulated to the elution heater, carbon regeneration kiln and smelting furnace on a ring main.

17.4.11.10 Anti-scalant

Anti-scalant will be delivered to the plant in totes or bulk containers (IBC). Metering pumps will distribute anti-scalant directly from the IBC to the process water and elution circuits.

17.4.11.11 Reagents Storage

A minimum of 90 days stock of reagents will be stored on site to ensure that supply interruptions due to port, transport or weather delays do not restrict production.

17.4.12 Services

17.4.12.1 Raw Water

Raw water for the plant will be supplied from both the Marahoué River and mine dewatering pumps. The mine dewatering bore pumps will deliver water to a pit dewatering transfer tank, from where it will be transferred to the raw water pond by the duty / standby mine dewatering transfer pumps. Mine in-pit dewatering pumps will deliver water to a mine dewatering transfer tank, from where it will be transferred to the Water Storage Facility (WSF) before being pumped to the raw water pond.

At the Marahoué River, three pontoon mounted pumps will supply water to the river extraction water tank and be transferred to the site water storage facility by a series of river water extraction transfer and booster pumps, with intermediate break tanks as required. At the water storage facility, the water storage pumps will transfer water to the plant raw water pond.

The raw water pond will be a 7,500 m^3 lined pond with approximately nine hours of storage capacity. Water will overflow from the raw water pond into the process water pond for make-up if required. Dedicated duty / standby raw water pumps will be used to transfer raw water into the filtered water treatment plant.

Fire water for the process plant will be drawn from the base of the raw water pond. The suction for raw water pump will be at an elevated level to ensure a fire water reserve always remains in the raw water pond.

17.4.12.2 Process Water

The plant process water will consist of pre-leach and tailings thickener overflow and TSF decant return water, with raw water make-up as required. The process water pond will be situated adjacent to the raw water pond such that the raw water pond overflows to process water. With this arrangement the raw water pond can be kept full at all times. The filtered water treatment plant waste stream will also report to the process water pond. The process water pond will be a 30,000 m^3 lined pond, which will have a nominal capacity of six hours.

Duty / standby process water pumps will be provided for the plant water supply. Anti-scalant will be added to the process water to reduce scaling of pipelines, spray nozzles and screen decks.

The process water system will be configured such that bulk water for the milling circuit such as HPGR wet screens, ball mill feed and cyclone feed dilution will be supplied directly from the pre-leach thickener overflow tank. This ensures that if the overflow clarity of the pre-leach thickener is poor, solids in the process water pond can be minimized and screen spray blockages can be avoided. Process water for the ball mill trommel sprays will be supplied directly from the process water tank, along with flocculant dilution and hose-up water.

Tailing thickener overflow will be preferentially used for leach feed dilution, to take advantage of any residual cyanide in this stream. However, the facility to add process water directly from the process water pumps will also be provided for start-up and in the event of upset conditions.

17.4.12.3 Filtered Water

Filtered water for the process plant will be produced by treating raw water in the filtered water treatment plant. The treatment plant will be a containerised system consisting of auto backwashing multimedia filters and pH adjustment.

Filtered water will report to the filtered water tank and will be pumped to distribution points around the plant for use in primary crushing, mills cooling water, reagent mixing, carbon regeneration, elution, gold room, and some hose points.

Waste reject from the water treatment plant will report to the process water pond.

17.4.12.4 Gland Water

Gland water will be supplied from the filtered water tank. Duty / standby LP gland water pumps will distribute gland seal water around the plant. The duty / standby HP gland water pumps will be used to supply gland seal water to the tailings transfer pumps which require higher pressure supply.

17.4.12.5 Fire Water

Fire water for the process plant will be drawn from the base of the raw water pond.

The fire water pumping system will contain:

- An electric jockey pump to maintain fire ring main pressure
- An electric fire water delivery pump to supply fire water at the required pressure and flowrate
- A diesel driven fire water pump that will automatically start in the event that power is not available for the electric fire water pump or that the electric pump fails to maintain pressure in the fire water system
- Fire hydrants and hose reels will be placed throughout the process plant, fuel storage and plant offices at intervals that ensure complete coverage in areas where flammable materials are present.

17.4.12.6 Potable Water

Mine bore dewatering pumps will supply water to a centralised dewatering transfer tank. Water will be transferred to the centralised potable water treatment plant at the camp. Potable water produced will be distributed from the camp potable water storage tank to the camp, and to the plant via potable water transfer pumps.

At the processing plant, a further water treatment facility will include chlorination and ultra-violet sterilisation. Potable water will be stored in the plant potable water tank and will be reticulated to the site ablutions, buildings and the mine services area. A dedicated safety shower water tank will allow the safety shower and drinking fountain water to be reticulated on a ring main system to assist in keeping the potable water at a suitable temperature for use.

Waste from the camp potable water treatment plant will be discharged to grade neat the camp.

17.4.12.7 High Pressure Air

High pressure air at 750 kPa(g) will be provided by two high pressure air compressors, operating in a lead-lag configuration. The entire high pressure air supply will be dried and can be used to satisfy both plant air and instrument air demand. Dried air will be distributed via the plant air receiver, crushing area air receiver, grinding area air receiver and tailings valves air receiver. The tailings valves air receiver will enable the tailings valves at the processing plant to fail close in the event of power loss as the large size of these pneumatically actuated valves precludes the use of spring return for typical fail close functionality.

17.4.12.8 Oxygen

Oxygen, for use within the leach circuit, will be supplied by two duty oxygen pressure swing adsorption (PSA) plants. A standby plant will also be provided. Oxygen will be generated to a supply pressure of 500 kPa(g). Oxygen will be distributed to the required leach tanks.

17.5 Control System

The general approach to automation and control for the plant will be one with a moderate level of complexity offering the option of local control and remote monitoring or control from a central control room. Instrumentation will be provided within the plant to measure and control key process parameters to minimize operator intervention in standard start-up functions and to provide key monitoring and control to minimize process excursions and maintain steady operation.

18.0 PROJECT INFRASTRUCTURE

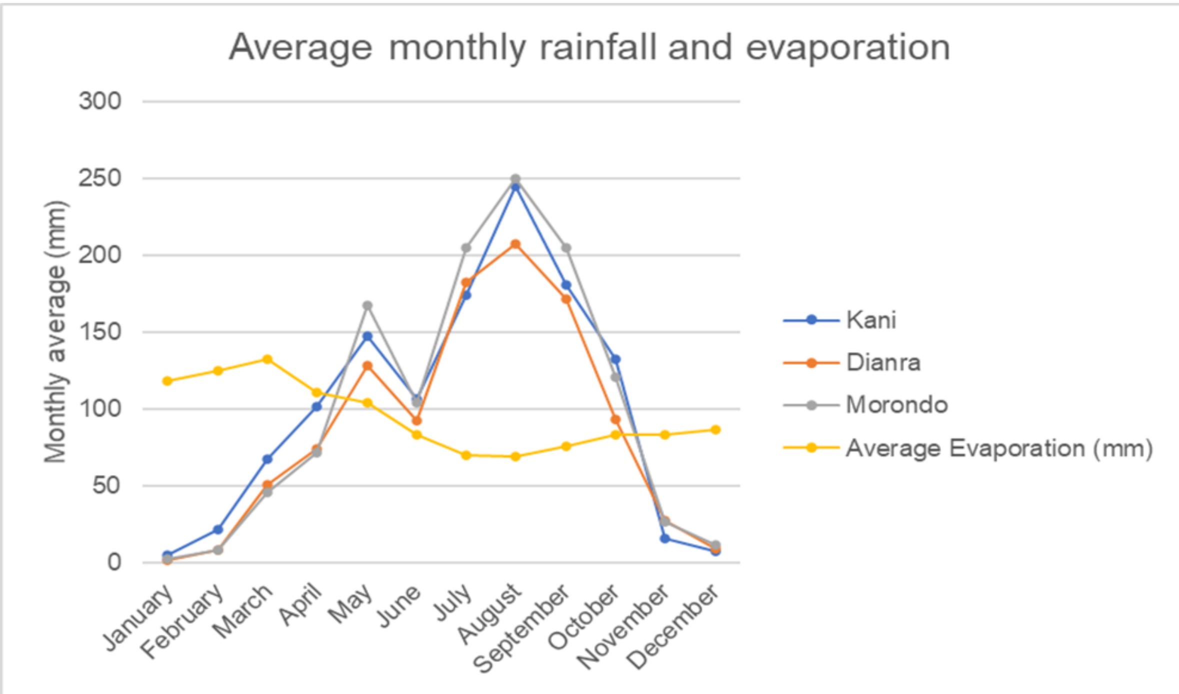
18.1 Water Supply

18.1.1 Preliminary Surface Water Assessment

18.1.1.1 Climate and Rainfall

The area falls within a Wet Semi Equatorial Climatic Zone. The climate is typically warm and humid with a mean-monthly temperature of 27°C. Three seasons can be distinguished, namely warm and dry (November to March), hot and dry (March to May) and hot and wet (June to October). Average daytime maximum temperatures range from 22 to 32˚C. The average annual rainfall data were derived from three weather stations2 located in the Koné region and are depicted in Figure 18-1. Average estimated lake evaporation is in the order of 1,140 mm per annum, which is slightly lower than the average rainfall of 1,212 mm per annum.

Figure 18-1 Monthly rainfall and evaporation for Koné



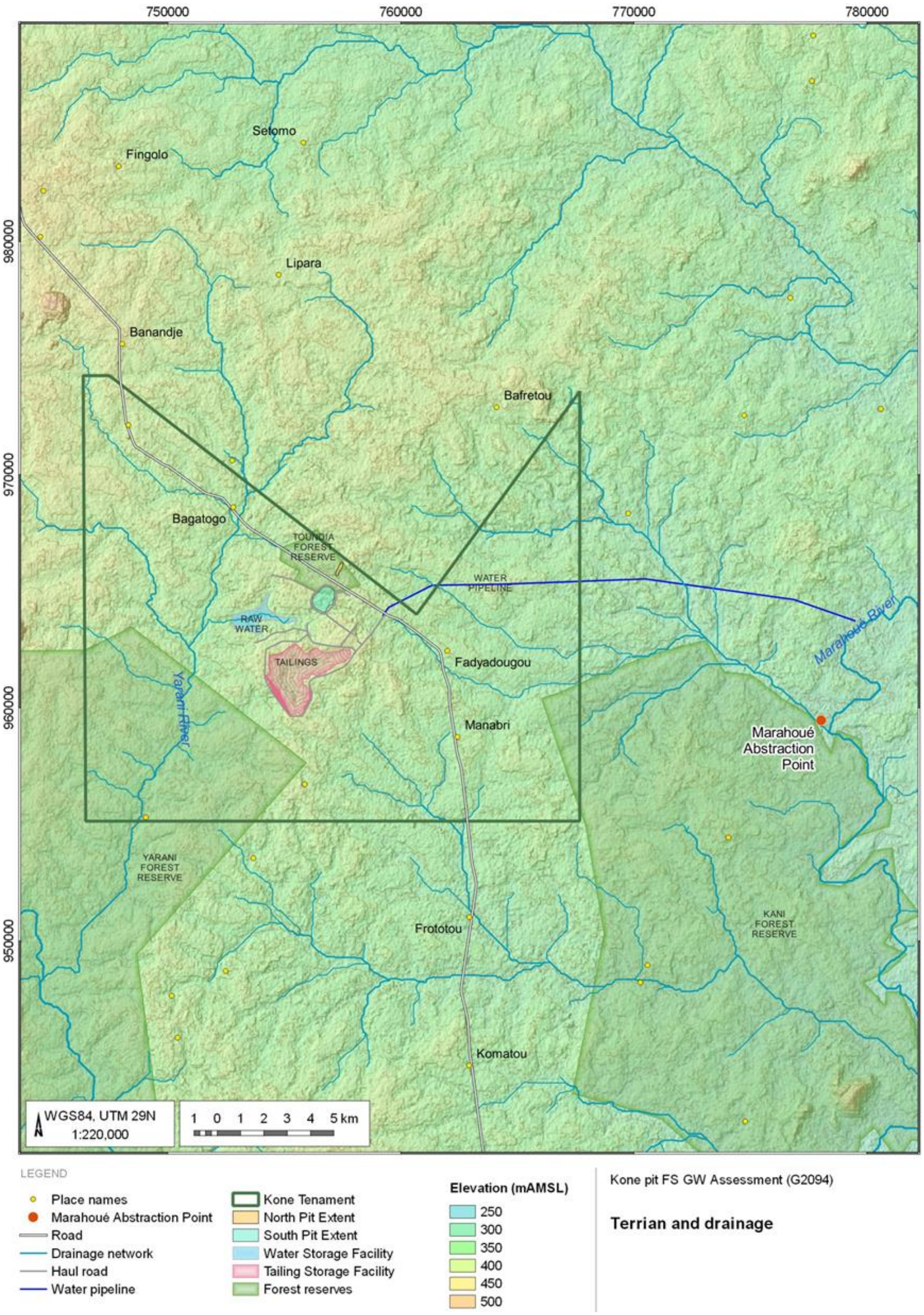
18.1.1.2 Terrain, drainage and land use

The Project area is characterized by moderate relief between 300 and 400 metres above mean sea level (m AMSL) (Figure 18-2). The Marahoué and Yarani rivers are the main surface drainage features in the Project area. The bulk of the Project area is drained by shallow ephemeral streams that only show significant flow in the wet season.

The Project lies within the Guinean forest-savanna ecoregion of West Africa, a band of interlaced forest, savanna and grassland running from western Senegal to eastern Nigeria and dividing the tropical moist forests near the coast from the West Ivorian savanna of the interior.

The locations of the two nearby communities of Fadiadougou and Batogo, situated approximately 7 km to the south-east and north-west respectively of the Project site, are shown in Figure 18-2.

Figure 18-2 Site terrain and drainage



Source: AGE, November 2021

18.1.2 Groundwater Assessment

18.1.2.1 Dykes and structural geology

Multiple sets of dykes are observed through the deposit, displaying varying composition, orientation and deformation. Some of these can be traced across the deposit, such as the felsic, feldspar porphyry and main late green dykes. Others aren't continuous across drill fences and/or the deposit. Generally, the dykes are interpreted to be anastomosing and lack continuity across the deposit.

Two variations of mafic dykes (MDY) are observed, considered to vary in time of emplacement:

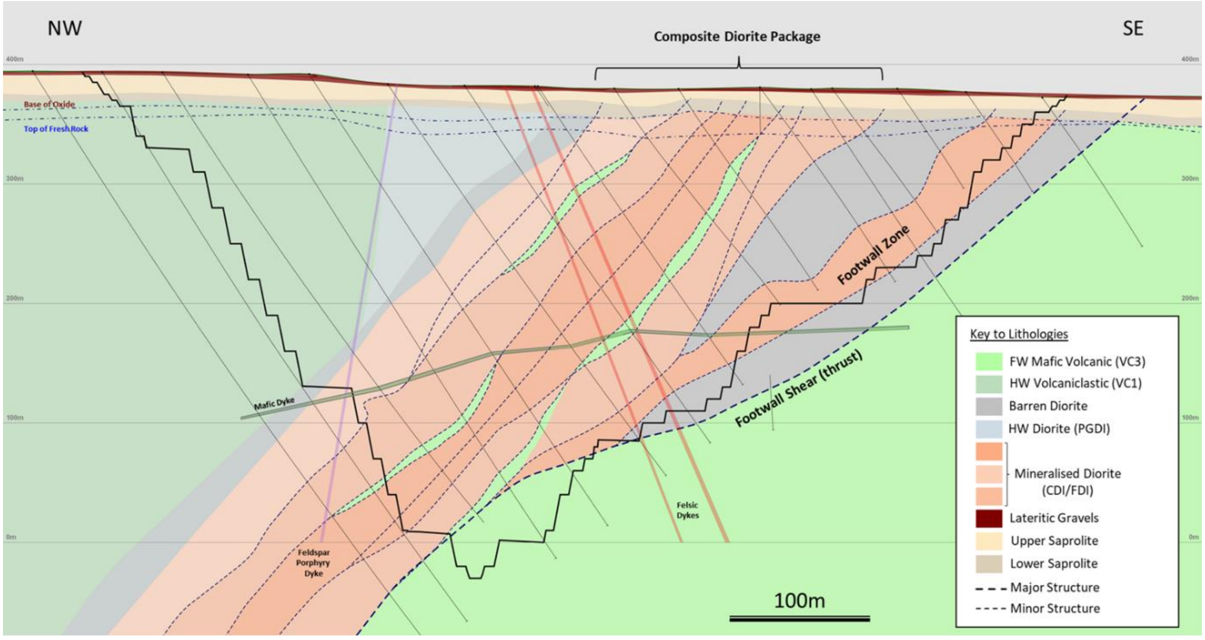
- The first dyke variation displays weak banding and deformation, with residual calcite. Further data needs to be collected to define the orientation, continuity and frequency of these early dykes

- The second dyke variation is a late sub-horizontal version (Figure 18- 4), lacking any foliation or deformation and runs consistently through the deposit, observed clearly cutting foliation.

Felsic dykes (FDY) are light grey in colour, aphanitic, massive and cross-cut the foliation at a high angle. These dykes are not magnetic and are volumetrically subordinate to the other types of dykes. They are consistent on and between sections and can be modelled easily, striking ~015o through the deposit and dipping ~75-80o. Two main felsic dykes have been logged in the main pit, with increased frequency seen to the north (Figure 18-3 and Figure 18-4).

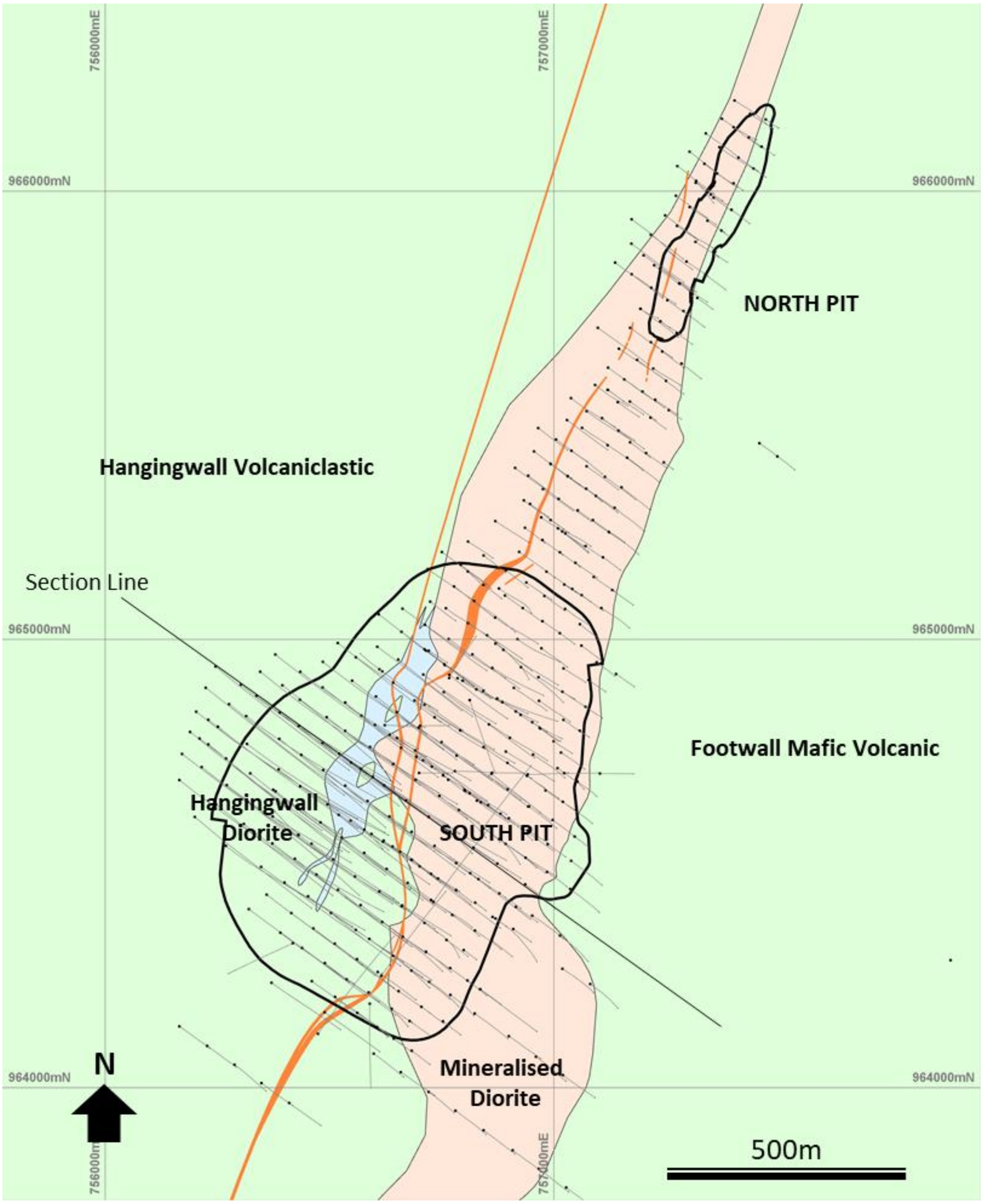
Koné is interpreted to have formed as part of a stacked thrust-shear under a compressional tectonic regime. The major thrust faulting is located at the footwall of the diorite. Very few more significant faults, clays/breccias, are observed within the deposit, with the majority of faulting observed being minor faults displaying both normal and reverse movement.

Figure 18-3 Major geological units – Section



Source: Montage, 2022

Figure 18-4 Major geological units – Plan



Source: Montage, 2022

18.1.2.2 Hydrocensus

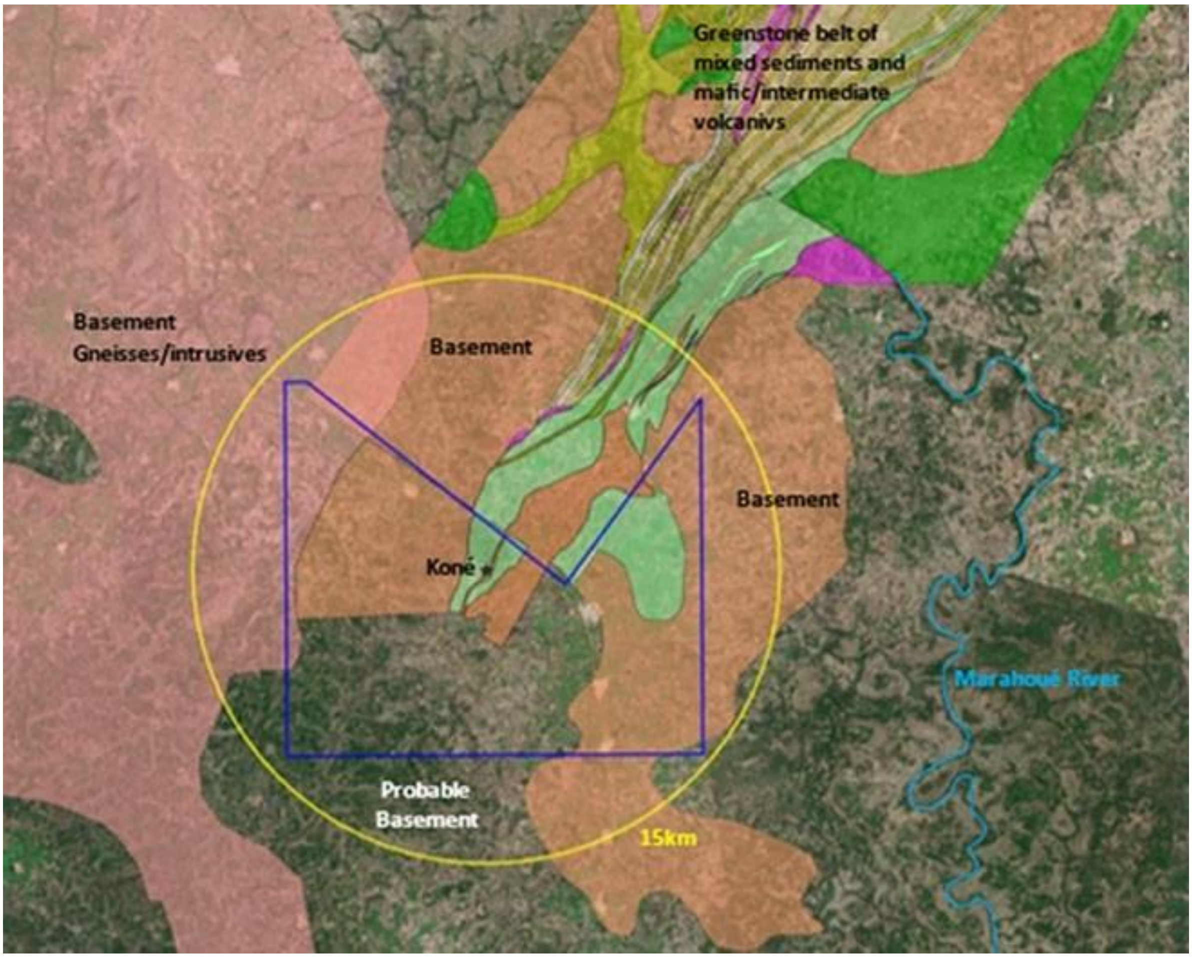
A hydrocensus was completed in 2020 (GCS, 2021) within a 15km radius of the Koné exploration site, from which the following was derived:

- groundwater supply infrastructure is generally out of order and comprises hand and / or foot pumps.
- most of the boreholes equipped with pumps are completed within the transition zone.
- shallow boreholes (wells) are not equipped with pumps and instead, groundwater is obtained through a rope-and-bucket system.
- static groundwater levels range from ~15 metres below ground level (m BGL) to 2 m BGL.
- groundwater samples obtained show neutral pH (6.3 to 6.6) and Electrical Conductivity (EC) ranging from 38 to 65.5 µS/m.
- groundwater is mainly used for domestic and stock watering.

18.1.2.3 Geology and proposed mining

The prospect is located 5-10 km south of the southern end of the Boundiali Basin, probably on the southern continuation of a merged Fonondara / Gbéou Shear Zone (Figure 18-5).

Figure 18-5 Simplified Geological Setting

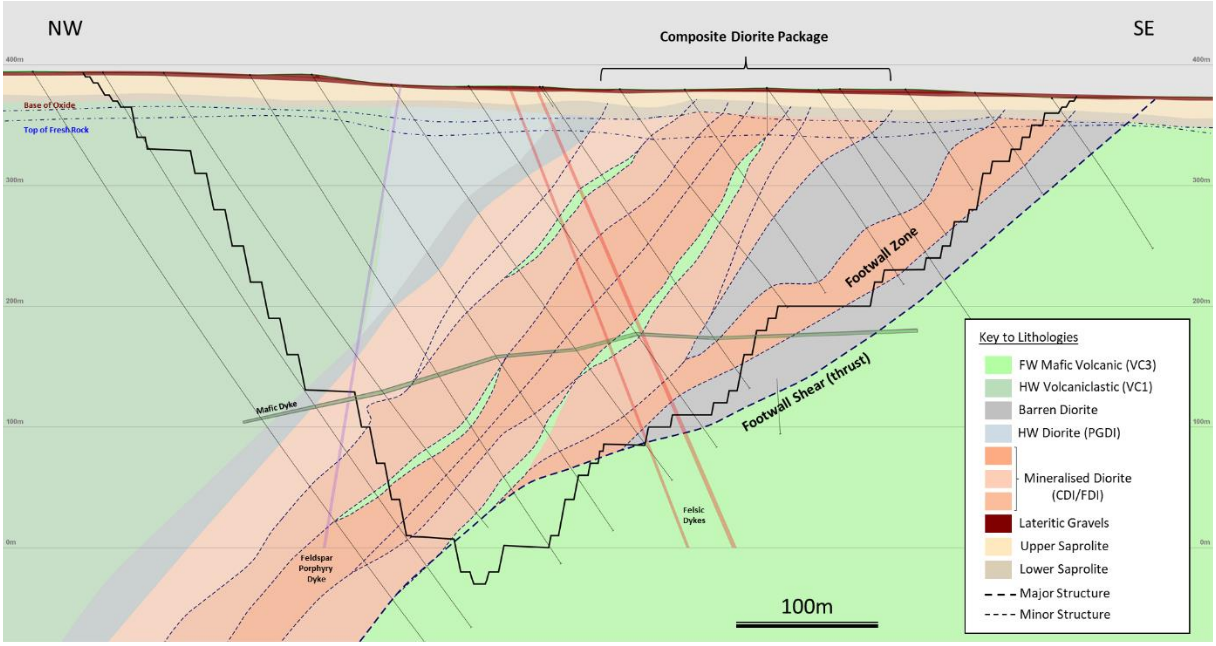


Source: Montage, June 2020

Drilling indicates that the mineralized zones strike NNE and dip approximately 40° towards the west. Drill holes plunge 50° towards an azimuth of 120°, perpendicular to the orientation of mineralized zones in the core of the prospect.

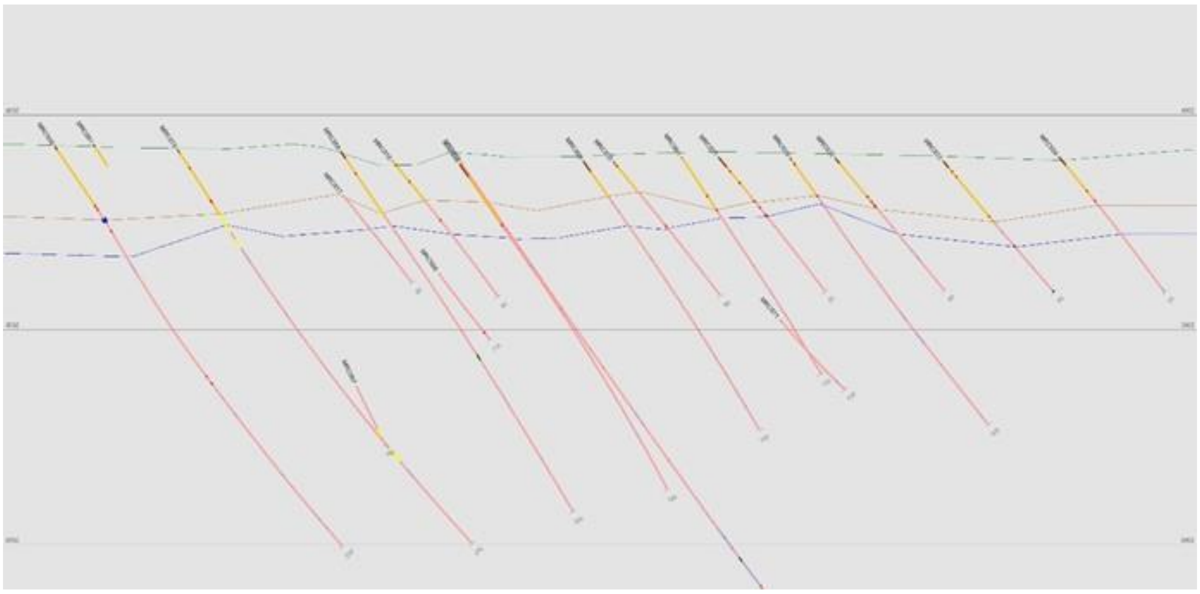
Pit depths will be in the order of 470m (Figure 18-6).

Figure 18-6 Preliminary example of pit dimensions at cross section



The surface area is covered in a saprolitic regolith overburden (soil and clays) layer above the fresh rock which varies in depth. The saprolitic material is presented in Figure 18-7 as the brown line and top of fresh rock as the blue line, the zone in between is a transition zone. The level is fairly consistent, and the thickness varies according to topography, from 0 and 25m below surface.

Figure 18-7 Example of saprolite thickness



18.1.2.4 Aquifer Characteristics

The most widespread aquifer type in Côte d'Ivoire is weathered and fractured basement rocks, which underlie most of the country. Gold projects in the region similar to the Koné Gold Project reported on predicted groundwater inflows into pits ranging between 25 to 60 l/sec.

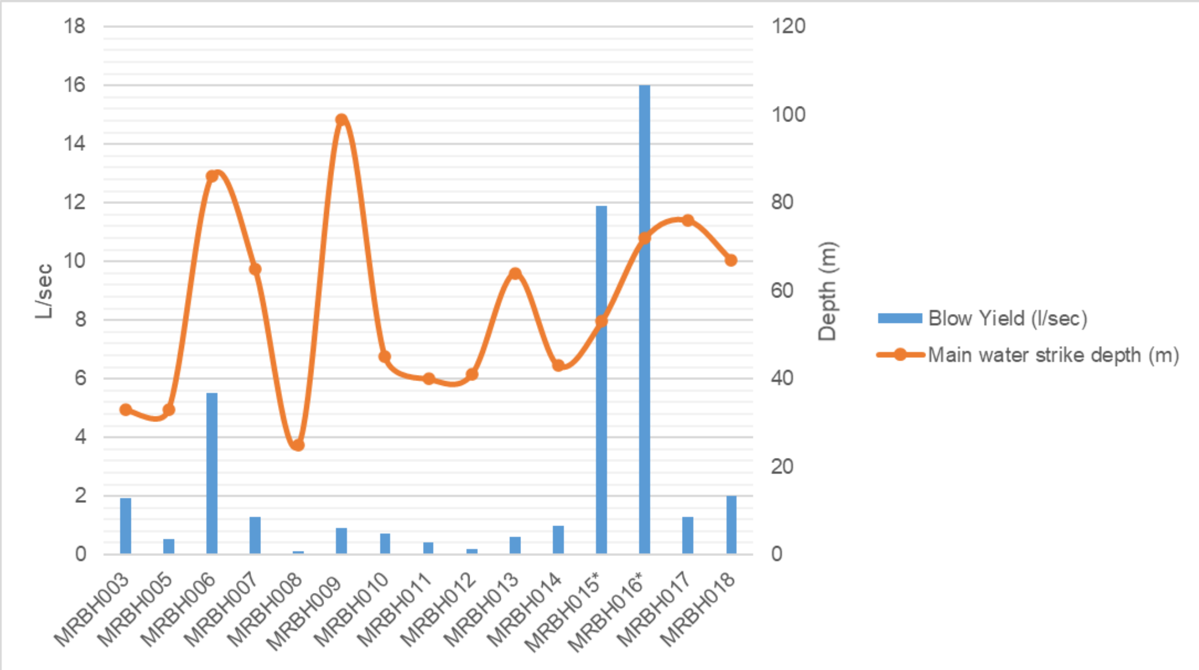
Three hydrostratigraphic units, comprising the shallow (regolith), intermediate (transitional weathered interface with fresh rock) and deep (fresh rock) systems extend across the proposed pit area. The intermediate and deep fractured rock hydrostratigraphic units generally have higher groundwater exploitable potential when comparing to the shallow regolith system. However, shallow systems associated with alluvial zones along surface drainage and rivers have a higher groundwater exploitable potential when comparing to the shallow regolith aquifers.

Three paired bore (shallow and deep) and two additional bores installed in exploratory boreholes were completed within the proposed South Pit area during the initial 2020 groundwater assessment. The shallow bores were completed in the saprolite and served as observation bores for the deep bores situated in the transitions and deep fractured system.

Seven deep test bores were drilled in July and August 2021 targeting the deeper fractured aquifer zones. Observations during the drilling revealed the following:

- the upper saprolite / laterite contain significant clay content and t generally very low inflow
- groundwater inflow generally occurs within two zones representing the majority of groundwater intersections during drilling:
  - Zone 1: Total of 10 water strikes; approximately ~40 m BGL, representing transition, dyke intersections and fractured diorite.
  - Zone 2: Total of 6 water strikes; ~90 m BGL, representing fractured diorite and dyke intersections.
  - only three boreholes had groundwater intersected deeper than 120 m BGL, representing limited fracturing.
- blow yields (airlift yields) obtained during drilling range between 0.1 and 16 litres per second (l/sec) as presented on Figure 18-8 with:
  - three boreholes providing yields greater than 5 l/sec.
  - four boreholes providing yields between 1 and 2 l/sec.
  - eight boreholes providing yields less than 1 l/sec.
- two test boreholes intersected the FDY (16 l/sec) and LGD (14 l/sec), respectively;
- the FDY, LGD and MDY dykes intersected by four boreholes had limited groundwater flow observed.

Figure 18-8. Blow Yields Observed During Drilling



Source: AGE, November 2021

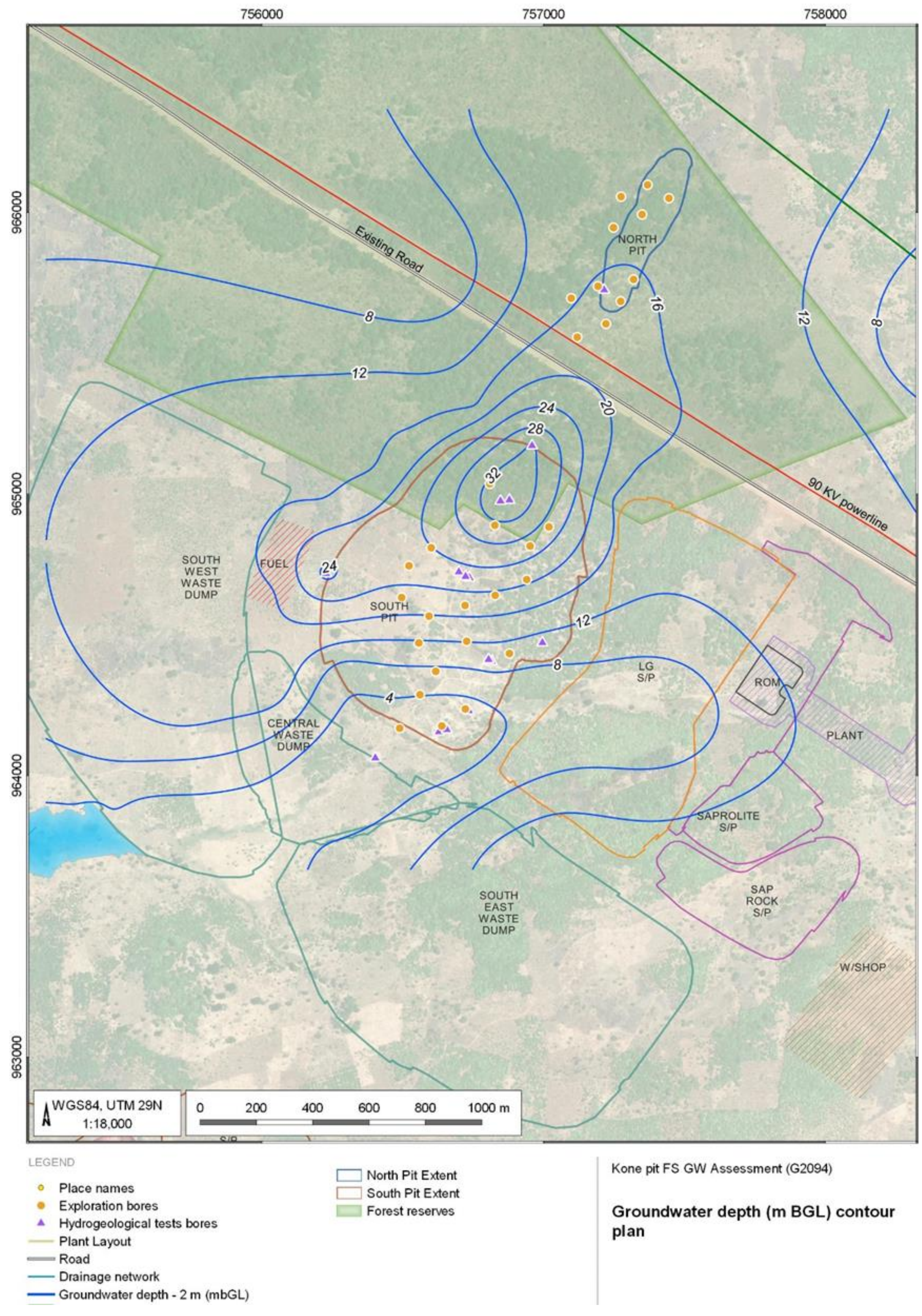
18.1.2.5 Groundwater Levels

The hydrocensus bores show the transition and deep aquifer zones are confined by the saprolite aquitard. The static groundwater levels indicate that:

- the minimum groundwater level is: 0.0 mbgl.
- the maximum groundwater level is 36 mbgl.
- the average groundwater level is 16 mbgl.

Groundwater level contours for the Project area were interpolated from bore water level data by the Kriging method. Groundwater depth below ground surface is presented in Figure 18-9. The groundwater elevation contour plan shows a gentle gradient towards the drainage line south of the South Pit and towards the low topographical zone along the middle section of the North Pit area. The northern portion of the South Pit shows deeper groundwater levels, and the southern portion shows shallow groundwater levels.

Figure 18-9 Groundwater Depth



Source: AGE, November 2021

18.1.3 Aquifer properties

Aquifer hydraulic parameters were derived from the aquifer testing program, which mainly comprised conventional pumping tests. Packer testing was completed through the mine pit and TSF geotechnical assessment programs. Transmissivity ranges between 0.2 and 40 m^2/day and hydraulic conductivity between 0.002 and 1.35 m/day. The Pump tests were analysed using the Cooper and Jacob analytical method through the Aqtesolv software. The transmissivity (T) presented were calculated from the pump boreholes and not from the observation boreholes. The hydraulic conductivity (K) values were derived from the transmissivity by applying the aquifer thickness.

The majority of the hydraulic conductivity estimates are below 0.1 m/day representing the transition zone and various degrees of fractured diorite. Higher hydraulic conductivity estimates above 0.2 m/day are evident from four boreholes representing multiple fracture zones associated with dyke contact zones, fractured footwall diorite, and the footwall shear zone.

The hydraulic conductivity estimates obtained from the packer testing indicates:

- The upper saprolite to be in the order of 0.001 to 0.03 m/day.
- Fractured diorite ranging between 0.06 to 0.1 m/day.
- Diorite matrix between 0.003 to 0.01 m/day.

Aquifer storativity (S) were derived using the observation data. Storativity or the storage coefficient is the volume of water released from storage per unit decline in hydraulic head in the aquifer, per unit area of the aquifer. Storativity is a dimensionless quantity and is always greater than zero. For a confined aquifer or aquitard, storativity is the vertically integrated specific storage value (Ss). Specific storage is the volume of water released from one unit volume of the aquifer under one unit decline in head. This is related to both the compressibility of the aquifer and the compressibility of the water itself.

18.1.4 Groundwater quality

The groundwater has been sampled in 2020 and 2021 and generally, the groundwater quality is within the 2017 WHO Guidelines for Drinking Water Quality. A summary of the available groundwater quality data is tabulated in Table 18-1.

Table 18-1 Groundwater Quality Data

DETERMINANT	UNIT	Ave	Min	Max	2018 WHO Guidelines for Drinking Water Quality
pH		6.7	6.1	7.46	*n/s*
Electrical Conductivity	**mS/m**	46.2	29.6	67.1	*n/s*
Calcium	**mg/l**	62.5	7.769	128.3	*n/s*
Magnesium	**mg/l**	13.2	2.64	22.64	*n/s*
Sodium	**mg/l**	18.9	9.52	32.91	200
Potassium	**mg/l**	5.2	1.17	8.64	*n/s*
Iron	**mg/l**	1.2	0.05	5.36	0.30
Sulphate	**mg/l**	29.5	<5	144.10	250
Chloride	**mg/l**	6.2	<5	18.06	250
Manganese	**mg/l**	0.6	0.03	2.05	0.4
Arsenic	**mg/l**	0.0099	<0.005	0.018	0.01
Nitrate	**mg/l**	0.37	<0.177	2.2	3.00
Fluoride	**mg/l**	0.3	0.1	0.8	1.50
Total Dissolved Solids	**mg/l**	250.4	135	429.2	*n/s*
Aluminium	**mg/l**	0.3	<0.05	3.41	0.2

18.1.5 Hydrogeological Conceptual Model

The key conceptual information of the Study Area groundwater system used to construct the numerical groundwater model is summarized as follows:

- Recharge takes place in the wetter months and the rate depends on the saprolite thickness and rock type. Average recharge values for the Koné site are expected to be in the range of 0.5 to 3% of mean annual precipitation (MAP). The MAP is in the order of 1212 mm/annum, Table 18-2. The relatively low permeability of the clay rich saprolite retards recharge to deeper strata

- The Project area consists of generally low hydraulic conductivity bedrock and groundwater flow is predominantly controlled by local to intermediate-scale strata domains and geological structures leading to zones of higher groundwater flow from secondary porosity and fracturing

- The average groundwater elevation across the proposed mine pit area is 360 m AMSL with a depth around 20 m. Although the groundwater gradient is relatively flat across the mine pit, there is a slight topographical influence

- The average depth of weathering is ~30 m, approximately 10m below the groundwater table. Saturation of the upper saprolite or weathering zone varies between 40 to 5m

- Groundwater was consistently intersected in most of the boreholes at the transition aquifer zone and is on average 45 m deep.

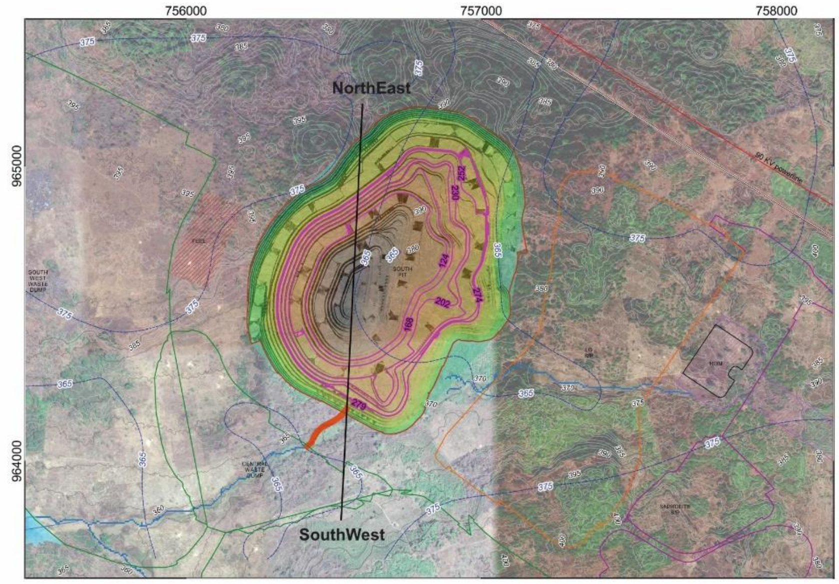
Table 18-2 Weather Station Data

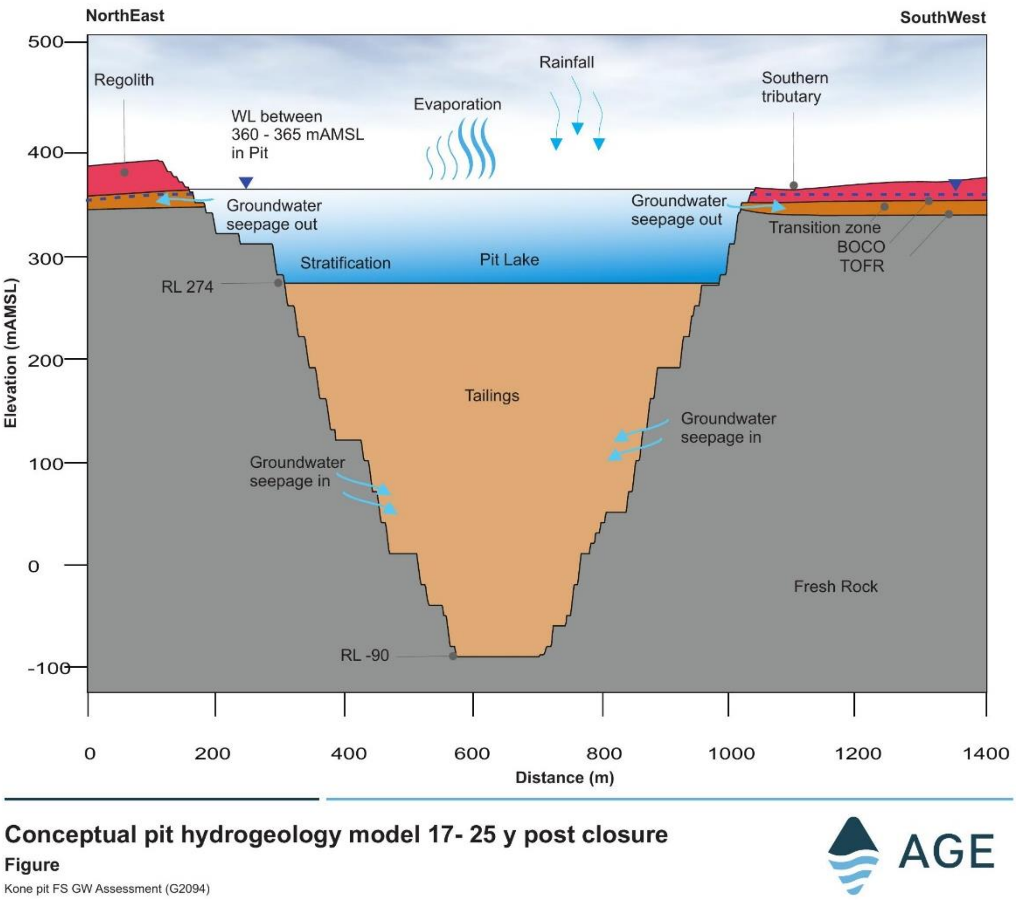
Weather station	Distance from Project (km)	Data period	Average annual rainfall (mm)	Max annual rainfall (mm)	Min annual rainfall (mm)	Max daily rainfall (mm)
KANI	29	1977-1996	1281	1513	1039	90
DIANRA	43	1980-1996	1064	1313	937	117
MORONDO	29	199-1996	1290	1559	1159	141
Average			**1212**	**1462**	**1045**	**116**
Average annual evaporation (mm)						
Annual Average Pan Evaporation			1629			
Annual Average Lake Evaporation*			1140			

Groundwater flow is from recharge areas, often geological structures and outcrops, towards the groundwater outflow boundaries. These outflow boundaries could be local point features, such as springs or diffuse seepage points often associated with drainage lines.

Geozones were identified, and these were converted into hydro-stratigraphic units for the site (Figure 18-10).

Figure 18-10 Conceptual Pit Hydrgeology model





Source: AGE, 2022

18.1.6 Numerical Groundwater Model

The numerical model was developed out of a conceptual model supported by the field observations. The groundwater model for the Project was developed with the following objectives:

- numerically replicate the mine scheduling and backfill process for groundwater dewatering simulation
- assess the area of influence of mine dewatering
- predict the inflow to mines during dewatering
- propose the dewatering wells – positions and pumping rates
- simulate the long-term particle travel from the mine pit representing groundwater movement and direction from pit lake as a source.

Numerical modelling has been undertaken using the MODFLOW 6[3] code (Panday et. al. 2015). Groundwater occurrence, distribution, movement, and properties are influenced by the site's geology and key system stresses. System stresses include inputs (i.e. rainfall recharge and river leakage) and outputs (i.e. upward and downward leakage, interception through mine dewatering, and baseflow to surface drainages). Based on the mining plan and considering the south pit and north pit have different lifespans, a total of eight models are constructed to simulate two general scenarios.

MODFLOW 6 with DISV grid structure[4] was used to accommodate the local refinement on significant hydrogeological features, such as the mine pits, the TSF and WSF, dykes, streams, etc. (Figure 18-11).

The surface area covered by both planned mine pits is approximately 0.98 km^2. The proposed model domain covers an area of 125.4 km^2. The western boundary is the Yarani River which is a perennial river flowing from north to the south and located 4.4 km from the mine site. The northern and southern boundaries were delineated from natural watershed boundaries, assumed as no-flow boundaries, and are approximately 5.6km from the proposed mine pits. The eastern boundary was delineated to include part of the surface drainage in the east and a no-flow boundary was applied.

Several surface water drainage lines are included in the model extent and the River Package (RIV) is used to simulate the groundwater and surface water interaction for the perennial rivers. The non-perennial drainage lines that feeds the main rivers are simulated using the drain package (DRN)

Mine dewatering is simulated by a relatively high hydraulic conductance value of 100 m^2/day.

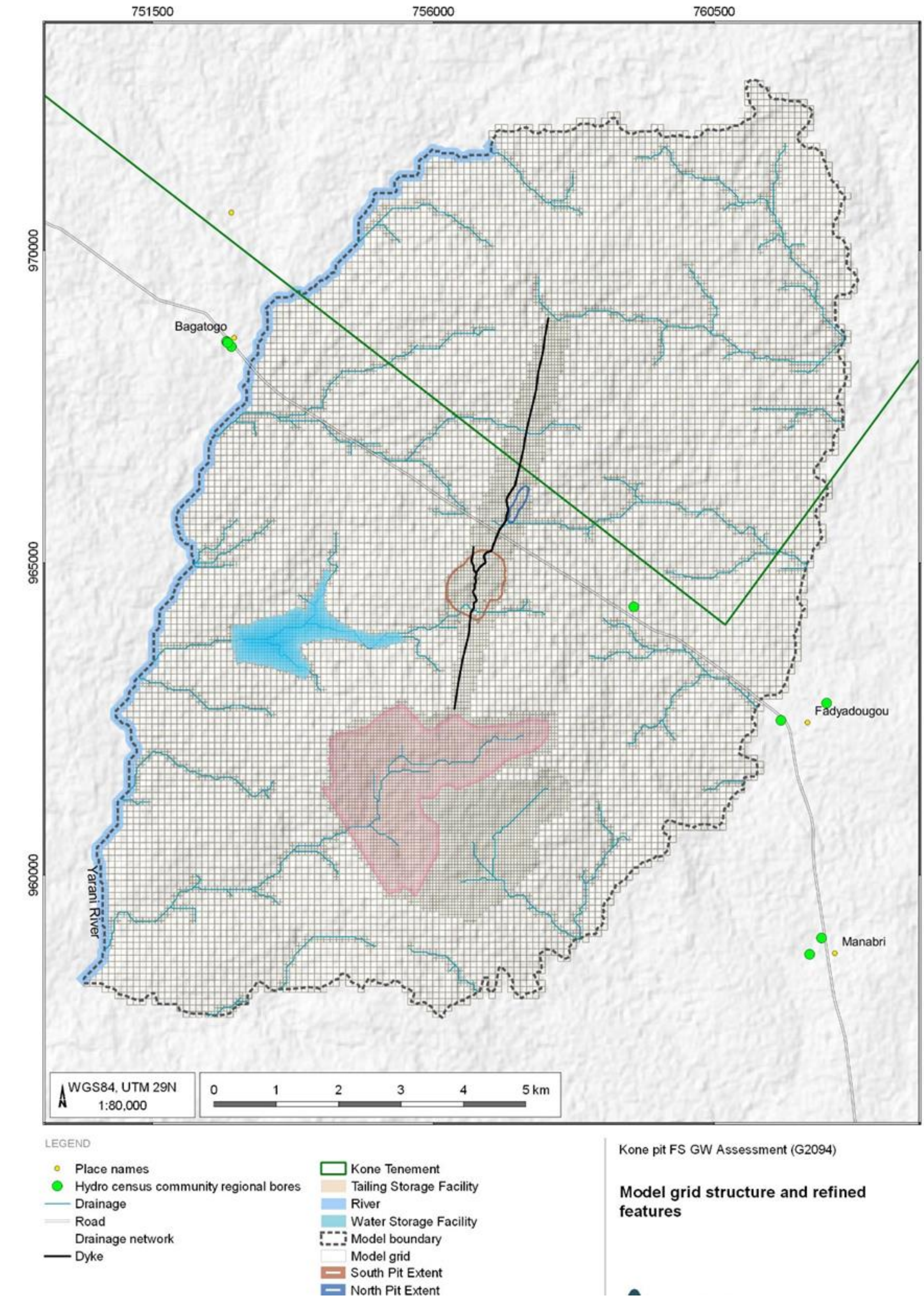
Following the completion of the mining in year nine, tailings will be deposited into the pit via four spigots located around the perimeter of the pit. This will allow the tailings deposition to be rotated to avoid interruption with decant pump, which will need to be maintained close to the pit ramp to facilitate access for maintenance. The pumps will be moved progressively up the ramp as the tailings level increases. Water will be extracted from the decant pond using floating intake lines to position the pumps above the pond elevation. The pond volume will be at its highest at the first year as the TSF pond will be pumped to the pit to let the TSF commence the closure process. The pond will be gradually pumped back to process plant and the pond will be smaller in the final years of operation.

There will be no underdrainage system to recover water from the base of the deposited tailings. The spigots will comprise solid dropper pipes down to the lowest crossing with the pit ramp. Tailings will then discharge from the open end of the pipe. The dropper pipes will need to be trimmed to each ramp crossing as the tailings level increases. This is to ensure that the pit ramp remains accessible throughout the deposition

At the conclusion of mining in the south pit the maximum depth will be -75 m AMSL. Tailings disposal will start from year nine and the elevation for the rate of rise is shown in Figure 18-12. At the time of the mine closure, tailings will have reached 279 m AMSL, which is ~55 m below the transition aquifer and ~75 m below the average static groundwater level. After closure, the water level is expected to rise fairly rapidly due to the positive water balance.

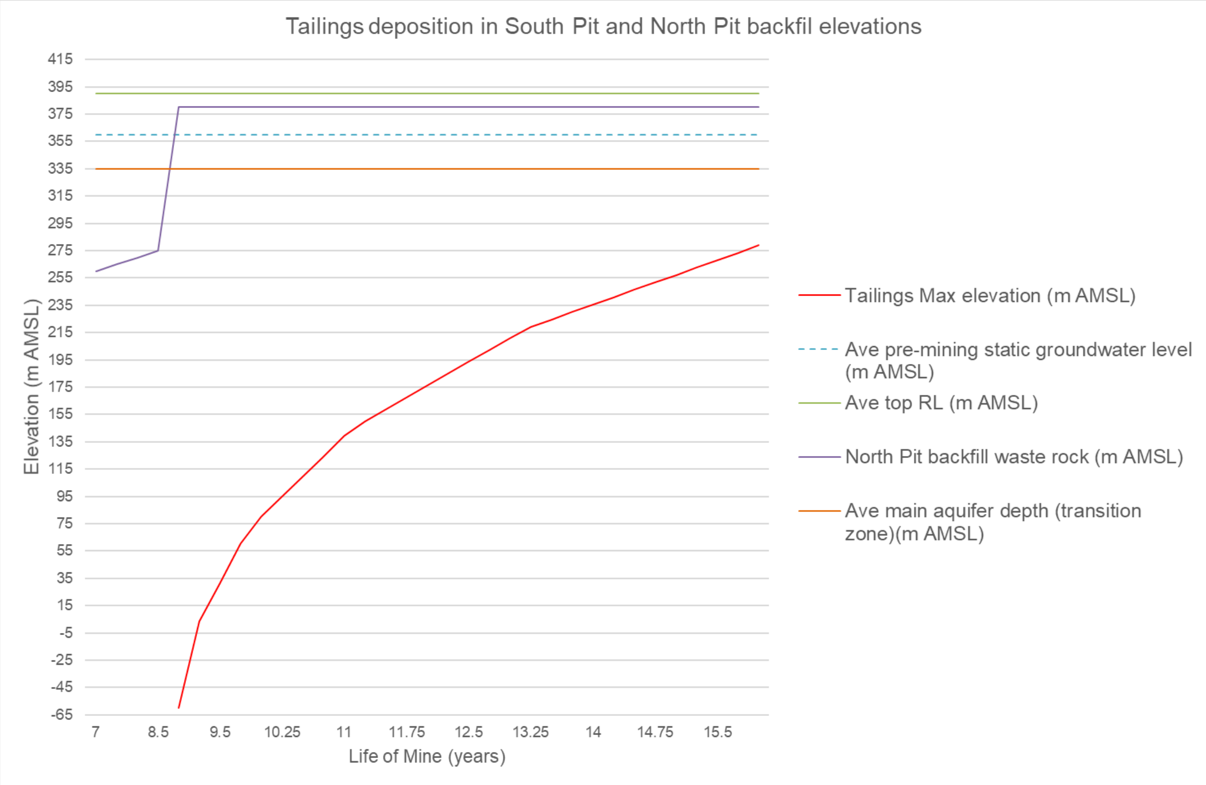
The dominant mechanism for recharge to the groundwater system is diffuse rainfall infiltration through the soil profile and subsequent drainage to underlying groundwater systems.

Figure 18-11 Model Grid Structure



Source: AGE, November 2021

Figure 18-12 Tailings Disposition Elevation in South Pit

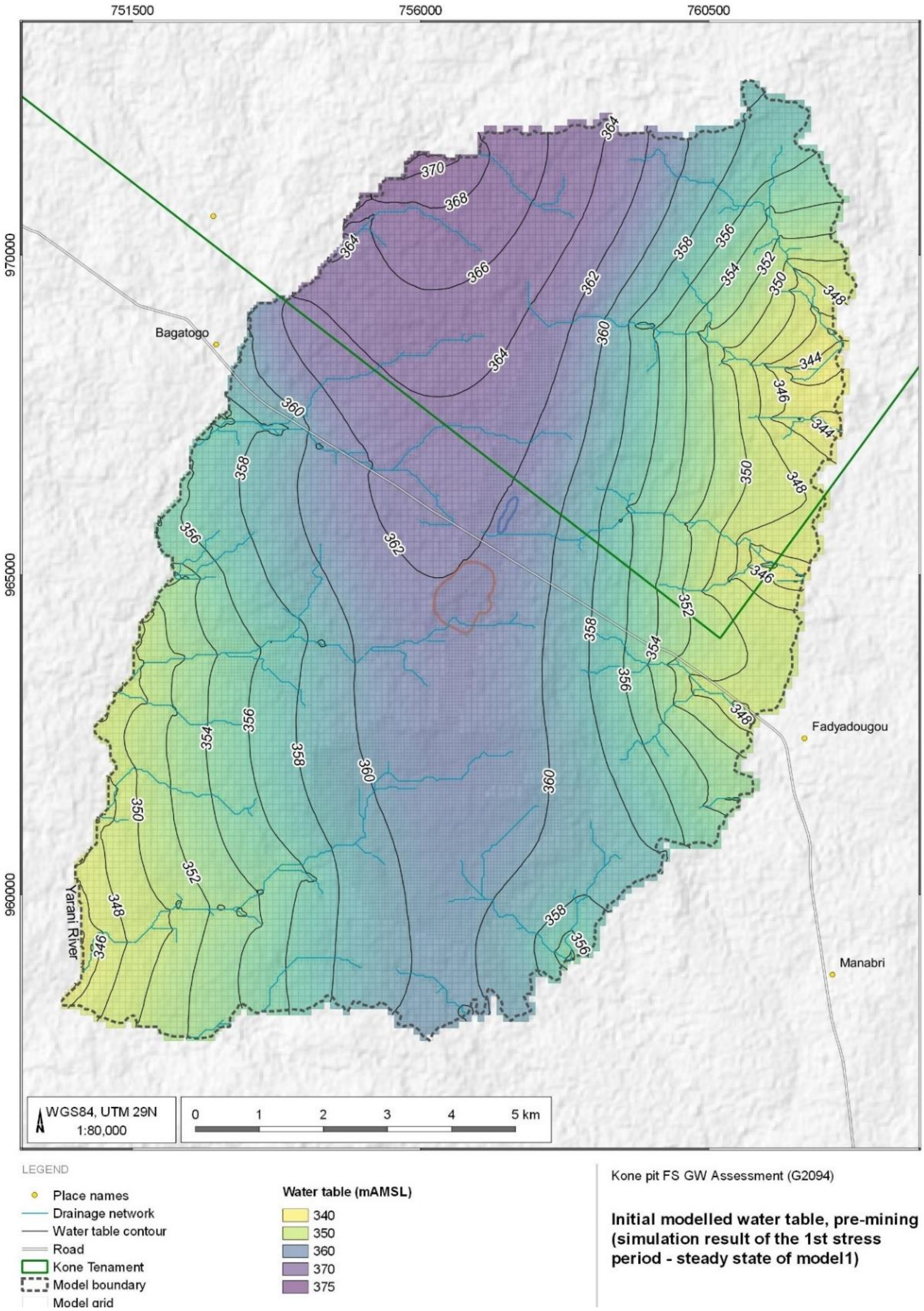


Source: AGE, November 2021

The following groundwater level maps present the different stages of pre-mining and operational groundwater levels:

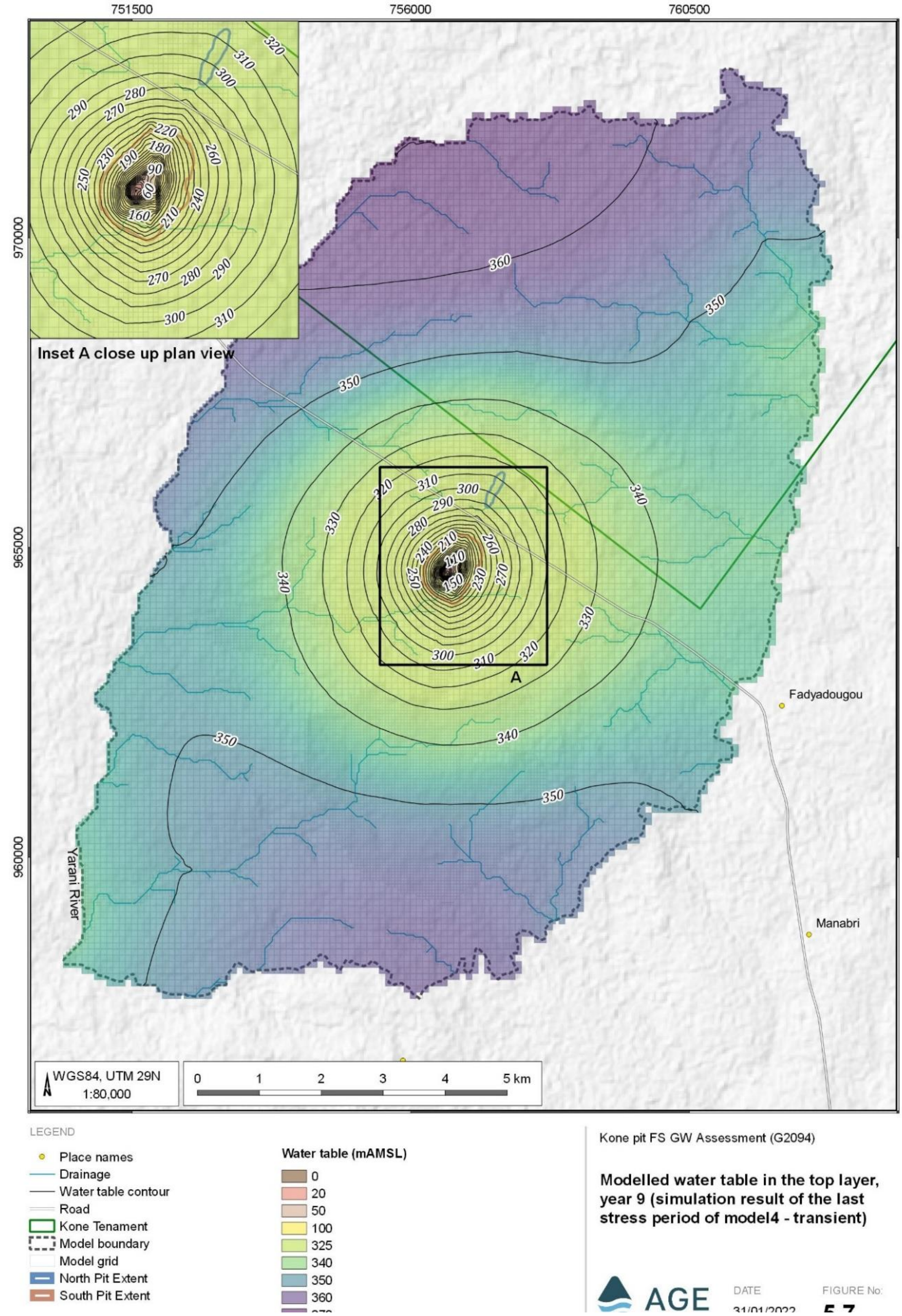
- Figure 18-13 shows the steady-state groundwater level for the model domain under the natural state
- Figure 18-14 shows the maximum extent of de-watering in year nine
- The cumulative drawdown by the end of mining is shown in Figure 18-15 which shows the predicted 2 m drawdown zone of influence is approximately 4.5 km from the mine and does not include any known community groundwater supply assets.

Figure 18-13 Pre-mining Water Table



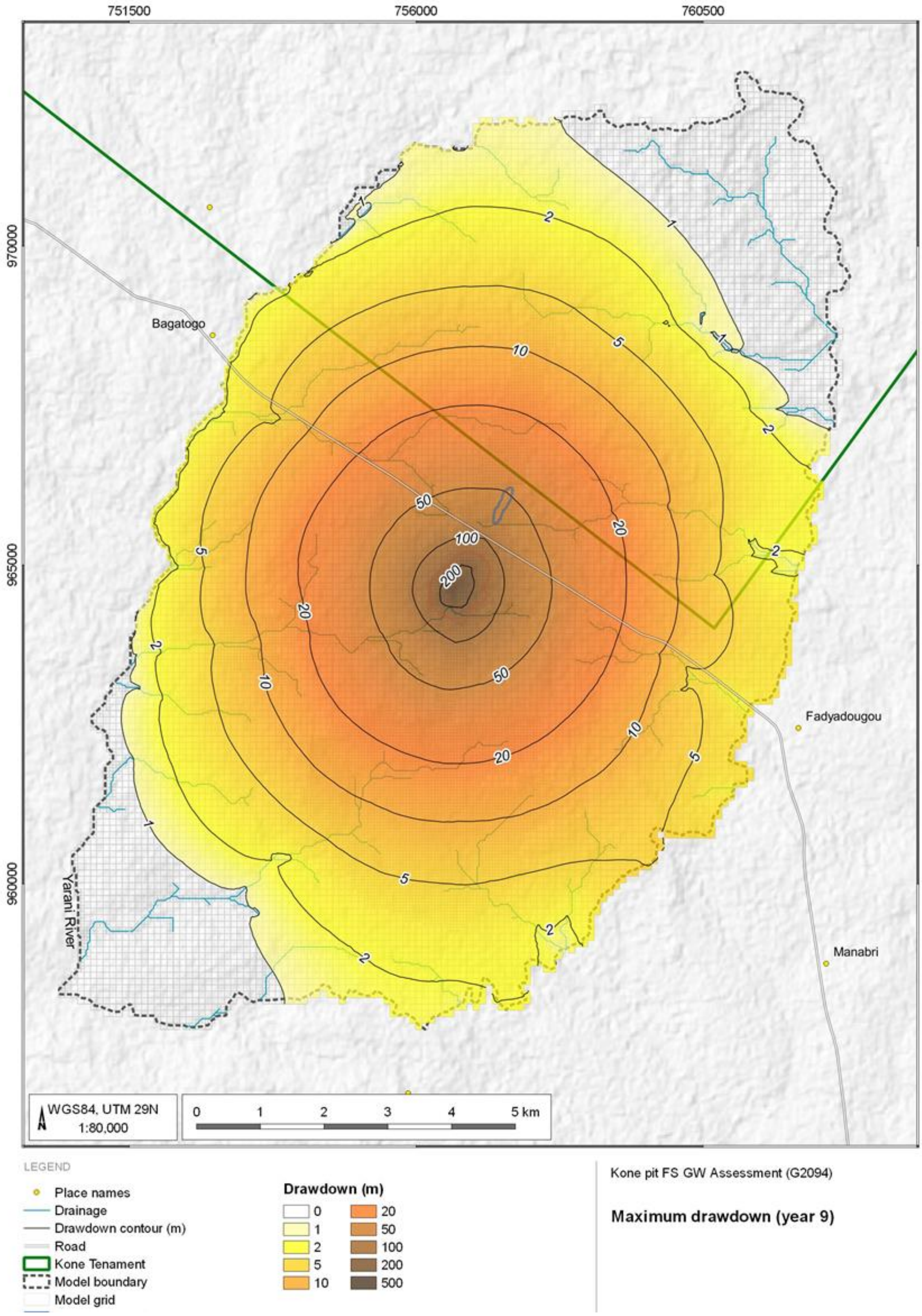
Source: AGE, November 2021

Figure 18-14 Post mining Water Table



Source: AGE, November 2021

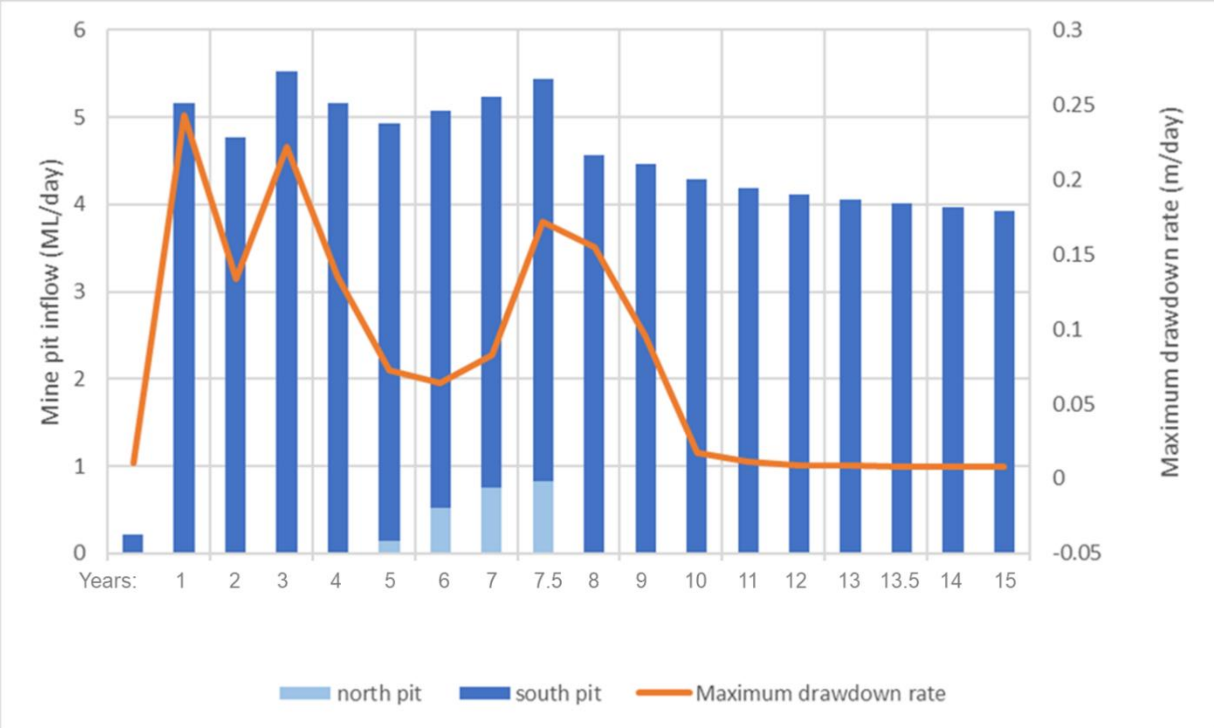
Figure 18-15 Cumulative Water Table Drawdown



Source: AGE, November 2021

The maximum drawdown rate (m/day) and the simulated mine pit inflow for each transient stress period are plotted in Figure 18-16. For all models and stress periods, the total and maximum inflow is in the range of 4 to 5.5Ml/day.

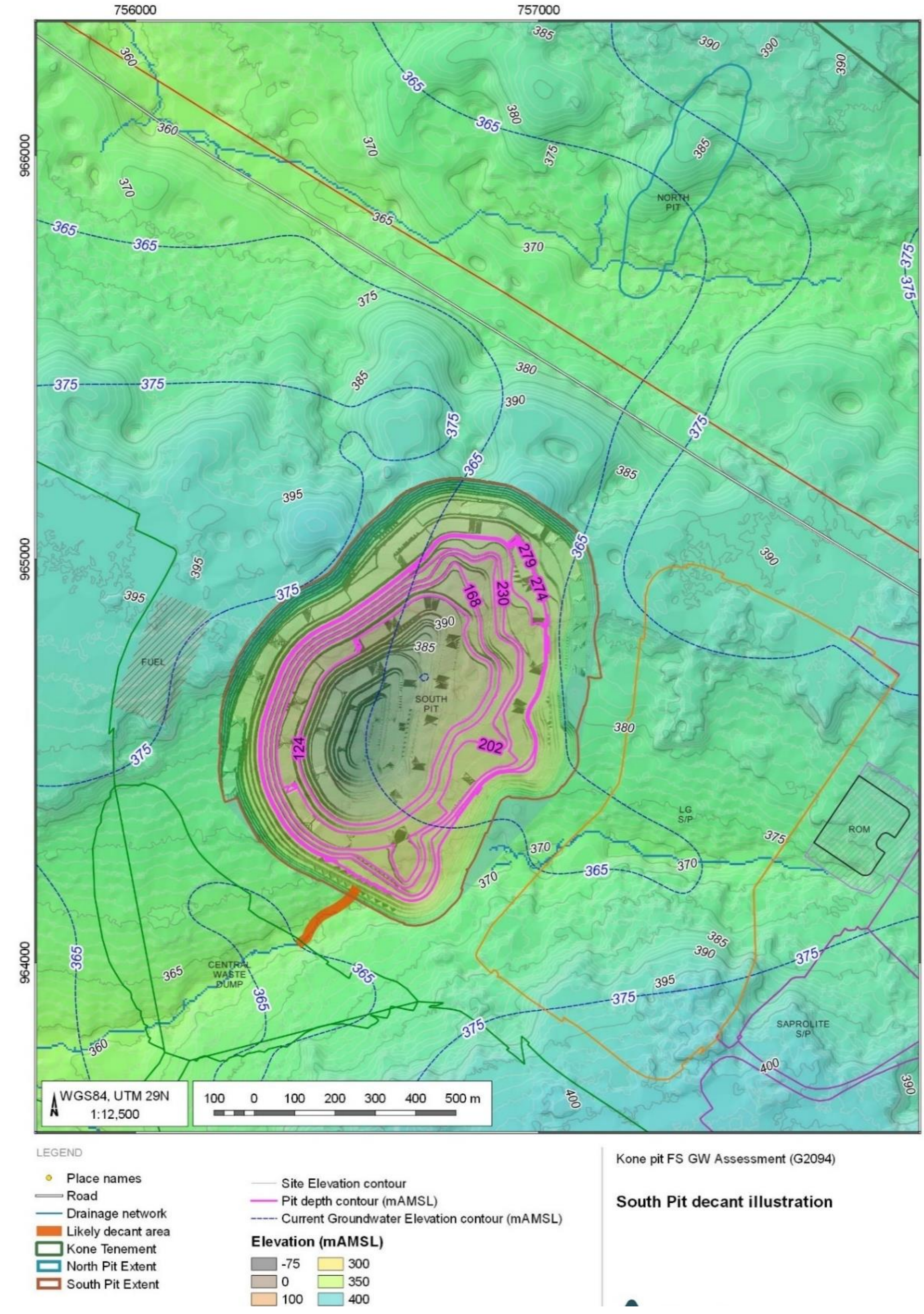
Figure 18-16 Predicted ground water inflow



Source: AGE, November 2021

The in pit tailings disposal takes seven years to reach the planned elevation of 279 m AMSL. The pre-mining groundwater level of ~361 m AMSL is reached between 17 to 25 years after dewatering is stopped. Due to the positive water balance, the groundwater level rises above the original level as the mine pit becomes a water source from this period onwards. This will lead to the decanting of the open pit after closure, Figure 18-17.

Figure 18-17 South Pit decant



Source: AGE, November 2021

18.1.7 Tailings disposal in South pit

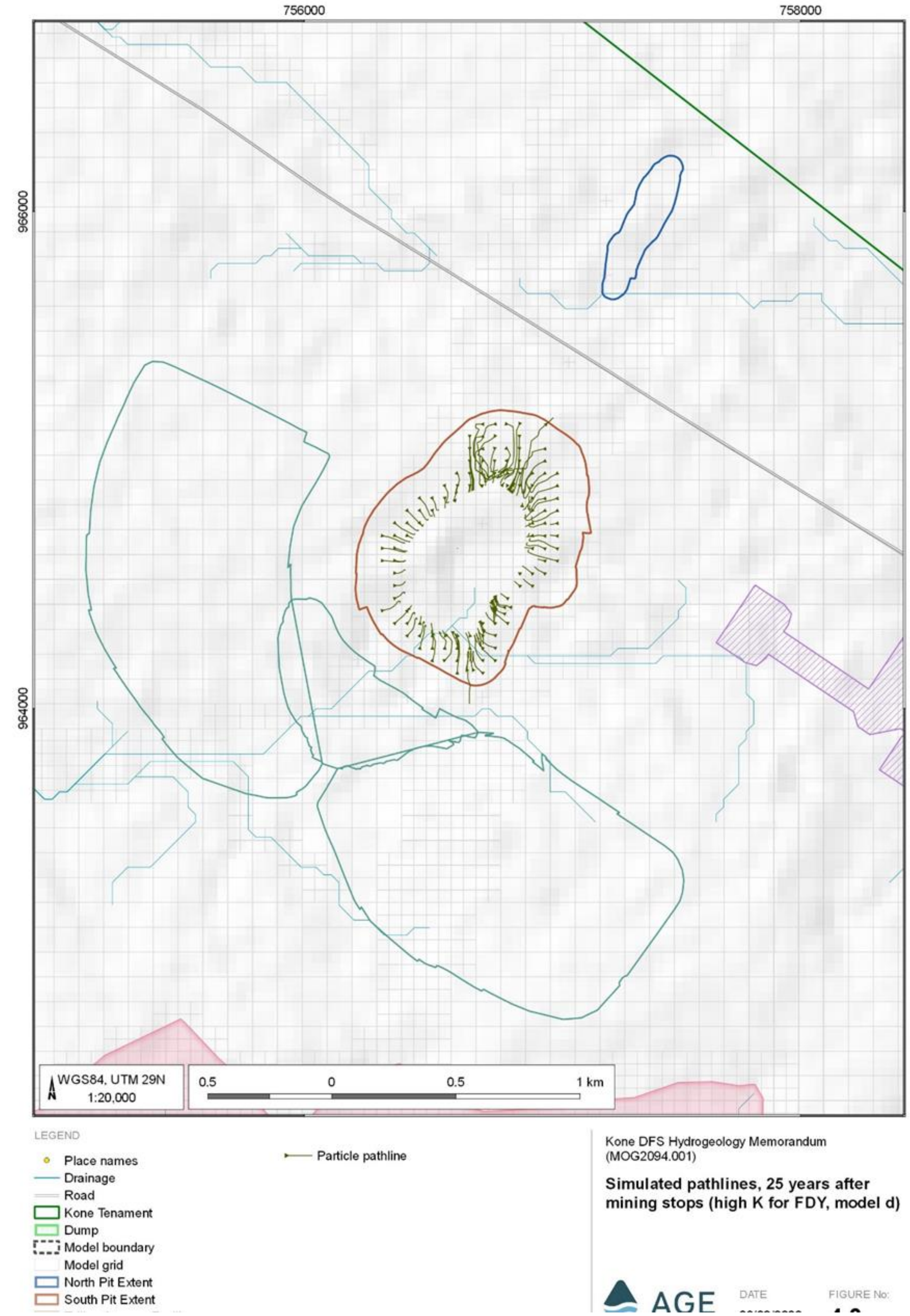
At the time of deposition into the pit the processed tailings may contain elevated values of pH, copper, iron, nitrate and cyanide. However, the detailed modelling of the groundwater environment through the mine life and after closure shows clearly that due to natural degradation and dilution with surface and groundwater, by the time groundwater starts to flow away from the pit area the concentrations of cyanide will be reduced to be within identified guideline values

A geochemical model was developed to simulate the likely range of cyanide concentrations that will remain in the pit water after mine closure. The model results were then used in a mixing model to present a dataset that undergoes mixing and dilution with natural sources. The model predicted that nearly all of the cyanide would oxidise over a 35 day period. Oxidation of cyanide by UV light is a significant degradation pathway.

The mixing modelling has been undertaken in two stages. First, rainwater and groundwater are mixed based on the median volumetric proportions estimated by the water balance modelling and the median concentrations of solutes analysed in groundwater. Second, the mixture of rainwater and groundwater is mixed/reacted with the median volume of "supernatant release water". The cyanide that will be transported will be within environmentally acceptable ranges. Additional test work and modelling will be carried out to confirm the concentrations of other parameters.

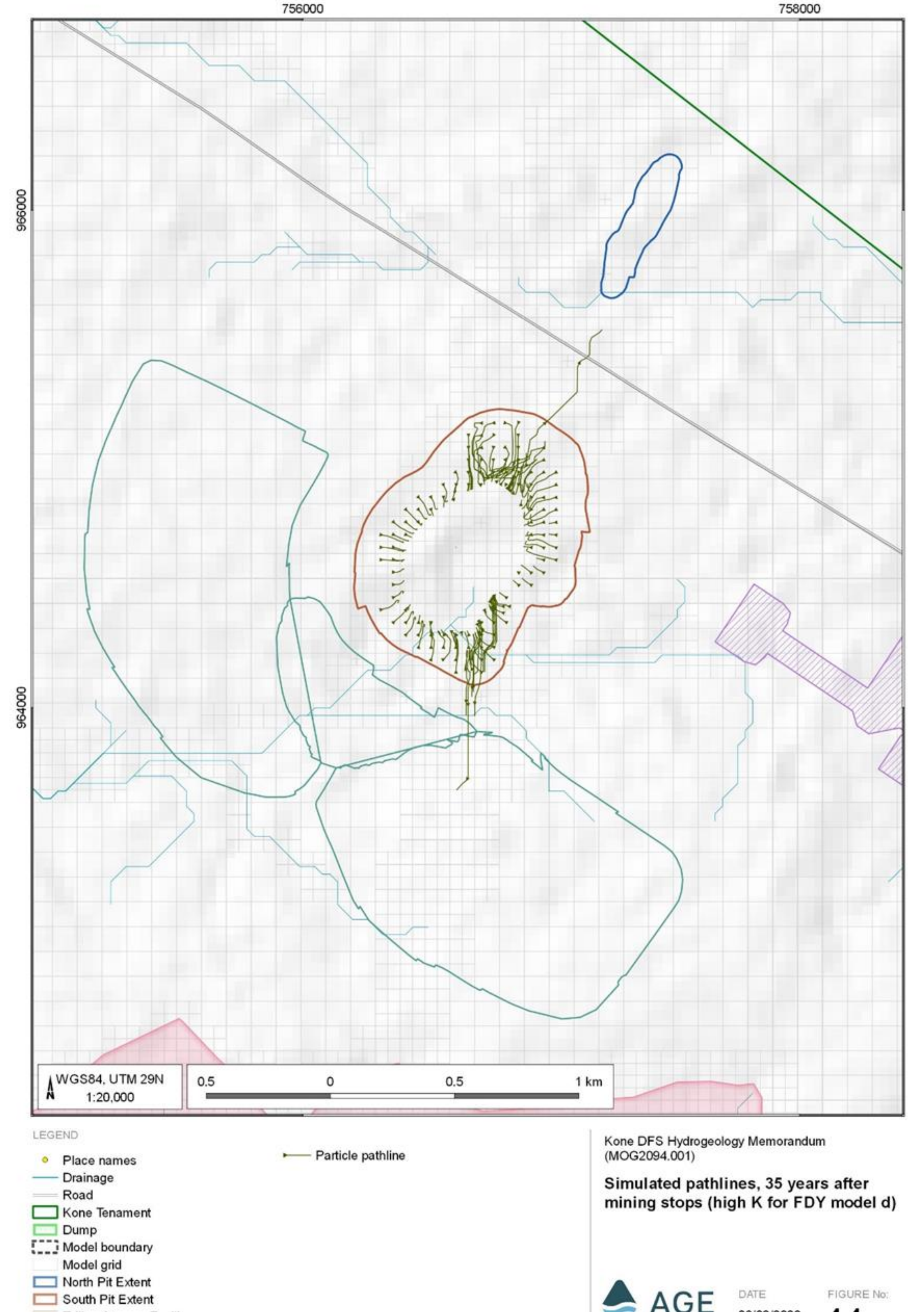
The model generated pathlines for 25 years, 35 years, and 100 years are shown in Figure 18-18, Figure 18-20 and 18-20. By 35 years, the first particles are projected to move outside the open pit confines. By 100 years, the preferential dyke flow may take effect with the pathlines expand in the N-S direction. Figure 18-22 shows the difference between the average and maximum projected permeabilities found through the testwork.

Figure 18-18 Simulated pathlines - Year 25



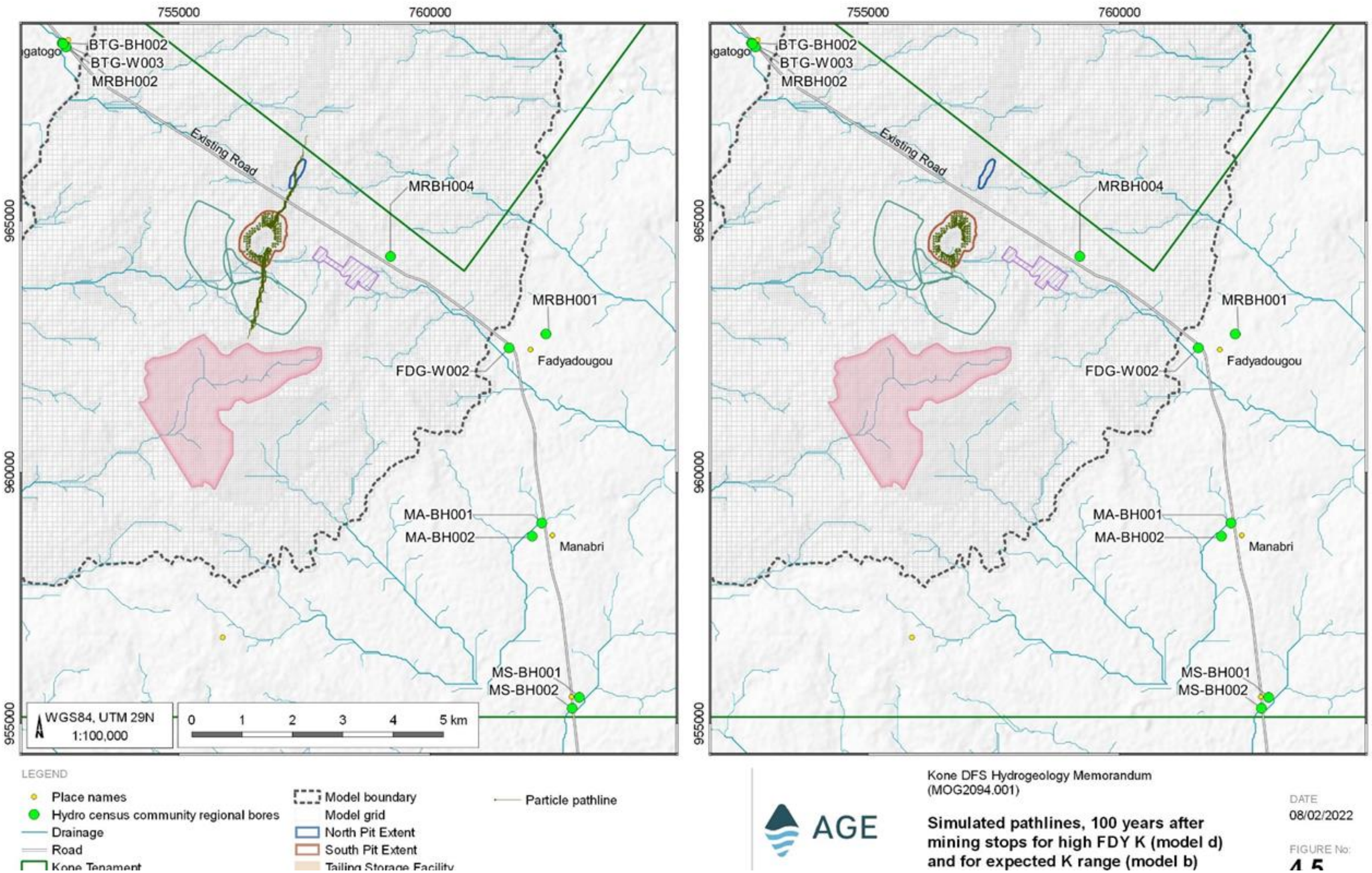
Source: AGE, 2022

Figure 18-19 Simulated pathlines - Year 35



Source: AGE, 2022

Figure 18-20 Simulated pathlines - Year 100



Source: AGE, 2022

18.1.8 Environmental Assessment

The numerical groundwater model and work completed has not identified the potential environmental impacts to any identified groundwater asset (community supply borehole) or groundwater dependant ecosystems from either mine drawdown or the potential for transport of any constituents of concern via the identified shallow and deep aquifer systems.

The South Pit will become a steady state pit lake after about 11 to 19 years after the end of operations and will decant during the wet season.

The decant water will be captured in the WSF and is unlikely to have any chemical constituents above the Ivory Coast discharge guidelines. Additional work is underway to model the likely range of concentrations of chemical elements.

18.1.9 Mine Pit De-Watering

Pit dewatering for the Koné FS Project is proposed in two stages:

- Pre- mining, using dewatering boreholes, targeting the high yielding areas depicted from recent drilling results to provide additional time for drainage of the less permeable geology units;
- operational dewatering through conventional dewatering methods.

The saprolite material has relatively high groundwater storage compared to the underlying unweathered bedrock with poor drainage properties. Regional groundwater levels in the saprolite are sustained by recharge and seasonal infiltration of surface waters.

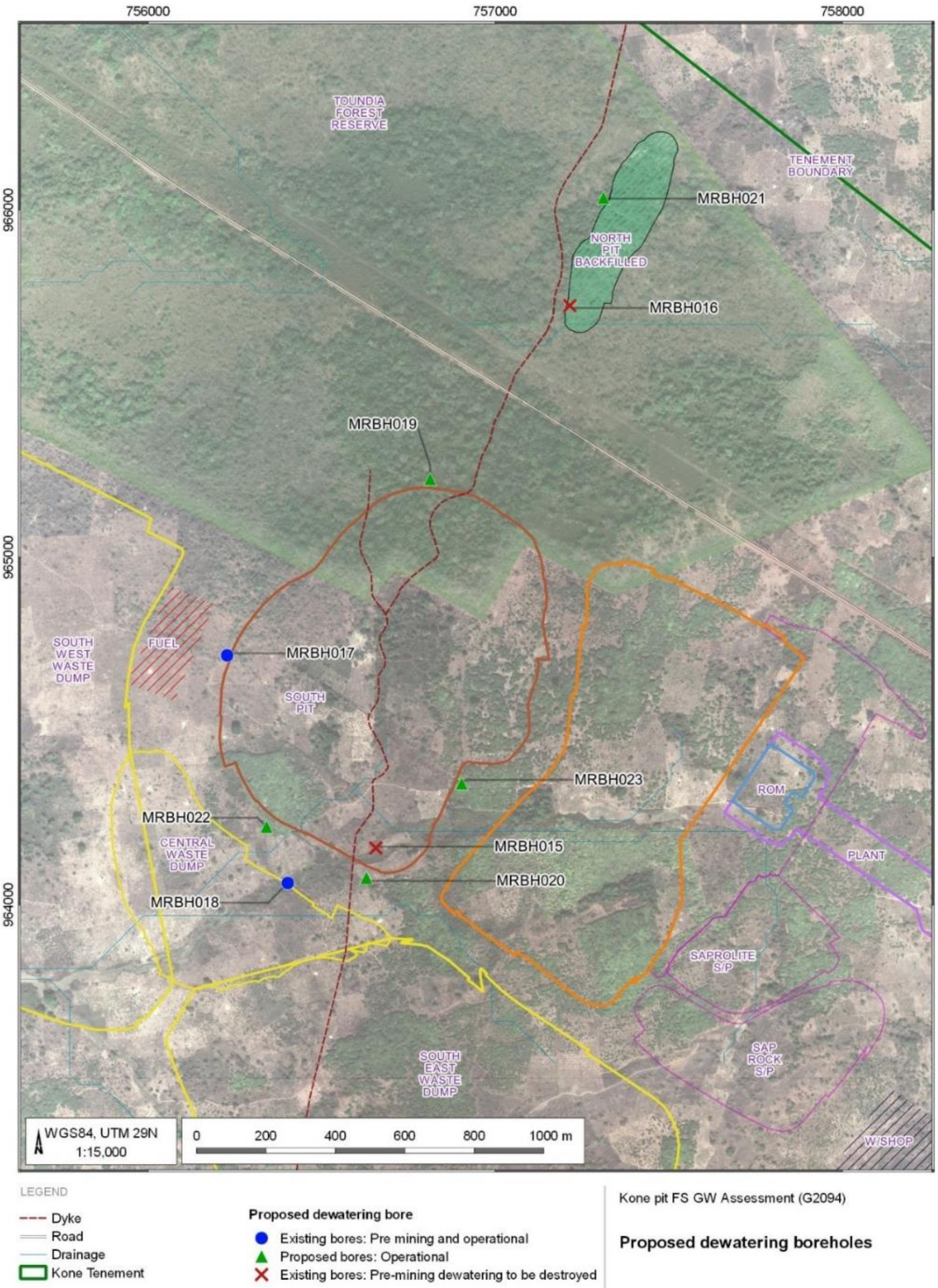
Four of the test boreholes from the 2020 and recent hydrogeological drilling program will operate as dewatering bores in the six to twelve months prior to mining starts. The groundwater abstracted should be discharged and stored in the WSF for mine construction usage. Approximately 1,800 m^3/day be pumped from the boreholes.

Five additional dewatering boreholes will be added and in total, seven (7) dewatering bores, Figure 18-21, should be operated to abstract approximately 1,700 m^3/day. The water can be directly discharged into the southern drainage line, which reports to the WSF.

Groundwater inflow and rainwater runoff into the open pit will drain towards the lowest part of the open pit and be collected in an in-pit sump from where it will be pumped to the WSF. It is recommended that the minimum in-pit pump capacity should be ~8 000 m^3/day (250 m^3/hr).

Approximately 70 to 80% of dewatering will be from the in-pit sump and 20 to 30% from the external dewatering boreholes.

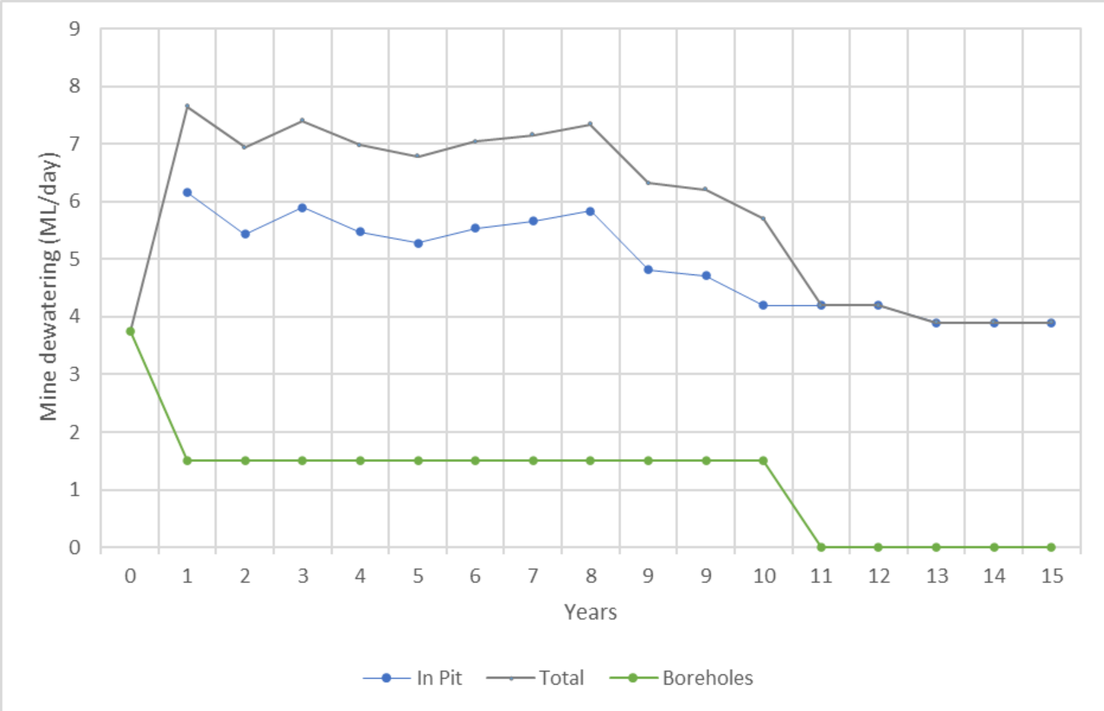
Figure 18-21 Dewatering Borehole Locations



Source: AGE, November 2021

The total in pit and out of pit dewatering is presented in 18-22.

Figure 18-22 Dewatering Volumes



Source: AGE, November 2021

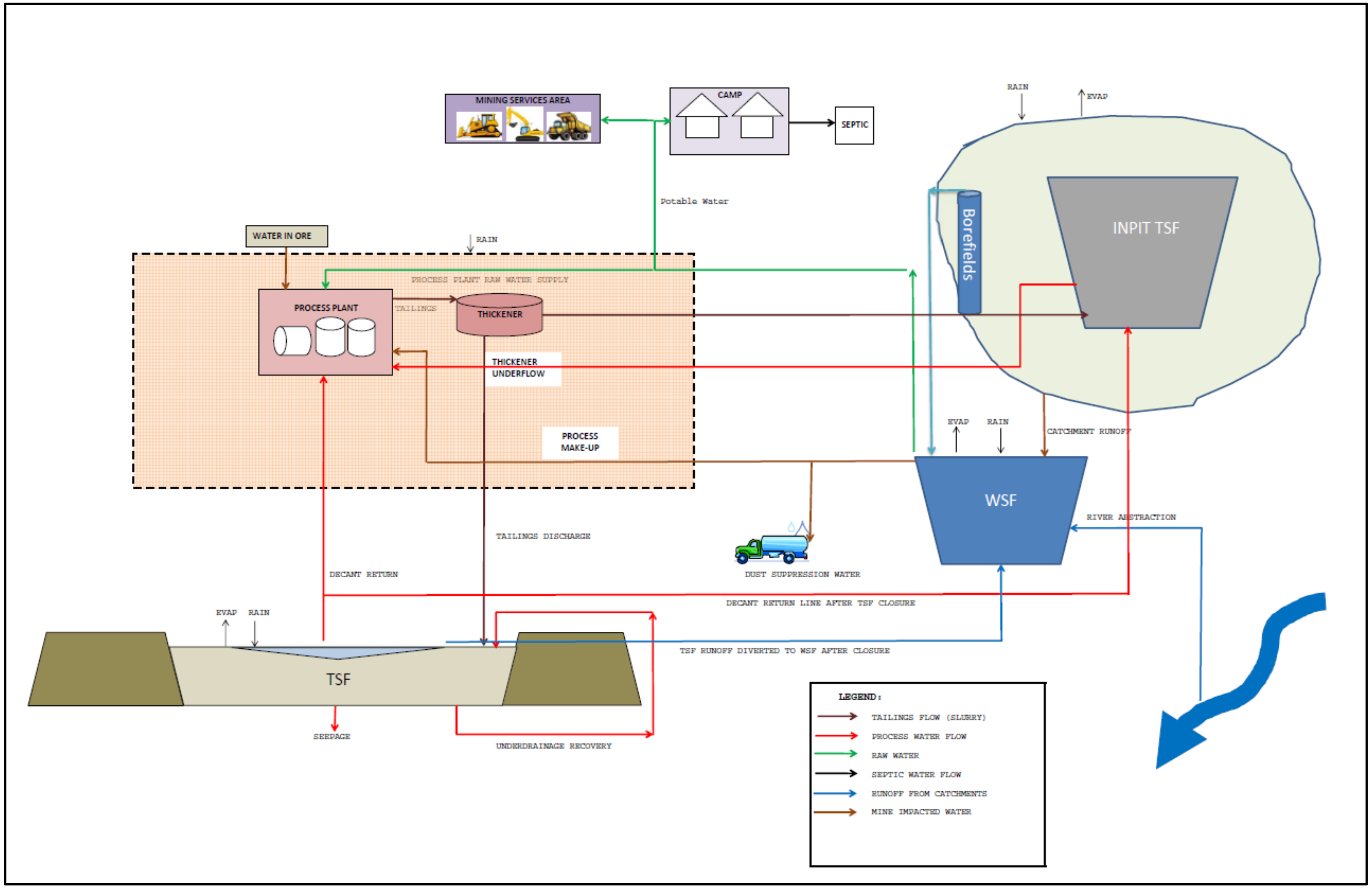
18.1.10 Water Balance Modelling

Water demand for the mineralization processing, potable water and dust suppression demands will be supplied to the project from tailings storage facility recycle, river harvesting, pit dewatering, harvesting of runoff from the mining disturbance areas and a supplementary borefield for potable water supply. A preliminary water balance model has been established to estimate the water demands for the project, to assess the availability of water to meet the demands and to size the various component of the water management system.

A detailed probabilistic water management model has been established to estimate the water demands for the project, to assess the availability of water to meet the demands and to size the various components of the water management system. The water balance model has been run using 38 years of historical rainfall and river flow data and has been run as a Monte Carlo simulation using calibrated Australian Water Balance Model (AWBM) parameter for the river flow and probabilistic climate data with 100 iteration runs for each year of operation.

For clarity only average conditions are presented with this report, however the modelling run indicate sufficient water is available in all runs to meet the projects demand. The schematic flow diagram for the model in shown in Figure 18-23.

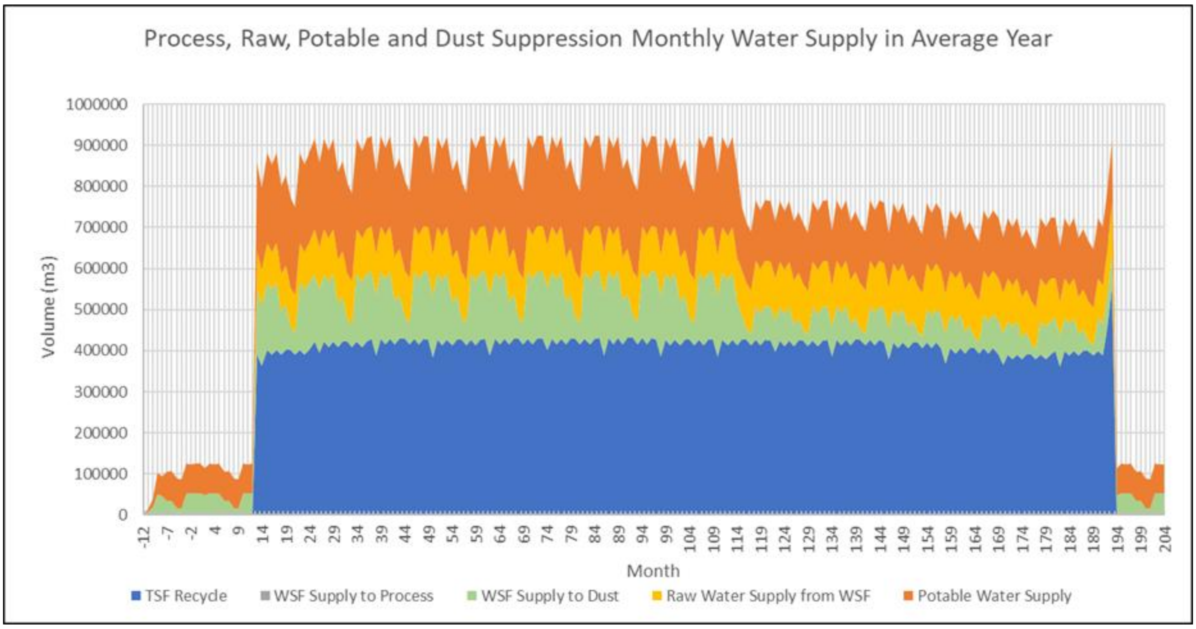
Figure 18-23 Water Balance Schematic

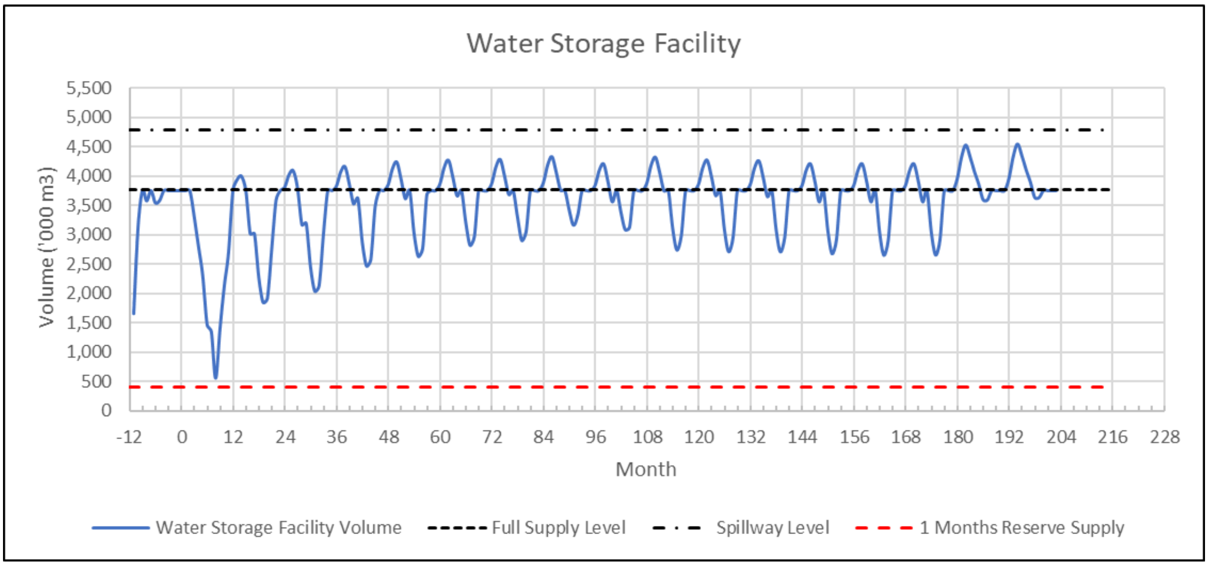


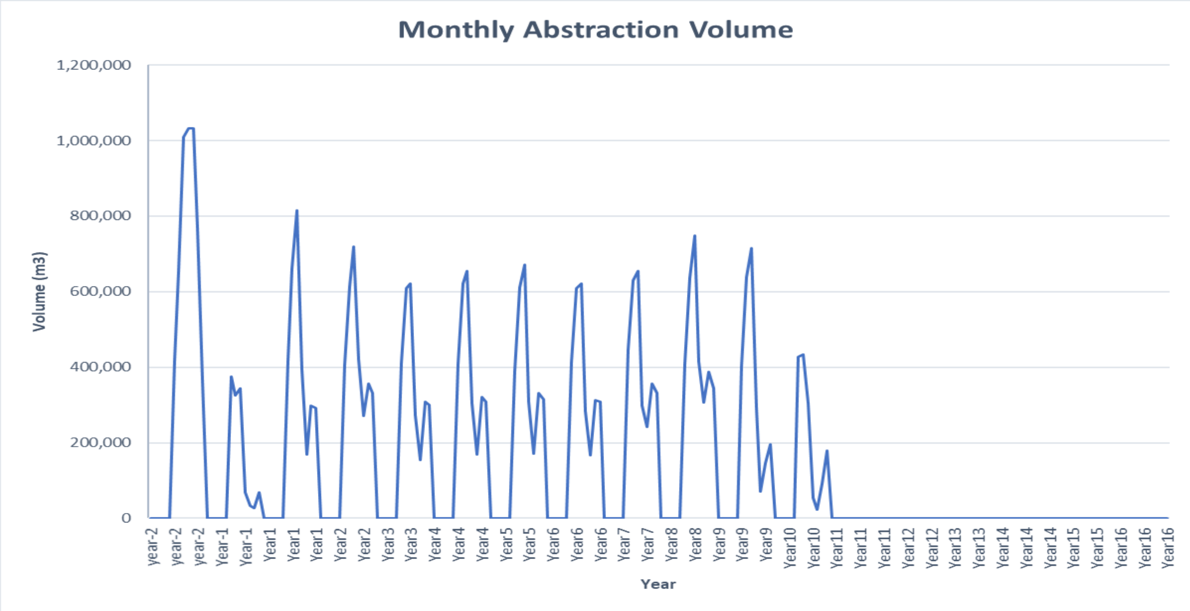
Source: KP, November 2021

The model indicates that sufficient water is available to meet process, potable and dust suppression demand under average and extreme dry conditions, although the early years of operation are the most critical in terms of water supply and demand. These can be accommodated by the proposed water supply system. The water balance results under average conditions are summarized in Figure 18-24.

Figure 18-24 Water Balance Results (Average Conditions)







Source: KP, November 2021

Table 18-3 shows the projected water supply and demand under average rainfall conditions.

Table 18-3 Koné Water Supply and Demand in Average Conditions

	Water Demand from WSF					Water Supply to WSF							TSF Water Supply and Demand			
Result:	Evaporation	Seepage	Potable Water Demand	Dust Demand	Process RawWater Demand	Runoff and Rainfall	Marahoue River Abstraction	TSF Closure Pumped to WSF	Pit Runoff	Borefields	Shortfall	Spillway	Process Water Demand	Process Water Supply from TSF	Process Water Supply from Inpit	Process Shrtfall from WSF
Unit:	Mm3	Mm3	Mm3	Mm3	Mm3	Mm3	Mm3	Mm3	Mm3	Mm3	Mm3	Mm3	Mm3	Mm3	Mm3	Mm3
Yr-2	0.54	0.18	0.63	0.39	0.00	1.52	5.74	0.00	0.27	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Yr-1	1.54	0.43	0.85	0.53	0.00	1.51	1.00	0.00	0.35	0.82	0.00	0.00	0.00	0.00	0.00	0.00
Yr1	1.44	0.40	2.58	1.60	1.12	1.51	2.87	0.00	0.35	2.08	0.00	0.00	4.73	4.73	0.00	0.00
Yr2	1.43	0.40	2.59	1.61	1.30	1.51	2.94	0.00	0.35	2.54	0.00	0.00	5.00	5.00	0.00	0.00
Yr3	1.44	0.40	2.59	1.60	1.30	1.44	2.49	0.00	0.71	2.70	0.00	0.00	5.05	5.05	0.00	0.00
Yr4	1.44	0.40	2.59	1.60	1.31	1.44	2.63	0.00	0.71	2.55	0.00	0.00	5.02	5.02	0.00	0.00
Yr5	1.43	0.40	2.59	1.60	1.30	1.44	2.70	0.00	0.71	2.48	0.00	0.00	5.05	5.05	0.00	0.00
Yr6	1.44	0.40	2.59	1.61	1.30	1.45	2.63	0.00	0.71	2.57	0.00	0.00	5.06	5.06	0.00	0.00
Yr7	1.43	0.40	2.59	1.60	1.30	1.44	2.97	0.00	0.30	2.61	0.00	0.00	5.07	5.07	0.00	0.00
Yr8	1.43	0.40	2.59	1.60	1.30	1.44	3.21	0.00	0.16	2.49	0.00	0.00	5.03	5.03	0.00	0.00
Yr9	1.42	0.40	2.16	1.26	1.30	1.44	2.66	0.00	0.16	2.27	0.00	0.00	5.03	2.54	2.50	0.00
Yr10	1.48	0.41	1.74	0.80	1.30	1.45	2.04	0.00	0.16	2.09	0.00	0.00	5.02	0.00	5.02	0.00
Yr11	1.60	0.45	1.73	0.80	1.30	1.44	0.00	6.59	0.16	2.08	0.00	3.22	5.03	0.00	5.03	0.00
Yr12	1.73	0.48	1.73	0.80	1.30	1.44	0.00	6.64	0.16	2.08	0.00	4.28	4.95	0.00	4.95	0.00
Yr13	1.73	0.48	1.73	0.80	1.26	1.44	0.00	6.63	0.16	1.97	0.00	4.22	4.78	0.00	4.78	0.00
Yr14	1.73	0.48	1.74	0.80	1.22	1.45	0.00	6.63	0.16	1.98	0.00	4.23	4.62	0.00	4.62	0.00
Yr15	1.73	0.48	1.73	0.80	1.12	1.44	0.00	6.62	0.16	1.97	0.00	4.31	4.77	0.00	4.77	0.00
Yr16	1.76	0.49	0.93	0.56	0.13	1.44	0.00	6.60	0.16	1.97	0.00	6.02	0.56	0.00	0.56	0.00

18.1.11 Water Storage Facility

A WSF will be constructed downstream of the mining and processing area to act as the main water storage facility and sediment control dam. The facility will have a capacity of 7.2 Mm^3 (up to the spillway invert level) with a pond area of 158 Ha. The water storage facility embankment will be a maximum of 15 m high and have a length of 660 m. A spillway will be provided to safely release excess water from the facility. Water will be recovered from the facility by a floating pontoon mounted pump. The stability assessment indicates that WSF embankment meet the minimum FOS requirements recommended by ANCOLD 2019 under undrained, drained, and post-seismic loading conditions.

18.1.12 Water Harvesting / River Abstraction Facility

The river abstraction facility will be constructed adjacent to the Marahoué River at a location approximately 26 km east of the WSF. The facility will comprise a sump to capture and allow for harvesting of water. Water will be reclaimed from the facility by a pump mounted on a floating pontoon. A pipeline alignment has been nominated between the river abstraction location and WSF, with an access road located adjacent to the pipeline to allow for inspection and maintenance.

18.2 Power Supply

Electric power consumption for the Koné project is estimated to be:

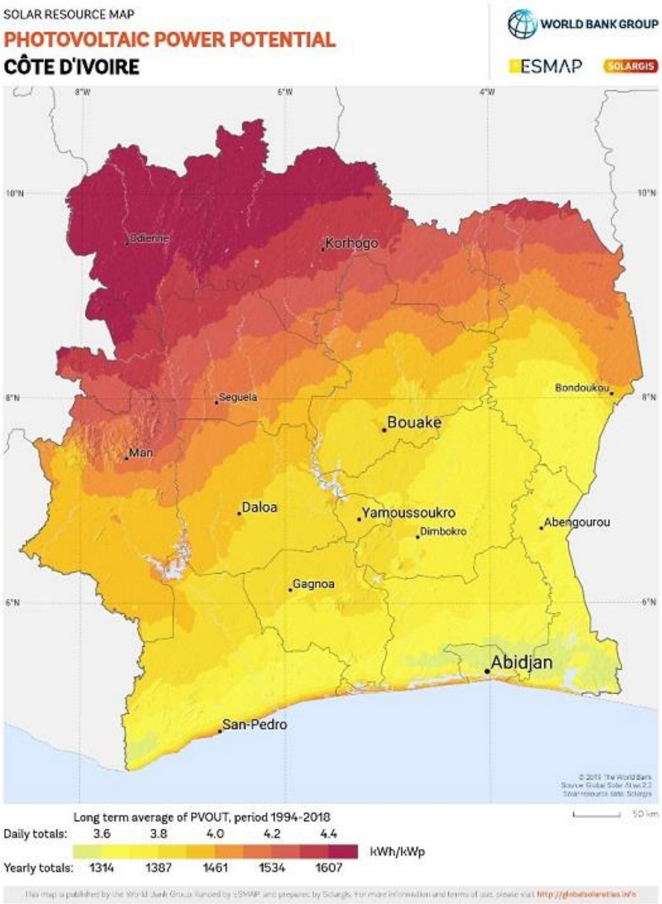
- Connected load 65.3 MW
- Maximum Demand 44.8 MW
- Average annual demand 37.0 MW (at a load power factor of 0.95 lagging)
- Energy consumption 303 GWhr/yr
- LNG delivered price $11.2/GJ.

The thermal power plant will be supplemented with photovoltaic (PV) power generation capacity and a Battery Energy Storage System (BESS) which will stabilize (filter) the power generated by the power station (and the load on the thermal power generation equipment) and to prevent sudden dips in power generation capacity resulting from fluctuations in PV system output. The solar farm will generate 22.7 MW of solar PV coupled with 8.8 MW of BESS.

A comparison of the estimated capital cost and operating costs for an LNG/Solar/BESS hybrid option was undertaken by ECG based on proposals from West African power providers and reported in their report "KON-G-RP-0002-0 - Power Supply Options Review".

The performance of the solar hybrid power station has been analysed based on the solar irradiance provided from the NASA surface meteorological data for the Koné Gold Mine site, Figure 18-25.

Figure 18-25 Côte d'Ivoire Solar Irradiance Map



The power provider will be responsible for the ongoing operation and maintenance of the equipment and guarantee the power supply at a standardised cost per kWhr at the outgoing feeders for the 5 year life of the contract. This includes the following;

- Operate and maintain the power station and solar plant including meeting all key performance indicators – fuel, gas and solar yield guarantee
- Scheduling and performing routine maintenance of overall asset and equipment as per manufacturer's recommendations
- Engine overhauls as per manufacturer's recommendations
- Procurement and holding of onsite spares necessary to maintain the power station and solar plant availability
- Containment and handling of waste oil (incinerator or other)
- Provide and maintain at least two or more qualified operators and maintenance technicians onsite at any given time.

Following completion of the capital payments in year 6, the operation of the power supply system will be undertaken by KGP.

The LNG supply will be procured from Prestea GCP in Prestea, Ghana. The feed gas for the Prestea GCP is natural gas sourced from Ghana's gas fields via a pipeline infrastructure in Ghana and sourced from the domestic gas sources at Jubilee, TEN and Sankofa gas fields. LNG will then be trucked in bulk road vehicles (BRVs) to Koné for power generation.

LNG from Prestea can be trucked quickly and effectively using specialized LNG tank trucks of 23.67MT (typical industry capacity requirement for LNG tank trucks) by road covering a total distance of 725km. A minimum of seven trucks per day will be required to meet daily fuel consumption requirements and LNG storage facilities will be installed at both the Koné site and at the Prestea GCP to serve as a buffer for the LNG supply chain and to minimise the risk of gas supply shortages to the Koné power station.

The cost of LNG delivered to Koné site consists of the gas supply cost in Ghana, liquefaction, storage and transportation costs. The cost of gas supply within Ghana is estimated at $6.70/MMBtu, with a total delivered to site cost of $11.20/MMBtu.

The LNG/Solar/BESS hybrid power station solution has been selected as the preferred power supply option for KGP. This selected option has a unit operating price of $0.0998/kWhr with sustaining capital payments of $100M (5 year contract payments) with an estimated capital transfer fee of $38M transfer payment at the conclusion of the contract term.

18.3 Tailings Storage Facility Design

The design of the tailings storage facility (TSF) was undertaken, to international standards, to provide a facility to safely contain the tailings and reduce the potential effect thereof on the environment in the form of dusting, seepage, or run-off from the tailings surface during operation and post closure. Provision was made for the effects of seismic events and probable maximum precipitation events during operation and post closure. To support the design and improve the safety of the facility, seepage analysis and stability analysis were conducted on the embankments. A water balance model was prepared to determine the water volumes retained in the TSF and the available recycle volumes to the plant. If built and operated in accordance with the principles and design concepts outlined in this document, this facility would contain the tailings generated from the project and the effects on the environment would be within acceptable limits as defined by international standards.

The tailings management arrangement comprises one tailings storage facility (TSF) confined by a cross valley embankment and in-pit deposition when mining in south pit is completed. Initially the TSF will be constructed to store the tailings and will be raised annually until the mining in south pit is completed (after 8.5 year). Tailings will be deposited in south pit for the final 6.5 years of processing. The TSF will be closed and rehabilitated after deposition transferred to pit.

18.3.1 LOM Capacity

The TSF has been designed to store tailings capacity of approximately 92.5 Mt which will be generated by the process plant over a period of 8.5 years at a rate of approximately 11 Mtpa after the initial ramp up period.

South pit will store approximately 68.5 Mt and will be utilised from year 8.5 up to end of year 15.

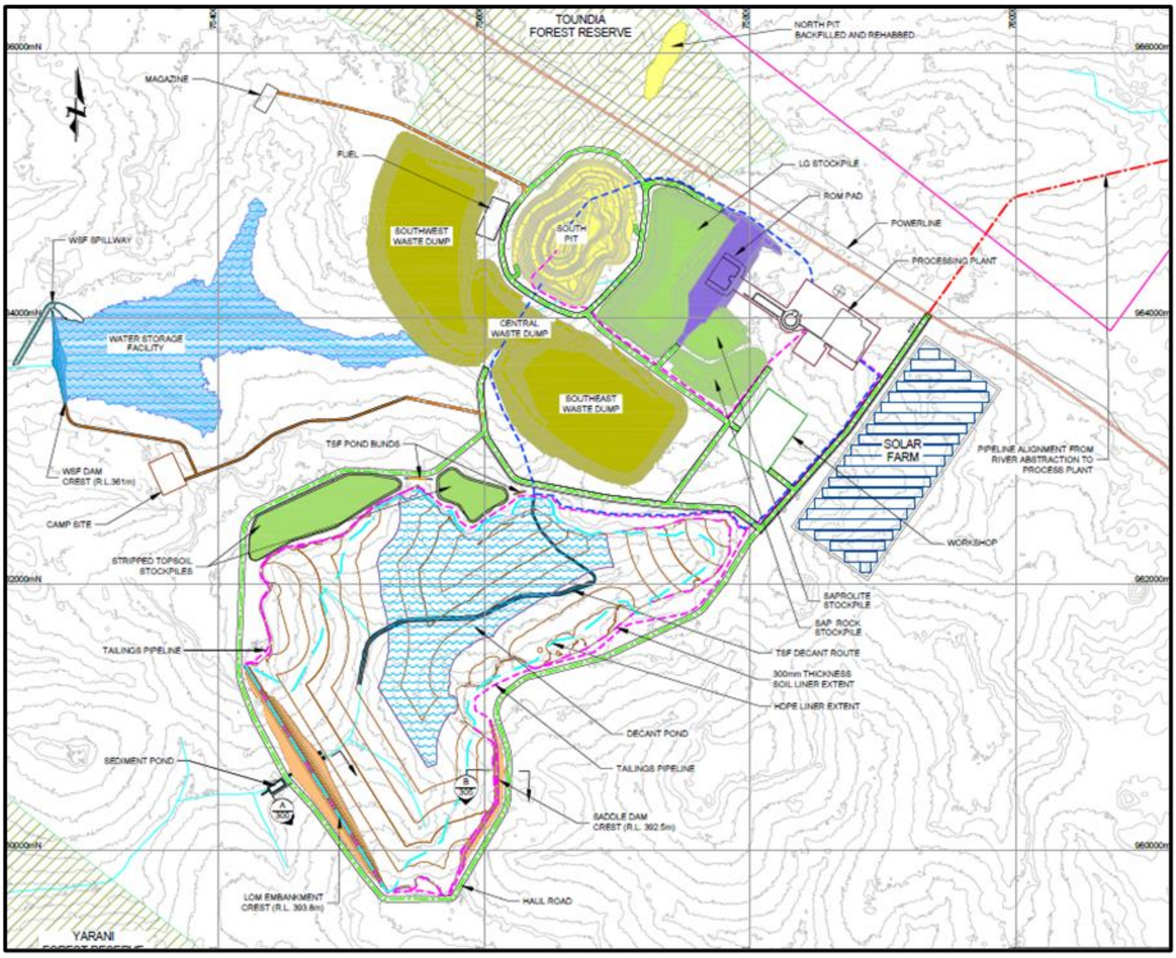
The tailings storage facility comprises one cell confined by a cross valley embankment and a saddle dam. Initially the main embankment will be constructed, with the saddle dam constructed later in the mine life to provide enough capacity for the first stage of tailings management.

The geometry of the valley offers efficient storage for tailings with deposition points required along the main embankment, saddle dam and, subsequently, the ridgelines to the east. This enables the pond to be migrated to the northeast away from the embankment and towards the closure spillway location.

Maximum embankment height will be of 41 m with maximum saddle dam height of 12.5 m. The tailings beach surface at full capacity will cover an area of approximately 522 ha.

The general arrangement of the LOM embankment is shown on Figure 18-26.

Figure 18-26 Tailings Storage Facility General Arrangement



18.3.2 Tailings Physical Characteristics

The testing has been undertaken at approximately 63%, 58% and 36% solids w/w for the Fresh, Transition and Oxide tailings, which are in the design target for the operation.

The testing indicated that the supernatant release from the Fresh sample is the quickest, taking hours to complete. The release of supernatant from Transition and Oxide samples are comparably slower, taking about 2 days and one week to complete, respectively.

The expected supernatant release would be in the range of 16 to 28% of the water in slurry for the Fresh tailings, 14% to 33% for the Transition tailings and 22% to 32% for the Oxide tailings, not accounting for rainfall and evaporation but incorporating the loss of water to re-saturate lower tailings layers for the operating tailings.

Underdrainage release is relatively quick for the Fresh sample, taking less than a day to complete. In comparison, it takes 2 days and one week for the Transition and Oxide samples respectively to complete the underdrainage release. Underdrainage could be as high as 20% of the water in slurry, however, a recovery rate between 5% and 10% could be expected depending on the arrangement of underdrainage collection and basin treatment.

The test results indicate that there is little improvement on dry density achieved by air drying compared with sedimentation tests for the Fresh sample, moderate improvement for the Transition sample and significant improvement for the Oxide sample.

This suggests that having an exposed tailings beach will still overall benefit the dry density for the Transition and Oxide tailings and should be targeted. With suitable underdrainage and air drying of the tailings slurry, settled densities of approximately 1.00 t/m^3 for the Oxide tailings, 1.30 t/m^3 for the Transition tailings and 1.30 t/m^3 for the Fresh tailings are expected in the facility.

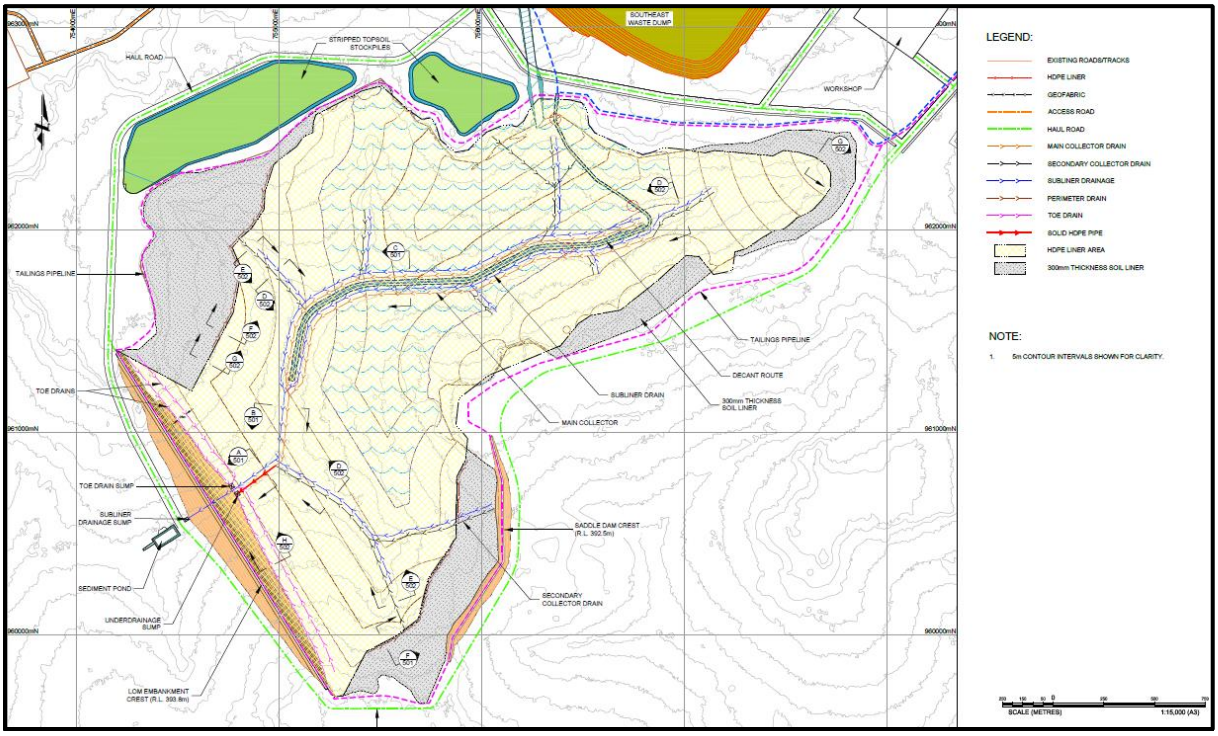
18.3.3 Tailings Geochemical Characteristics

The supernatant extracted from the tailings slurries was analysed to assess the water quality which may be encountered in the facility during operations. The total cyanide concentration from the laboratory testwork range from 275 to 562 mg/l, however in normal operations cyanide levels will be managed which are estimated to result in concentrations of 130mg/l. The following seepage control measures have been incorporated in the tailings management design:

- Profiling of the in-situ fine grained soils and capping of any areas where granular, high permeability basin soils are encountered
- Placement of a HDPE liner across the normal operating pond extents with a compacted soil liner elsewhere
- Construction of an above liner underdrainage collection system to reduce the hydraulic head acting on the liner
- Construction of a sub-liner seepage recovery system
- Placement of a 6m wide low permeability zone on the upstream batter of both confining embankments
- Placement of a HDPE liner on the upstream batter of both confining embankments (main embankment and saddle dam).

The proposed partial basal liner, above liner drainage and sub-liner drainage systems are shown on Figure 18-27.

Figure 18-27 Tailings Storage Facility Liner and Drainage System



18.3.4 Dam Failure and Environmental Spill Consequence Category

Three scenarios were considered in dam break assessment to model failure of the main embankment, saddle dam and water dam. Breach assessment in each scenario has been conducted. Embankment failures were modelled for when the TSF and WSF are at their ultimate height and capacity, where the potential volumetric outflow is the largest and inundation area is the greatest. This is considered the critical case for the assessment. As per the ANCOLD 2019 Guidelines, the consequence category assessments for the tailings and water storage facilities were undertaken to assess the Population at Risk (PAR) in the event of failure of the facilities which will define the dam failure consequence categories. The consequence category results are summarised in **Error! Reference source not found.**4.

Table 18-4 Summary of Consequence Category Assessment

Description	ANCOLD Consequence Category
Main Embankment Dam Failure Consequence Category	Extreme
Saddle Dam – Dam Failure Consequence Category	Extreme
TSF – Environmental Spill Consequence Category	Extreme
Water Storage Facility – Dam Failure Consequence Category	High B

18.3.5 Embankment Configurations

The TSF embankment will be approximately 2 km in length at the final stage and is orientated northwest to southeast at a natural constriction in a valley. Containment for the remainder of the facility will be provided by a saddle dam (approximately 1.2 km in length) and the natural terrain. The facility has a catchment area of 716 ha.

All stages of the embankment will be constructed by downstream raise construction techniques. Embankment will be HDPE lined and comprise a 6 m wide upstream low permeability zone (Zone A), a 1.5 m wide transitional zone (Zone B) and downstream structural zone (Zone C).

Figure 18-28 and Figure 18-29 show the typical details of the TSF embankment and saddle dam.

Figure 18-28 Main Embankment – Typical Section

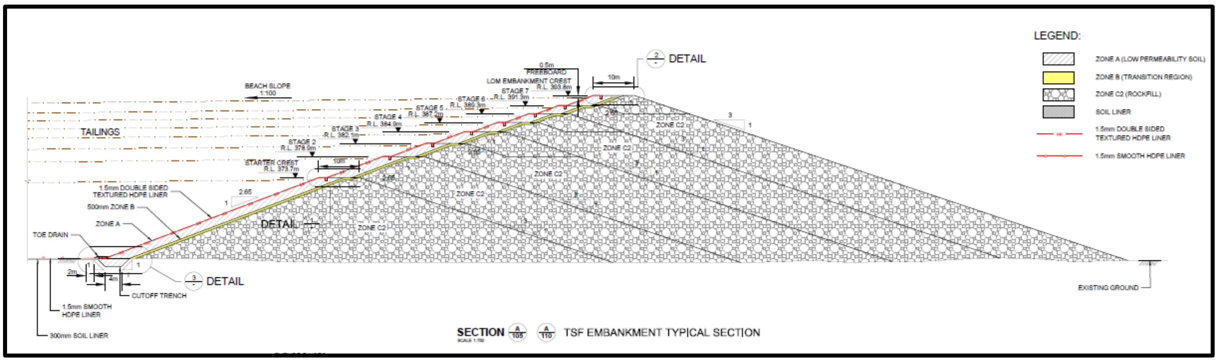
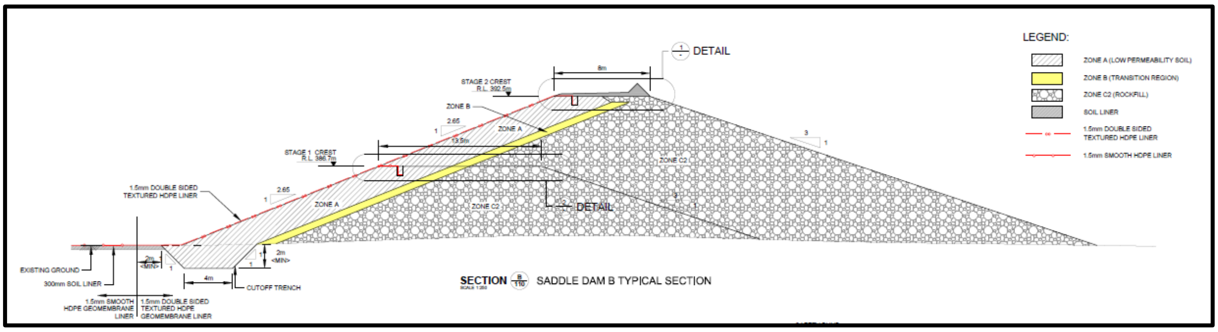


Figure 18-29 Saddle Dam - Typical Section



18.3.6 Embankment Construction

Construction of the embankment will be staged over the LOM, with the aim of deferring the capital and sustaining capital costs involved with the embankment construction. The water balance model indicated that supernatant pond under average climatic conditions will be located below the tailings level at the embankment during the life of mine. As such, only 0.5 m operational freeboard was considered when defining the embankment crest RLs.

Embankment will be constructed with a 21m high starter embankment and be raised annually in 7 stages

18.3.7 Geotechnical Analysis

Geotechnical analysis of the facility has been conducted to assess the stability and seepage rates of the TSF and saddle dam embankments. The stability assessment indicates that TSF and saddle dam embankments meet the minimum FOS requirements recommended by ANCOLD 2019 under undrained, drained, and post-seismic loading conditions.

The transient seepage analysis of the facility indicates that the embankment will be fully drained with a seepage rate from the facility basin of approximately 1.4 m^3/ha/day at the final stage of operation.

A high level deformation assessment indicated that the maximum embankment crest settlement is less than the design freeboard of 0.5 m. Therefore, the maximum design earthquake (MDE) event is unlikely to lead to loss of containment.

18.3.8 Tailings Deposition

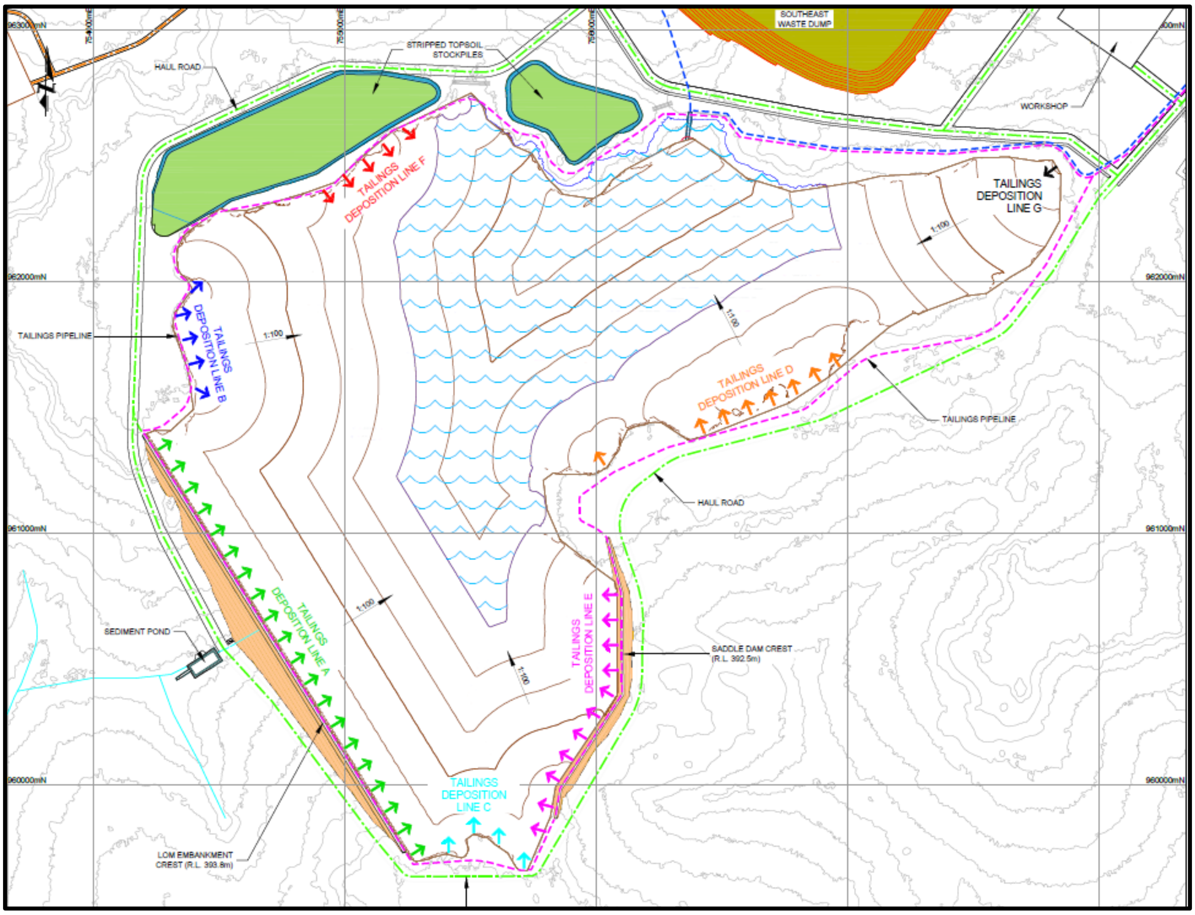
Tailings will be deposited subaerially with the supernatant pond located away from the embankment. Water will be recovered from the supernatant pond by a suction pump with floating intake located in a channel excavated adjacent to an access causeway.

Sub-aerial deposition allows for the maximum amount of water removal from the facility by the formation of a large beach for drying and draining. Together with keeping the pond size to a minimum, sub-aerial deposition will increase the settled density of the tailings and, hence, maximise the storage potential and efficiency of each facility.

Tailings slurry will be pumped to the facilities via a welded high-density polyethylene (HDPE) pipe. This pipe will be located in a bunded corridor adjacent to the haul roads to contain any spill should the pipeline fail.

Deposition commences from main embankment and northwest ridgeline in Stage 1 to push the pond away from embankment. From Stage 3 deposition from south ridgeline will start to guide the pond to north, close to its final location. In Stage 4 and after construction of saddle dam, deposition line from saddle dam will be added to the deposition plan. North line will be depositing in Stage 5 and in Stage 8 eastern line will be added. Deposition arrangement is presented in Figure 18-30.

Figure 18-30 Tailings Deposition Arrangement



18.3.9 Monitoring and Instrumentation

A monitoring programme will be designed for the tailings facilities to detect a number of potential problems which may arise during operation. The facilities will be equipped with monitoring equipment to measure the performance of the facility. The monitoring will include:

- Vibrating wire piezometers to measure the phreatic surface within the embankment and foundation materials
- Monitoring bores to measure the groundwater quality around the facility
- Flow meters to measure underdrainage and decant return flows
- Prisms to measure deformation of the embankments.

Regular drone surveys of the tailings beach should be conducted to measure the beach slope, pond area and allow for the achieved density of the tailings to be calculated.

18.3.10 Water Balance

A water management model has been established to estimate the water demands for the project, to assess the availability of water to meet the demands and to size the various components of the water management system. The model indicates that sufficient water is available to meet process, potable and dust suppression demand under average and extreme dry conditions, although the early years of operation are the most critical in terms of water supply and demand these can be accommodated by the proposed water supply system.

Error! Reference source not found.3 shows the projected water supply and demand under average r ainfall conditions.

18.4 Geotechnical Investigation

A geotechnical investigation was conducted as part of the study to evaluate the site conditions and geotechnical design parameters of the plant site, TSF and WSF footprints including their abutments and basins. The investigations conducted to date comprised the following scope of work:

- Drilling of thirty-four (34) boreholes to depths of between 18.9 m and 52.7 m
- Excavation or drilling of sixty-eight (68) test pits / auger holes
- Laboratory testing of soil samples.

18.4.1 TSF Geotechnical Investigation

The geotechnical investigation indicated that the foundation materials mainly include a layer of residual soil (1 - 5 m), underlain by extremely weathered (XW) Granite/Basalt (5 - 35 m) and finally distinctly to slightly weathered Granite/Basalt/ Schist. The foundation oxide materials indicated having low potential to liquefy under a seismic event.

18.4.2 WSF Geotechnical Investigation

The geotechnical investigation indicated that the foundation materials mainly include a layer of residual soil (1 - 8 m), underlain by extremely weathered (XW) Granite/Basalt (8 - 25 m) and finally distinctly to slightly weathered Granite/Basalt/Schist.

18.4.3 Plant Site Geotechnical Investigation

The geotechnical investigation indicated that the foundation materials mainly include a layer of residual soil (1 - 6 m), underlain by extremely weathered (XW) Granite / Basalt / Schist (6 – 12m) and finally distinctly to slightly weathered Granite / Basalt / Schist. The interpretation of basic testing results of foundation materials (i.e., particle size distributions and Atterberg limits) indicated having a higher potential to liquefy under a seismic event compared with the TSF / WSF sites. Since the plant site equipment may produce dynamic loads higher than the MCE event, additional care needs to be maintained when designing the footings.

18.4.4 Water Storage Facility

A water dam will be constructed downstream of the mining and processing area to act as the main water storage facility and sediment control dam. The facility will have a capacity of 7.2 Mm^3 (up to the spillway invert level) with a pond area of 158 Ha. The water storage facility embankment will be a maximum of 15 m high and have a length of 660 m. A spillway will be provided to safely release excess water from the facility. Water will be recovered from the facility by a floating pontoon mounted pump. The stability assessment indicates that WSF embankment meet the minimum FOS requirements recommended by ANCOLD 2019 under undrained, drained, and post-seismic loading conditions.

18.4.5 Water Harvesting / River Abstraction Facility

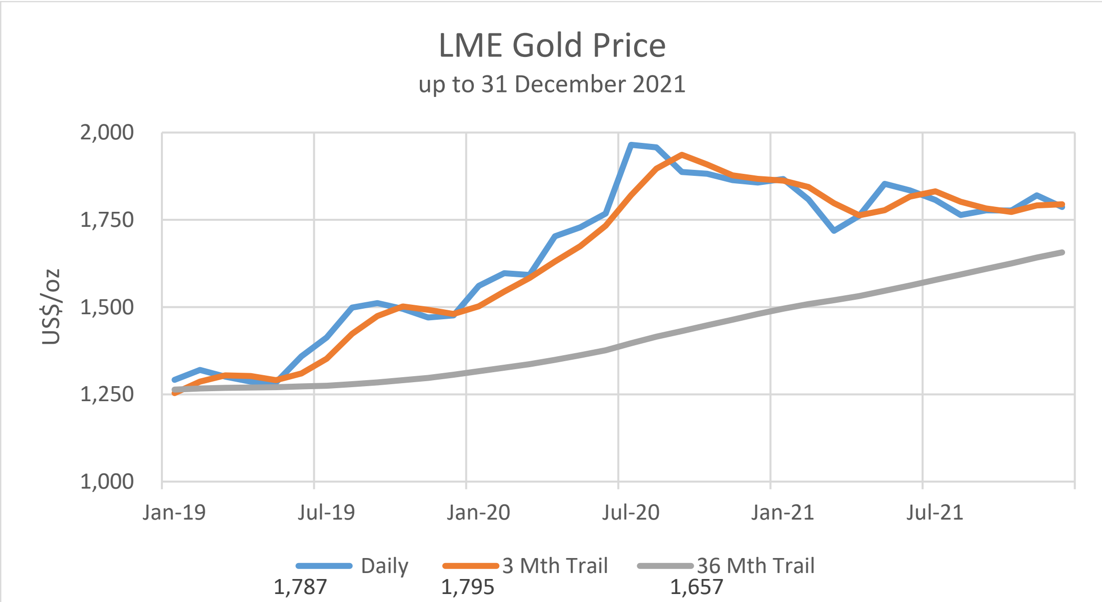
The river abstraction facility will be constructed adjacent to the Marahoué River at a location approximately 26 km east of the WSF. The facility will comprise a sump to capture and allow for harvesting of water. Water will be reclaimed from the facility by a pump mounted on a floating pontoon. A pipeline alignment has been nominated between the river abstraction location and WSF, with an access road located adjacent to the pipeline to allow for inspection and maintenance.

19 MARKET STUDIES AND CONTRACTS

No Market Studies were carried out for this study. The final product of the Koné project will be gold / silver doré bars, which will be shipped to a refinery for processing. The refined gold can either be sold by the refinery or bullion returned to the company. Preliminary quotations have been received from a refinery and transport provider.

Gold bullion sells on several international markets, the most well-known being the London Metals Exchange or LME (Figure 19-1).

Figure 19-1 LME Gold Price



No material contracts have been entered into as of the date of this report. Construction and mining contracts will be negotiated in the future.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

Mineesia Ltd, a UK based consultancy, has supported the environmental management of the Project activities of the Koné Gold Project, including supporting the development of the environmental impact assessment by CECAF, an Ivorian environmental consultancy. The primary environmental and social requirements include:

- Characterization of all the project's potential impacts;
- Finalise schedule of environmental and social other permitting requirements;
- Evaluate project setting for potentially significant environmental and social permitting constraints
- All requisite environmental data for project are identified and reviewed
- site sampling and analyses are complete
- detailed review of the type, scope and schedule for producing environmental and social government reports, including regulatory inspections, waste handling practices; management plans
- comprehensive gathering and evaluation of baseline environmental and social conditions; Social, training, and health/safety programs confirmed
- Draft Environmental and Social Impact Assessment (ESIA) submitted to regulatory authorities initiated
- Environmental characteristics defined and used in project design
- Finalise the following:
  - environmental plans and monitoring programs.
  - sediment and erosion control plan.
  - management plans for tailings and waste rock.
  - management plan for solid and hazardous wastes.
  - impact mitigation plan.
  - geotechnical stability analysis of all major facilities.
  - closure plan.

    - analysis of acid rock drainage.

    - spill and emergency response plan.

- Complete environmental monitoring plan.

- Detailed evaluation of all pertinent authorizations and permitting requirements and schedule for obtaining operating license.

In addition, a preliminary Environmental and Social Management Plan is required, as well as ongoing community relations and stakeholder engagement plans.

20.2 Côte d'Ivoire Legal Setting – Environmental

The Côte d'Ivoire Constitution (2000) addresses environmental protection with Article 19 guaranteeing each person's right to a healthy environment and Article 27 imposing a duty of environmental protection on the community and all-natural persons and legal entities. This is reiterated in Article 33 of the principal environmental legislation, the Environment Code, which states that everyone has the fundamental right to live in a healthy environment. Other environmental legislation that may impact upon mining projects includes the Water and the Forestry Codes. Environmental issues are administered by the Ministry of Environment and Sustainable Development and by the National Environmental Agency (Agence Nationale de L'Environnement (ANDE)).

The Environment Code requires that every project must be subject to an Environmental Impact Assessment prior to granting of any authorization. It applies to mining installations and includes the minimum environmental impact study requirements. Decree No. 96-894 (8 November 1996) details the relevant rules and procedures for environmental and social impact assessments for development projects. This decree specifies mining operations as Annexe 1 projects, which require Environmental Impact Assessment. The Mining Code requires that all mining title applicants (excluding artisanal) submit an Environmental and Social Impact Study (EIES, in French) to the General Directorate of Mines and Geology (DGMG) and ANDE and all other institutions as required by the Mining Decree. The Mining Code also includes provisions regarding mine closure. To ensure environmental protection, mining titleholders must open an escrow account in a leading Ivoirian financial institution at the beginning of mining operations, to be used to cover costs related to the environmental management and mine closure plans.

20.3 International Requirements and Guidelines

Côte d'Ivoire is a Member State of the West African Economic and Monetary Union (WAEMU), which enacted a mining code in 2003 (the WAEMU Mining Code). This Mining Code governs any mining operation related to prospecting, exploration, exploitation, detention, traffic, transport, treatment, trade and transformation of minerals within the WAEMU Member States' territories.

Côte d'Ivoire has been a member of the Extractive Industries Transparency Initiative (EITI) since 2008, when the government issued Presidential Decree 2008 25 establishing the EITI multi stakeholder group (known as the National Council (Comité National) for implementation of EITI Principles. Although the EITI Standard does not require or encourage disclosures regarding environmental management, EITI Principles emphasise that natural resource wealth should be an engine for sustainable economic growth. Côte d'Ivoire is one of the countries that have included information related to environment as part of their EITI reporting. The Mining Code also requires adhesion to good governance principles, including the Equator Principles and the Extractive Industries Transparency Initiative principles. As such, mining titleholders must issue EITI reports.

The Koné Project is classified as a Category A development in accordance with International Finance Corporation (IFC) Guidelines, due to the scale and type of operation. The IFC Sustainability Framework, as revised in 2012, with associated Performance Standards on Environmental and Social Sustainability, provide the basis for most impact assessments. Additional guidance is provided by the Equator Principles, which provide an approach to determine, assess and manage environmental and social risk in project financing.

20.4 Project Permitting

The development of the Project will be subject to receiving environmental approval of its design, environmental management programme and appropriate mitigation measures where required. Based on the provisions of the various legal requirements and sectoral laws as well as policies of different departments, the impacts of any proposed project will need to be assessed and appropriate mitigation measures recommended where appropriate.

Under the Mining Code, all applicants for an exploitation licence must submit an EIES to ANDE, which is the environmental authority in charge of supervising, validating and controlling environmental impact studies. The EIES will include an Environmental and Social Management Plan and a site rehabilitation plan. The Environment Code provides the minimum requirements for environmental impact studies, with the purpose of evaluating the environmental effects of an activity and proposing measures to eliminate, reduce or mitigate potential adverse environmental impacts. As a minimum, the EIES must include:

- A description of the proposed activity.
- Description of the environment likely to be affected, including the specific information needed to identify or assess the effects of the proposed activity on the environment.
- List of products used where appropriate.
- Description of the alternative solutions, if any.
- Assessment of the likely or potential effects of the proposed activity and other possible solutions on the environment, including direct, indirect, cumulative effects in the short.
- medium and long term.
- Identification and description of measures to mitigate the effects of the proposed activity and other possible solutions on the environment, and an assessment of these measures.

- Summary of the closure plans and cost estimate for closure.
- Indication of the knowledge gaps and uncertainties encountered in developing the necessary information.
- Indication of the environmental risks in transboundary issues due to the proposed activity or other possible solutions.
- Description of health and safety measures that will be implemented.
- Brief summary of the information provided under the previous headings.
- Definition of the procedures for the regular monitoring and follow-up of environmental indicators before (initial state), during the construction phase, during the operation of the structure or, if applicable, after the end of the operation (restoration or redevelopment of the premises).
- A financial estimate of the measures recommended to prevent, reduce or offset the negative effects of the project on the environment and regular monitoring and control measures of relevant environmental indicators.

The rehabilitation plan must take into account several aspects such as the cleaning of the site, the dismantling and removal of mining installations, the post-rehabilitation surveillance of the site, and suggestions on how the site could be reconverted. Such operations must start during the exploitation period and not only at the end of operations. After the closure of the mine, any exploitation permit holder remains liable under civil law for damages and accidents on the site that could be caused by the former installations during the five years following closure.

Some protected areas, such as classified forests, places of worship or cultural sites, cannot be subject to mining activities without the prior consent of the owners, occupants and concerned communities, as well as authorisation from the Minister of Mines.

In addition, the Code follows modern African mining legislation, which increasingly aims at protecting the rights of local populations. The Code guarantees a right to a fair indemnity for the land's occupants and legal owners in the event of occupation of the land. Such indemnity will be paid following the signing, under the supervision of the mining administration, of a memorandum of understanding by the exploitation companies, the occupants and the legal owners. The mining code requires setting aside 0.5% of revenue to fund a local mining development committee (CDLM) which is created formally (with Mining ministry, state Admin, etc.) at the local level (prefect, other state services, communities) to handle this fund for local development.

20.5 Project Layout

The proposed Project layout is described in previous sections and shown in Figure 20-1. In summary, the main components of the operation are anticipated to comprise the following:

- Open pit mines – two open pits to be developed using standard open pit mining techniques, treating approximately 11 Million tonnes/yr.

- Processing Plant – ore crushed prior to adsorption of gold onto activated carbon through carbon-in-pulp (CIP) extraction methods. The process plant will be located near the Koné deposit.

- Tailings Storage Facility (TSF) – with capacity of 75 million tonnes and including tailings water drainage system, a recovery water basin and pipeline connecting the TSF to the plant.

- Waste rock dumps – for disposal of overburden and waste material from the open pits.

- Water supply and treatment – Pipe work will be required to supply water from the Marahoué River to a raw water storage facility on the mine site.

- Power will be sourced from an on site LNG / Solar Hybrid power station.

- Associated infrastructure – including haul roads, ROM pads, offices, workshops, domestic waste facility for non-mineral wastes, ablutions and sewage treatment systems, explosives storage and a minerals laboratory.

- A deviation of the national road passing between the 2 pits.

- Accommodation camps for construction workers and mine employees.

Figure 20-1 Site Layout

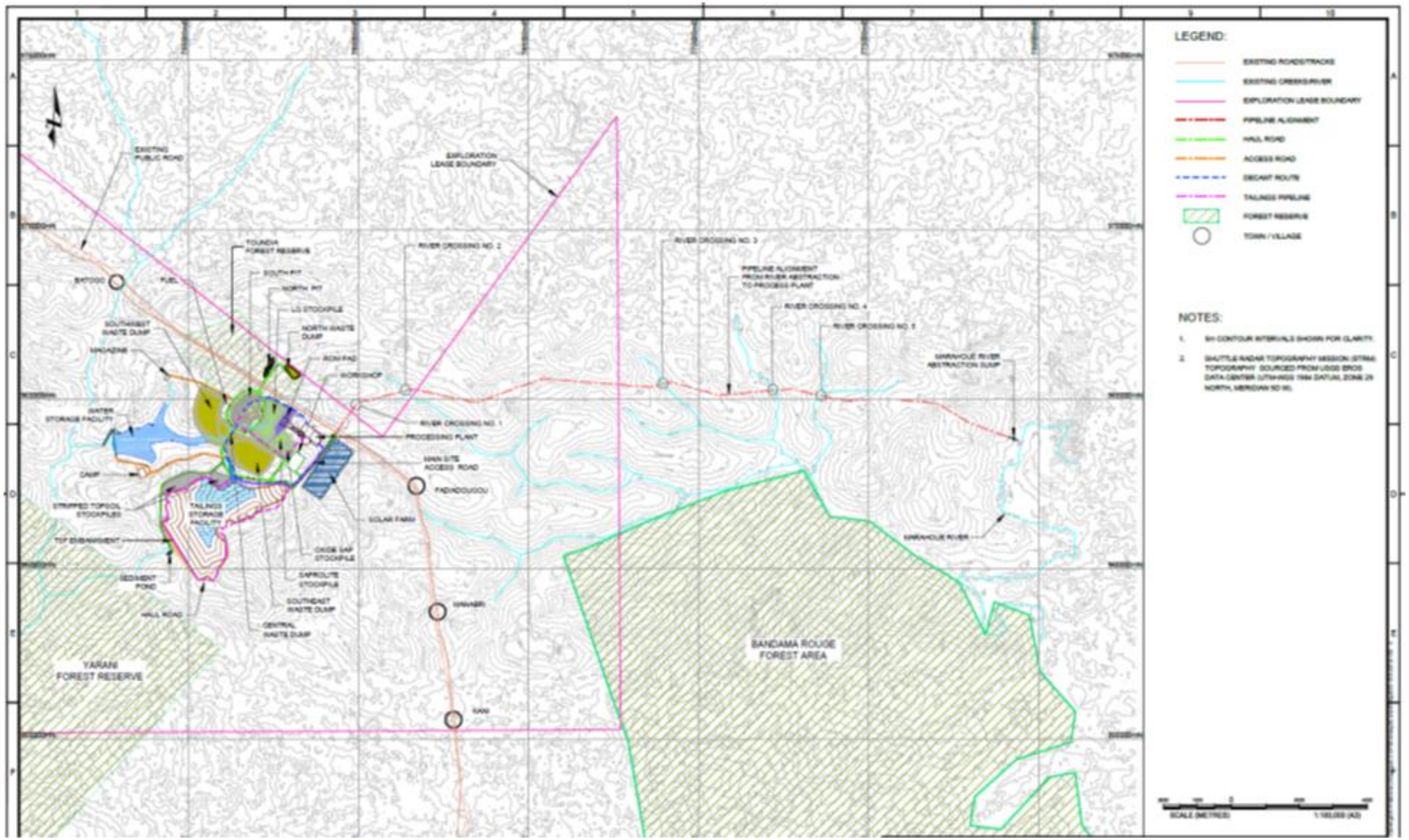
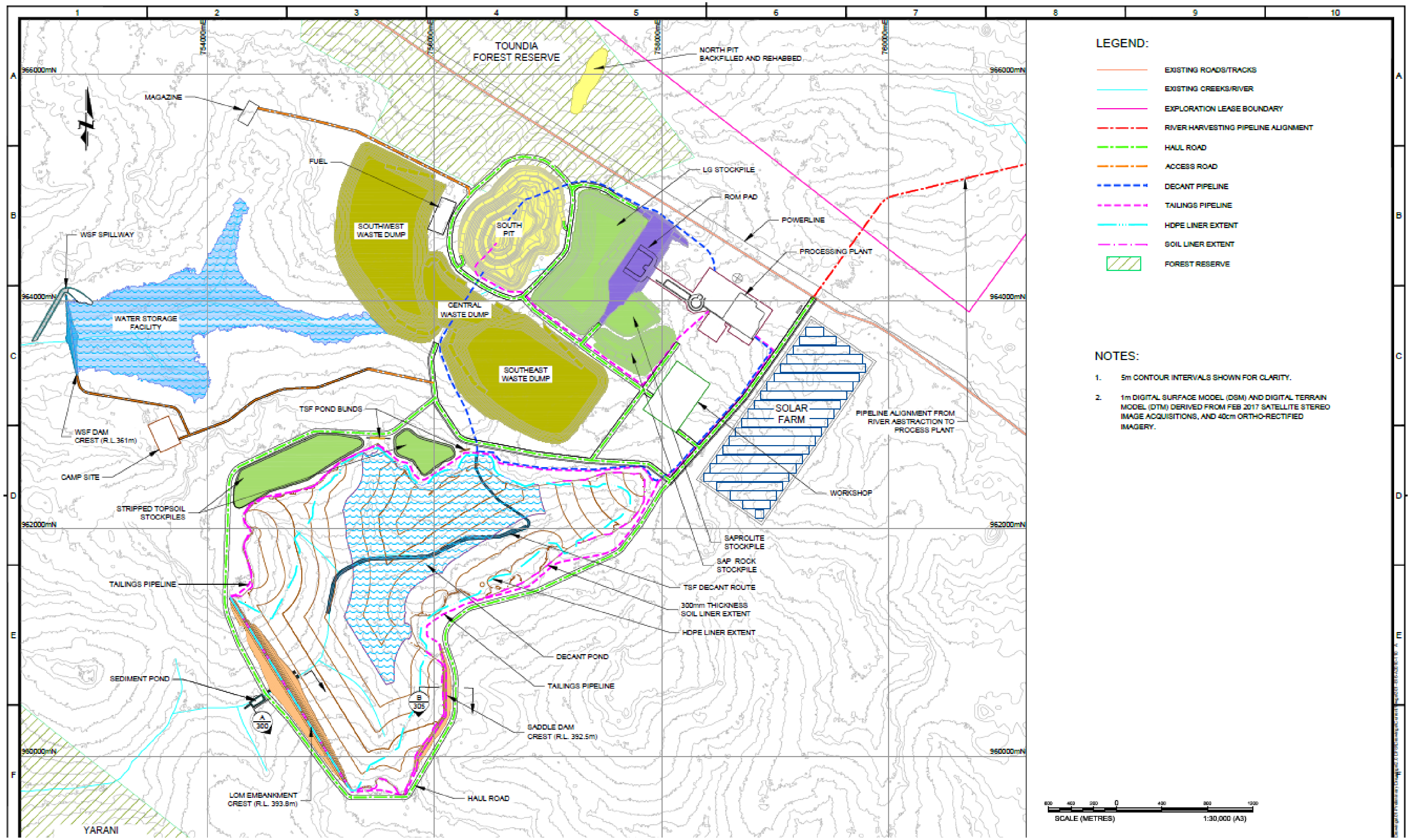


Figure 20-2 Central Site Layout



20.6 Baseline Environmental and Social Setting

Côte d'Ivoire is the most biodiverse country in West Africa, but unlike other countries of the region, its diversity isn't concentrated along the coast, but rather in the interior. More than 1,200 animal species and 4,700 plant species have been recorded.

20.6.1 Project Location

The Project is located midway between the villages of Batogo to the north-west and Fadiadougou to the south-east, along the Séguéla-Boundiali national road joining these 2 villages. Part of the footprint lies on Manabri lands.

The smaller northern pit and the northern edge of the larger pit are located within the Toudian classified forest. Permission is being sought to facilitate mining activities in this forest.

20.6.2 Protected Areas

There are protected forest reserves affected by and adjacent to the Project. These forest reserves are portions of state lands where commercial harvesting of wood products is restricted depending upon classification.

In the northern part of the Koné project area, and covering parts of its extraction pits, is the Toudian Classified Forest Reserve (identified in the UNEP-WCMC database as Classified Forest Reserve Name Unknown (CIV) No. 14) with an area of 527 ha. It is listed as an IUCN Category V protected area by UNEP-WCMC, where the interaction of people and nature over time has produced an area of distinct character with significant ecological, biological, cultural and scenic value (Dudley, 2008). In these areas, safeguarding the integrity of this interaction is vital to protecting and sustaining the area and its associated nature conservation and other values.

To the southwest and outside the project footprint is the Yarani Classified Forest Reserve. This is an IUCN Category IV protected area (UNEP-WCMC, 2019), aiming to protect particular species or habitats and management reflects this priority (Dudley, 2008). Many category IV protected areas need regular, active interventions to address the requirements of particular species or to maintain habitats, but this is not a requirement of the category.

To the east and outside project footprint is the Kani-Bandama-Rouge or Bandama Rouge (UNEP-WCMC, 2021) Classified Forest Reserve, with an area of 1,055 km^2.

20.6.3 Baseline Environmental Setting

The climate of Ivory Coast is generally warm and humid, ranging from equatorial in the southern coasts to tropical in the middle and semiarid in the far north. There are three seasons: warm and dry (November to March), hot and dry (March to May), and hot and wet (June to October). Temperatures average between 25 and 32 °C (77.0 and 89.6 °F) and range from 10 to 40 °C (50 to 104 °F).

In the southern half of the country, rainfall is higher and the soils more productive, making it the center of production of most of the export crops, such as coffee and cacao. Palm, coconut trees, and rubber tree plantations also occur mostly in the southern and central parts of the country. In the northern half of Côte d'Ivoire, subsistence and cash crops such as cashew, cotton, sugar, starches, and rice have greatly increased, fragmenting large expanses of woodland and savannas.

The Project area occurs within the Guinean forest-savanna ecoregion of West Africa, a band of interlaced forest, savanna, and grassland running from western Senegal to eastern Nigeria and dividing the tropical moist forests near the coast from the West Sudanian savanna of the interior. Agricultural expansion is a key factor compromising forest cover. In general, farmers still use slash and burn techniques to clear land for agriculture. This practice is destructive, lays waste to large amounts of land, and undermines reforestation efforts.

Given the lack of industry within 10 km of the Project, there are no obvious sources of anthropogenic air emissions, noise and vibration. Intermittent use by road traffic, agricultural and forestry machinery are the most discernible sources of such. The nearest towns are Fadiadougou, Batogo and Manabri. These towns have light industry supporting the communities, but no heavy industry. Road traffic generates dust which is captured by roadside vegetation. The majority (about 73%) of power in Côte d'Ivoire is from power stations fired by natural gas, with the remaining 27% sourced from hydropower. The Project has assessed the use of a hybrid system, using liquefied natural gas combined with solar as a power source.

Water samples have been collected from a number of exploration holes through the site and from the Marahoué River. Results have been compared to previous sampling to develop the database. While the wells have not been developed as monitoring wells, all the results indicate that water quality is good. From the samples collected, water is generally turbid, with elevated levels of dissolved iron, selenium and manganese, but these are aesthetic parameters rather than health related. Water from the well installed at Fadiadougou indicated that the water was of good quality.

Figure 20-3 Water Tanks at Fadiadougou

Source: Montage

Following the flora and fauna studies no protected species have been identified within the Project footprint area. The flora and fauna inventory will be included in the environmental impact study. The area is predominantly agricultural and comprises modified habitat. Trees have been cleared and habitat is fragmented (Figure 20-4 to Figure 20-6). Monkeys and cattle have been observed in the early images captured by the passive cameras.

Figure 20-4 Panoramic of pit area, showing level terrain

Figure 20-5 Panoramic of area between TSF and WRD, in south of project

Figure 20-6 View from camera trap #2

20.6.4 Baseline Social Setting

The Koné project area is located relatively close to the communities of Batogo, Fadiadougou and Manabri, but is sufficiently remote that environmental impacts on these communities are likely to be minor. Preliminary investigations indicate that the local community is overwhelmingly positive towards the company. Montage staff have engaged them regularly and the nearest communities have received specific support, such as beds for the clinic and construction materials for the school. The local community provides labour for construction. Water wells have been provided in Fadiadougou and Batogo, along with water tanks to ensure suitable supply. A clinic in Batogo has been completed, including housing for the nurse, and similar mini-project is under study for Manabri.

Human receptors are cotton and cashew farmers, with potentially some maize crops, hunters and sand miners also affected. The site has been well maintained by exploration staff, with bags emptied and destroyed. Natural revegetation has occurred in areas cleared for roads.

The communities are located far enough away from the site to suggest there would be few potential receptors for landscape and visual impacts. The topography is relatively level, with differing degrees of vegetation. Land use is mainly farming, held according to customary law. Land users include sharecroppers. It is anticipated that 25 households may require physical resettlement as a result of the Project.

The Project area is primarily farmland, utilized for cotton and cashew. No sites of archaeological interest have been identified on the site, but this will need to be verified through the EIES process. In the meantime, the Project has a chance finds procedure to mitigate against any impacts from their exploration activities. The Chance Finds Procedure defines a series of steps to minimize physical impacts to cultural heritage by providing a process for conducting archaeological look ahead-survey; monitoring of ground disturbing activities; and responding to any tangible cultural heritage encountered unexpectedly during exploration.

20.7 Potential Environmental Impacts

National environmental consultants will be used to gather data and produce a potential environmental and social impacts assessment, with support from international consultants as required. The consultant will commission and complete studies and work necessary to produce the ESIA in accordance with national regulatory provisions and IFC performance standards. They will also facilitate engagement with ANDE and other stakeholders relevant to the Project.

Terms of reference for the ESIA have been approved by ANDE. Key components of these terms of reference are expected to include:

- Project description and context of the ESIA and the institutional framework of Côte d'Ivoire.
- Description of the baseline conditions of the project area.
- Identification of study area and area of influence of the Project.
- Description of the physical environment, such as climate, air quality, acoustic environment, geology, geomorphology, topography, pedology, hydrogeology, surface hydrology, etc..
- Description of the biological environment, such as fauna, floras, rare or endangered species, natural habitats and sensitive habitats, terrestrial and aquatic environment.
- Description of the social environment, such as administrative, socio-economic, land, cultural and archaeology, ecosystems services, involuntary displacement, nuisances and contributions of the mine to local development.
- Identification and assessment of impacts, and definition of mitigation measures.
- Specific chapters of the ESIA, such as:

- Health, safety and emergency management.
- Environmental and social management plan.
- Conceptual framework for mine closure.
- Resettlement policy framework.
- Public participation and consultation.

The Project is likely to give rise to a range of environmental and social impacts. However, these impacts are considered manageable and controllable through reasonable mitigation practices and therefore would enable effective environmental and social development, operation and closure of the Project

20.8 Environmental Management Plan

The Project is following Montage's environmental and social policies and has developed an Exploration Environmental and Social Management Programme (ESMP) to guide environmental and social management as well as stakeholder and community relations. The Project will aim to conform to the environmental and social requirements of the IFC International Finance Corporation Performance Standards, its associated Environmental Health and Safety guidelines, International Council of Metals and Mining and Equator Principles where they are relevant to the Project.

An Environmental Management Plan (EMP) has been developed for exploration work and this is designed to be developed through the life of the Project. The key priorities of this management are:

- Protect the health of workers, the public, flora and fauna
- Manage all waste generated by exploration operations in a responsible manner
- Minimize emissions generated by exploration, particularly dust.

The Exploration Management Plan being implemented on site will continue to be developed to inform the Koné Project ESMP. The purpose of the ESMP is to ensure that appropriate control and monitoring measures are in place to deal with all significant impacts of the Project. The ESMP has been designed so that it can be regularly reviewed and updated according to company policies. The ESMP includes details of the area of impact, objectives to reduce negative or enhance positive impacts, specific targets adopted to achieve those objectives and definition of responsibilities for implementing the programme. It is a live document that can be reviewed and updated on a systematic basis, in-line with the principles of continual improvement.

Records are being maintained during exploration to monitor all activities and engagement. This includes interaction with local communities, observations of wildlife and environmental conditions and location of boreholes, including those to be abandoned. Procedures for monitoring baseline data have been developed. All exploration programs will be under the control and responsibility of a designated qualified representative of the company and audited to ensure that requirements are met.

20.9 Closure Plans

Closure and rehabilitation of the mine site will commence once mining is complete and a detailed Closure Plan will be developed and finalized prior to closure to guide these activities. Progressive reclamation will be carried out during normal mine operations where circumstances allow. Côte d'Ivoire has a standard format for estimating closure costs, with established tasks and unit rates used. The schedule of costs to be incurred are presented in Table 20-1, below.

Some closure activities can be completed during mine operations. The northern pit, norther dump and TSF will all be closed prior to the end of processing ceases in year 14. These costs are incorporated as sustaining capital costs, with residual closure costs of approximately $5.2 million incurred post-processing (Years 15-16).

Table 20-1 Schedule of Closure Costs

$Million	8	9	10	11	12	13	14	15	16	Total
Infrastructure, Revegetation	0.00	0.00	1.23	0.97	0.97	0.97	0.00	1.43	1.49	7.05
Dumps, Roads	1.51	0.73	5.96	8.72	0.44	0.20	0.00	2.30	0.00	19.87
TSF	0.00	0.00	14.87	14.87	0.00	0.00	0.00	0.00	0.00	29.74
Total Closure Costs	1.51	0.73	22.06	24.57	1.41	1.17	0.00	3.73	1.49	56.67

Post-closure management and maintenance objectives will be to ensure that the site achieves a sustainable and maintenance-free status. The proposed overall strategy for the decommissioning and closure of the Project is as follows:

- Decontaminate, dismantle and demolish, as far as practicable, all installations, structures and infrastructure not identified for retention and hand over to another entity
- Safe disposal of all contaminated materials removed during decontamination, dismantling and demolition activities
- Salvage for sale and/or allocation to other operations, all equipment, mechanical and electrical plant, identified in the asset register as having a residual value or useful life
- Removal from the site as scrap (if economically viable) or dispose as solid waste of all equipment, plant and structures not deemed suitable for future refurbishment and/or re-use
- Apply closure design options which are effective, practical and cost effective
- Ensure the site is left in a safe condition
- Where practical, undertake phased closure of the facilities making allowance in the implementation timeframe for retention of facilities required to support the closure process and subsequent post closure monitoring activities
- Address any potential residual environmental impacts, where appropriate, resulting from Project activities
- Minimize residual impacts requiring on-going monitoring post closure of the facility.

Closure activities commence during the construction period with pre-stripping of topsoil and dumping onto topsoil stockpiles. Revegetation of the area is planned with predominantly indigenous species, establishing an on-site nursery and seed harvesting of local species. The waste rock and tailings are considered benign and non-hazardous; no acid rock drainage (ARD) or metal leaching is expected during operations or post-closure.

Initial closure activities will focus on the rehabilitation of the waste rock dumps (WRD) and tailings storage facility (TSF) five years prior to the mine closure. Benches on the WRD will be cut and filled to produce a landform in keeping with the surrounding landscape. Topsoil will be placed over the WRD to a depth of about 0.15 m. The combination of the shallow slope, compaction of the waste rock and revegetation will minimize the infiltration of precipitation into the WRD, and maximize water runoff and evapotranspiration.

Open pits will be made safe for closure during operations, with slopes being made stable and access points controlled. Waste rock will be backfilled into the northern pit once exhausted, commencing in year 8. The southern pit will cease operations in year 9 and will be used for tailings storage until the end of the mine life. The south pit will fill with rainfall post-closure, flooding the tailings and the majority of exposed surfaces and will start decanting 15 to 20 years after closure.

Closure of the TSF will commence with the placement of approximately 0.5m of waste rock over the surface. Topsoiling and revegetation will follow the same methodology as the WRD. The placement and levelling of the waste rock will promote water runoff and minimize ponding.

Buildings and structures other than the camp and solar farm will be dismantled. This will include the process plant, conveyors, workshops, offices and other ancillary structures. The building structures will be dismantled, and the materials removed from the site for sale, reuse, recycling, or disposal at a registered waste site. All oil, fuels, and processing chemicals will be drained from the equipment and disposed of at a licensed off-site disposal facility. The processing equipment and conveyor structures will be removed from site and sold or recycled. All the disturbed areas will be ripped or ploughed (to increase water infiltration and reduce the potential for surface erosion and instability), levelled and covered with about 0.15 m of topsoil (except the concrete structures). Revegetation will be as for the WRD. Concrete foundations will remain in situ and covered with about 0.40 m of topsoil, either from stockpiles or imported as necessary. The tailings and water supply pipelines will be removed and disposed of off-site. Any roads that will not be required for post-closure management will be decommissioned, unless otherwise requested and agreed with local communities.

Active site management and maintenance is expected to continue for five years after closure. This will entail inspections at appropriate intervals to ensure that any soil erosion is repaired, vegetation density is maintained, the integrity of water control structures is maintained, and the ecology of the area achieves the required status. Passive closure is anticipated to continue for a further five years with inspection intervals reduced appropriately. Maintenance will be carried out on an as-required basis. Closure monitoring will be undertaken to document the progression of the mine site from the operational phase to relinquishment.

20.10 Health and Safety

The exploration works are being conducted in accordance to best practice for labour safety. All personnel are subject to site-specific health and safety training prior to commencing work. Only suitably trained personnel are allowed to operate machinery.

Appropriate clothing is required to be worn, including personal protective equipment. Alcohol consumption is banned before and during working hours. As a result, the health and safety culture is being developed during exploration, and this will be extended through construction and life of operation.

20.11 Monitoring

The Project has initiated an environmental and socially related baseline data collection programme to determine the current conditions of the potential exploitation area. Initial data collection will include collection and analysis of surface water and groundwater quality, installation of weather recording and air quality instrumentation, recording wildlife type and movements, and identification of important environmental and cultural sites in the Project area.

Montage has developed and implemented an environmental and social monitoring plan, including appropriate sampling procedures. Currently, baseline monitoring of weather data, water sampling and ecology are underway. Passive infrared camera traps are being used to capture wildlife, as the density of vegetation and abundance of both water and food, combined with the presence of farmers, means that wildlife is shy and difficult to count.

Groundwater levels are recorded on a quarterly basis, and water quality samples are to be collected and sent for analysis on a six-monthly basis. Interactions with local communities are recorded in a daily diary, along with wildlife observations and any other items of environmental interest. The Marahoué river water quality is being monitored quarterly, with river levels monitored using dataloggers and manual verification.

The baseline assessment studies will be used to develop a more detailed environmental and social assessment and determine any additional further monitoring requirements and planning. The objective of the studies is to identify receptors of potential impacts that the Project may have on the surrounding environments (biophysical and social) and which should be examined and assessed in more detail as the Project develops. The EIES is a multi-disciplinary and iterative process, and these baseline studies provide the first stage of this process. Monitoring programmes will continue to inform important activities through the life of the Project to observe any changes in the environment.

20.12 Public Consultation

There are currently no objections to the development of the Project. As part of the environmental assessment, public consultation and disclosure is required. In order to ensure that the Project is developed and operated in an appropriate manner, Montage Gold will incorporate the concept that effective engagement with its stakeholders is an essential component of the assessment process and its ongoing "licence to operate". Montage is committed to a proactive program of communications with all relevant stakeholders.

The Project has few stakeholders, with the closest people being the towns of Fadiadougou, Batogo and Manabri. Meetings have been held with the leaders of these communities already (Figure 20-7 and

Figure 20-8) and ongoing meetings are planned. A record of all meetings is being maintained, summarizing the numbers of people engaged with, their activities and any issues or concerns they may have with the Project.

Figure 20-7 Meeting with Fadiadougou chief and elders

Source: Montage

Figure 20-8 Meeting with Batogo chief and elders

Source: Montage

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The overall study capital cost estimate was compiled by Lycopodium and is presented here in summary format. The various elements of the Project estimate have been subject to internal peer review by Lycopodium and have been reviewed with Montage for scope and accuracy.

The capital cost estimate was developed to an accuracy level range of ±15% to cover engineering, procurement, construction, and start-up of the mine and processing facilities, as well as the ongoing sustaining capital costs. The capital cost estimates were developed for a conventional open pit mine, CIP process plant and supporting infrastructure for an operation capable of treating 11.0 Mtpa of material. For the purpose of this DFS, provision of the power plant and the operations camp will be constructed under a 5 year Build Own Operate Transfer arrangement. Mining will be carried out by a contractor.

The estimate covers the direct costs of purchasing and constructing the CIP facility and infrastructure components of the project. Mining related infrastructure has been assumed to be provided by the mining contractor and no allowance for these costs has been made in the financial model.

Indirect costs associated with the design, construction and commissioning of the new facilities, owner's costs, and contingencies have also been estimated, based on percentages of the direct capital cost estimate. Risk amounts are specifically excluded from this estimate. A breakdown of the capital cost estimates is shown in Table 21-1.

All costs are estimated in United States dollars (US$) as at 4Q21.

21.2 Capital Cost Summary

The capital estimate is summarized in Table 21-1 and 21-2. The initial project capital cost is estimated at US$543.9M, including a contingency allowance of US$50.5M.

Table 21-1 Capital Estimate Summary (4Q21, ±15%)

Main Area	US$M
Mine	39.6
Treatment Plant	320.7
TSF	50.6
Camp	3.3
Resettlement	9.5
EPCM	39.4
Owners Costs	30.3
Subtotal	**493.3**
Contingency	50.5
Grand Total	**543.9**

The total LOM cost is estimated at US$835.6M including sustaining capital costs of US$291.7M, as shown in Table 21-2. The LNG power plant and the camp will be financed under 5 year Build Own Operate Transfer (BOOT) contracts.

Table 21-2 Sustaining Capital Estimate Summary (4Q21, ±15%)

Main Area	US$M
Camp	5.7
TSF	59.4
Power	138.0
Process Plant	31.9
Closure	56.7
Grand Total	**291.7**

21.2.1 Capital Costs – Mining

Due to the use of mining contractors, who will provide the mining fleet, the capital costs include the provision of water and power to the contractor workshop and reticulation for the pit dewatering US$0.9M has been included in Year -1. In addition $38.7M of pre-strip costs are incurred in Year -1.

21.2.2 Capital Cost – Process Plant and Infrastructure

To develop the process plant and infrastructure cost estimate a mechanical equipment list was compiled, based on the process requirements, and major equipment pricings were obtained from technology / equipment suppliers or from a database of similar size projects and factored as required for the project capacity.

Project infrastructure includes mine infrastructure as itemised in Section 18.6.2.

The EPCM estimate was factored based upon Lycopodium's recent experience with similar type and size of projects. Expenses such as catering and accommodation for the Engineer's site personnel, as well as site telecommunications costs, are included in the estimate.

A contingency allowance is included to make specific provision for uncertain elements of cost within the project scope. Contingencies do not include allowances for scope changes, escalation, or exchange rate fluctuations. Contingency has been applied to all parts of the process plant estimate.

21.2.3 Capital Cost – TSF

Construction quantities have been determined based on the design to an overall accuracy of ±15%. A contingency of 7.5% has been applied to engineering items, with 10.0% applied to preliminary and general items and 15% for earthworks and liners. Rates have been based on budget costing provided by seven earthworks contractors located within the region who have prior recent experience of construction of similar TSFs in Côte d'Ivoire.

The closure costs are incurred in Year 10 and 11 once the tails deposition in the south pit have commenced. Summarised costs are provided in Table 21-3.

Table 21-3 TSF Capital Estimate Summary (4Q21)

	Pre-Production	Sustaining	Closure	LOM
Engineering & Design	2.0	5.1	1.2	8.3
Preliminary & General	9.7	5.4	3.3	18.4
TSF	29.9	38.9	19.8	88.6
WSF and River Abstraction	6.6	0.0	0.0	6.6
Fuel	2.4	2.8	1.7	6.9
Total	50.6	52.2	26.1	128.9
Total + Contingency	57.6	59.4	29.8	146.7

21.3 Operating Cost Summary

21.3.1 Operating Costs – Mining

As the bulk of the mining costs would be related to a mining contract, the mine operating costs were derived from several existing mining contracts using similar equipment awarded in West Africa and in some instances were validated against equipment purchase and parts supply studies conducted in 2020 in Europe and the CIS.

Fixed mining costs were calculated for the assumed fleet and dependant on material type (Crusher Feed/Waste and Oxide + Transitional/Fresh) and destination. These included components for:

- Loading costs.
- Fixed hauling costs.
- Drill & Blast costs.
- Ancillary costs.
- Mine admin costs.

Fixed mining costs do not include any time haul trucks spend travelling up or down in-pit ramps.

Table 21-4 Fixed Mining Costs

Ore/Waste	Feed	Feed	Feed	Waste	Waste	Waste
Material	Oxide	Transitional	Fresh	Oxide	Transitional	Fresh
Loading	0.32	0.34	0.36	0.32	0.34	0.36
Fixed Hauling	0.34	0.34	0.34	0.36	0.36	0.36
Drill & Blast	0.89	0.89	1.00	0.74	0.74	0.91
Ancillary	0.50	0.50	0.50	0.50	0.50	0.50
Mine Admin	0.14	0.14	0.14	0.14	0.14	0.14
Grade Control	0.26	0.26	0.26	0.00	0.00	0.00
Total Fixed Cost	2.45	2.47	2.60	2.06	2.08	2.28

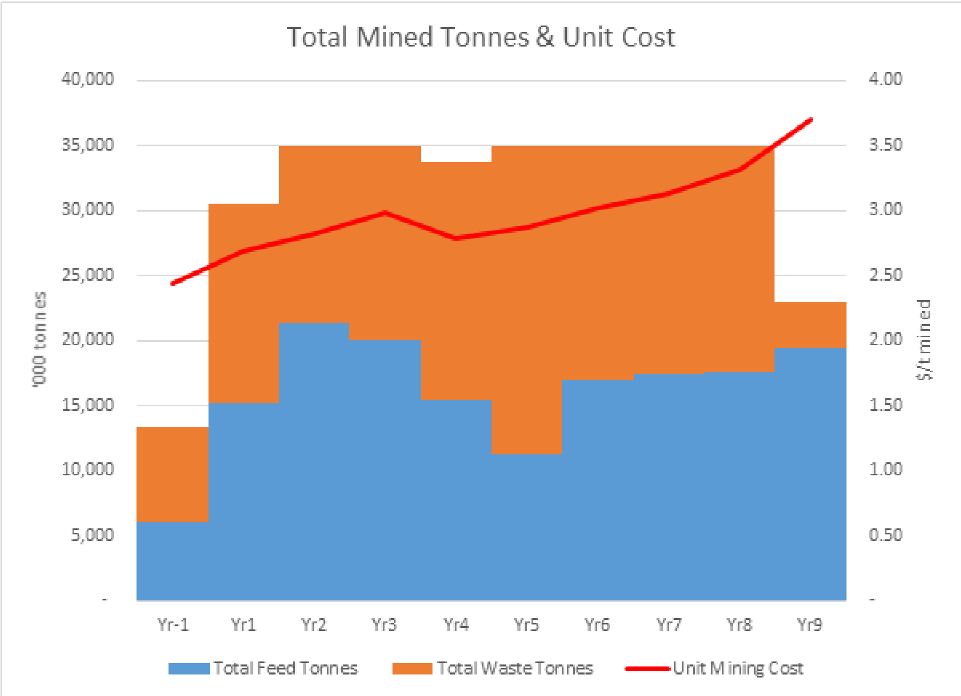
Incremental mining costs were determined for the fleet and included a fuel and non-fuel component. The non-fuel component covered costs such as operator salary, maintenance costs and other running costs associated with the time spent on ramps. The Incremental Mining Cost was determined to be $0.027/t/10m vertical lift based on a reference RL of 375 m.

The fuel price used for mining optimization calculations was $0.85/litre.

Rehandle costs were not included in the optimization as it was anticipated that the amount of rehandle would be quite low. Rehandle was costed at $0.93/tonne in the financial model.

Figure 21-1 shows the unit mining cost for the life of the operation compared to the total tonnes mined in each period. The unit costs increase gradually as the depth of the pit increases.

Figure 21-1 Unit Mining Cost



21.4 Operating Cost – Plant and Infrastructure

The Operating Cost Estimate (OPEX) for the plant and infrastructure has been divided into multiple cost centres, with Fixed and Variable costs calculated for each cost centre for each different material type. The operating cost estimate is presented in Table 21.3 and is deemed to have an accuracy of ±15% based on pricing as at 4Q21. The process operating cost includes all direct costs to produce gold doré for the Project.

In general, costs have been built up from first principle estimates, with quotations obtained for major reagents and consumables and consumption rates based on metallurgical testwork, calculations or modelling. Minor reagents, laboratory, expatriate labour rates and a number of G&A costs have been sourced from the Lycopodium database. Power consumption has been calculated from the gross power required to achieve the desired grind size on each material type, based on OMC comminution modelling, plus the remaining installed power from the mechanical equipment list, with suitable drive efficiency and utilization applied and factored for the design throughput. The total power draw was used to calculate power costs based on a IPP power price of US$0.0998/kWhr.

Total fixed general and administration ("G&A") costs are estimated at $10.1M annually.

Table 21-5 South Pit Operating Cost per Material Type

COST CENTRE	Total				LOM	
	Oxide	Transition	Fresh	HW Fresh	Fix	Variable
	US$/t	US$/t	US$/t	US$/t	US$'000/yr	US$/t
Power (exc. crush & grind)	0.49	0.49	0.49	0.49	7,549	0.49
Power (crushing/ore storage)	0.03	0.04	0.06	0.06		0.06
Power (grinding)	0.51	1.02	1.53	1.53		1.44
Operating Consumables	3.83	3.24	3.79	3.80		3.77
Maintenance	0.09	0.13	0.13	0.13	5,724	0.13
Total Processing (Variable)	4.95	4.92	6.01	6.02	13,273	5.89
Laboratory	0.14	0.14	0.14	0.14		0.14
Process & Maintenance Labour					4,176	
Total Labour, Lab (Fix)	0.14	0.14	0.14	0.14	4,176	0.14
TOTAL	**5.08**	**5.06**	**6.15**	**6.16**	17,448	**6.03**

Table 21-6 North Pit Operating Cost per Material Type

COST CENTRE	Total				LOM	
	Oxide	Transition	Fresh	HW Fresh	Fix	Variable
	US$/t	US$/t	US$/t	US$/t	US$'000/yr	US$/t
Power (exc. crush & grind)	0.49	0.49	0.49		7,549	0.49
Power (crushing/ore storage)	0.03	0.04	0.06			0.04
Power (grinding)	0.51	1.02	1.53			0.89
Operating Consumables	3.83	3.55	4.13			3.85
Maintenance	0.09	0.13	0.13		4,743	0.11
Total Processing (Variable)	4.95	5.23	6.35		12,292	5.38
Laboratory	0.14	0.14	0.14			0.14
Process & Maintenance Labour					4,176	
Total G&A, Labour, Lab (Fix)	0.14	0.14	0.14		4,176	0.14
TOTAL	**5.08**	**5.37**	**6.48**		16,467	**5.52**

21.5 Exclusions

The following items have been excluded from the operating cost estimate:

- All sunk costs.
- Government monitoring and compliance costs.
- All Montage head office costs and corporate overheads.
- Withholding taxes and other taxes, such as GST or VAT.
- Escalation.
- Financing costs.
- Foreign exchange fluctuations.
- Interest charges.
- Political risk insurance.
- All costs associated with areas beyond the battery limit of the study.
- Land compensations costs which are included in pre-production capital.
- Subsidies to local communities which are included in Community royalty @ 0.5%.
- Licence fees.
- Royalties which are included in the financial model @ Government 4.0% and Meverix 2.0%.
- Contingency.
- All mining and exploration costs, except power costs for mining services within Lycopodium scope of work, Montage owned mine light vehicle costs and grade control sample assay costs.
- Maintenance costs of all mine, haul and plant access roads which are included in mining costs.
- Gold refining costs and Doré transport and insurance costs are included in the financial model @ $4.71/oz.
- Tailings storage costs, including future lifts and rehabilitation, which are included in sustaining capital.
- Tailings dust suppression costs.
- Any rehabilitation or closure costs which are included in sustaining capital.

22 ECONOMIC ANALYSIS

22.1 Introduction

The economic analysis is based on Inferred Resources and mine schedule as per Table 22-4.

An economic analysis has been carried out for the project using a cash flow model. The model is constructed using annual cash flows by taking into account annual mined and processed tonnages and grades for the CIP feed, process recoveries, metal prices, operating costs and refining charges, royalties and capital expenditures (both initial and sustaining).

The financial assessment of the project is carried out on a "100% equity" basis and the debt and equity sources of capital funds are ignored. No provision has been made for the effects of inflation. Current Côte d'Ivoire tax regulations are applied to assess the tax liabilities, duties and other levies. All amounts in this section are presented in US$. Discounting and IRR calculations have been applied mid year from the first year of operation and pre-production capital is deducted on an undiscounted basis.

The model reflects the base case and technical assumptions as described in the foregoing sections of this report.

22.2 Model Inputs and Assumptions

The model inputs and assumptions used in the economic analysis are summarized in Table 22-1 and unless otherwise stated, is used in the model.

Table 22-1 Model Inputs and Assumptions

Model Inputs	Unit / Value
Base Currency	US$
Base Date	4th Quarter 2021
Côte d'Ivoire Royalty @$1,600/oz (charged against Revenue)	4.0%
Maverix Royalty (charged against Revenue)	2.0%
Community Royalty (charged against Revenue)	0.5%
Côte d'Ivoire Tax Rate	25%
NPV Discount Rate	5%
Metal Price – Fixed for LOM	US$1,600/oz
Refining Payability	99.9%
Refinery Charges & Shipping	US$4.71/oz
Assumptions	
Capex excludes Finance Charges & Fees	
Capex excludes Pre-production Investigations	
Capex Amortisation/Depreciation makes no allowance for potential salvage value	
Capex excludes escalation	
Tax paid on an Annual mid year basis in the following year	

22.2.1 Capital Costs

Pre-production capital expenditures are defined in Table 22-2. Sustaining capital for the Plant, Mining and TSF expansion costs have been phased over the life of the project and detailed in Table 22-3.

Table 22-2 Pre-production Capital Expenditure

Item	Unit	Total	Year		
			-3	-2	-1
Mine	US$M	39.6			
Process Plant	US$M	320.7			
TSF	US$M	50.6			
Camp	US$M	3.3			
Resettlement	US$M	9.5			
EPCM	US$M	39.4			
Owner	US$M	30.3			
Construction Sub Total	**US$M**	**493.3**			
Contingency	US$M	50.5			
Construction Total	**US$M**	**543.9**	**22.7**	**173.2**	**348.0**

Table 22-3 Sustaining Capital Expenditure

Item	Unit	LOM
Camp	US$M	5.7
TSF	US$M	59.4
Power	US$M	138.0
Plant	US$M	31.9
Mine Closure	US$M	56.7
Sustaining Total	US$M	**291.7**

22.2.2 Revenue

Revenue has been calculated allowing for 0.1% refinery loss.

22.2.3 Royalties

Royalties at 6.5% have been included for the LOM and charged against the revenue.

22.2.4 Cost of Sales

Cost of Sales includes freight and refining costs. A value of US$4.71/oz gold recovered has been allowed for in the model.

22.2.5 Depreciation

Depreciation is calculated using a straight line depreciation over 5 year starting with first year of production and can be summarized as follows:

- Initial pre-production capex depreciated over the first five years of production
- Capitalised pre-production costs (i.e. cumulative exploration and study costs) to date depreciated over the total LOM, using estimated total capitalised pre-production costs of US$22.4M to December 31, 2021
- The annual plant sustaining capital is assumed to be largely repairs and maintenance
- Remaining sustaining capital items (i.e. TSF, Generators, Camp) each depreciated separately based on 5 year straight line from start of use.

22.2.6 Inflation

Inflation has not been included in the cash flow analysis.

22.2.7 Operating Costs

Annual fixed and variable costs, as per Sections 21.5 and 21.6, are included in the cash flow.

22.2.8 Financial Model

Figure 22-1 shows the pre-tax and post –tax cumulative cash flow for the project over the LOM; the payback period corresponds to when the cumulative cash becomes positive for the pre-tax and the post-tax model. Figure 22-2 shows the annual and cumulative post-tax cash flow.

The pre-tax and post-tax financial results of the project are summaries in Table 22-4. On a pre-tax basis, the project has a Net Present Value (NPV) of US$991.6M at a discount rate of 5%, an Internal Rate of Return (IRR) of 39.6%; on a post-tax basis the NPV is US$746.2M at a discount rate of 5%, the IRR is 34.8% and the payback period is 2.7 years following commencement of production.

Figure 22-1 Cumulative Cash Flow

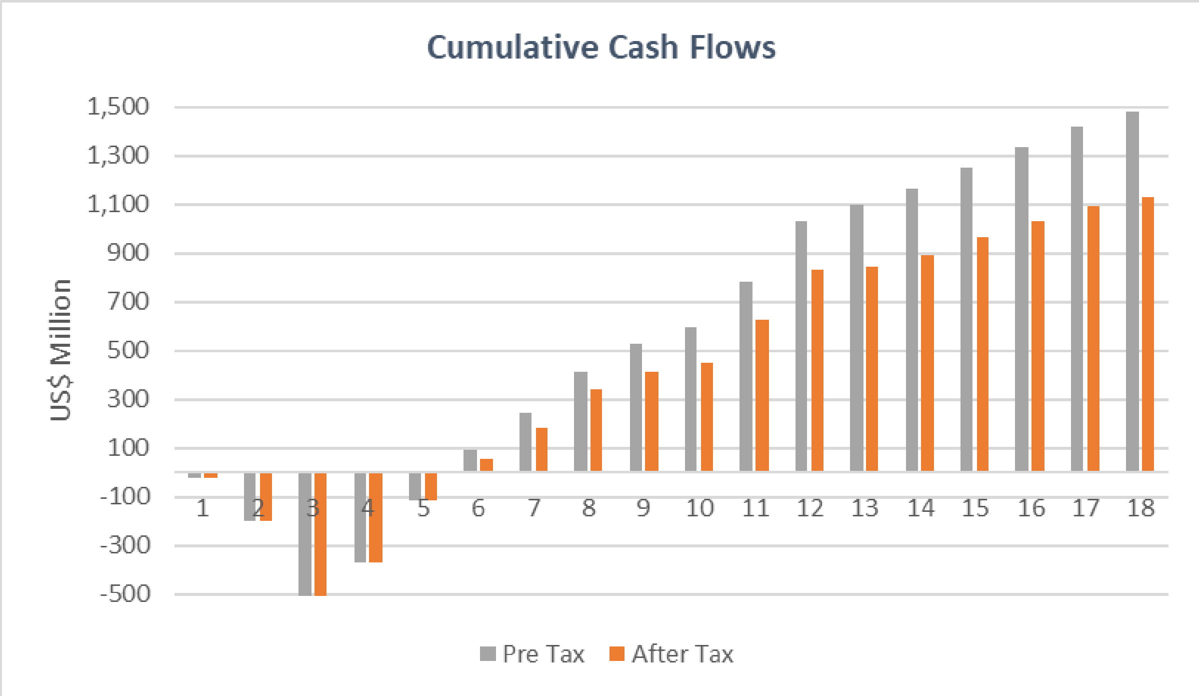


Table 22-4 Mine and Process Schedule

Kone - DFS Financial Model vs 1s.xlsx

Description	Unit	LOM Total	Yr-3	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15
Mining																				
North Pit Tonnes	Mt	1.9						0.2	0.4	0.5	0.6	0.1								
North Pit Grade	Au g/t	0.50						0.49	0.46	0.50	0.52	0.56								
South Pit Tonnes	Mt	159.1			6.7	18.8	23.0	17.5	13.4	18.8	15.1	13.0	21.3	11.7						
South Pit Grade	Au g/t	0.66			0.61	0.66	0.73	0.68	0.62	0.64	0.65	0.55	0.64	0.81						
Total Tonnes	Mt	161.1			6.7	18.8	23.0	17.7	13.8	19.3	15.7	13.1	21.3	11.7						
Total Grade	Au g/t	0.66			0.61	0.66	0.73	0.68	0.62	0.64	0.65	0.55	0.64	0.81						
North Pit Waste Tonnes	Mt							0.6	0.6	0.5	0.4	0.1								
South Pit Waste Tonnes	Mt				11.1	16.2	12.0	16.4	20.6	15.2	18.9	21.8	9.2	2.0						
Total Waste Tonnes	Mt	145.7			11.1	16.2	12.0	17.1	21.2	15.7	19.3	21.9	9.2	2.0						
Strip Ratio	W:O	0.90			1.66	0.86	0.52	0.96	1.54	0.81	1.23	1.67	0.43	0.17						
Processing																				
Stockpile Rehandle	Mt	71.3				0.3	0.7	0.7	2.5	0.5	0.7	0.7	0.7	0.7	11.0	11.0	11.0	11.0	11.0	8.6
Oxide Tonnes	Mt	10.7				0.6	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.6
Oxide Grade	Au g/t	0.56				1.04	0.96	0.94	0.94	0.75	0.42	0.39	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38
Transition Tonnes	Mt	8.3				2.7		0.2	0.1	0.1	0.1							0.3		4.9
Transition Grade	Au g/t	0.56				0.91		0.82	0.75	0.67	0.52							0.40		0.35
Fresh Tonnes	Mt	124.6				5.7	10.0	8.9	8.5	9.0	9.1	8.6	9.6	9.8	10.2	10.3	10.3	2.8	8.6	3.2
Fresh Grade	Au g/t	0.69				0.89	1.00	0.89	0.77	0.84	0.78	0.60	0.86	0.88	0.44	0.44	0.44	0.44	0.43	0.43
FW Fresh Tonnes	Mt	17.5				0.4	0.3	1.1	1.7	1.2	1.1	1.7	0.6	0.5	0.0			7.2	1.7	
FW Fresh Grade	Au g/t	0.56				0.87	0.95	0.78	0.72	0.70	0.68	0.58	0.66	0.56	0.51			0.42	0.41	
Total Processed Tonnes	Mt	161.1				9.5	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	8.6
Total Processed Grade	Au g/t	0.66				0.91	0.99	0.88	0.78	0.82	0.75	0.58	0.82	0.83	0.44	0.44	0.43	0.42	0.42	0.38
Total Process Recoveries	%	89.3%				91.2%	91.3%	90.7%	90.0%	90.3%	89.7%	88.2%	90.3%	90.5%	85.8%	85.8%	85.7%	86.9%	85.8%	88.4%
Total Recovered	000 ozs	3,059				251	320	283	247	261	238	182	261	267	133	133	132	130	128	94

Table 22-5 Financial Model

Kone - DFS Financial Model vs 1s.xlsx

Description	Unit	LOM Total	Yr-3	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16
Pre-Production Capex	$M	(543.9)	(22.7)	(173.2)	(348.0)																
Sustaining Capex	$M	(291.7)				(31.7)	(33.8)	(33.0)	(30.9)	(29.1)	(49.0)	(8.0)	(7.3)	(3.8)	(24.6)	(27.5)	(4.0)	(3.7)	(0.0)	(3.8)	(1.5)
Revenue	$M	4,889.9				401.5	511.8	452.1	394.8	416.6	380.0	291.4	416.9	426.9	213.1	212.9	210.4	207.5	204.1	150.1	
Selling Costs	$M	(14.4)				(1.2)	(1.5)	(1.3)	(1.2)	(1.2)	(1.1)	(0.9)	(1.2)	(1.3)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.4)	
Royalties	$M	(317.8)				(26.1)	(33.3)	(29.4)	(25.7)	(27.1)	(24.7)	(18.9)	(27.1)	(27.7)	(13.8)	(13.8)	(13.7)	(13.5)	(13.3)	(9.8)	
Op Cost Mining	$M	(836.9)			(38.8)	(83.1)	(92.3)	(89.7)	(89.4)	(96.0)	(96.0)	(100.1)	(98.6)	(49.8)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	
Op Cost Process Fixed	$M	(257.8)				(17.4)	(17.4)	(17.4)	(17.4)	(17.4)	(17.4)	(17.4)	(17.4)	(17.4)	(17.4)	(17.4)	(17.4)	(17.4)	(17.4)	(13.7)	
Op Cost Process Variable	$M	(1,036.8)				(54.9)	(67.5)	(67.2)	(69.1)	(67.3)	(67.5)	(67.6)	(67.5)	(67.5)	(77.1)	(77.1)	(77.1)	(77.0)	(77.1)	(55.2)	
G&A	$M	(150.0)				(10.1)	(10.1)	(10.1)	(10.1)	(10.1)	(10.1)	(10.1)	(10.1)	(10.1)	(10.1)	(10.1)	(10.1)	(10.1)	(10.1)	(8.0)	
Operating Profit	$M	2,315.0				208.7	289.6	236.9	181.9	197.4	163.1	76.3	194.9	252.9	93.4	93.2	90.9	88.3	85.0	62.5	
Net Cash Flow before tax	$M	1,479.4				177.1	255.8	203.9	151.0	168.3	114.0	68.4	187.6	249.1	68.8	65.7	86.9	84.5	84.9	58.8	(1.5)
NPV Pre Tax	$M	991.7																			
IRR Pre Tax	$M	39.6%																			
Depreciation	$M	(769.4)				(208.7)	(150.3)	(132.3)	(137.9)	(30.0)	(32.8)	(27.3)	(21.3)	(15.9)	(10.5)	(1.5)	(0.4)	(0.2)	(0.1)	(0.2)	
Other Sustaining / Closure	$M	(88.6)					(0.6)	(2.5)	(3.1)	(2.5)	(3.1)	(2.5)	(6.6)	(3.2)	(24.6)	(27.1)	(3.9)	(3.7)		(3.7)	(1.5)
Taxable Profit	$M	1,457.0					138.7	102.1	40.9	165.0	127.1	46.6	167.0	233.8	58.3	64.7	86.6	84.3	84.8	58.6	(1.5)
Tax	$M	(364.6)						(34.7)	(25.5)	(10.2)	(41.2)	(31.8)	(11.6)	(41.7)	(58.5)	(14.6)	(16.2)	(21.6)	(21.1)	(21.2)	(14.7)
Net Cash Flow after tax	$M	1,114.8				177.1	255.8	169.2	125.5	158.1	72.8	36.6	176.0	207.3	10.3	51.1	70.7	62.9	63.8	37.6	(16.1)
NPV Post Tax	$M	746.2																			
IRR After Tax		34.8%																			
Cash Cost	$/pay oz	838				764	690	757	858	837	909	1,176	847	647	894	895	904	915	929	929	
AISC	$/pay oz	933				908	814	892	1,002	967	1,134	1,238	894	680	1,097	1,120	953	962	948	988	
Payback Period	yrs	2.7																			

22.2.9 Financial Summary

The results of the financial model are summarized in Table 22-5.

Revenue generated per domain is shown in Figure 22-2.

A breakdown of the total cash costs is shown in Figure 22-3. Table 22-6 shows the breakdown of the LOM cash costs and unit costs per tonne processed.

Table 22-6 Financial Model Summary @ $1,600

Description	Units	LOM
Feed Tonnage	Mt	161.1
Waste Rock	Mt	145.7
Total Mined	Mt	306.7
Strip Ratio	W:O	0.90
Feed Grade Processed (average)	g/t	0.66
Gold Recovery (average)	%	89.3%
Gold Production	'000 oz	3,059
Annual Gold Production (average)	'000 oz/y	207
Pre-production Capital Cost	US$M	(544)
Sustaining Capital Cost	US$M	(292)
Total Capital Cost	US$M	(836)
Net Revenue	US$M	4,890
Selling Costs	US$M	(14)
Royalties	US$M	(318)
Total Operating Costs	US$M	(2,281)
EBITDA	US$M	2,315
Tax	US$M	(365)
Net Cash Flow After Tax	US$M	1,115
$NPV_{5\%}$ After Tax	US$M	746
IRR	%	34.8%
Cash Cost	US$ /pay oz	838
AISC	US$ /pay oz	933

Figure 22-2 Revenue Generated per Material Type

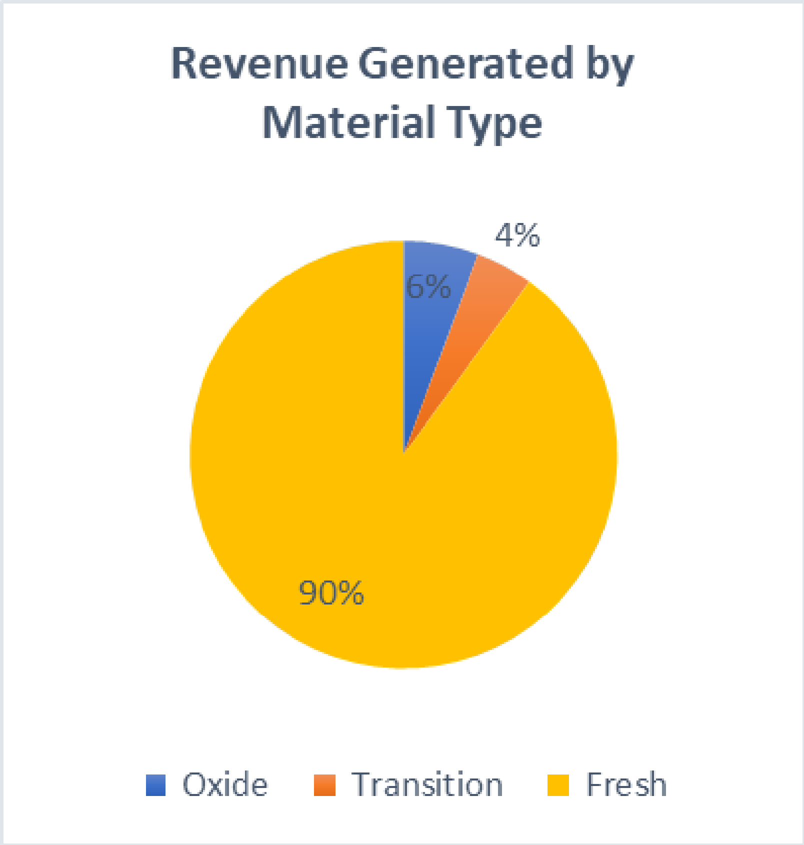


Figure 22-3 Operating Expense Split

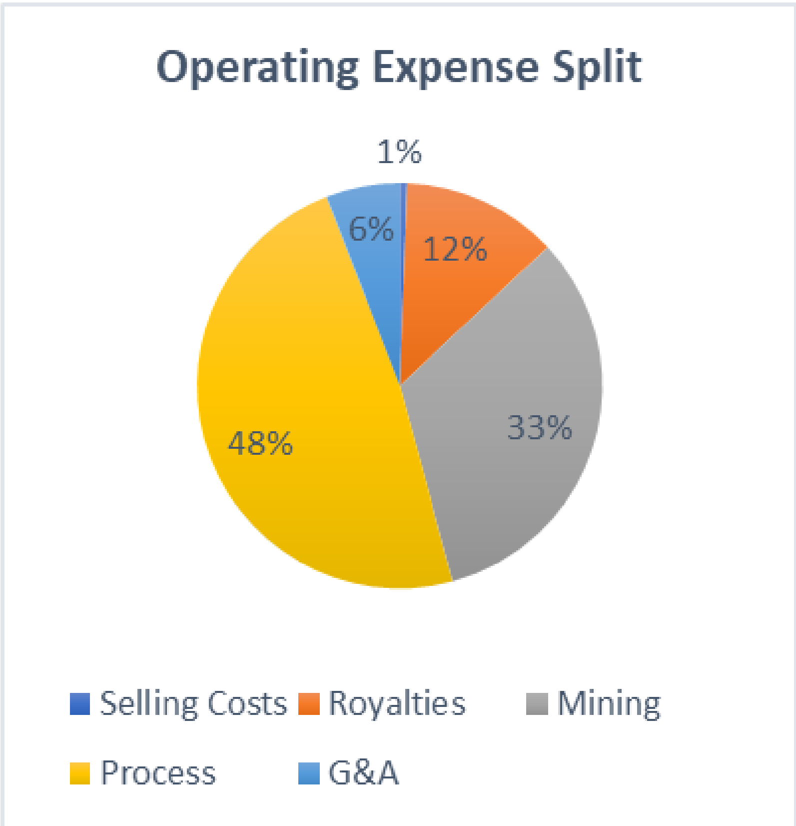


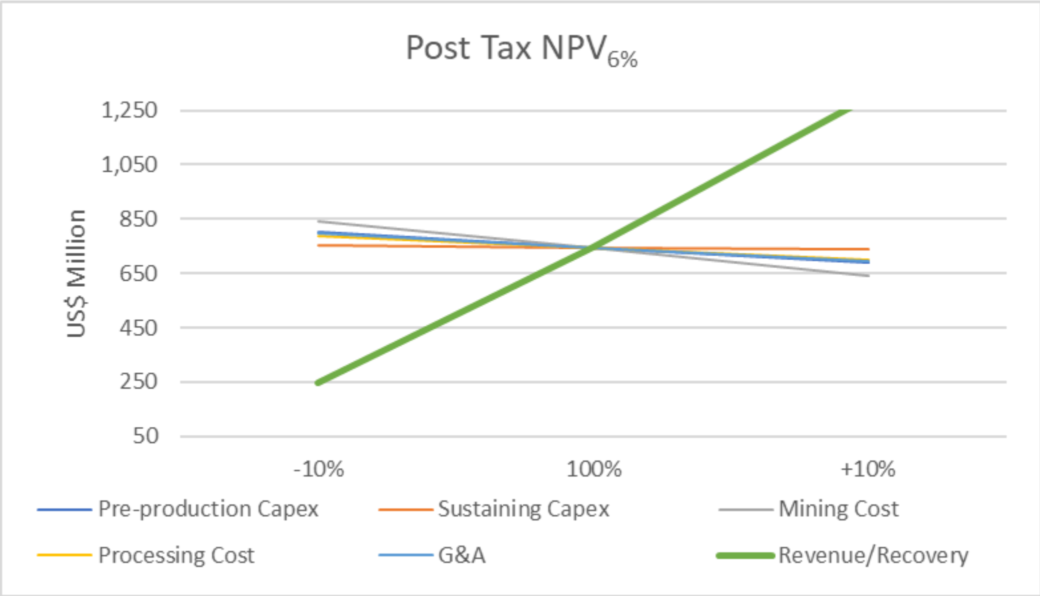
Table 22-7 Cash Cost and Unit Cost Summary

Description	LOM ($/oz)	LOM ($/t processed)
Mining	261	4.95
Processing	424	8.04
G&A	49	0.93
Royalties	104	1.97
Total Cash Cost	838	15.89
Sustaining Capital	77	1.46
Closure	19	0.35
All-in Sustaining Costs	933	17.71

22.2.10 Single Parameter Sensitivities

Figure 22-4 shows the changing post-tax $NPV_{7\%}$ and IRR for varying single parameter sensitivities for revenue, pre-production and sustaining capital costs, mining, plant and G&A operating costs and revenue / gold recovery. Figure 22-4 also shows the post-tax IRR sensitivity to parameters that the NPV is most sensitive revenue / recovery.

Figure 22-4 NPV and IRR Sensitivity



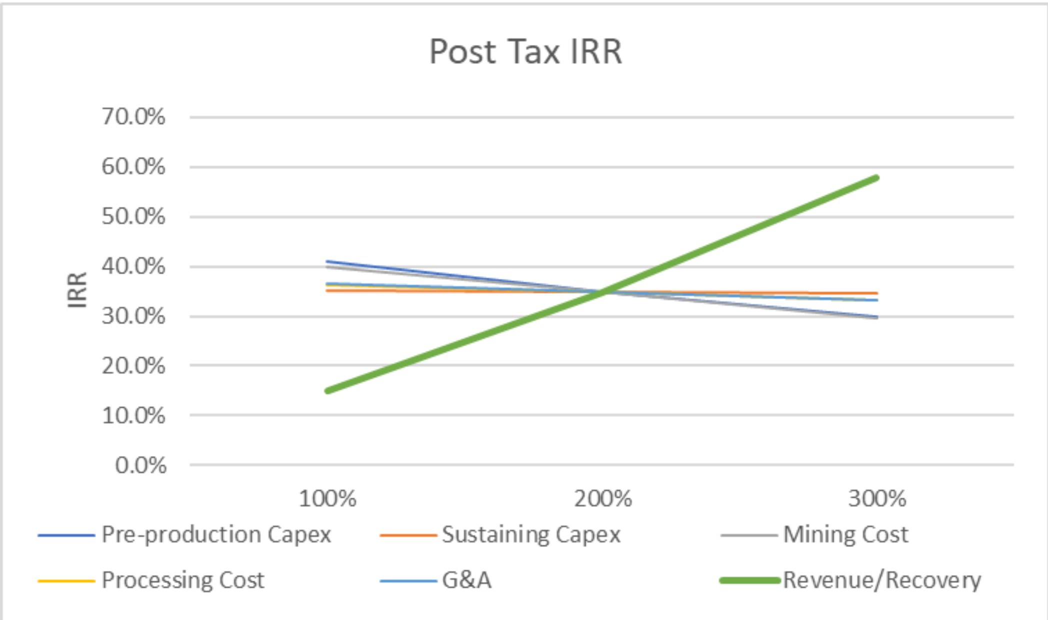


Table 22-8 shows the sensitivity of the NPV and IRR with gold price and discount rate.

Table 22-8 NPV and IRR Sensitivity

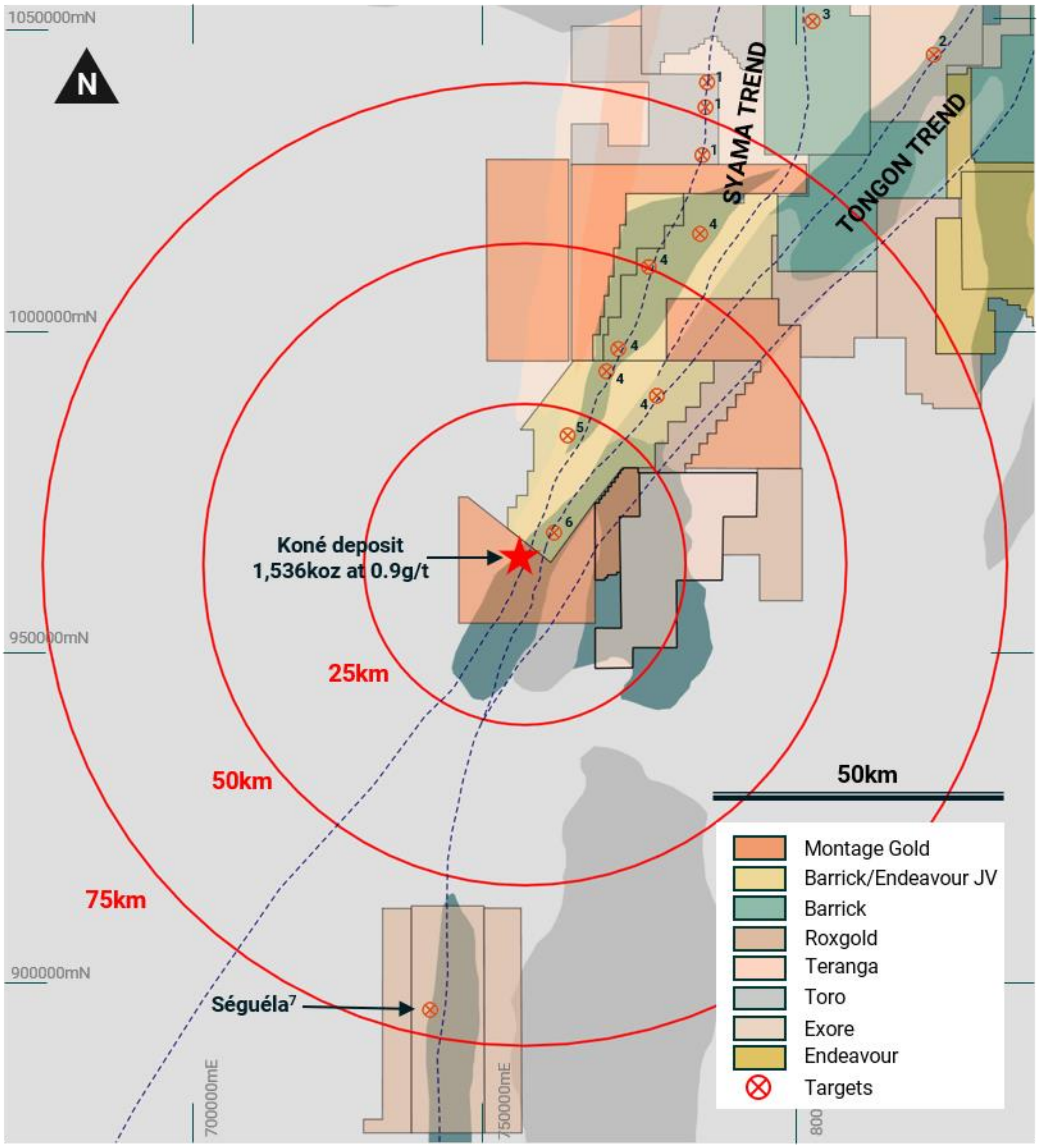
Gold Price	1,400	1,500	**1,600**	1,650*	1,700	1,850	2,000
NPV $_{5\%}$	417	582	746	799	881	1,043	1,367
IRR	21.8%	28.3%	34.8%	37.0%	40.3%	47.0%	60.9%
Cash Cost	825	831	838	858	861	869	884
AISC	920	927	933	953	957	964	979
Payback	3.8	3.1	2.7	2.5	2.4	2.1	1.8

*** Three-year trailing average (31 December, 2021)**

23 ADJACENT PROPERTIES

Figure 23-1 shows tenements held by other owners in the region of the Koné Gold Project. This figure is derived from the Côte d'Ivoire Ministry of mines and geology's mining cadastral (Côte d'Ivoire Ministry of mines, 2020). Immediately to the north of the Koné Exploration Permit lies the Mankono Joint Venture held by Barrick Gold and Endeavour Mining. To the east of the Koné Exploration Permit is the Dianra Exploration Permit that is held by Teranga Gold Corp.

Figure 23-1 Adjacent properties



Source: Montage August 2020

24 OTHER RELEVANT DATA AND INFORMATION

There is no additional information or explanation is required in order to make this report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geological setting and assessment status

The Indicated Mineral Resource Estimates for the Koné deposit are based on 102,249 m drilling (54,703.4 m of core and 45,545.3 m of RC). The deposit has been tested by 50 m spaced traverses of generally 50 m, and rarely 25 m spaced holes with drilling on each traverse extending to vertical depths of between 60 m and 560 m.

The handling, sampling, transport, analysis, geological logging and storage of sample material along with documentation of analytical results is consistent with the author's experience of good, industry standard practise.

The author considers that quality control measures adopted for the Koné drilling and exploration sampling have established that the sampling is representative and free of any biases or other factors that may materially impact the reliability of the sampling.

The author considers that quality control measures adopted for sampling and assaying have established that the field sub-sampling, and assaying is representative and free of any biases or other factors that may materially impact the reliability of the sampling and analytical results. The author considers that the sample preparation, security and analytical procedures adopted for the Koné drilling provide an adequate basis for the Mineral Resource estimates.

Mineral Resource estimates include an Indicated Mineral Resource of 225 Mt grading 0.59 g/t for 4.27 Moz and an Inferred Mineral Resource of 22 Mt grading 0.45 g/t for 0.32 Moz, both at 0.20 g/t cut-off.

There do not appear to be any other factors (including environmental, permitting, legal, title, taxation, socio-economic, marketing or political) which could materially affect the mineral resource estimates.

25.1.1 Mineral Processing and Metallurgical Testing

A comprehensive comminution testwork programme has been carried out over all studies consisting of 65 JK Tech SMC, 67 Bond Ball Mill Work Index, 17 Abrasion Work Index and 12 Bond Low Energy Impact tests and 130 leach variability samples using the optimum conditions from the PEA study.

Table 25-1summarizes the comminution testwork results. The predominant fresh mineralization zone is moderately hard in terms of resistance to SAG milling and crushing but soft in terms of resistance to ball milling and has medium abrasivity.

Table 25-1 Comminution Testwork

		JK Tech SMC A x b			Ballmill Work Index		Abrasion Index		Crusher Work Index	
Ore Type	Deposit % Tonnes	No Samples	Relative Density	JK SMC A x b	No Samples	Bond BWi kwhrs/t	No Samples	Bond Ai g	No Samples	Bond CWi kwhrs/t
Fresh	87.4	53	2.75	31.3	54	11.4	10	0.419	11	17.0
Trans	5.5	9	2.69	76.5	9	7.8	4	0.152	1	8.5
Oxide	7.0	3	1.57	*	4	3.9	3	0.115		
Total	100.0	65	2.66	34.0	67	10.7	17	0.383	12	15.4
	* Oxide Data - Off JK Tech Scale									

The metallurgical tests included oxide, transition and fresh mineralization with results indicating that all material types are amenable to direct tank carbon in pulp (CIP) cyanide leaching.

Gravity concentration was evaluated, but not discarded due to the fine gold grain sizes observed.

Forecast gold recoveries were estimated based on predicted residue grades for the feed grade, a solution loss of 0.005 mg/l and carbon fines loss of 0.15%. Table 25-2 estimates the gold recoveries based on the average deposit grades, which are good considering the low head grades. Cyanide consumptions are all low to very low and lime consumptions are low for the predominant fresh zone (88%), but higher for the less dominant transition (5%) and oxide (7%) zones.

Table 25-2 Metallurgical Testwork Summary

# Samples	Domain	Tonnes Processed (x 000)	Total Average Au Recovery,%	kgs/t NaCN	kgs/t CaO
53	**South HW Fresh**	124,107	89.10	0.26	0.55
12	**North HW Fresh**	469	78.13	0.37	0.43
13	**FW Fresh**	17,337	87.65	0.23	0.45
17	**South Transition**	7,894	91.23	0.18	0.99
5	**North Transition**	387	88.06	0.35	0.75
21	**South Oxide**	9,807	93.79	0.18	2.50
9	**North Oxide**	917	94.17	0.13	2.79
130	**LOM**	**160,918**	**89.30**	**0.25**	**0.70**

The good gold recoveries, low reagent consumptions and medium-low resistance to grinding provide favourable processing economics.

25.1.2 Mining

The mining of the Koné deposit has been shown to be technically feasible through conventional open pit mining methods and that it contains economically viable material under the assumed economic and physical parameters. Using available geotechnical information and a series of pit optimizations and mining schedules, the study has shown that the project can support an 11 Mtpa processing plant for a little over 15 years. The estimated crusher feed under this scenario is 161.1 million tonnes at a gold grade of 0.66 g/t. This is comprised entirely of Indicated material.

By implementing an elevated cut-off grade strategy, processing material above 0.5 g/t and stockpiling lower grade material, mining is completed in 10 years, with the remaining low grade material being stockpiled for processing at the end of the mine life, the NPV of the project has been increased considerably.

25.1.3 Processing

The plant design is based on a robust metallurgical flowsheet designed for optimal precious metal recovery. The flowsheet chosen is based on unit operations that are well proven in the industry. The metallurgical testwork conducted to date, has confirmed that the gold contained in the Koné mineralization is amenable to recovery via conventional cyanidation techniques and carbon adsorption.

The key project design criteria for the plant are:

- Nominal throughput of 11.0 Mtpa with a grind size of 80% passing (P80) 75 µm
- Overall process plant availability of 91.3% supported by the selection of standby equipment in critical areas, reputable western vendor supplied equipment and connection to an onsite LNG fired power station
- Sufficient automated plant control to minimize the need for continuous operator interface but allow manual override and control if and when required.

25.1.4 Hydrology

Subject to final approval by government authorities, water will be sourced from the nearby Marahoué river, from pit dewatering and a supplementary borefield. Hydrological assessment of the river catchment indicates that the river will have flow in excess of total water demand for 7 months of the year.

The preliminary numerical groundwater model simulations concluded that pit de-watering will require abstraction in the order of 3,000 to 6,000 m^3/day (35 l/sec to 70 l/sec).

Harvested river water, pit de-watering and supplementary borefield water will be pumped to an off-stream water storage facility (WSF), adjacent to the process plant. Surface runoff from the mining area, ROM pad and stockpiles will gravity flow to this WSF. The WSF will have a capacity of approximately 7.2 Mm^3 and will enable accumulation of water during the wet season and a gradual drawdown in the dry season. In addition, water will be recycled from the tailings storage facility to the process water pond.

The processing, potable and dust suppression water requirements will be in the order of 30,000 m^3/day. The site water balance indicates that sufficient water will be available for the duration of the life of mine with the proposed WSF, river harvesting, pit de-watering and supplementary borefield.

25.1.5 Power

A Build Own Operate Transfer (BOOT) contract is the preferred commercial arrangement for the power station supply and an LNG/Solar Hybrid has been assessed as the preferred power station. The power plant will be operated by KGC after the 5 year contract is concluded.

The Koné Plant is estimated to have a maximum demand of 44.8 MW, an average annual demand of 37 MW with an expected energy consumption of 303 GWhr/yr. The solar farm will generate 22.7 MW of solar energy which will be coupled with 8.8 MW of BESS. Solar generation will provide 16% of the demand.

The capital cost estimate for this LNG/Solar hybrid power station estimated at US$138M with five annual repayments of $20.0M and a transfer payment of $38M in year six. The operating cost is estimated at $0.0998/kWhr. The solar PV and Battery Energy Storage Systems integration is expected to save in the order of 23,000 tonnes/year of CO_2 emissions compared to the stand-alone LNG power plant.

25.1.6 Environment and Permitting

There are currently no objections to the development of the Project. The Project has completed baseline data collection and commenced ESIA write-up, in order to inform environment management plans. This will be an ongoing process, considering that Côte d'Ivoire is the most biodiverse country in West Africa. The Toudian Classified Forest Reserve is a protected forest reserve affected by and adjacent to the Project. To the southwest of Koné is the Yarani Forest Classified Reserve, and to the east is Kani-Bandama Rouge Classified Forest Reserve; neither are directly impacted by the Project footprint. The protection criteria of each of these forests will be assessed during the impact assessment process.

The Project is located relatively close to the communities of Batogo, Fadiadougou and Manabri, but is sufficiently remote that environmental impacts on these communities are likely to be minor. Montage Gold provides support to local communities and exploration geologists engage frequently with the local people. Preliminary investigations indicate that the local community is positive towards the company. The company records all contact with local communities through monthly records, including support provided. An environmental management plan has been developed for exploration work, which is designed to be developed through the life of the Project and used to inform the impact assessment and subsequent Environmental and Social Management Plan (ESMP). The ESMP will include details of the area of impact, objectives to reduce negative or enhance positive impacts, specific targets adopted to achieve those objectives and definition of responsibilities for implementing the programme. Records shall be accurately maintained during exploration to monitor all activities and engagement. All exploration programs will be under the control and responsibility of a designated qualified representative of the company and audited to ensure that requirements are met.

Montage Gold is committed to managing the impacts of its operations, in conformance with recognized international best practice. The Project aims to conform to the environmental and social requirements of the IFC International Finance Corporation Performance Standards, its associated Environmental Health and Safety guidelines, International Council of Metals and Mining and Equator Principles where they are relevant.

26 RECOMMENDATIONS

26.1.1 Geology

Future resource and definition drilling programs at the Koné Gold Project, consistent with Montage's planned work program should reflect the following:

- Koné mineralization is open at depth and along strike and, in the author's opinion, additional drilling is warranted to define the limits of potentially economic mineralization

- A section of the Indicated Mineral Resource should be drilled to a higher confidence level representing the early years of production

- District scale exploration should continue to investigate then potential for satellite deposits that could be trucked to Koné with the aim of enhancing the economics of the project.

26.1.2 Mining

As part of the geotechnical review, SRK Consulting (UK) Ltd recommended that the development of a 3D deposit-scale structural model to assist with the spatial prediction of local/inter-ramp scale structures significant to geotechnical analysis. Given the competent nature of the rock mass, structures could have influence on overall pit slope stability depending on their orientation relative to the slope. Further investigations should consider geophysics data and field mapping to assist with characterising potential fault zones.

26.1.3 Metallurgical Testwork

The sizing and performance guarantees associated with the installation of HPGRs will require further laboratory and pilot scale vendor testing.

26.1.4 Water

A comprehensive water monitoring program will be continued to monitor surface water and groundwater, to provide reference data and to quantify the impact of the mine on local water resources.

To further develop the understanding the mobility of cyanide and other elements following tailings deposition in the pit:

- Evaluate how the cyanide concentration and speciation may change due to mixing with surface water inflows and groundwater

- Calibrate existing models to predict other cyanide decay products.

It is recommended that three piezometers be installed around the South Pit and two around the North Pit and equipped with continuous groundwater level sensors. The data are essential to calibrate the numerical groundwater model. It is recommended that the pit-de-watering model be updated on an annual basis.

26.1.5 Tailings Storage Facilities and Water Management

To advance the design to the next phase of study the following activities are recommended to be included in the scope of the definitive feasibility study:

- Expanding topography to include areas potentially impacted by a dam break
- Sterilization of infrastructure footprints
- Site inspection visit by KP project manager, COVID-19 permitting
- Update of the design based on the findings of the above investigations.

26.1.6 Electric Power Supply

Further options for the LNG supply chain, including gas storage are to be explored in the next phase.

26.1.7 Environmental

By initiating the impact assessment process early, results have been used to improve the design, increasing the benefits of the study without incurring excessive costs. To support the Environmental and Social Management Plan, the following activities are recommended to continue:

- Ongoing monitoring of wildlife presence in the Project area
- Monitoring of impacts on each of the classified forest reserves
- Recording of community engagement, including information sharing as well as support initiatives and infrastructure development
- Maintaining a grievance procedure to identify and pre-empt potential issues.

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28 QP CERTIFICATES

I **Jonathon Robert Abbott** hereby state:

1. I am a Consulting Geologist, with the firm of MPR Geological Consultants Pty Ltd, 19/123A Colin Street, West Perth, WA 6005, Australia.
2. This certificate applies to the technical report with an effective date of 14th February, 2022, titled "Koné Gold Project, Côte d'Ivoire Definitive Feasibility Study".
3. I am a practising a practising Geologist and registered Member of the Australian Institute of Geoscientists.
4. I graduated with a Bachelor of Applied Science in Applied Geology from the University of South Australia in 1990. I am a member of the Australian Institute of Geoscientists. I have worked as a geologist for a total of 30 years since my graduation from university. My experience includes mine geology and resource estimation for a range of commodities and mineralization styles. I have been involved in preparation and reporting of resource estimates in accordance with JORC guidelines for 25 years, and NI43 101 guidelines for approximately 17 years.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I have been involved with the Morondo Gold Project since July 2018 and visited the project site on the 23rd and 24th August 2018.
7. I am responsible for Sections 1.7, 12 and 14 of the Technical Report.
8. I am independent of the Issuer pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Montage or any associate or affiliate of such company.
10. I have had prior involvement with the Morondo Gold Project. Between August and November 2018, I prepared Mineral Resource estimates for Orca Gold and authored a Technical Report titled "Mineral Resource Estimation for the Koné gold deposit Morondo Gold Project Côte d'Ivoire NI 43-101 Technical Report with an effective date of the 3rd of October 2018. During August and September 2019, I was co-author of an updated Technical Report titled "NI 43-101 Technical Report for the Morondo Gold Project, Côte d'Ivoire" with an effective date of the 17th of September 2020. In January 2021 I was co-author of a Technical Report titled "NI 43-101 Technical Report for the Morondo Gold Project, Côte d'Ivoire" with an effective date of the 27th of January 2021. In June 2021 I was co-author of a Technical Report titled "Preliminary economic assessment for the Koné gold deposit, Côte d'Ivoire" with an effective date of the 25th of May 2021.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th day of March 2022

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Jonathon Robert Abbott

I, **Sandra (Sandy) Hunter,** hereby state:

1. I am employed as a Principal Process Engineer, with the firm Lycopodium Minerals Pty Ltd, Level 2, 60 Leichhardt Street, Spring Hill, Queensland 4000 Australia.
2. This certificate applies to the technical report with an effective date of 14th February, 2022, titled "Definitive Feasibility Study, Koné Gold Project, Côte d'Ivoire".
3. I am a practising Process Engineer and registered Chartered Professional (Metallurgy) and Fellow of the Australian Institute of Mining and Metallurgy.
4. I am a graduate of Murdoch University with a Bachelor of Science with Honours in Mineral Science (Extractive Metallurgy) 2001. I have worked as a metallurgist, metallurgy manager and process engineer continuously since 1996. My relevant experience includes process engineer for studies and projects in numerous commodities around the world, with particular focus on gold projects for detailed design and commissioning in Africa and Asia Pacific. Additional relevant experience includes my operational roles to the level of metallurgy manager for gold processing plants in Australia.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I have not visited site.
7. I am responsible for sections 1.10, 1.11.2, 1.14, 17, 18.2, 21, 22 (overview), 25.1.3, 26.1.3 and 26.1.6.
8. I am independent of the Issuer pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Montage or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI 43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th day of March 2022

Sandra Hunter

AusIMM
THE MINERALS INSTITUTE
CHARTERED PROFESSIONAL
METALLURGY
Sandy Hunter

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Sandra (Sandy) Hunter, FAusIMM(CP)

I **Michael Peter Hallewell** hereby state:-

1. I am a consulting Metallurgist, with UK registered company named MPH Minerals Consultancy Ltd, 8 The Gluyas, Falmouth, Cornwall, TR11 4SE.
2. This certificate applies to the technical report with an effective date of 14th March, 2022, and titled "Definitive Feasibility Study, Koné Gold Project, Côte d'Ivoire".
3. I am a practising Metallurgical Consultant and a Fellow of the South African Institute of Mining & Metallurgy (RSA), a Fellow of the Institute of Materials, Minerals and Mining (London, UK) and a Chartered Engineer.
4. I am a graduate with a B.Sc (Engineering) degree in Minerals Engineering from the University of Birmingham, UK.
5. I am a Consulting Metallurgist with 41 years practical experience in Minerals Processing as Plant Manager, Consulting or Senior Metallurgist in precious metals, base metals and ferrous metals industry. I work with Mining Companies and am actively involved in the flowsheet development, design and optimisation for greenfield and brownfield projects.
6. I have read the definition of "Qualified Person" (QP) set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
7. I have not visited the project site.
8. I am responsible for Sections 1.6, 1.16.4, 13, 25.1.1 and 26.1.3 of the report.
9. I am independent of the issuer as described in section 1.5 of NI 43-101.
10. I do not beneficially own, directly or indirectly, any securities of Montage or any associate or affiliate of such company.
11. I have had prior involvement with the Kone Gold Project as QP for the Technical Report titled "Preliminary Economic Assessment of the Koné Gold deposit" with an effective date of 25th May 2021.
12. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
13. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th day of March 2022

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Michael Peter Hallewell

I **Pieter Ferdinandus Labuschagne** hereby state:-

1. I am a consulting Hydrogeologist, with the firm of AGE (Pty) Ltd and situated in 15 Mallon Street, Bowen Hills, Queensland 4006 Australia.
2. This certificate applies to the technical report with an effective date of 14th February, 2022 and titled "Definitive Feasibility Study, Koné Gold Project, Côte d'Ivoire".
3. I am a practising Hydrogeologist and registered Member of the South African Council for Natural Scientific Professions – SACNASP (Pr.Sci.Nat.400386/11).
4. I am a graduate of the University of the Free State, Bloemfontein, South Africa with a Master's of Science degree in Hydrogeology (2004). I have practiced my profession continuously since 1998 and completed more than 50 mining related hydrogeological studies.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I Have not visited the Project site.
7. I am responsible for sections 1.11.1, 1.16.5 (part), 18.1, 25.1.4 and 26.1.4.
8. I am independent of the Issuer pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Montage or any associate or affiliate of such company.
10. I have had prior involvement with the Kone Gold Project as QP for the Technical Report titled "Preliminary Economic Assessment of the Koné Gold deposit" with an effective date of 25th May 2021.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th day of March 2022

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Pieter Ferdinandus Labuschagne

I **Carl Steven Nicholas** hereby state:-

1. I am a Chartered Environmental Consultant, with the company of Mineesia Limited, 4 Mace Farm, Cudham, Kent, TN14 7QN, UK.
2. This certificate applies to the technical report with an effective date of 14th February, 2022, titled "Definitive Feasibility Study, Koné Gold Project, Côte d'Ivoire".
3. I am a practising Environmental Consultant and registered Member of the Institute of Materials, Minerals and Mining.
4. I am a graduate of Imperial College, London, UK with a Masters in Environmental Diagnosis, with a Bachelor of Science (Honours) degree in Biodiversity Conservation and Environmental Management. I have practiced my profession continuously since 2005, and have 12 years practical experience in Environmental Impact Assessments for mining projects.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I visited the Koné Gold Project between 13th March and 18th March 2021. The purpose of the visit was to review the baseline conditions and establish priorities for environmental management for the project.
7. I am responsible for sections 1.13, 1.16.2, 20, 25.1.6 and 26.1.7.
8. I am independent of Issuer pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Montage or any associate or affiliate of such company.
10. I have had prior involvement with the Koné Gold Project as QP for the Technical Report titled "Preliminary Economic Assessment of the Koné Gold deposit" with an effective date of 25th May 2021.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th day of March 2022

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Carl Steven Nicholas

I **Joeline McGrath** hereby state:-

1. I am a Chartered Mining Engineer and Principal Consultant with Carci Mining Consultants, 21-23 Croyden Road, Caterham, Surrey CR3 6PA, England.
2. This certificate applies to the technical report with an effective date of 14th February, 2022, and titled "Definitive Feasibility Study, Koné Gold Project, Côte d'Ivoire".
3. I am a practising Mining Engineer with over 15 years of relevant experience in open pit mining operations, 15 of which have been in open pit gold mines. I have over 20 years mining engineering experience spanning gold mines both underground and open pit. The initial 6 years being within Australia and the following 14 years within various overseas countries. In 2014 I was involved in the commissioning a new gold mine in Cote D'ivoire which included designing and building site layouts and infrastructure, open pit and waste dump designs and mining, community relations in terms of interactions between the local landholders at the mine site boundaries.
4. I am a graduate of the Curtin University of Technology, Australia with a Bachelor of Engineering degree in Mining Engineering. I have practised my profession continuously since 2002.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I visited the Project site between 17th to 19th November, 2021.
7. I am responsible for sections 1.8, 1.9, 1.16.3, 15, 16, 21.2.1, 21.3.1, 25.1.2 and 26.1.1.
8. I am independent of the issuer pursuant to Section 1.5 of NI 43-101.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th day of March 2022

Jo McGrath

I **Timothy Rowles** hereby state:-

1. I am employed as Regional Manager, with the firm Knight Piésold Pty Limited, Level 1, 36 Cordelia Street, Brisbane, QLD 4101, AUSTRALIA.
2. This certificate applies to the technical report with an effective date of 14th February, 2022, titled "Definitive Feasibility Study, Koné Gold Project, Côte d'Ivoire".
3. I am a current Fellow of the Australian Institute of Mining and Metallurgy (No 227249) and have been awarded the status of Chartered Professional (CP) in the field of Environmental Engineering.
4. I am a Registered Professional Engineer of Queensland (RPEQ) and in good standing with the Board of Professional Engineers of Queensland, Australia (No 10166).
5. I am a current member of the Australian Institute of Geoscientists (No. 8161) and a current member of the Australian National Committee on Large Dams.
6. I graduated from the Royal School of Mines, Imperial College, London with a Bachelor of Science in Environmental Geology in 1996 and from the University of Manchester with a Masters Degree in Earth and Environmental Science in 1998. I was awarded a Professional Certificate in Tailings Management by AusIMM in 2021.
7. I have practiced my profession continuously within the mining industry since 1999 and have been responsible for the design, construction, operation and closure of tailings management systems and water dams throughout that time. I have experience in the design of waste dumps, surface water / sediment management systems, geotechnical & hydrogeological investigation and geochemical characterisation of mine waste and tailings. This experience includes mine sites in Australasia, Asia, Europe, South America and Africa, with specific West African experience having previously worked on projects in Côte d'Ivoire, Ghana, Republic of Guinea, DRC, Mali, Burkina Faso and Cameroon.
8. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
9. I have not visited the project site.
10. I am responsible for sections 1.11.3, 1.16.5 (part), 18.1.7, 18.1.9, 18.1.10, 18.1.11, 18.3, 18.4 and 26.1.5.
11. I am independent of the Issuer pursuant to Section 1.5 of NI 43-101.
12. I do not beneficially own, directly or indirectly, any securities of Montage or any associate or affiliate of such company.
13. I have had prior involvement with the Koné Gold Project. I provided technical review of sections of a Technical Report titled "Preliminary Economic Assessment of the Koné Gold deposit" with an effective date of 25th May 2021.
14. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
15. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th day of March 2022

Timothy Rowles