Montage Gold

Koné Gold Project, Côte D'Ivoire

Updated Feasibility Study

National Instrument 43-101 Technical Report

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The author has undertaken an extensive review of Montage Gold's Koné Gold Project, incorporating the Koné and Gbongogo Main ('Gbongogo') open pits, technical and economic data and has reviewed Montage Gold's contributions to this report.

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Signed, this 15 February, 2024.

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1.0 SUMMARY

1.1 Introduction

This independent Technical Report comprises an Updated Feasibility Study ('UFS') for Montage Gold Corp.'s ('Montage' or 'Company') Koné Gold Project incorporating the Koné and Gbongogo Main ('Gbongogo') open pits ('KGP' or 'Project') in Côte d'Ivoire. The UFS has been prepared by Lycopodium Minerals Pty Ltd ('Lycopodium') 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 ('KGP') covers 1,801 square kilometres (km^2) in northwest Côte d'Ivoire, 470 kilometres (km) northwest of the commercial capital Abidjan.

The Project comprises six exploration permits (PR's 262,748, 842, 879b, 919 and 920 748) covering 1,801 km^2 and two exploration permit applications covering a further 458 km^2.

The Project area straddles the Departments of Kani, Dianra and Boundiali in the Worodougou and Savanes regions of Côte d'Ivoire. The communities of Fadiadougou, Batogo and Manabri are located 4 to 6 km from the proposed Koné plant site, and the village of Gbongogo is located 3 km from the proposed Gbongogo satellite open pit.

A part of the Toudian Forest Reserve lies 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 minimises incursion in the forest area. Discussions with the Ministry of Water and Forests have commenced to obtain authorisation and replacement planting will be undertaken as part of future programmes.

1.2.2 Ownership

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 ('Kinross') 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 (TSX) in October 2020.

A summary of the exploration permits that comprise the KGP is shown below in Table 1.2.1. The exploration permits are held by three wholly owned subsidiaries of Montage: Shark Mining CDI SARL ('Shark Mining'), Orca Gold CDI SARL and Mankono Exploration Ltd. ('Mankono'). The key permits are the Koné Exploration Permit that hosts the Koné Mineral Resource and Reserve, and the Gbongogo Exploration Permit that hosts the Gbongogo Mineral Resource and Reserve.

Table 1.2.1 KGP Exploration Permits

Permit	Company	Area	Decree No	Permit No	Decree Date	Status
Koné	Shark	225.00	2013-198	262	22/03/2013	Exceptional Renewal ends 22/03/2024
Gbongogo	Mankono	399.97	2022-686	919	06/09/2022	In initial period
Sissédougou	Mankono	387.00	2019-681	842	24/07/2019	In first renewal period
Sisséplé	Mankono	105.85	2022-687	920	06/09/2022	In initial period
Farandougou	Orca	361.51	2021-306	748	16/06/2021	In initial period
Sisséplé North	Orca	321.60	2022-027	879b	12/01/2022	In initial period

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, in 2013.

In March 2016 and March 2019, the Koné Exploration Permit was renewed for three years and in March 2022 was granted an exceptional renewal for a period of two 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.

In 2021 and 2022, Orca was issued the Farandougou and Sisséplé North exploration permits. In November 2022, Montage completed the acquisition of Mankono from Barrick Gold Corporation ('Barrick') and Endeavour Mining ('Endeavour'). Mankono holds the Gbongogo, Sissédougou and Sisséplé Exploration Permits.

Montage is currently in the process of transferring the Gbongogo permit from Mankono Exploration Permit to Shark Mining to facilitate the issue of the mining permits to a single entity.

In addition, the Company holds two exploration permit applications covering a further 458 km^2.

Under the terms of the exploration permits, 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 during the exploration phase.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The KGP is located 470 km northwest of Abidjan and is accessible by an established network of asphalt roads from the capital. An asphalt road, linking the area to the nearest major centre at Séguéla, 80 km to the south and the town of Boundiali to the north, runs through the Koné Exploration permit. The communities of Fadiadougou, Batogo and Manabri lie on the asphalt road close to the proposed Koné Plant site, and there are numerous small villages within the wider project area connected by a network of dirt roads.

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 degrees Celsius (˚C).

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

The Project area is characterised by moderate relief, lying between 200 m and 420 m above sea level (mASL). 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 that are only suitable for grazing.

1.4 Geology and Mineralisation

The KGP lies within the Birimian Baoulé-Mossi domain, which in the Project area comprises metamorphosed intrusives, sediments, volcaniclastics and volcanics flanked to the west by basement tonalite and diorites. Much of the Project area is covered by duricrust and lateritic soils with only rare outcrop and deep weathering.

In the area of the Koné Mineral Resource, local stratigraphy comprises a moderately westerly dipping sequence of mafic volcanics, which are intruded by an approximately 250 metre (m) thick package of quartz diorites. Gold mineralisation 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.

The Gbongogo gold deposit is a lithologically constrained deposit, hosted within a single, north plunging (50 degrees (°)), quartz diorite intrusion which is an approximately elliptical cylinder. The quartz diorite is intruded along the westerly dipping contact of two stratigraphic packages, the fine grained, volcanic to volcaniclastic hanging wall package, and the medium to coarse grained siliciclastic and volcanic sediments of the footwall group. Gold mineralisation in the Gbongogo quartz diorite is associated with a stockwork of quartz / tourmaline / quartz-tourmaline veins, and their associated orthoclase-albite alteration haloes. The deposit is believed to controlled by the rheological contrast of the quartz diorite host and surrounding volcaniclastics, where accommodation space for mineralisation formed by brittle deformation of the quartz diorite.

1.5 Exploration and Resource Definition

During 2009, an 800 metres (m) x 50 m spaced soil sampling and subsequent local infill to 400 m x 50 m and 200 m x 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 2021, the Koné mineralization has been tested by 102,249 m of drilling (54,703 m of core and 45,545 m of RC). Drilling has been based on 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. This drilling was used as the basis for the Mineral Resource Estimate (MRE) undertaken in August 2021. The December 2023 MRE is reported within an optimal pit shell generated at a gold price of US$1,800 /oz and updated input costs in line with this report.

Between 2016 and 2021, under the supervision of Barick, the Gbongogo mineralisation was tested by 6,022 m of drilling (4,827 m of core and 1,195 m of reverse circulation (drilling) (RC). Following acquisition of the Project, Montage completed a further 12,254 m of drilling (7,330 m of core and 4,924 m of RC) between November 2022 and July 2023. Drilling has been based on 50 m spaced traverses of generally 50 m and rarely 25 m spaced holes extending to vertical depths of between 50 m and 250 m. This drilling was used as the basis for the MRE undertaken in September 2023. The December 2023 MRE is reported within an optimal pit shell generated at a gold price of US$1,800 /oz and updated input costs in line with this report.

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 representivity of the Koné and Gbongogo RC and diamond drilling includes RC sample condition logs, recovered RC sample weights, and core recovery measurements. Geological logging and storage of sample material along with documentation of analytical results is consistent with good industry standard practise.

The reliability of sample preparation and assaying include results for coarse blanks and reference standards, along with inter-laboratory repeat and duplicate assays.

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

The sample preparation, security and analytical procedures adopted for the 2010 to 2023 drilling programmes provide an adequate basis for the MRE and exploration activities.

1.6 Metallurgical Testing

A comprehensive comminution testwork programme has been carried out consisting of 68 JK Tech SMC, 68 Bond Ball Mill Work Index, 17 Abrasion Work Index and 14 Bond Low Energy Impact tests and 146 leach variability samples using the optimum conditions identified in the PEA study.

Table 1.6.1 summarises the comminution testwork results. The predominant fresh mineralisation zone is moderately hard in terms of resistance to semi-autogenous grinding (SAG) milling and crushing but soft in terms of resistance to ball milling and has medium abrasivity. One Koné fresh ore sample was used for a full suite of high pressure grinding roll (HPGR) testing, with a further two footwall fresh ore samples submitted for static pressure testing (SPT). The results (bulk sample, 13.2 kilowatt hours per tonne (kwh/t), SPT samples, 13.4 and 14.5 kwh/t) show that the fresh ore types tested are 'medium' in terms of their high pressure grinding index values.

Table 1.6.1 Comminution Testwork

Ore Type	Deposit %	JK Tech SMC A x b			Ball Mill Work Index		Abrasion Index		Crusher Work Index	
		No. of Samples	Relative Density	JK SMC Axb	No. of Samples	Bond BWi kWh/t	No. of Samples	Bond Ai g	No. of Samples	Bond CWi kWh/t
Fresh	83%	52	2.74	31.9	52	11.8	11	0.5	13	16.4
FW Fresh	7%	3	2.77	31.1	3	9.7	-	-	-	-
Trans	4%	9	2.69	76.5	9	7.8	4	0.2	1	8.5
Oxide	6%	4	2.54	488.9	4	3.9	2	0.2	-	-
Total	**100%**	**68**	**2.68**	**59.6**	**68**	**11.0**	**17**	**0.4**	**14**	**14.0**

The metallurgical tests included oxide, transition and fresh mineralisation with results indicating that all material types are amenable to direct tank carbon-in-pulp (CIP) cyanide leaching.

Gravity concentration was evaluated but not incorporated in the flowsheet due to the fine gold grain sizes observed.

Forecast gold recoveries have been based on predicted residue grades for the head grade with allowances for a solution loss of 0.005 milligrams per litre (mg/L) and carbon fines loss of 0.15%. Table 1.6.2 shows the estimated gold recoveries by pit and domain based on the average deposit grades. Cyanide consumptions are all low to very low and lime consumptions are low for the predominant fresh zone (89%), but relatively higher for the less dominant transition (5%) and oxide (6%) zones.

Table 1.6.2 Metallurgical Testwork Summary

No. Samples	Deposit	Domain	Processed '000 t	Processed Au g/t	Au Recovery %	kg/t NaCN	kg/t CaO
53	South	Fresh	129,510	0.69	89.1	0.22	0.47
13	South	FW Fresh	15,776	0.58	87.7	0.37	0.43
12	North	Fresh	416	0.51	77.1	0.23	0.45
8	GB	Fresh	9,427	1.46	86.1	0.42	0.55
17	South	Transition	6,957	0.60	91.3	0.18	0.99
5	North	Transition	425	0.44	88.0	0.35	0.75
4	GB	Transition	523	1.09	91.2	0.21	1.06
21	South	Oxide	9,628	0.59	93.9	0.18	2.50
9	North	Oxide	943	0.47	93.2	0.13	2.79
4	GB	Oxide	742	1.36	92.8	0.29	2.60
146	**Total**	**LOM**	**174,345**	**0.72**	**89.0**	**0.24**	**0.62**

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

1.7 Mineral Resource Estimate

Matix Resource Consultants ('Matix') estimated Mineral Resources for KGP based on the basis of RC and diamond drilling data supplied by Montage for the Koné deposit in August 2021 and for the Gbongogo deposit in August 2023.

Estimates for the Koné deposit tested by drilling spaced at around 50 m x 50 m are classified as Indicated, with Inferred estimates based on generally 100 m spaced drilling. The Gbongogo Mineral Resource is classified as Indicated, and is based on 50 m spaced drilling.

Recoverable resources were estimated for both deposits by Multiple Indicator Kriging (MIK) of two metre downhole composited gold grades from RC and diamond drilling. Estimated resources include a variance adjustment to provide estimates of recoverable resources above gold cut-off grades for selective mining unit dimensions of 5 m x 10 m x 5 m (east, north, vertical) and are reported within optimal pit shells generated at a gold price of US$1,800 per ounce (/oz) using cost inputs in line with the UFS. The estimates have an effective date of 19 December 2023.

The MRE has been classified and reported in accordance with NI 43-101 and classifications adopted by CIM Council in May 2014.

Table 1.7.1 shows the combined MRE based on the base case cut off grades for each deposit. The figures in this table are rounded to reflect the precision of the estimates and include rounding errors. Mineral Resources that are not Mineral Reserves do not necessarily demonstrate economic viability. The Indicated Mineral Resources are inclusive of Mineral Reserves. The Inferred Mineral Resources are additional to Mineral Reserves.

Table 1.7.1 Combined Indicated and Inferred Resources (December 2023)

Deposit	Cut-off	Indicated			Inferred		
	Au g/t	Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz
Koné	0.2	229	0,59	4.34	25	0.5	0.4
Gbongogo	0.5	11	1.48	0.52	-	-	-
Total		**240**	**0.63**	**4.87**	**25**	**0.5**	**0.4**

1.8 Mineral Reserve Estimate

The Mineral Reserve estimate was undertaken by Carci Mining Consultants Ltd ('Carci') using Deswik mine planning software and demonstrated that mining of the deposits is practical and economically viable. 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.

The MRE is shown in Table 1.8.1, which is based on the December 2023 MRE. The figures in this table are rounded to reflect the precision of the estimates and include rounding errors.

Table 1.8.1 Summary of KGP Mineral Reserves

Description	Classification	Oxide			Transitional			Fresh			Total		
		Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz
South Pit	Probable	9.6	0.59	0.18	7.0	0.60	0.13	145.3	0.68	3.18	161.9	0.67	3.49
North Pit	Probable	0.9	0.47	0.01	0.4	0.44	0.01	0.4	0.51	0.01	1.8	0.47	0.03
Gbongogo	Probable	0.7	1.36	0.03	0.5	1.09	0.02	9.4	1.46	0.44	10.7	1.43	0.49
Total	**Probable**	**11.3**	**0.63**	**0.23**	**7.9**	**0.63**	**0.16**	**155.1**	**0.73**	**3.62**	**174.3**	**0.72**	**4.01**

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 ('SRK'). Pit optimizations were completed based on slope angle recommendations from SRK:

- For Koné South 48° for oxide, 68° for transition and 68° for fresh rock; the overall slope angle inclusive of ramps and berms is approximately 55°.
- For Gbongogo 32° for oxide, 40° for transition, and from 43° to 55° for fresh rock, which is reduced in areas of unfavourable bedding to 35°; the overall slope angle inclusive of ramps and berms is approximately 43°.

Pit optimizations were run using processing cost and recovery data at a gold price of US$1,550 /oz. Mining costs were categorised by 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 90 t rigid body haul trucks with suitably sized loading units.

The Koné deposit will be exploited through two pits - a smaller North Pit, which reaches a depth of 70 m and a larger South Pit, which extends to a depth of 495 m. The strip ratio for the North Pit is 1.19:1 and 1.01:1 for the South Pit. Based on the assumed mining equipment, a bench height of 5 m in the oxide, 10 m in the transition, and 10 m in the fresh rock have been designed, although geotechnical conditions allow for up to three 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.

The Gbongogo deposit will be exploited through a single pit, which extends to a depth of 220 m. The overall strip ratio is 3.77:1. Based on the assumed mining equipment, a bench height of 5 m in the oxide, 10 m in the transition, and 10 m in the fresh rock was designed.

A ramp up period of 12 months has been included in the pre-strip period. The scheduled mining rates are 39 million tonnes per annum (Mtpa) and 15.5 Mtpa at Koné and Gbongogo respectively. The ore target for Year 1 was 9.5 million tonnes (Mt), with all subsequent years targeting 11.0 Mtpa. Mining dilution and recovery were not included in the schedule, as these had been included in the MRE.

Table 1.9.1 shows the mine production by pit with the extraction commencing with a pre-strip year followed by eight years of operations.

Table 1.9.1 Mine Production Schedule

Description	Unit	LOM Total
GB Pit Tonnes	Mt	10.7
GB Grade	Au g/t	1.43
North Pit Tonnes	Mt	1.8
North Grade	Au g/t	0.47
South Pit Tonnes	Mt	161.9
South Grade	Au g/t	0.67
Total Tonnes	**Mt**	**174.3**
Total Grade	**Au g/t**	**0.72**
GB Waste Tonnes	Mt	40.3
North Pit Waste Tonnes	Mt	2.1
South Pit Waste Tonnes	Mt	162.8
Total Waste Tonnes	**Mt**	**205.3**
Total Strip Ratio	**W:O**	**1.18**

Table 1.9.2 shows the processing schedule with the highest grade ore processed first through the use of an elevated cut-off grade which creates low grade stockpiles for processing in the later years of the life of mine (LOM). Tailings will initially be disposed in the tailings storage facility (TSF) until the completion of the South Pit mining when it will be used to deposit the remaining stockpiled lower grade ore processed.

Table 1.9.2 Mine Processing Schedule

	Unit	LOM Total	Year 1 to 8	Year 8 to 16
Tails Deposit Location			TSF	In Pit
Stockpile Rehandle	Mt	108.9	26.3	82.6
Oxide Tonnes	Mt	11.3	5.9	5.4
Oxide Grade	Au g/t	0.63	0.85	0.39
Transition Tonnes	Mt	7.9	3.2	4.7
Transition Grade	Au g/t	0.63	0.95	0.40
FW Fresh Tonnes	Mt	15.8	8.8	7.0
FW Fresh Grade	Au g/t	0.58	0.68	0.46
Fresh Tonnes	Mt	139.3	73.0	66.4
Fresh Grade	Au g/t	0.74	0.99	0.48
Total Processed Tonnes	**Mt**	**174.3**	**90.9**	**83.5**
Total Processed Grade	**Au g/t**	**0.72**	**0.95**	**0.46**
Total Process Recovery	**%**	**89.0%**	**90.0%**	**86.8%**
Total Recovered	**'000 oz**	**3,570**	**2,489**	**1,081**

1.10 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 mineralisation 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 (P_{80}) 75 microns (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 the National Grid.

- Sufficient automated plant control to minimise 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 P_{80} of approximately 31 mm. 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 (wt), 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 grind size of P_{80} 75 µm.

- Pre-leach thickening to increase the slurry density feeding the leach and CIP circuit to minimise 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 (h) leach residence time.

- A Kemix Pumpcell CIP circuit consisting of eight CIP tanks for recovery of gold onto carbon, to minimise carbon inventory, gold in circuit, and operating costs. The CIP and elution circuit design is based on daily carbon harvesting.

- 20 t split Anglo American Research Laboratories (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 TSF.

1.11 Project Infrastructure

1.11.1 Gbongogo Haul Road

The Gbongogo haul road is 38.1 km in length and transverses a sparsely populated area between the two sites and has been designed to avoid villages, defined forest areas and minimise interactions with existing public roads. The road incorporates a pedestrian corridor leading to underpasses along the alignment. Access to the road will be restricted by construction of safety berms along the entire length of the road. Traffic control will be provided at all intersections with the public roads.

The haul road alignment has been designed to limit the number of water courses impacted by the road with culverts provided at all main water intersections, and a bridge to be constructed at the crossing of the Marahoué River.

1.11.2 Water

Water supply will be sourced from the nearby Marahoué River and supplemented by pit dewatering and rainfall harvesting. Hydrological assessment of the river catchment indicates that the river will have flow in excess of total water demand for seven months of the year, when pumping will take place provided sufficient flow conditions are met.

Both the Koné and Gbongogo sites are underlain by a low yielding aquifer system with an overall average groundwater piezometric level of 20 metres below ground level (mbgl). Overall, groundwater monitoring data indicated a fairly flat groundwater table within the pit areas.

Nineteen and four hydrogeological test boreholes were drilled and tested to determine the aquifer characteristics at the Koné and Gbongogo Pits respectively. Aquifer pump tests were conducted and interpreted to derive aquifer parameters for three aquifer systems, including a shallow low permeable saprolite system, a higher permeable transition system at the base of the saprolite, and a low permeable fresh rock unit. The aquifer parameters obtained suggest overall low aquifer transmissivity with higher transmissivity associated with fracturing along geological structures and along the transition zone. In addition, data was also sourced from exploration, geotechnical and regional groundwater supply boreholes.

The numerical groundwater model simulations concluded that:

- Koné pit de-watering will require abstraction in the order of 3,000 to 6,000 cubic metres per day (m^3/d) (35 litres per second (L/s) to 70 L/s). 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.

- Gbongogo Pit dewatering will peak at around 3,000 m^3/d (35 L/s) and used at Gbongogo for small-scale use and dust suppression; the rest will be discharged back into the Marahoué River catchment.

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 and pit dewatering will be pumped to a water storage facility (WSF). Surface run-off from the mining area, ROM pad and stockpiles will gravity flow to this WSF. The WSF will have a capacity of approximately 6.4 million cubic metres (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 TSF to the process water pond. Dewatering of the Koné pit will start 2.5 years prior to production to maximise the WSF stored volume prior to the commencement of processing.

The processing, potable and dust suppression water requirements will be in the order of 30,000 m^3/d. The site water balance indicates that sufficient water will be available for the duration of the LOM with the proposed WSF, river harvesting rainfall run-off and pit dewatering.

1.11.3 Power Supply

The earlier definitive feasibility study (DFS) evaluated hybrid power supply options. However, local supplies of LNG cannot be guaranteed and so power will be supplied from the national grid via a new 225 kilovolt (kV) transmission line.

The processing plant is estimated to have a maximum demand of 45 megawatt (MW), an average annual demand of 38 MW and an expected annual electricity energy consumption of 305 gigawatt-hours per year (GWh/y).

1.11.4 Tailings Storage Facility

The tailings management arrangement comprises a single 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 mining in the South Pit is completed (after Year 8). Tailings will be deposited in the South Pit for the final eight years of processing.

The TSF basin will be lined with high density polyethylene (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 8, 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 is transferred to the 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 will be gold / silver doré bars, which will be transported 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 Environment and Permitting

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 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 Assessment (ESIA) 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. 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 adherence to good governance principles, including the Equator principles and the EITI principles. Mining titleholders must issue EITI reports.

The Project has completed baseline data collection, to inform environment management plans. There are protected forest reserves affected and adjacent to the Project, which have been assessed in the ESIA report submitted to ANDE. The Project is located relatively close to the communities of Batogo, Fadiadougou, Manabri, and Gbongogo. There are currently no objections to the development of the Project, and recent engagement indicates that these communities are generally positive towards the Company.

Montage is committed to managing the impacts of its operations in conformance with recognised international best practice. The Company has completed the impact assessment process, including submitting terms of reference for the impact assessment, completing baseline studies, and assessing potential impacts arising from the Project. Results of the assessment were used to avoid impacts where possible and improve the design, as well as maximise the benefits without incurring excessive costs. In accordance with continual improvement processes, there are several strategies that have been 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.
- Maintaining a grievance procedure to identify and pre-empt potential issues.

Mining of the Koné North Pit will affect less than 9% of the Toudian Forest Reserve, and discussions with the Ministry of Water and Forests have commenced to obtain authorisation. The impact of the Project on the forest reserve has been assessed during the ESIA, and the UFS includes provision for the backfilling and rehabilitation of all but 14 hectares (ha) during operations. This will be complemented by a forest management plan, in conjunction with the relevant government agency, to upgrade and protect the forest reserve.

The development of the Project will be subject to the following permitting process:

1. Approval of the ESIA by ANDE.

2. Receipt of environmental approval of its design and environmental management program.

3. Application for and receipt of a Mining Permit (valid for 10 years and then renewable).

4. Negotiation of a Mining Convention.

ANDE hosted a public enquiry in late December 2023 and expects environmental validation of the Project to be completed in the first quarter of 2024 (1Q24). In parallel with this process, Montage is preparing the Mining Permit application and Mining Convention with assistance from local advisors. Based on current expectations, Montage believes it will be possible to receive final permits and approvals in 3Q24.

1.14 Capital and Operating Costs

1.14.1 Capital Cost

The initial capital cost is estimated at US$712.1M, including a contingency allowance of US$65.3M, and is summarised in Table 1.14.1.

Table 1.14.1 Captital Estimate Summary (4Q23, +15%/-10%)

Main Area	Koné US$M	Gbongogo US$M	Total US$M
Resettlement	7.4	2.0	9.4
Camp	6.4		6.4
Pre-Production Mining	45.2	11.9	57.1
Gbongogo Haul Road		27.4	27.4
Gbongogo Surface Water		3.3	3.3
Grid Connection	26.1		26.1
Process Plant	338.4		338.4
Infrastructure	26.5		26.5
TSF	41.4		41.4
WSF	13.6		13.6
EPCM	46.4		46.4
Owners Costs	49.3	1.4	50.7
Subtotal	**600.8**	**46.0**	**646.8**
Contingency	61.3	4.0	65.3
Grand Total	**662.1**	**50.0**	**712.1**

The duration of the detailed design and construction phase of the Project has been estimated to be 31 months, commencing with the construction of the Marahoué haul road, bridge and pump station and the WSF to ensure sufficient water is available for processing. It is estimated that the plant will take 27 months to construct. The Mining contractor will mobilise 12 months prior to the start of processing.

The total LOM capital cost is estimated at US$878.0M, including sustaining capital costs of US$165.3M, as shown in Table 1.14.2.

Table 1.14.2 Sustaining Capital Estimate Summary (4Q23, +15%/-10%))

Main Area	US$M
Camp	4.4
TSF	65.0
Process Plant	34.4
Closure	61.6
Grand Total	**165.3**

1.14.2 Operating Cost Mining

Contract open pit mining costs were derived from a tender process involving several West African mining contractors who were provided with a detailed mining plan. The average unit operating costs are shown in Table 1.14.3.

Table 1.14.3 Mining Costs

	Ore US$ /t	Waste US$ /t	Total US$ /t
Total	3.49	2.86	3.22

A diesel price of US$1.00 per litre (/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 run-of-mine (ROM) feed into the primary crusher (ROM loader by Mining), production of doré in the goldroom 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 4Q23. Process operating costs have been developed for each major domain. Operating costs were developed using the plant parameters specified in the process design criteria. A National Grid connection will supply the power at a cost of US$0.1145 /kWh. Table 1.14.4 presents the operating cost summary.

Table 1.14.4 Process Operating Cost (4Q23, ±15%)

Cost Centre	Fixed US$M /y	Variable Processing Costs US$ /t			LOM
		Oxide	Transition	Fresh	Fix & Var US$ /t
Total	19.3	5.42	5.65	6.71	8.35

G&A costs have been estimated at US$12.1M /y.

Table 1.14.5 shows the LOM cash cost and unit cost (excluding pre-production mining). In addition to the processing costs, LOM rehandle costs equate to US$0.59 /t processed.

Table 1.14.5 Cash Cost and Unit Cost Summary (at US$1,850 /oz)

Description	LOM Total US$M	LOM AISC US$ /payable oz	LOM US$ /t processed
Mining	1,164	326	6.67
Gbongogo Road Haulage	68	19	0.39
Processing	1,456	408	8.35
Stockpile Rehandle	103	29	0.59
G&A	171	48	0.98
Royalties	495	139	2.84
Total Cash Cost	**3,457**	**969**	**19.82**
Sustaining Capital	104	29	0.60
All-in Sustaining Costs	**3,561**	**998**	**20.42**

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$1,850 /oz (three year trailing average). 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. The results of the financial model are summarised in Table 1.15.1. A breakdown of the annualised operating and economic details can be found in Table 22.2.4 and Table 22.2.5.

Table 1.15.1 Financial Model Summary at US$1,850 /oz

Description	Units	LOM
Feed Tonnage	Mt	174.3
Waste Rock	Mt	205.3
Total Mined	Mt	379.6
Strip Ratio	W:O	1.18
Feed Grade Processed (average)	g/t	0.72
Gold Recovery (average)	%	89.0%
Gold Production	'000 oz	3,570
Annual Gold Production (average)	'000 oz/y	223
Pre-production Capital Cost	US$M	(712)
Sustaining Capital Cost	US$M	(165)
Net Revenue	US$M	6,598
Selling Costs	US$M	(17)
Royalties	US$M	(495)
Total Operating Costs	US$M	(3,019)
EBITDA*	US$M	3,068
Tax	US$M	(547)
Net Cashflow after Tax	US$M	1,700
$NPV_{5\%}$ After Tax	US$M	1,089
IRR	%	31.0%
Cash Cost	US$/oz	969
AISC	US$/oz	998

***EBITDA is a non-GAAP financial measure**

Table 1.15.2 shows the project sensitivity of the NPV, IRR, cash cost and AISC with gold price.

Table 1.15.2 Project Sensitivity

Gold Price	1,650	1,750	1,850[1]	1,950	2,0502
NPV 5% (US$M)	721	906	1,089	1,273	1,456
IRR (%)	22.6	26.9	31.0	35.2	39.3
Cash Cost (US$)	954	962	969	977	984
AISC (US$)	983	991	998	1,006	1,013
Payback (y)	3.2	2.8	2.6	2.3	2.2

[1]Three year trailing average (31 December 2023).
[2]Spot 31 December 2023.

1.16 Recommendations

1.16.1 Geology

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

1.16.2 Environmental

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

In 1Q24, SRK will develop 3D deposit-scale structural models for the pits to assist with the spatial prediction of local / inter-ramp scale structures significant to geotechnical analysis. Further investigations will 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 be confirmed by pilot scale vendor testing in 1Q24.

1.16.5 Infrastructure

Water

Investigations will be carried out to establish the availability of groundwater supplies in the Koné area to reduce the volume of water pumped from the Marahoué River in the initial years. A geophysical survey and exploration drilling will be completed to identify potential aquifers, which would be investigated by pump tests from successful exploration boreholes.

Tailings Storage Facilities and Water Management

To advance the design to the next phase of study the following activities will be included in the scope of the front end engineering design (FEED) study:

- Expanding topography to include all areas potentially impacted by a dam break.
- Sterilisation of infrastructure footprints.
- Update of the design based on the findings of the above investigations.

2.0 INTRODUCTION

The Project lies within the sous-prefectures of Kani and Fadiadougou, 470 km northwest of Abidjan. In February 2017, Orca announced that it had executed a share purchase agreement with two wholly-owned subsidiaries of Kinross 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 TSX in October 2020.

The Project comprises an open pit mining operation with the process plant, WSF and TSF located near the pit.

2.1 Basis of Technical Report

This Technical Report has been compiled by 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 Updated Feasibility Study.

The QPs responsible for Sections in this Technical Report are as follows:

- Jonathon Abbott (Matrix Resource Consultants), responsible for report Sections: 1.7, 12.1, 14, and 25.1.2.

- Sandy Hunter (Lycopodium Minerals Pty. Ltd.), responsible for report Sections: 1.10, 1.11.3, 1.14, 1.15, 17, 18.2, 21 (except 21.2.1, 21.2.3, and 21.3.1), 22 (overview), 25.4, 25.7 and 26.7.

- Michael Hallewell (MPH Minerals Consultancy Ltd.), responsible for report Sections: 1.6, 1.16.4, 12.2, 13, 25.3, and 26. 3.

- Pieter Labuschagne (AGE Pty. Ltd.), responsible for report Sections: 1.11.2, 1.16.5 (part), 12.4, 16.3, 16.5 (part),18.1 (except 18.1.7, 18.1.9, 18.1.10, and 18.1.11), 25.6, and 26.4.

- Carl Nicholas (Mineesia Ltd.), responsible for report Sections: 1.13, 1.16.2, 20, 25.8, and 26.8.

- Joeline McGrath (Carci Mining Consultants Ltd.), responsible for report Sections: 1.8, 1.9, 1.14.2, 1.16.3, 12.3, 15, 16, 21.2.1, 21.3.1, 25.3, and 26.2.

- Timothy Rowles (Knight Piésold Pty. Ltd.), responsible for report Sections: 1.11.1, 1.11.4, 1.16.5 (part), 18.1.7, 18.1.9, 18.1.10, 18.1.11, 18.3, 18.4,18.5, 21.2.3, 25.5, 26.5, and 26.6.

2.2 Qualified Person Site Inspection

A summary of the QP site visits is detailed in Table 2.2.1.

Table 2.2.1 Summary of QP Site Visits

Qualified Person	Site Visit
Jonathon Abbott	23.08.18 to 24.08.18; 25.09.23 to 28.09.23
Carl Nicholas	13.03.21 to 18.03.21
Joeline McGrath	17.11.21 to 19.11.21; 10.07.23 to 12.07.23
Pieter Labuschagne	13.01.23 to 16.01.23
Tim Rowles	13.01.23 to 16.01.23

2.3 Effective Date

The Effective Date of this report is 16 January 2024. 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

AGE	Australasian Groundwater and Environmental Consultants Pty Ltd
Barrick	Barrick Gold Corp.
Carci	Carci Mining Consultants Ltd.
Endeavour	Endeavour Mining
Gbongogo	Gbongogo Main
KGP, 'the Project'	Koné Gold Project
Kinross	Kinross Gold Corporation
KP	Knight Piésold Pty Ltd.
Lycopodium	Lycopodium Minerals Pty Ltd.
Mankono	Mankono Exploration Ltd.
Matix	Matix Resource Consultants
Montage, the 'Company'	Montage Gold Corp.
OMC	Orway Mineral Consultants
Orca	Orca Gold Inc.
Red Back	Red Back Mining (Côte d'Ivoire) SARL
Shark Mining	Shark Mining CDI SARL
Sirocco	Sirocco Gold Côte d'Ivoire SARL
SRK	SRK Consulting
Triple Flag	Triple Flag Precious Metals Corporation
$....k	thousand dollars
$....M	million dollars
#	number (quantity)
"	inch(es)
%	percentage

% (w/w)	percent weight by weight
/	per
µm	micron
µS/m	microsiemens per metre
000 t	thousand tonnes
a	annum
AARL	Anglo American Research Laboratories
AAS	Atomic Absorption Spectrometry
Ag	Silver
Ai	abrasion index (Bond)
ALS	ALS Global
ANDE	Agence Nationale de L'Environnement
ARD	Acid Rock Drainage
Au	Gold
AWBM	Australian Water Balance Model
Axb	JKMRC determined ore impact parameter
bcm	bank cubic metre
BLEG	LeachWELL Bulk Leach Extractable Gold
BSD	Black Silicious Diorite
BV	Bed Volumes
BWi	Ball mill Work Index
CDI	Coarse Grained Diorite
CDLM	Committee of Development and Local Mining
CIP	Carbon-in-Pulp
CIUC	Consolidated Undrained Triaxial Compression
cm	centimetre
Cu	copper
°	degree (angle)
°C	degree Celsius
DFS	Definitive Feasibility Study
DGMG	General Directorate of Mines and Geology
DGPS	Differential Global Position System
DO	Dissolved oxygen
EC	Electrical conductivity
EIA	Environmental Impact Assessment
EITI	Extractive Industries Transparency Initiative
EGD	Early Green Dykes
EMP	Environmental Management Plan

ESIA	Environment and Social Impact Assessment
ESPM	Exploration Environmental and Social Management Programme
FDI	Fine Grained Diorite
FDY	Felsic Dykes
Fe	iron
FEED	Front End Engineering Design
FPR	Feldspar Porphyry Dyke
Fortuna	Fortuna Silver Mines Inc.
FS	Feasibility Study
g	gram
g/L	grams per litre
g/t	grams per tonne
G&A	General and Administration
GAT	Gravity Amenability Test
Geostats	Geostats Pty Ltd, WA
GPS	Global Positioning System
GRG	Gravity-Recoverable-Gold
GWh	Gigawatt-hour
h	hour or hours
ha	hectare
HCN	Hydrogen cyanide
HDPE	High Density Polyethylene
HPGR	High Pressure Grinding Rolls
HPi	High Pressure Grinding Index
HQ	Exploration drill size (96 mm OD)
IDY	Intermediate Dyke
IFC	International Finance Corporation
Intertek	Intertek Minerals Ltd.
IRR	Internal Rate of Return
IQD	Quartz Diorite
JV	Joint Venture
k	thousands (kilo)
km	kilometre
km^2	square kilometres
koz/y	kilo ounce per year
kPa(g)	kilopascal gauge
kV	kilovolt
kWh	kilowatt hour

kWh/t	kilowatt hours per tonne
L	Litre
LGD	Late Green Dykes
LME	London Metals Exchange
LOM	Life of Mine
L/s	litre per second
m	metre
M	millions (mega)
m^2	square metre
m^3	cubic metre
Ma	mega annum
mAMSL	metres Above Mean Sea Level
MAP	Mean annual precipitation
mASL	metres Above Sea Level
mbgl	metres below ground level
MDE	Maximum Design Earthquake
MDY	Mafic Dykes
mg/L	milligrams per litre
MIK	Multiple Indicator Kriging
mi	mile
min	minute
mL	millilitre
mm	millimetre
mm^2	Square millimetre
Moz	million ounces
MPa	megapascal
MRE	Mineral Resource Estimate
mS/m	milliSiemens per metre
Mt	million tonne
Mtpa	million tonne per annum
MW	megawatt
No.	number
NPAG	Non-Potentially Acid Generating
NPV	Net Present Value
NQ	Exploration drill size (75. 5mm OD / 47.6 mm ID)
NSR	Net Smelter Return
n/s	no sample
Opex	Operating Cost Estimate

OREAS	Ore Research & Exploration Pty Ltd., Australia
oz	ounce (troy = 31.10348 grams)
P_{80}	80% product passing size (in microns)
PAG	Potentially Acid Generating
PAR	Population at Risk
PEA	Preliminary Economic Assessment
PGDI	Pale Green Diorite
pH	hydrogen ion exponent (measure of acidity of alkalinity)
PLS	Pregnant Leach Solution
ppb	Parts per billion
ppm	parts per million
PQ	Exploration drill core size (122.6 mm OD)
PSA	Particle Size Analyser
Q	quarter
QA/QC	Quality Assurance / Quality Control
QP	Qualified Persons
RC	Reverse Circulation
RF	Revenue Factor
RL	Reference Level
Rocklabs	Rocklabs Ltd., Auckland, New Zealand
ROM	Run-of-Mine
RQD	Rock Quality Designation
SAG	Semi-Autogenous Grinding
SANAS	South Africa National Accreditation System
Saprock	Saprolite Rock
SG	Specific Gravity
S/L	solid / liquid
SPT	Static Pressure Testing
t	metric tonne (1,000 kg)
tpd	tonnes per day
tph	tonnes per hour
TOS	Trade-off Study
TSF	Tailings Storage Facility
TSX	Toronto Stock Exchange
TTG	Tonalite-Trondhjemite-Granodiorite
UFS	Updated Feasibility Study
UTM	Universal Transverse Mercator (coordinate system)
US$	United States dollar

VTEM	Versatile Time Domain Electromagnetic
WAEMU	West African Economic and Monetary Union
WRD	Waste Rock Dumps
WSF	Water Storage Facility
wt	wet tonne
w/v	weight by volume
y	year

The coordinate system used throughout this report is World Geodetic System (WGS84) Zone 29 North coordinates.

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, 1.12, 4, 5, 6, 19, and 23.

Geology: the Author has relied on information provided by Montage for Sections 1.4, 1.5, 1.16.1, 7, 8, 9, 10, 11, 25.1.1, and 26.1.

Metallurgical Testwork: the Author has relied on information provided by MPH Minerals Consultancy Ltd for Sections 1.6, 1.16.4, 12.2, 13, 25.2, and 26.3. Lycopodium has reviewed the metallurgical testwork results and concurs with their interpretation.

Resources: the Author has relied on information provided by Matrix Resource Consultants for Sections 1.7, 12.1, 14, and 25.1.2.

Mining: the Author has relied on information provided by Carci Mining for Sections 1.8, 1.9, 1.16.3, 12.3, 15, 16, 21.2.1, 21.3.1, 25.3, and 26.2.

Hydrogeology: the Author has relied on information provided by AGE Pty. Ltd. for Sections 1.11.2, 1.16.5 (part), 12.4, 18.1 (except 18.1.7, 18.1.9, 18.1.10, 18.1.11), 25.6, and 26.4.

Tailings Storage: the Author has relied on information provided by Knight Piésold Pty. Ltd. for Sections 1.11.4, 1.16.5 (part), 18.1.7, 18.1.9, 18.1.10, 18.1.11, 18.3, 18.4, 21.2.3, 25.5, and 26.5.

Infrastructure: the Author has relied on information provided by Knight Piésold Pty. Ltd. for Sections 1.11.1, 1.16.5 (part), 18.5, and 26.6

Environment and Social: the Author has relied on information provided by Mineesia Limited for Sections 1.13, 1.16.2, 20, 25.8, and 26.8.

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

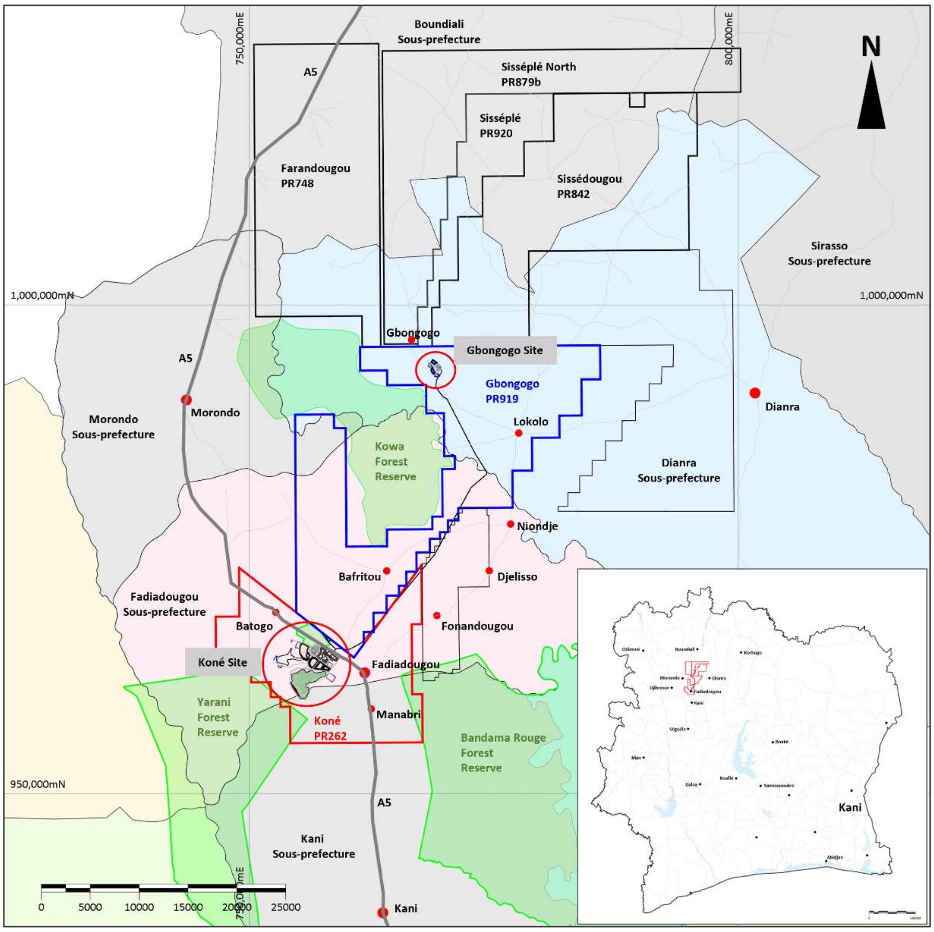
4.1 Property Location

The KGP covers 1,801 km^2 in northwest Côte d'Ivoire 470 km northwest of the commercial capital Abidjan (Figure 4.1.1).

The Project comprises six exploration permits (PR's 262,748, 842, 879b, 919 and 920 748) covering 1,801 km^2 and two exploration permit applications covering a further 458 km^2. The Project area straddles the Departments of Kani, Dianra and Boundiali in the Worodougou and Savanes regions of Côte d'Ivoire. The communities of Fadiadougou, Batogo and Manabri are located 4 to 6 km from the proposed Koné plant site, and the village of Gbongogo is located 3 km from the proposed Gbongogo satellite open pit.

A part of the Toudian Forest Reserve lies 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 has minimised its impact in the forest area. Discussions with the Ministry of Water and Forests have commenced to obtain authorisation for mining in a part of the forest reserve.

Figure 4.1.1 Project Location Map



Source Montage, January 2024.

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 four years, plus two renewals of three years with the possibility of a third renewal for two years in order to complete feasibility studies.
- 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, four year period, renewable.
- Artisanal Mining Licence - Ivorian Nationals or Ivorian majority co-operatives only, maximum of 25 ha. Two 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 be awarded 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.

4.2.2 Royalties

Mining royalties to the state of Côte d'Ivoire for gold extraction vary with gold price (Table 4.2.1).

Table 4.2.1 Summary of Royalties

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

There is an additional Royalty of 0.5% payable to a Community Development fund that will be constituted for the benefit of villages identified as 'affected localities' by the ESIA. This fund is annually credited and will be used to realise socioeconomic development projects. The terms of the local mining development committee (CDLM) are defined in the mining convention (not yet agreed as per the 'Effective Date' of this Report) and will be established by Ministerial decree. It may be assumed that the CDLM fund will be in place from the Date of First Commercial Production.

Triple Flag Precious Metals Corporation ('Triple Flag') holds a 2% Net Smelter Return (NSR) royalty on the Koné Exploration Permit.

Barrick and Endeavour hold a 2% NSR royalty (70:30 respectively) on the Gbongogo, Sissédougou and Sisséplé Exploration Permits.

4.2.3 Project Mineral Tenure and Ownership

In February 2017, Orca announced that it had executed a share purchase agreement with two wholly owned subsidiaries of Kinross 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 listed on the TSX in October 2020.

A summary of the exploration permits that comprise the KGP is shown below in Table 4.2.2. The exploration permits are held by three wholly owned subsidiaries of Montage: Shark Mining CDI SARL, Orca Gold CDI SARL, and Mankono Exploration SA. The key permits are the Koné Exploration Permit that hosts the Koné Mineral Resource and Reserve and the Gbongogo Exploration Permit that hosts the Gbongogo Mineral Resource and Reserve.

Table 4.2.2 KGP Exploration Permits

Permit	Company	Area	Decree No	Permit No	Decree Date	Status
Koné	Shark	225.00	2013-198	262	22/03/2013	Exceptional Renewal ends 22/03/2024
Gbongogo	Mankono	399.97	2022-686	919	06/09/2022	In initial period
Sissédougou	Mankono	387.00	2019-681	842	24/07/2019	In first renewal period
Sisséplé	Mankono	105.85	2022-687	920	06/09/2022	In initial period
Farandougou	Orca	361.51	2021-306	748	16/06/2021	In initial period
Sisséplé North	Orca	321.60	2022-027	879b	12/01/2022	In initial period

The Koné Exploration Permit number 262 (PR 262) was granted to Red, a wholly owned subsidiary of Kinross, in 2013.

In March 2016 and March 2019, the Koné Exploration Permit was renewed for three years and in March 2022 was granted an exceptional renewal for a period of two years. The local operating company's name Red Back Mining (Côte d'Ivoire) SARL was changed to Shark Mining CDI SARL in August 2018.

In 2021 and 2022 Orca Gold CDI SARL was issued the Farandougou and Sisséplé North exploration permits.

In November 2022, Montage completed the acquisition of Mankono Exploration SA from Barrick and Endeavour. Mankono holds the Gbongogo, Sissédougou and Sisséplé Exploration Permits.

Montage is currently in the process of transferring the Gbongogo Permit from Mankono to Shark Mining to facilitate the issue of the mining permits to a single entity.

In addition, the Company holds two Exploration Permit applications covering a further 458 km^2.

Under the terms of the Exploration Permits, 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 Project.

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 Exploration Permits and to the extent known, the Project is not subject to any environmental liabilities.

Under the terms of the 2014 Mining Code, the Company has the right to apply for an Exploitation Permit on completion of an ESIA and Feasibility Study (FS). The KGP ESIA was submitted for approval in December 2023 and the Company is seeking approval in 1Q24, at which time it will make the application for an Exploitation Permit.

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 230 km road between Abidjan and Yamoussoukro is a four-lane motorway with access by sealed road via Daloa, Séguéla, and Kani to the Company's main base in the village of Fadiadougou. The Company maintains an exploration camp close to the Gbongogo site, accessed via a network of unmetalled roads which cover much of the Project area and provides generally good wet and dry season access. Exploration activities can be undertaken throughout the year.

The Project site lies within 1 km of the main Séguéla-Boundiali asphalt road. The Gbongogo site will be accessed by a proposed dedicated haul road, linking it to the process facility at Koné.

The communities of Fadiadougou, Batogo and Manabri lie on the asphalt road close to the proposed Koné Plant site, and the village of Gbongogo lies 3.5 km north west of the Gbongogo site.

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, 80 km 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 miles (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 characterised by moderate relief between 200 m and 420 mASL (Figure 5.4.1). 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 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 5.4.1 Photograph of Koné Resource Area (Facing North)

Source: Montage.

6.0 HISTORY

Red Back applied for the Koné Exploration permit on 28 July 2008. An 'Autorisation de prospection' was issued on 22 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 summarises the main field exploration activities undertaken by previous tenement owners.

Table 6.1 Field Exploration Undertaken by Previous Owners

Activity	Red Back 2009 to 2010	Sirocco 2013 to 2014	Orca 2017 to 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 Koné resource area that may represent old workings of indeterminate age.

During the second half of 2009, an 800 m x 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 22 March 2013, the licence application was granted by Presidential Decree 198-2013 under Permit Number 262.

On 22 May 2013, Kinross 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 3,340 m 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 MRE completed in October 2018, which is described in a NI 43-101 Technical Report with an effective date of 3 October 2018 (Abbott, 2018). No other mineral resource estimates, including historic estimates have been produced for the Project.

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

On 13 July 2019, Orca's assets were transferred to newly created, private subsidiary, Montage. During the remainder of 2019 and much of 2020, Montage completed a programme of exploration in the wider Koné Exploration Permit and on diamond core drilling to test the depth extents of the Koné Deposit.

In October 2020, Montage Gold Corporation was listed on the Toronto Venture Stock Exchange and drilling activities ramped up considerably.

Montage first conducted a 20,000 m expansion drill program during 4Q20, targeting resource expansion below the depth extent of the October 2018 MRE. This drilling was successful in delineating a 4.00 Moz Inferred MRE in January 2021 (the 'January 2021 MRE'), which formed the basis of a preliminary economic assessment (PEA) published in May 2021

From January 2021 through to July 2021, Montage completed approximately 60,000 m of infill drilling to convert the Inferred Mineral Resource to the Indicated category. The outcome of the infill program was the initial 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 (at a 0.20 g/t cut-off grade).

On the basis of the Indicated Resource, Montage completed a DFS which was published in February 2022, demonstrating that the KGP had the potential to become a large, low cost gold operation producing over 200 koz/y for a period of 15 years.

In June 2022, Montage signed an agreement with Barrick and Endeavour to acquire the Mankono Joint Venture (JV) project, contiguous to the north of the Koné Exploration permit. This acquisition expanded the Project footprint and significantly increased the upside potential of the Project.

The acquisition of the Mankono-Sissédougou JV closed in November 2022 and, since that time, Montage has completed reconnaissance drilling, soil sampling, IP and other activities across a broad range of targets within the expanded land package. Notably, the Company was successful in upgrading and expanding the mineral resources at the Gbongogo satellite deposit with an Indicated Mineral Resource of 12.0 Mt grading 1.45 g/t for 559 koz (at 0.50 g/t cut-off grade).

Concurrent with this work, the Company also expanded and completed an ESIA which was submitted to the Agence Nationale de L'Environnement (ANDE) for approval in December 2023.

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

7.0 GEOLOGICAL SETTING AND MINERALISATION

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 7.1.1). 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. 2250 to 2120 mega annum (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. 2120 to 2090 Ma.

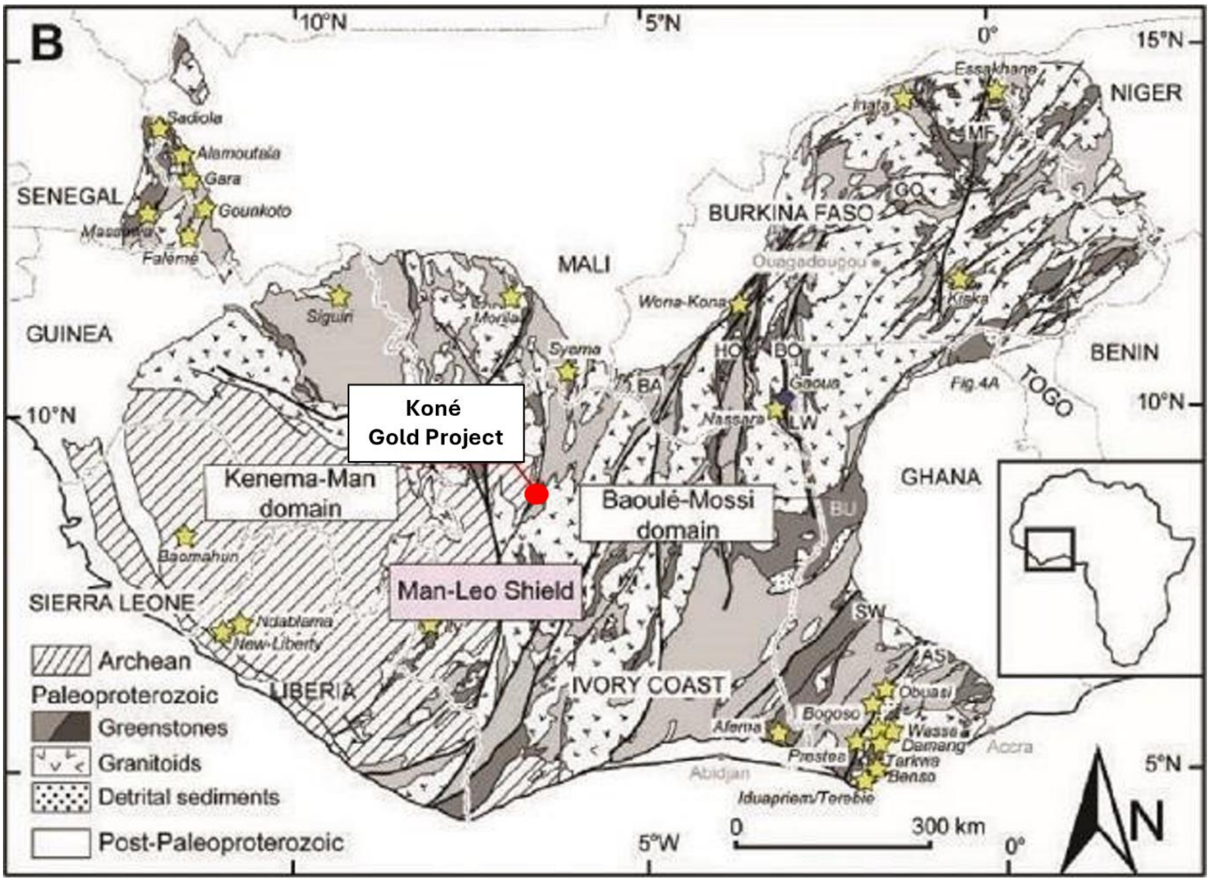
- Undeformed potassic granites, occasionally metaluminous or syenitic with abundant Kfeldspar often with a biotite association, amphibole is usually absent. 2110 to 2070 Ma.

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 remelting 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. 2130 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, mineralisation 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).

Figure 7.1.1 Geology of the Man Leo Shield



Source: Montage (after Goldfarb et al, 2017).

Structurally, most mineralisation is associated with the 'D2' phase of deformation where compressive stress shifted to transpression and transcurrent shearing / strike slip faulting. Gold mineralisation 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 mineralisation is structurally controlled and include volcanic rocks, sedimentary rocks and granites.

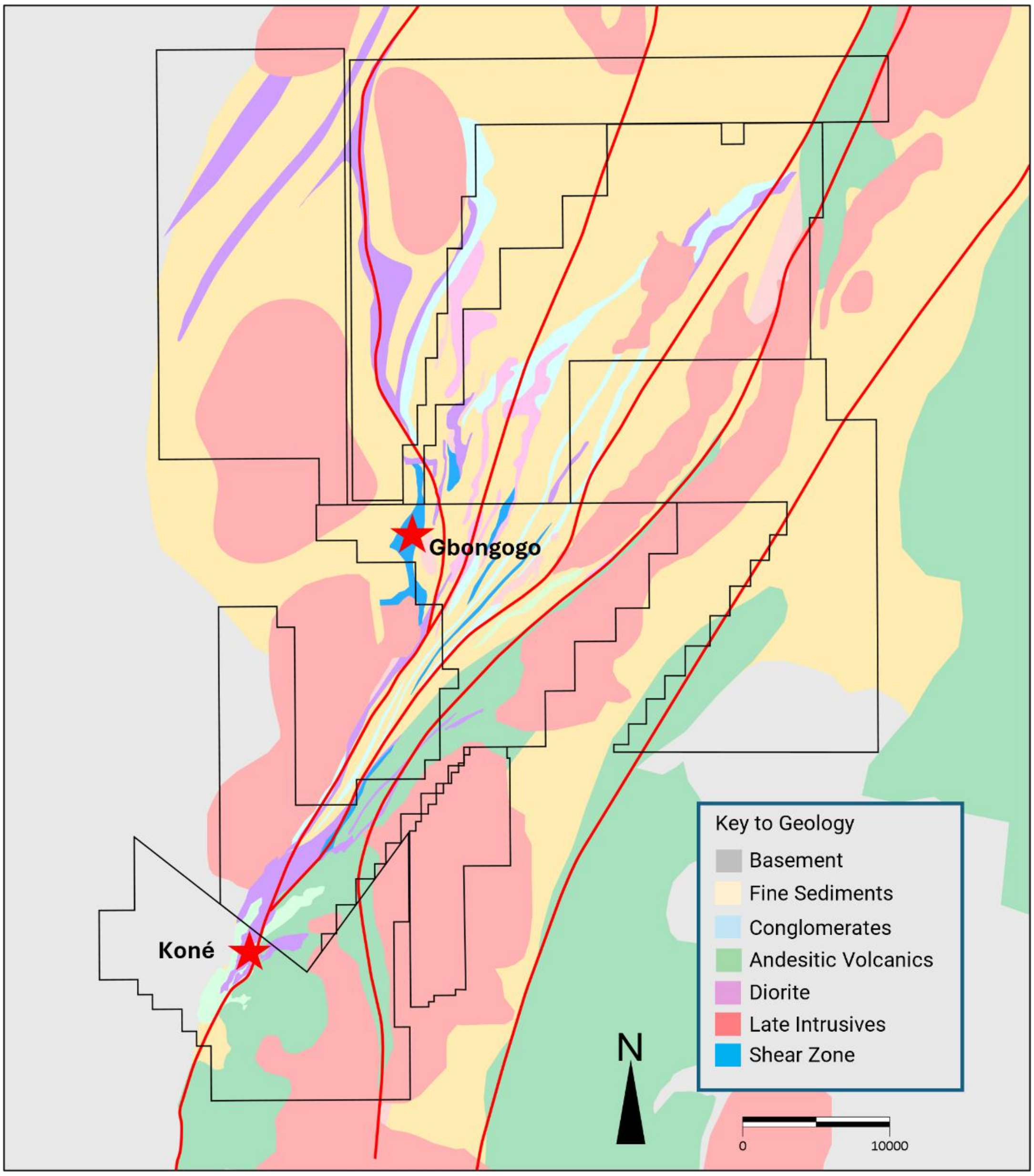
7.2 Koné Gold Project Geological Setting

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 7.2.1). The rocks have been metamorphosed to greenschist facies. Regional aeromagnetic data shows strong northeast–southwest trends interpreted to reflect the distribution of underlying rock units.

Within the Gbongogo Exploration Permit, geophysics and geological mapping indicate northeast–southwest and north–south trending Birimian sediments, volcaniclastics and volcanics flanked to the west by basement (Figure 7.2.1).

Figure 7.2.1 Geological Map of the Koné Gold Project



Source: Montage, January 2024.

7.3 Koné Deposit Geological Setting and Mineralisation

Koné is a mesothermal, structurally controlled gold deposit hosted within a north-south trending, westerly dipping (50°), composite package of sheeted 20 m to 30 m thick diorite intrusions which have been emplaced by multiple intrusive pulses (Figure 7.3.1). 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 350 m in true thickness and can currently be traced along strike for 2.4 km (Figure 7.3.2).

The diorite bodies at Koné have intruded into the contact zone between two different sequences of mafic volcaniclastic rocks which form the hanging wall 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 to 2100 Ma).

The hanging wall volcaniclastics are characterised 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.

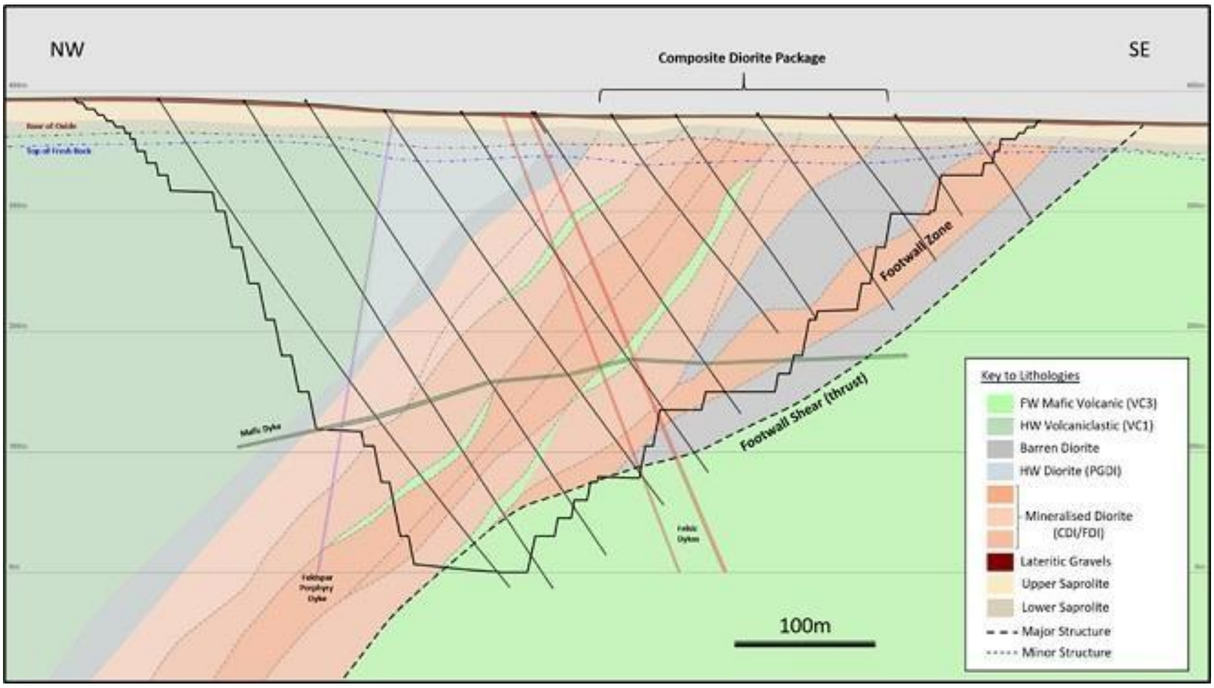
Koné is interpreted to have formed as part of a stacked thrust-shear under a compressional tectonic regime. The major thrust is located at the footwall of the diorite body where a 5 m to 20 m wide zone of shallowly plunging / horizontal tight folds can be observed within volcanic rocks immediately below thefootwall 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 mineralisation related to it.

Within the diorite, higher gold grades (>1 g/t) are associated with swarms of foliation parallel, 1 to 2 mm quartz + pyrite ± chalcopyrite veinlets which form distinct corridors of mineralisation. 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 (0.2 g/t to 1.0 g/t) are associated with disseminated pyrite mineralisation and, in general, the diorite package is mineralised over most of its width, averaging >200 m over the main part of the deposit, with a maximum of up to 330 m (MDD015B, 330.7 m grading 0.58 g/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.

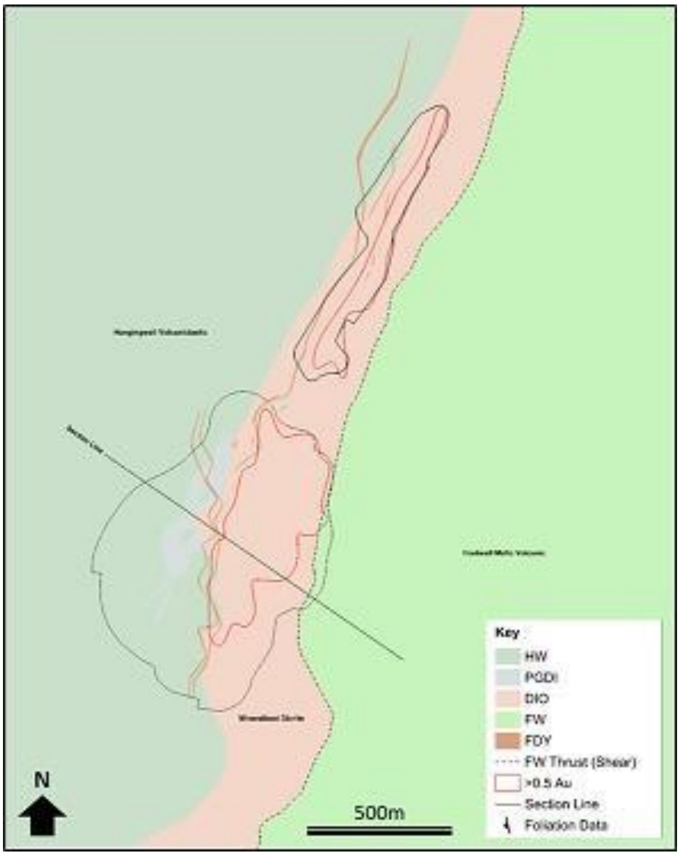
The deposit is intruded by multiple dykes with the majority post-dating mineralisation by as much as 50 Ma and seen cutting foliation. One set of dykes, known as the early green dyke, displays deformation and folding.

Figure 7.3.1 Section Through the Centre of the Koné Deposit



Source: Montage, January 2024.

Figure 7.3.2 Plan of the Koné Deposit



Source: Montage, January 2024.

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 mineralisation 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 7.3.3 shows a sharp contact between two diorite bodies highlighted by appearance of porphyritic texture in MDD017 at 214.15 m depth. The contact is dipping 53° towards 279° (striking 009°). Numerous observations of this nature indicate a complex diorite package with stacked diorite bodies.

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 mineralised to a similar extent.

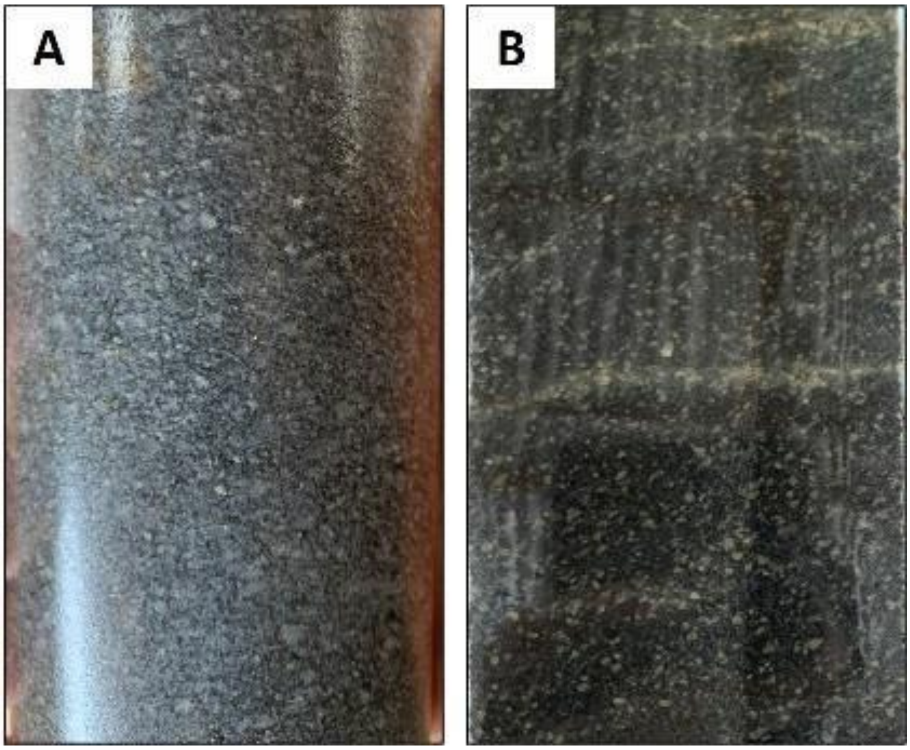
Figure 7.3.3 Example of Sharp Contact Between Two Diorite Bodies

Source: Montage, January 2024.

Coarse Grained Diorite (CDI)

Diorite with up to 2 mm 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 7.3.4). This unit hosts gold mineralisation, associated with 1 to 2 mm, 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 7.3.4 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, January 2024.

Fine Grained Diorite (FDI)

The fine diorite unit is very closely related to the coarse diorite (Figure 7.3.5). 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 mineralisation associated with 1 to 2 mm sulphide bearing veinlets and disseminated pyrite.

Figure 7.3.5 Fine Grained Diorite

Source: Montage, January 2024.

Black Silicious Diorite (BSD)

One distinctive fine grained diorite is observed. Characterized by its' very fine crystal size, dark grey / black colour and siliceous nature (Figure 7.3.6), this unit contains abundant magnetite and often hosts quartz-sulphide veinlets containing mineralisation. 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 7.3.6 Black Siliceous Diorite

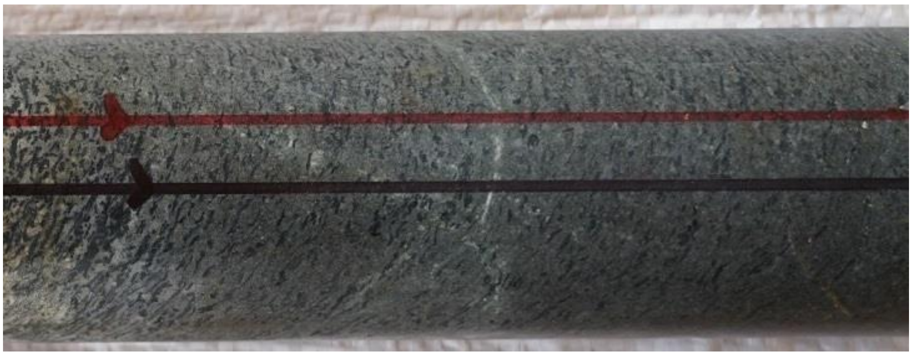
Source:Montage, January 2024.

7.3.2 Hanging Wall Geology

Pale Green Diorite (PGDI)

The pale green diorite is a rock of varying grainsize, characterised by its' distinctive pale green colour caused by large, foliation parallel amphibole crystals and abundant fine chlorite related to regional metamorphism (Figure 7.3.7). 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).

Figure 7.3.7 Pale Green Diorite



Source: Montage, January 2024.

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-mineralisation sulphides are observed (Figure 7.3.9) aiding the early intrusion interpretation.

Mafic Volcaniclastic Type 1 (VC1)

A poorly sorted volcaniclastic rock of polymictic clasts (<5 centimetres (cm) in length) within a fine groundmass composed of ultrafine plagioclase, chlorite and biotite (phlogopite). Some sections lack clasts displaying only planar foliation (Figure 7.3.8). Planar foliation wrapping round the clasts is highlighted by the chlorite and phlogopite (Figure 7.3.9). 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 7.3.8 Mafic Volcaniclastic Type 1

Source: Montage, January 2024.

Figure 7.3.9 Mafic Volcaniclastic Type

Source: Montage, January 2024.

7.3.3 Footwall Geology

Mafic Volcaniclastic Type 3 (VC3)

Strongly deformed (Figure 7.3.10), 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 0.7 g/t in this unit due to its proximity to the footwall shear controlling mineralisation.

Figure 7.3.10 Mafic Volcaniclastic Type 3

Source: Montage, January 2024.

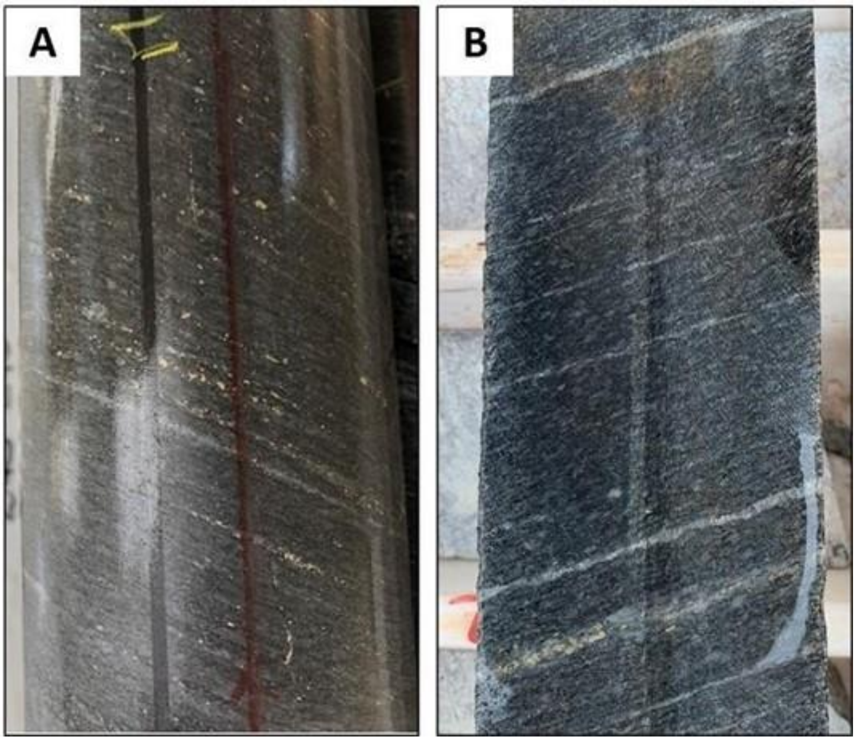
7.3.4 Koné Mineralisation

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

Higher grades (1 to 1.5 g/t) are associated with high density 'swarms' of 2 to 5 mm thick, foliation parallel translucent white to smoky quartz veinlets containing fine grained sulphide (Figure 7.3.11 and Figure 7.3.12). Lower grades are related to disseminated fine grained pyrite. Importantly, no significant silicification of the host rocks is associated with the mineralisation resulting in positive comminution characteristics.

Mineralisation at Koné is interpreted to be controlled by a major thrust shear at the footwall contact of the diorite contact (Figure 7.3.13). The volcaniclastic rocks in the footwall of the thrust have deformed plastically whereas the more rheologically 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 7.3.14 and Figure 7.3.15).

Figure 7.3.11 Mineralised Foliation



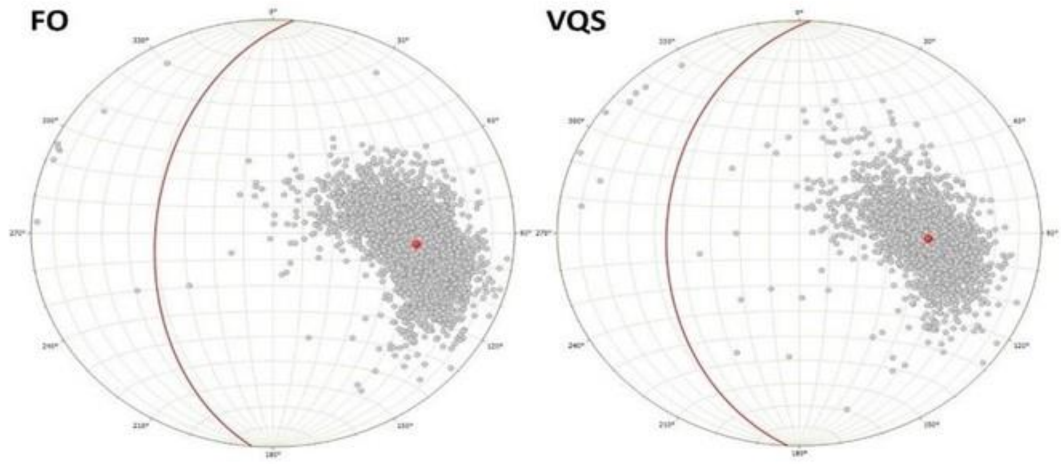
A – Mineralised Foliation **B – VQS Vein Swarm**

Source: Montage, January 2024.

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.

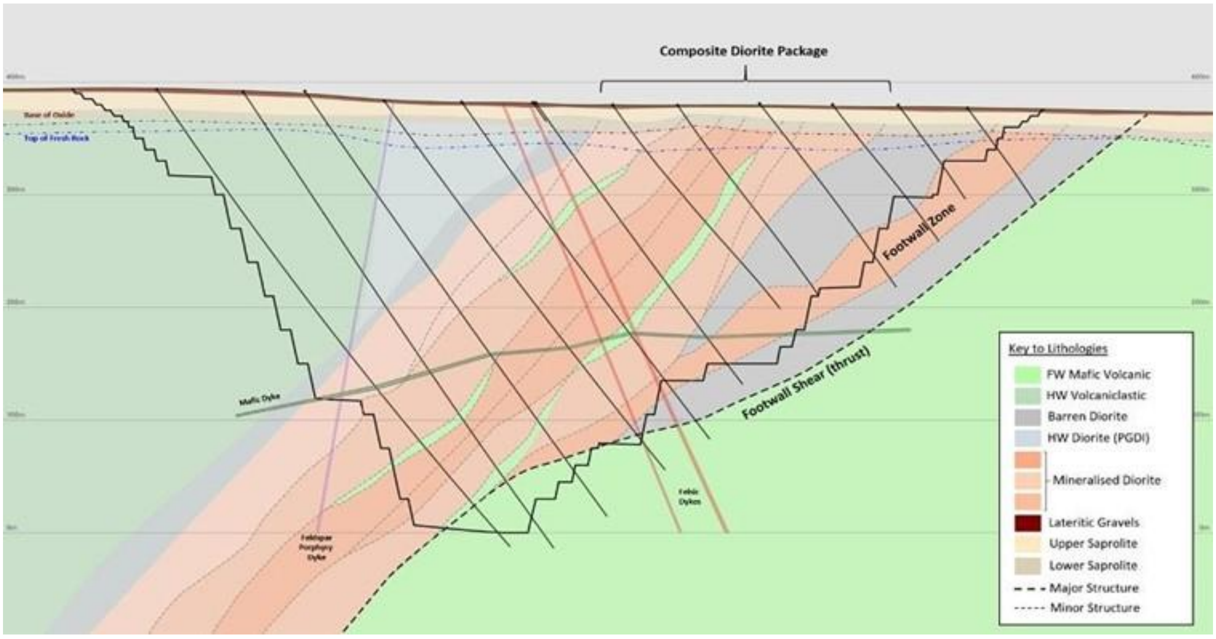
A phase of barren pyrite is present in the hanging wall that has a 1 to 2 mm grainsize and euhedral cubic in form. Pyrite related to the mineralisation is either <0.5 mm disseminated globules of ultrafine pyrite that appear to be replacing magnetite locally, or internally with the VQS veins.

Figure 7.3.12 Foliation Orientations



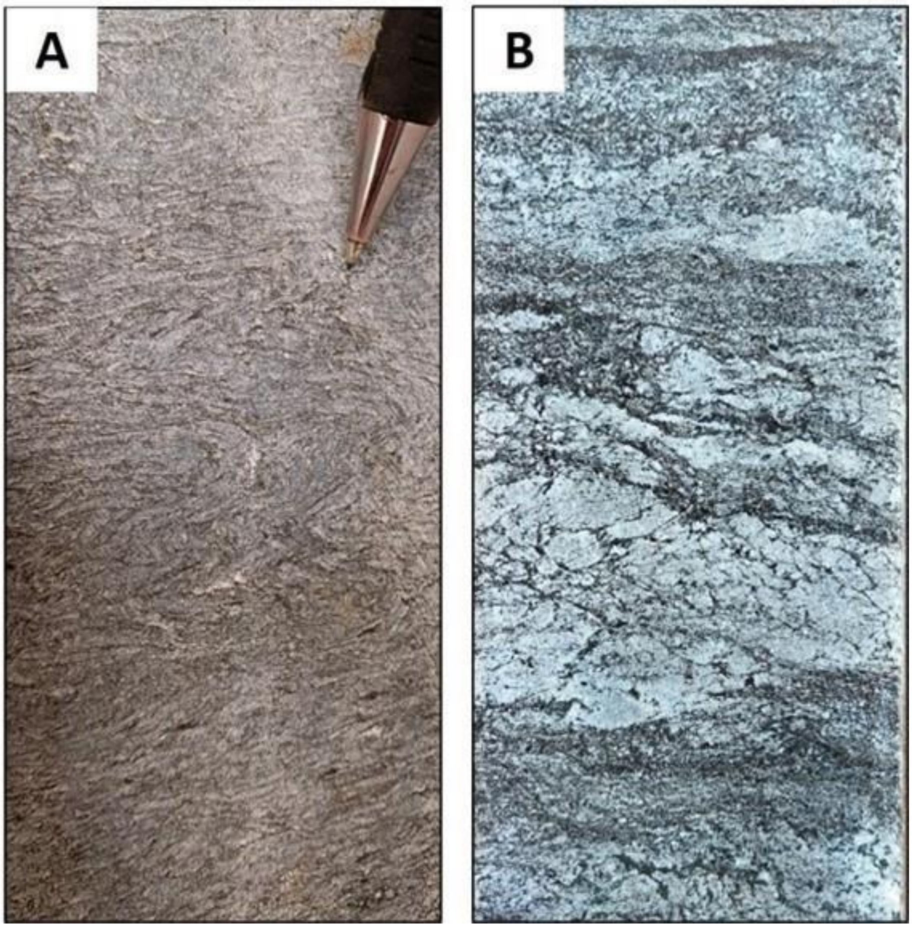
Source: Montage, January 2024.

Figure 7.3.13 Section Through the Centre of the Deposit



Source: Montage, January 2024.

Figure 7.3.14 Ductile Shear Within Diorite



A - Ductile Shear **B – Brecciated Zone**

Source: Montage, January 2024.

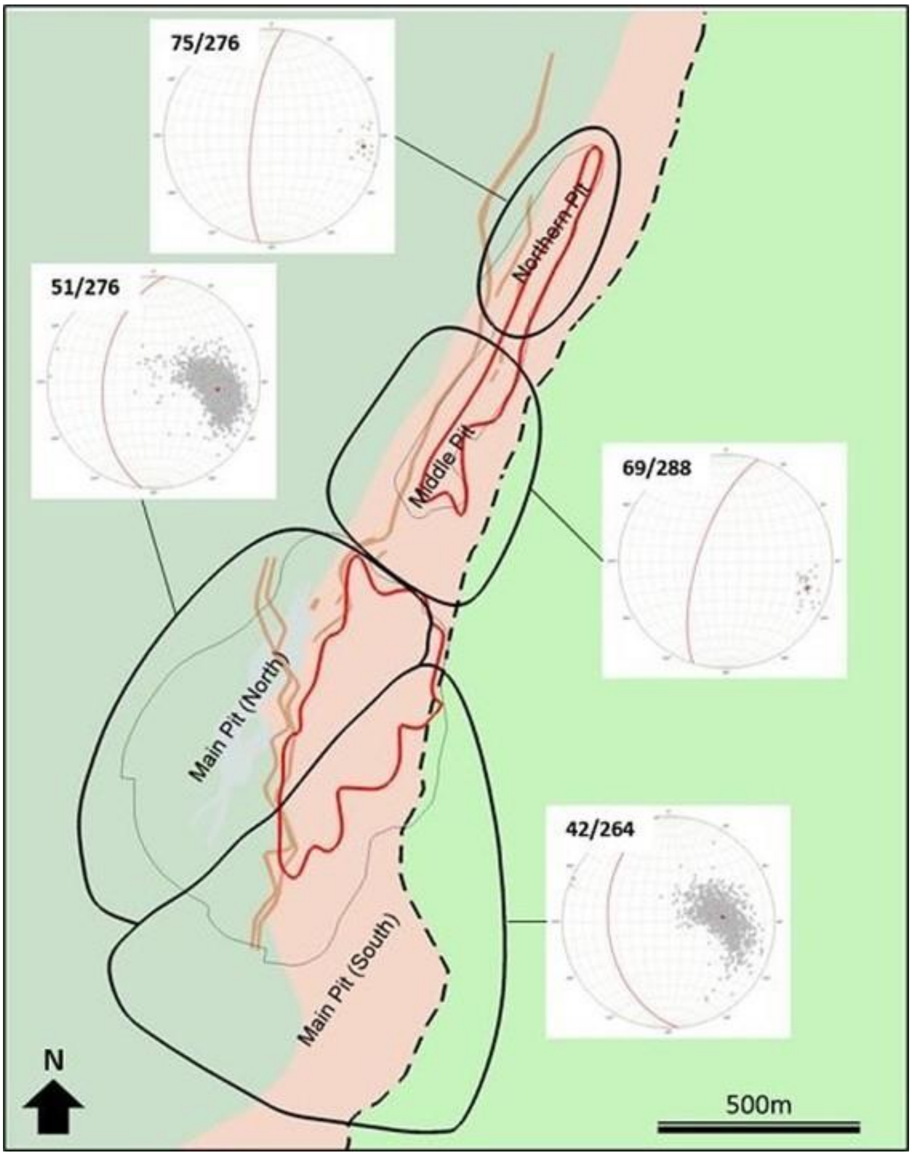
Figure 7.3.15 Buckle Folds Within the Diorite

Source: Montage, January 2024.

7.3.5 Structure and Deformation

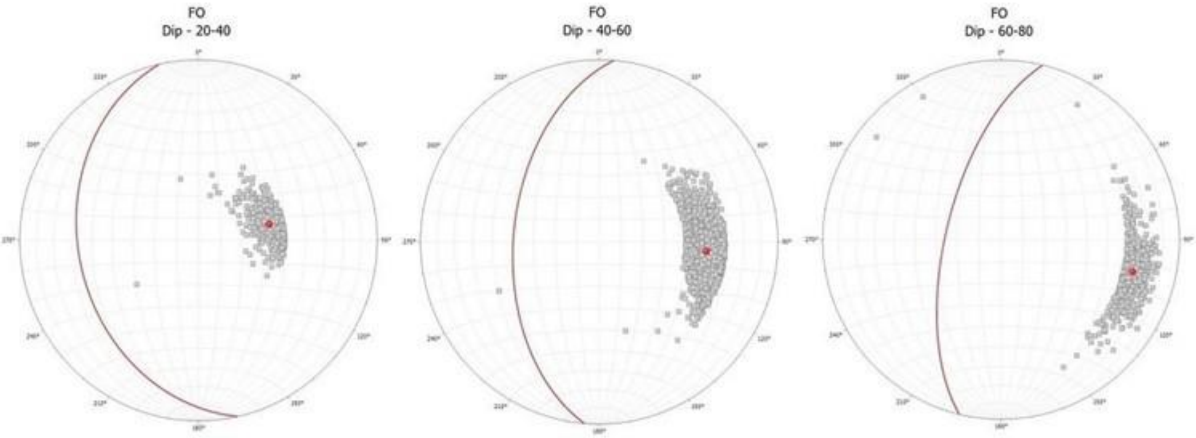
A correlation between the of dip the foliation and the strike of the host rocks can also be observed (Figure 7.3.16), with foliations dipping 20° to 50° averaging a strike of ~348° in the southern part of the deposit, and dipping 60° to 80° striking ~014° in the northern sector. This change in strike is thought to be related to a step in the regional, mineralised structure. This step is interpreted to be a controlling factor on mineralisation within this area, with the flattening and rotation of the diorite bodies by the structure allowing for fluid accommodation and subsequent mineralisation. 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-southeast (Figure 7.3.17).

Figure 7.3.16 Correlation Between the Foliation Dip and Host Rocks Strike



Source: Montage, June 2021.

Figure 7.3.17 Stereonets of Foliations



Source: Montage, January 2024.

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.

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 7.3.18). The foliation is often deformed and the dykes regularly return above detection limit for gold, meaning these dykes are interpreted to be pre- / syn-mineralisation. 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.

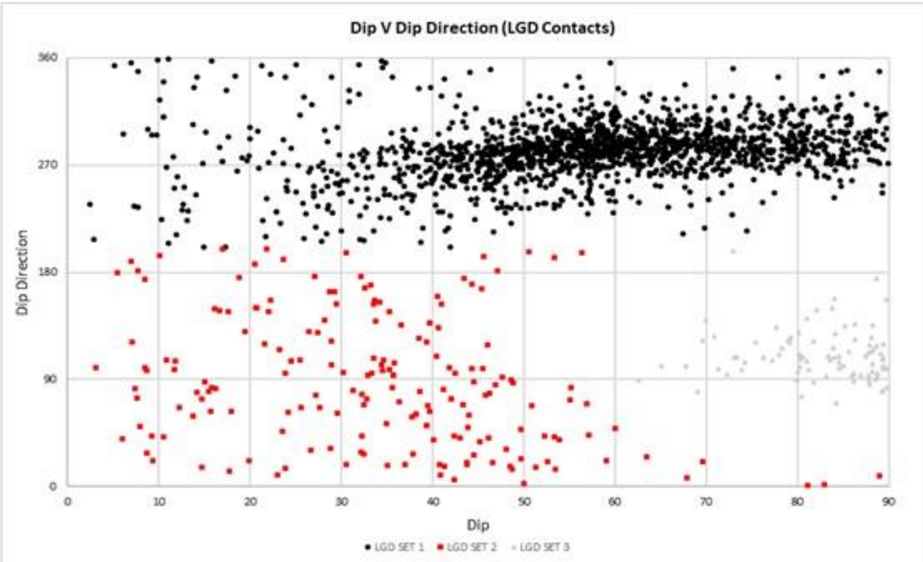
Figure 7.3.18 Chlorite / Biotite Altered Foliations

Source: Montage, January 2024.

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 7.3.19. 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 to 2 mm amphibole phenocrysts (Figure 7.3.20). Interpreted as late-stage, post mineral dykes shown by lack of deformation and alteration. Occasional late calcite veinlets are observed.

Figure 7.3.19 Late Green Dykes Dip and Dip Direction



Source: Montage, January 2024.

Figure 7.3.20 Late Green Dykes

Source: Montage, January 2024.

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 mineralisation (Figure 7.3.21).

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 7.3.21 Massive Mafic Dyke

Source: Montage, January 2024.

Felsic Dykes (FDY)

Felsic dykes are light grey in colour, aphanitic, massive and cross-cut the foliation at a high angle (Figure 7.3.22). They are intruded very late into the sequence and post-date the main deformation and mineralisation 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° to 80°. Two main felsic dykes have been logged in the main pit, with increased frequency seen to the north.

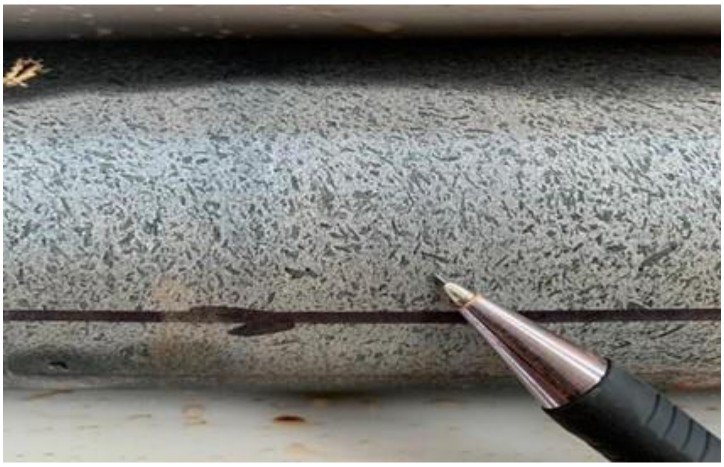
Figure 7.3.22 Felsic Dyke

Source: Montage, January 2024.

Intermediate Dyke (IDY)

Late massive, undeformed intermediate dyke. Characterised by large (<5 mm) randomly orientated amphibole crystals with strong magnetite and sub-vertical, sharp contacts (Figure 7.3.23). Located in the west of the deposit, striking near close to north-south.

Figure 7.3.23 Intermediate Dyke



Source: Montage, January 2024.

Feldspar Porphyry Dyke (FPR)

Massive, unaltered, porphyritic, intermediate dyke with distinctive round feldspar phenocrysts and moderate magnetism (Figure 7.3.24). 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 ± 4 Ma, approximately 50 Ma after the dated emplacement of the diorite.

Figure 7.3.24 Feldspar Porphyry Dyke

Source: Montage, January 2024.

7.3.7 Post Mineralisation Deformation

Faults

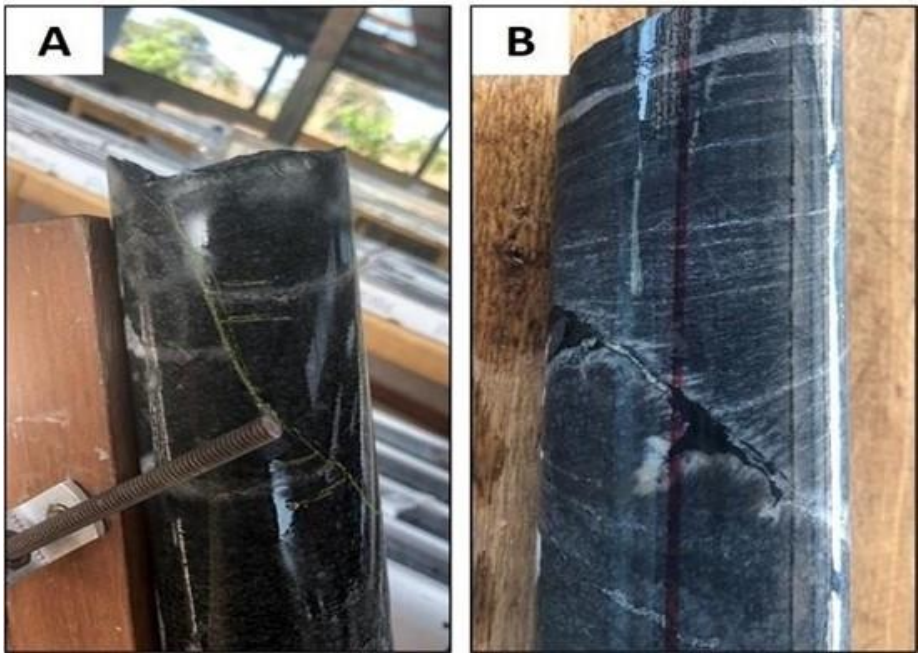
Very few larger fault, clays / breccias are observed within the deposit, with the majority of faulting observed being minor faults displaying both normal and reverse movement (Figure 7.3.25), leading to high core recoveries (average 98% across the entire deposit). Therefore, any displacement and offset of the mineralisation 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 7.3.26):

- 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 mineralised veins, therefore showing that faulting occurred post mineralisation and is offsetting it.

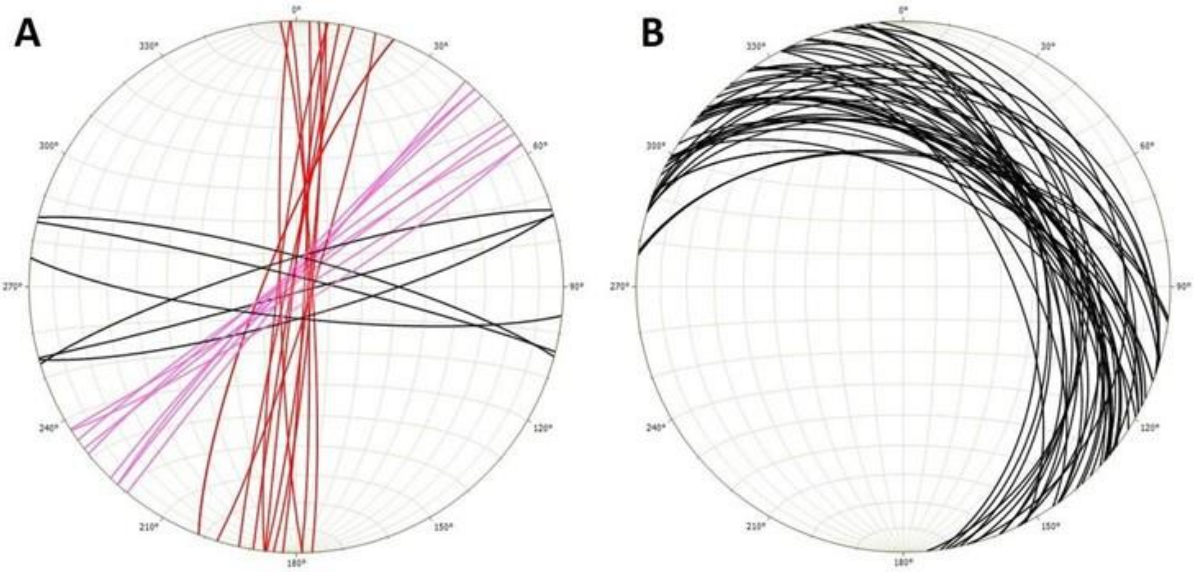
Figure 7.3.25 Minor Healed Faults



A - Minor Healed **B - Minor healed fault cutting across veins**

Source: Montage, January 2024.

Figure 7.3.26 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.

Source: Montage, January 2024.

Folding

In places, veins display non-linear, sinuate contacts showing post formation deformation (Figure 7.3.27). 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.

Figure 7.3.27 Folded VQS

Source: Montage, January 2024.

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 7.3.28). 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 7.3.28 Ductile Folding

Source: Montage, January 2024.

7.4 Gbongogo Deposit Geological Setting and Mineralisation

The Gbongogo gold deposit is a mesothermal, lithologically constrained deposit, hosted within a single, north plunging (50°), quartz diorite intrusion which is an approximately elliptical cylinder.

The intrusion has a surface expression of 120 m x 75 m and dimensions of 180 m x 100 m in section, perpendicular to plunge (Figure 7.4.1 and Figure 7.4.2). The intrusion has been intercepted in drilling to a vertical depth of 320 m and is open at depth.

The quartz diorite is interpreted to have been emplaced in a single event, based on its homogeneous mineral composition, low variability in pXRF analysis, and lack of internal contacts. The quartz diorite is intruded along the contact of two stratigraphic packages, the fine grained, volcanic to volcaniclastic hanging wall package, and the medium to coarse grained siliciclastic and volcanic sediments of the footwall group. The contact between these packages dips approximately 47° to 275°. The quartz diorite intrusion has been dated to 2094 Ma (± 4 Ma).

The quartz diorite is interpreted as being intruded, at least in part, along the contact of the eastern package of medium to coarse grained siliciclastic between the eastern domain coarse grained sedimentary, and western domain fine grained volcanic to volcaniclastic rocks. The contact of the quartz diorite and the wall rocks and are sharp, often accurately measurable.

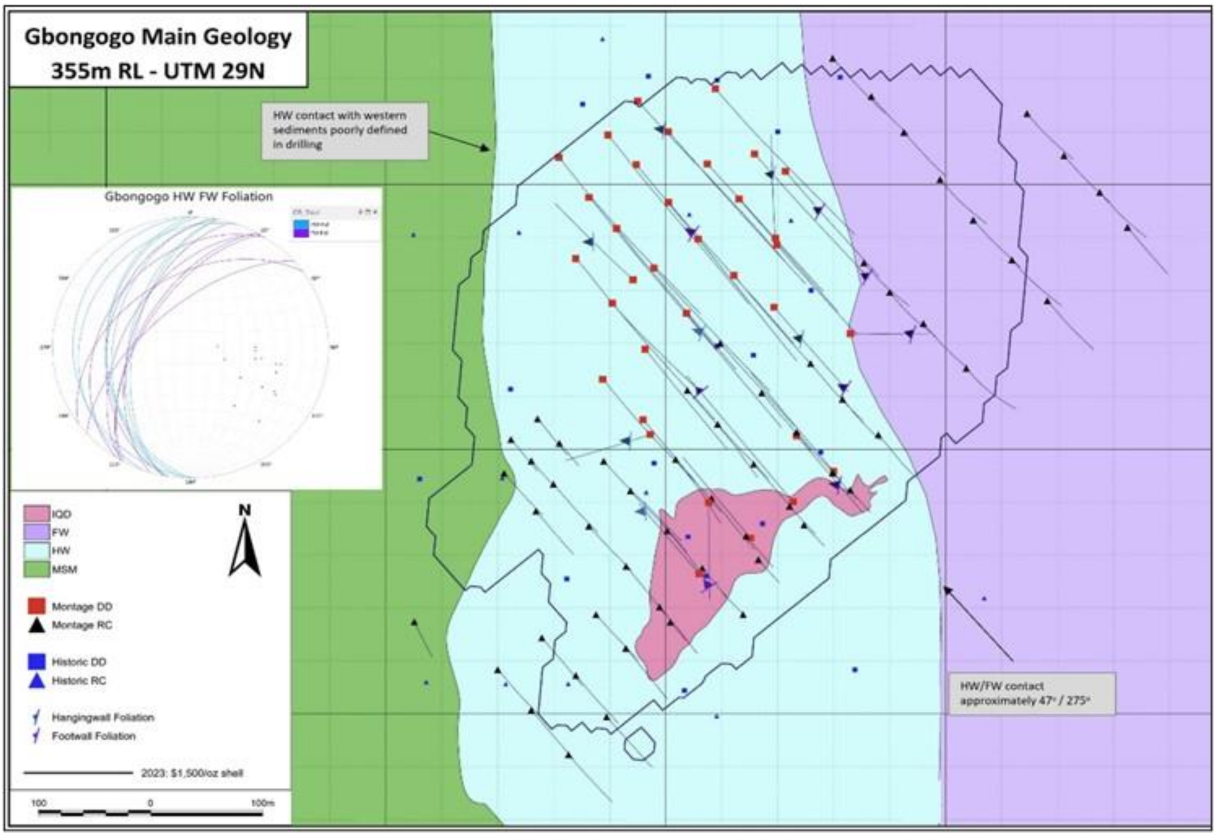
Two volcaniclastic domains form the footwall and hanging wall of the deposit.

The hanging wall package consists of very fine-grained rhyolitic rock, and calcite altered mafic dykes. The footwall unit is composed of coarse-grained volcanic sandstones, monomict and polymict conglomerates, and porphyritic diorites. The contact between these two domains is interpreted as a north-south striking thrust, with the older western hanging wall being thrust over the younger eastern footwall.

No cross cutting intrusions have been observed in the Gbongogo quartz diorite, with all cross cutting intrusive rocks constrained to the surrounding volcaniclastic rocks.

Gold mineralisation in the Gbongogo quartz diorite is associated with a stockwork of quartz / tourmaline / quartz-tourmaline veins, and their associated orthoclase-albite alteration haloes. The deposit is believed to be controlled by the rheological contrast of the quartz diorite host and surrounding volcaniclastics, where accommodation space for mineralisation formed by brittle deformation of the quartz diorite.

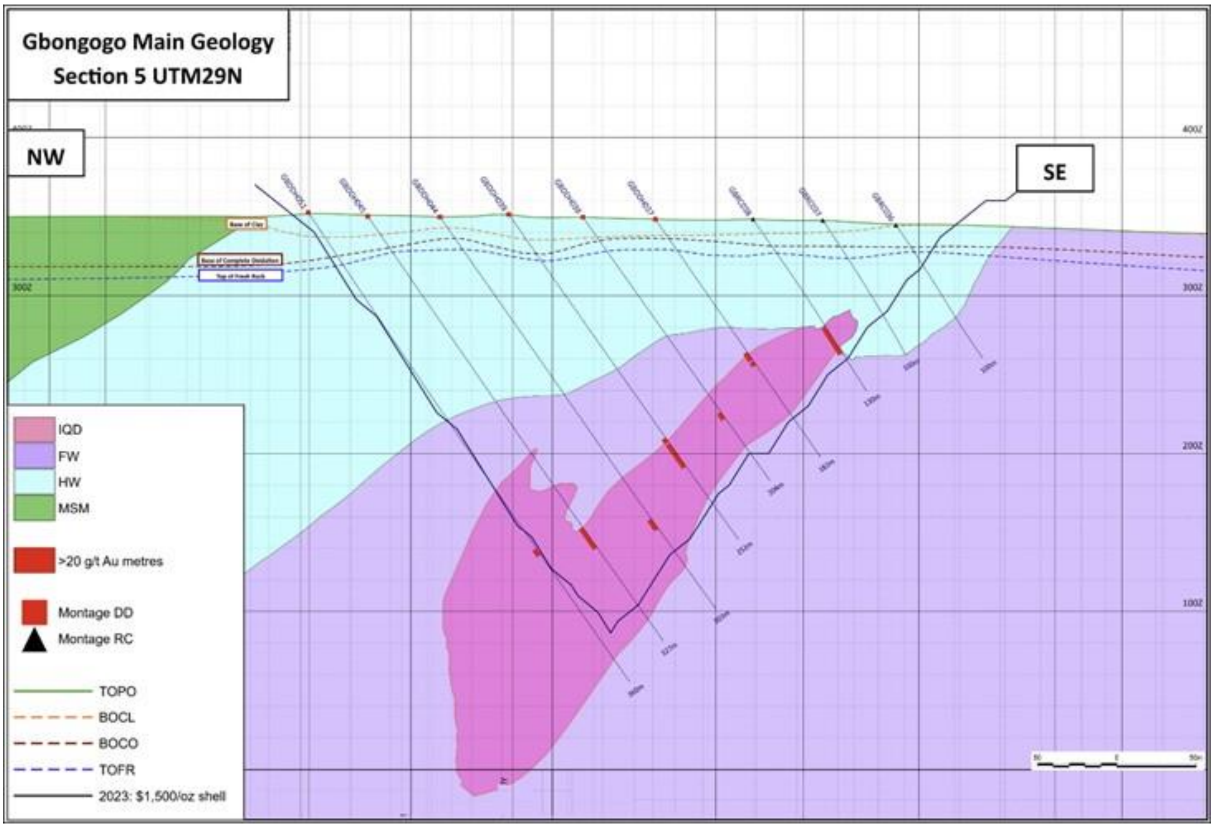
Figure 7.4.1 Gbongogo Surface Expression



Source: Montage, January 2024.

The Gbongogo quartz diorite is composed of a single north plunging, north-south trending intrusive plug. In perpendicular section, maximum dimensions of approximately 160 m x 110 m is tested to a vertical depth of 320 m, and is open at depth. The quartz diorite intrusion hosts the majority of gold mineralisation at Gbongogo. Drilling of the quartz diorite typically intersects the upper and lower contacts once, however small intersects occur on the hanging wall side of the body, believed to be small fingers of intrusion coming off the main body. Gold mineralisation is hosted within a network of quartz tourmaline veins.

Figure 7.4.2 Gbongogo Cross Section with Schematic Geology



Source: Montage, January 2024.

7.4.1 The Gbongogo Quartz Diorite Intrusion

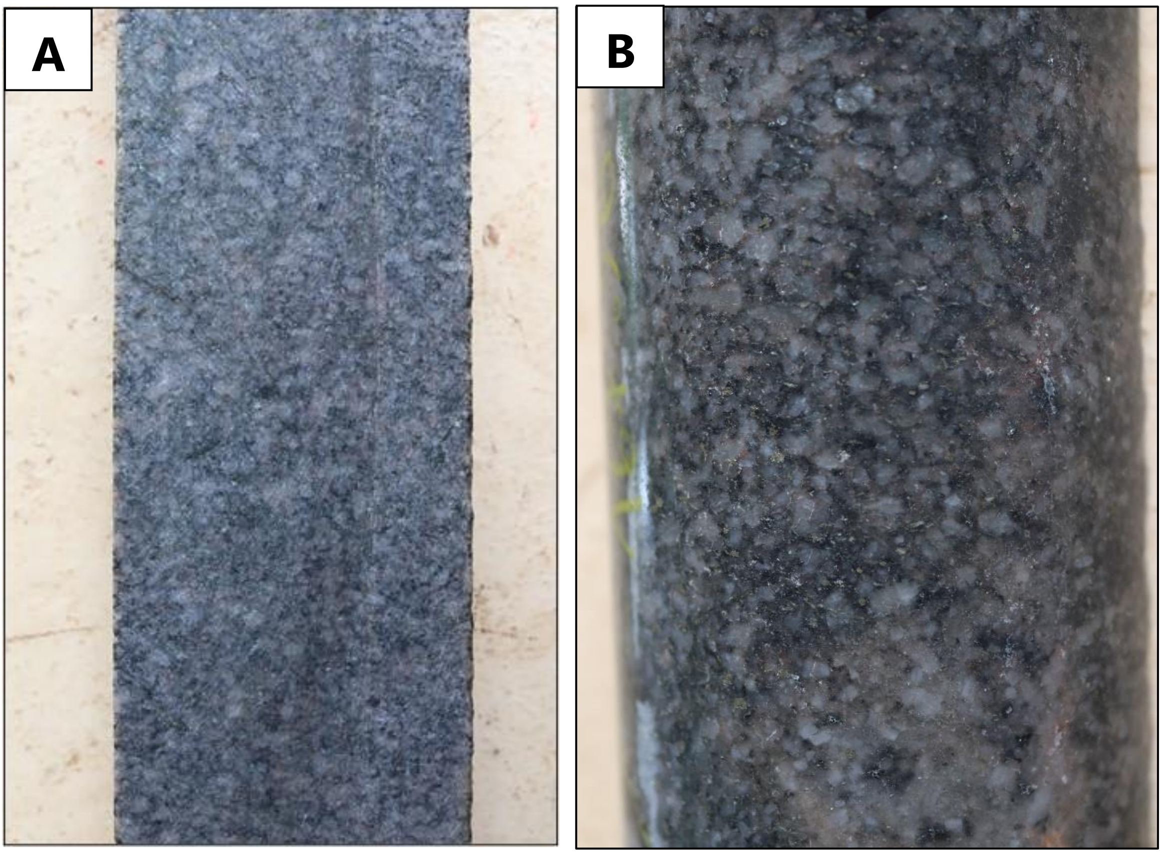
Quartz Diorite (IQD)

The quartz diorite is composed of 1 to 5 mm euhedral crystals of plagioclase feldspar (55%), quartz (25%), orthoclase feldspar (3 to 5%), white micas (3 to 5%), pyrite (2 to 5%), calcite (1 to 3%), and lesser mafic accessory minerals (Figure 7.4.3). Pale red to orange orthoclase feldspar is found in increased abundance in the alteration selvage of quartz tourmaline veins (Figure 7.4.3). The quartz diorite is unfoliated.

The intrusion is interpreted to have been emplaced in a single event as no internal contacts which would be indicative of a multi-phase intrusion have been observed, and multi-element pXRF analysis of drill core shows broadly homogeneous geochemistry and no sharp internal variations in the IQD.

Contacts with country rock are sharp and often confidently measurable. Analysis of these contacts indicates the surface of the intrusive is undulating and does not form a perfectly smooth surface.

Figure 7.4.3 Quartz Diorite, Gbongogo



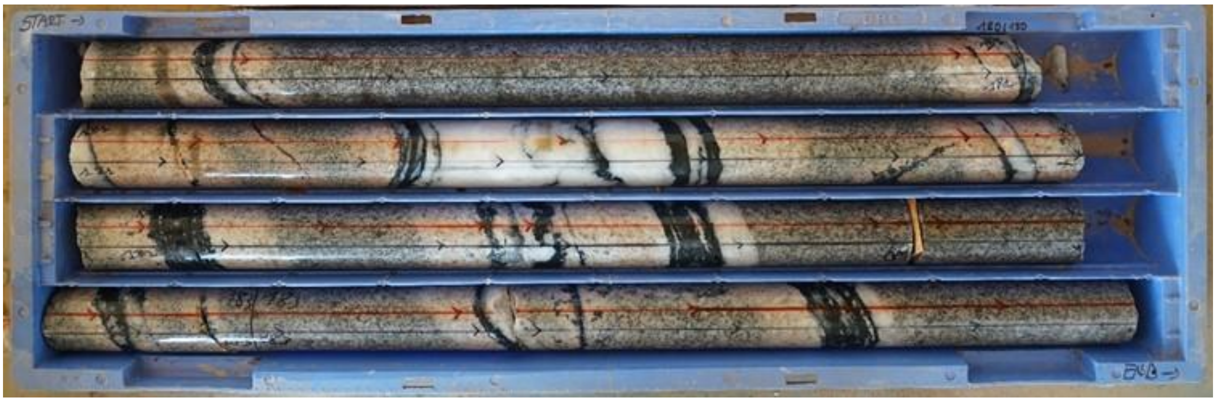
A. Quartz diorite **B. Phaneritic texture of the quartz diorite**

Source: Montage, January 2024.

Gbongogo Vein Stockwork

Gold mineralisation within the quartz diorite is closely associated with a stockwork of mm to cm scale quartz / tourmaline / quartz-tourmaline veins (Figure 7.4.4). The host diorite rock is unmineralised, and gold mineralisation directly correlates with the quartz tourmaline vein stockwork. Higher gold grades are associated with an increase in vein density.

Figure 7.4.4 Mineralised Quartz Diorite, Gbongogo



Source: Montage, January 2024.

7.4.2 Gbongogo Hanging Wall

Rhyolite – VRH

The dominant hanging wall lithology is a silicious rhyolite. It varies from mid to dark grey, to light to medium beige in colour, is extremely fine to fine grained, and often displays fine (sub mm) quartz laminations (Figure 7.4.5).

Small (0.5 to 1 mm) blue coloured quartz grains are common but not diagnostic. This unit is often moderately to strongly magnetic, and occasionally has visible magnetite crystals.

In core some intervals of the rhyolite are strongly deformed, folded, and occasionally display strong sericite–ankerite–hematite alteration. The rhyolite is generally unmineralised, with only small intervals of a few metres hosting above background grades.

Figure 7.4.5 Rhyolite, Gbongogo

Source: Montage, January 2024.

Mafic Dyke (IMD)

The subordinate hanging wall lithology are narrow mafic dykes. They are grey-green in colour, fine grained, commonly very strongly foliated / laminated with small scale (mm to cm) 'chaotic' deformation on a mm scale. In core it almost always has calcite veinlets (Figure 7.4.6).

The mafic dyke lithology is interbedded with the rhyolite on 5 cm to 5 m intervals. Contacts are typically sharp, upper and lower contacts generally parallel, and are sub parallel to foliation. It is not observed in the footwall domain, or within the IQD.

The mafic dyke unit is generally unmineralised, with only small intervals of a few metres hosting above background grades. The frequency and volume of IMD increases with proximity to hanging wall / footwall contact.

Figure 7.4.6 Mafic Dyke, Gbongogo

Source: Montage, January 2024.

7.4.3 Gbongogo Footwall

Porphyritic Diorite (IDIP)

The porphyritic diorite is dark grey to dark green in colour, 1 to 5 mm feldspar crystals in dark, fine grained matrix. It contains rounded to angular plagioclase crystals 65%, up to 10% quartz (Figure 7.4.7). This unit is only found in the footwall and can be interbedded with the SSC.

Figure 7.4.7 Porphyritic Diorite, Gbongogo

Source: Montage, January 2024.

Sandstone (SSC)

Medium to coarse grained sedimentary unit, variably beige to grey in colour siliciclastic with very occasional grains of blue quartz (not diagnostic) (Figure 7.4.8). Likely stratigraphically related to the SCO conglomerate. Upper contact with hanging wall is often sharp (<5 cm).

Figure 7.4.8 Sandstone, Gbongogo

Source: Montage, January 2024.

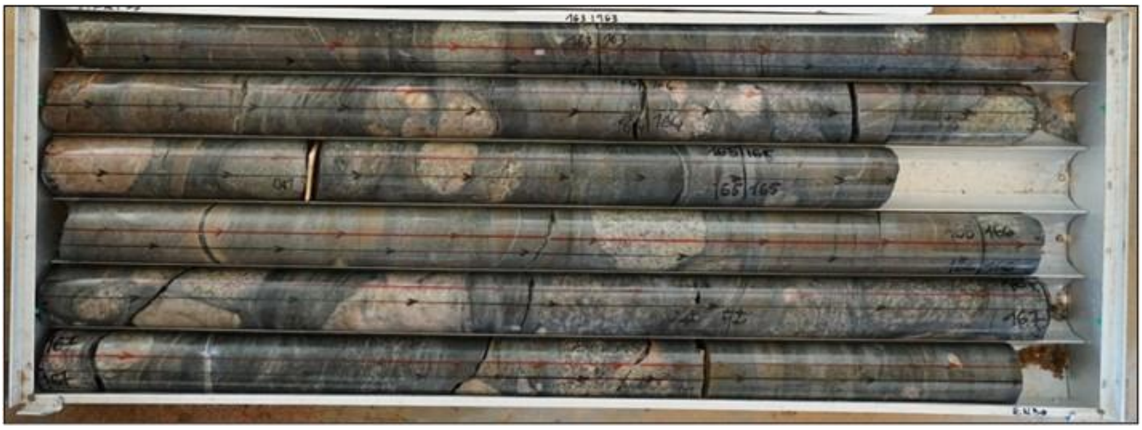
Conglomerate (SCO)

The footwall conglomerate is a matrix supported, polymict (occasionally monomict) conglomerate. Conglomeratic clasts are gravel to cobble size, sub-rounded to very well rounded (Figure 7.4.9 and Figure 7.4.10). Lithologies represented in the clasts include metasediments, possible crystal-ash tuffs, diorite and granite. The matrix is composed of fine to medium, poor to well laminated siliciclastics, often displaying foliation wrapping around the larger clasts. Occasionally blue quartz is visible in the matrix.

Figure 7.4.9 Conglomerate, Gbongogo

Source: Montage, January 2024.

Figure 7.4.10 Conglomerate, Gbongogo



Source: Montage, January 2024.

Diorite (IDI)

Grey to dark grey coloured groundmass with dark 1 to 5 mm, elongated clots of mafic minerals (biotite, chlorite) accounting for up to 20% (Figure7.4.11). Forms intervals 20 cm to 1 m. Often within the VRH but also in the SCO unit. Contacts in drill core are normally sharp. Volumetrically small in terms of the deposit.

Figure7.4.11 Diorite, Gbongogo

Source: Montage, January 2024.

7.4.4 Gbongogo Mineralisation

The majority of mineralisation of the Gbongogo deposit is constrained to within the Gbongogo quartz diorite intrusion (Figure 7.4.12). Minor gold mineralisation occurs within the volcano-sedimentary country rocks and along the contact with the intrusion.

Mineralisation correlates with the presence of a quartz and/or tourmaline stockwork and alteration selvage. An absence of vein stockwork and alteration is an indicator of lower or absent grade within the IQD. Mineralisation rarely extends into the volcano-sedimentary host rock, and when present is usually in the form of narrow tourmaline veinlets with pyrite and small alteration selvages.

Figure 7.4.12 Mineralised Quartz Diorite, Gbongogo

Source: Montage, January 2024.

8.0 DEPOSIT TYPES

The principal hosts of gold mineralisation in West Africa are the Lower Proterozoic greenstone belts, which comprise a collection of Paleoproterozoic metasedimentary and metavolcanic packages and associated intrusive complexes. The gold deposits are generally structurally-controlled, epigenetic lode or stockwork style mineralisation related to major shear zones.

The Koné deposit is considered to be a lode gold-style system, hosted by brittle ductile shearing within a quartz diorite / mafic volcaniclastic package. The Gbongogo gold deposit is a mesothermal, lithologically constrained deposit hosted by a plunging quartz diorite that has undergone brittle deformation. As such, both are typical of the region.

9.0 EXPLORATION

9.1 Exploration by Red Back Mining, Sirocco Gold, Orca Gold and Montage

Exploration completed by Red Back, Sirocco, Orca and Montage in the Project area since 2009 has been carried out using the same methodology and under the supervision of the same senior geological personnel. Table 9.1.1 shows a summary of all exploration completed by Montage.

Table 9.1.1 Exploration Activities to Date

Activity	Red Back 2009 to 2010	Sirocco 2013to 2014	Orca 2017 to 2019	Montage 2019 to 2023
Satellite Imagery Acquired				
Worldview Imagery (km^2)	230	-	-	3,400
Ground Geophysics				
Ground Magnetics (km^2)	4.68	-	-	46
Induced Polarisation (km^2)			104.7	50
Surface Sampling				
Soil Samples	5,095	-	2,469	15,208
Rock Chip Samples	61	2	99	-
Trenching (metres)	4,155	610	-	166
Pitting (m)			1,580	92

During the second half of 2009, Red Back completed 800 m x 50 m spaced soil sampling with subsequent local infill to 400 m x 50 m and 200 m x 50 m spacing identified a 2.7 km long + 75 parts per billion (ppb) gold in soil anomaly at Koné.

The anomaly was tested in 2010 by 200 m spaced hand dug trenches. 2 m channel samples were taken from the cleaned floor of the trenches and a 2.0 to 2.5 kg sample submitted to SGS (2009 to 2010) and Bureau Veritas (2013) for analysis. The results showed broad zones of mineralisation (best intercept 202 m grading 1.1 g/t) which justified exploratory drilling. In 2013, Sirocco completed three trenches for a further 610 m 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 resource area. Pits were dug at an average spacing of 50 m x 200 m to an average depth of 5 m and the north wall of the pit sampled. Samples were submitted to Bureau Veritas in Abidjan for analysis for gold by fire assay. Samples from only three pits returned gold assay grades of greater than 0.5 g/t.

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 8 km east of the Koné deposit.

In 2022, following the issue of the Sisséplé North and Farandougou permits, 11,083 soil samples were collected on these permits based on an 800 m x 50 m grid, infilling in places to 200 m x 50 m. This sampling, along with geological mapping and the location of several areas of artisanal workings, led to the delineation of the TZ1, TZ2, TZ3 and Korotou targets.

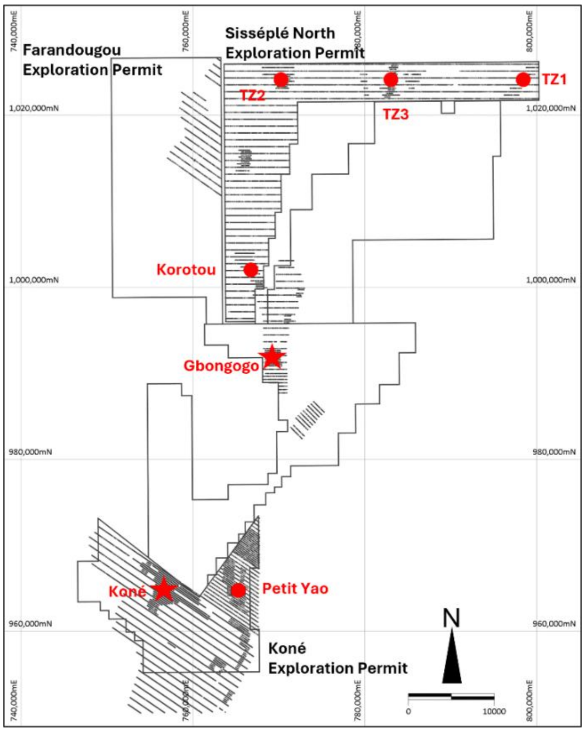
In 2023, exploration focus shifted to the newly acquired Gbongogo, Sissédougou and Sisséplé permits, where 2,103 soil samples were collected as checks against the historical soil sampling in those permits.

Soil sampling in 2022 and 2023 (Figure 9.1.1) was based on approximately 50 cm deep pits, from which a 1 kg sample was collected from the base. All samples were collected and transported to the field camp the same day under the supervision of a Field Geologist. Samples were analysed by Bureau Veritas in Abidjan.

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 QP 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.

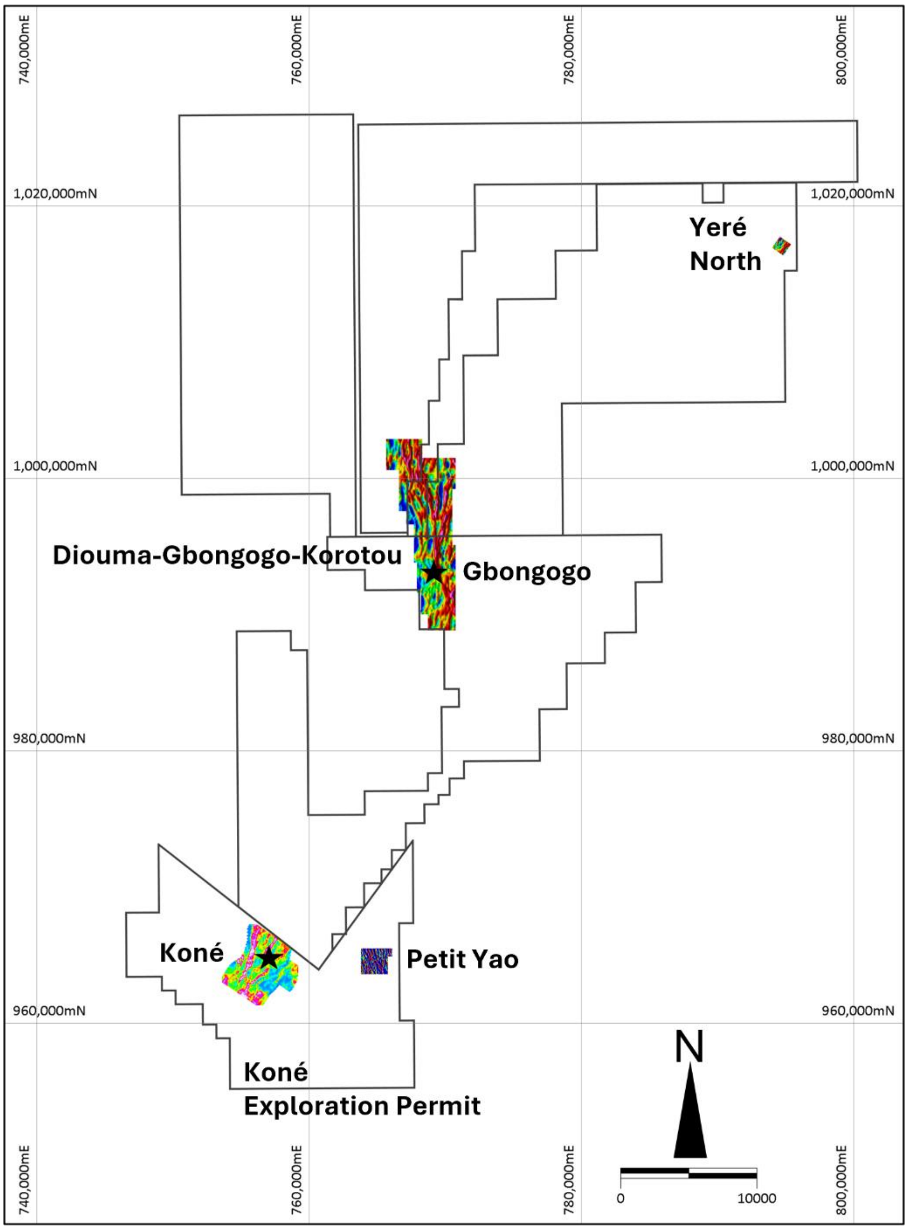
In early 2019, a gradient array induced polarisation survey was carried covering 104 line kilometres encompassing the Koné resource area. A second survey was completed over the Petit Yao target later that year and, in 2023, a larger survey was undertaken over the wider Gbongogo area (Figure 9.1.2) and the Yeré North prospect. The surveys used a line spacing of 100 m to 200 m and an electrode spacing of 25 m to 50 m. The surveys successfully aided in the mapping of the underlying stratigraphy.

Figure 9.1.1 Soil Sampling Distribution



Source: Montage, January 2024.

Figure 9.1.2 Induced Polarisation Surveys (Apparent Resistivity)



Source: Montage, January 2024.

9.2 Historic Exploration in the Gbongogo, Sissédougou and Sisséplé Exploration Permits 2012 to 2022

The Sissédougou and Sisséplé Exploration Permits were first granted to La Mancha Côte d'Ivoire, a precursor of Endeavour in the early 2000s, and the area explored intermittently until the formation of the Mankono JV in 2019.

The Gbongogo Exploration Permit area was first granted to Randgold Resources (now Barrick) in 2004, but exploration did not begin until 2013.

Barrick and Endeavour formed the Mankono JV in 2019 to explore the three contiguous permits. Exploration was carried out by Barrick.

Montage acquired the permits in November 2022 and exploration completed since that time is covered in Section 9.1.

The following section is based on a re-compilation by Montage of data provided by Barrick and Endeavour for the area.

Available information indicates that both Barrick and La Mancha / Endeavour employed industry standard methods for sampling and analysis (Table 9.2.1). With the exception of the exploration sampling carried out prior to 2013, adequate information has been retrieved to underpin the validity of the data.

Table 9.2.1 Historic Exploration Activities in the Mankono JV Area

Activity	La Mancha / Endeavour 2009 to 2017	Barrick 2014 to 2021
Satellite Imagery Acquired		
Worldview Imagery (km^2)	-	-
Airborne Geophysics		
Electromagnetic / Magnetic / Radiometric (km^2)	-	1,083
Ground Geophysics		
Ground Magnetics (km^2)	3.7	-
Induced Polarisation (km^2)	3.7	
Surface Sampling		
Soil Samples	16,590	39,787
Rock Chip Samples	73	5,206
Trenching (metres)	1,260.7	15,837.2
Trench Pits (metres)	216.1	4,809.7
Pitting (metres)	-	5,714.2
Auger (metres)		14,524

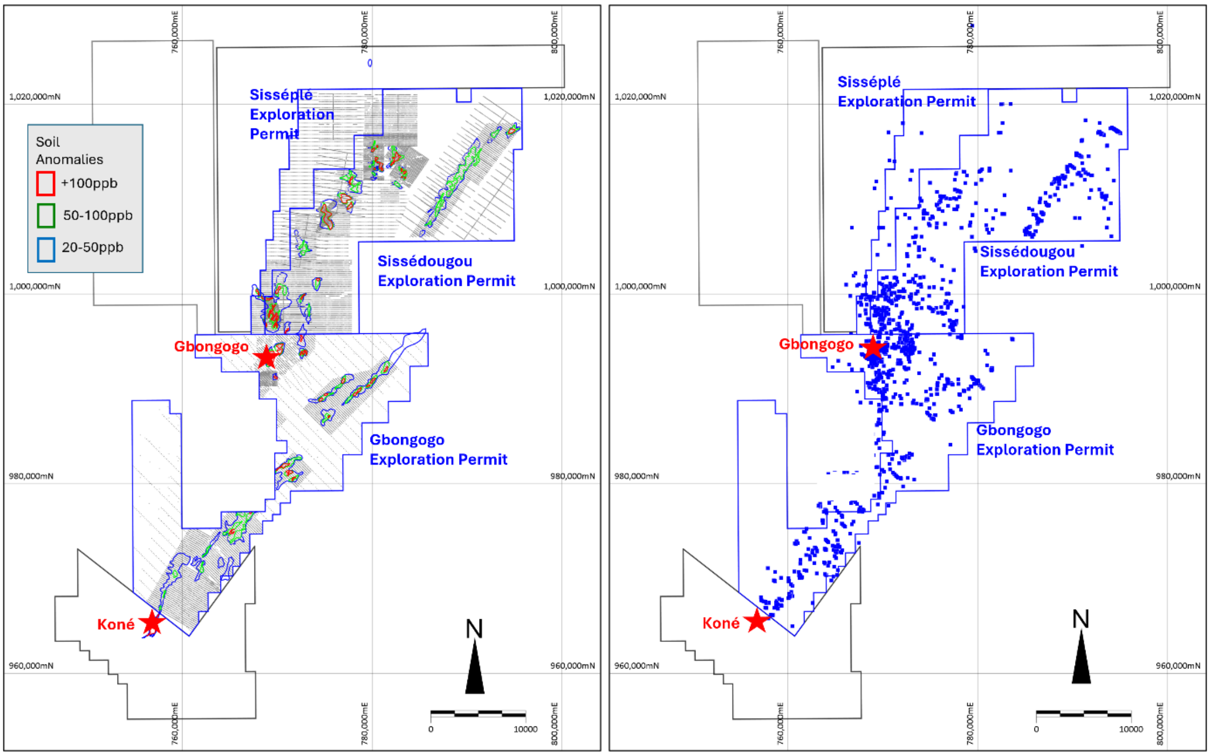
The Sisséplé and Sissédougou permits have been covered by numerous phases of soil sampling dating back to 2009 and carried out by La Mancha, Endeavour, and most recently under the joint venture supervised by Barrick geologists.

Sampling was based on a variety of grid spacings from 1 km to 200 m spaced lines with generally 50 m spacing along lines. Samples were taken from 50 cm deep pits and analysed by 50 g fire assay or aqua regia.

As shown in Figure 9.2.1 below, the programmes were successful in delineating the principal gold bearing zones within the Project area.

Rock chip sampling has been used extensively through the Project areas (Figure 9.2.1). Sampling was undertaken as a routine part of the soil sampling programmes, and also during reconnaissance and detailed geological mapping. Samples were analysed by fire assay and in some programmes for a multi-element suite.

Figure 9.2.1 Distribution of Historic Soil and Rock Chip Sampling within the Mankono JV Area

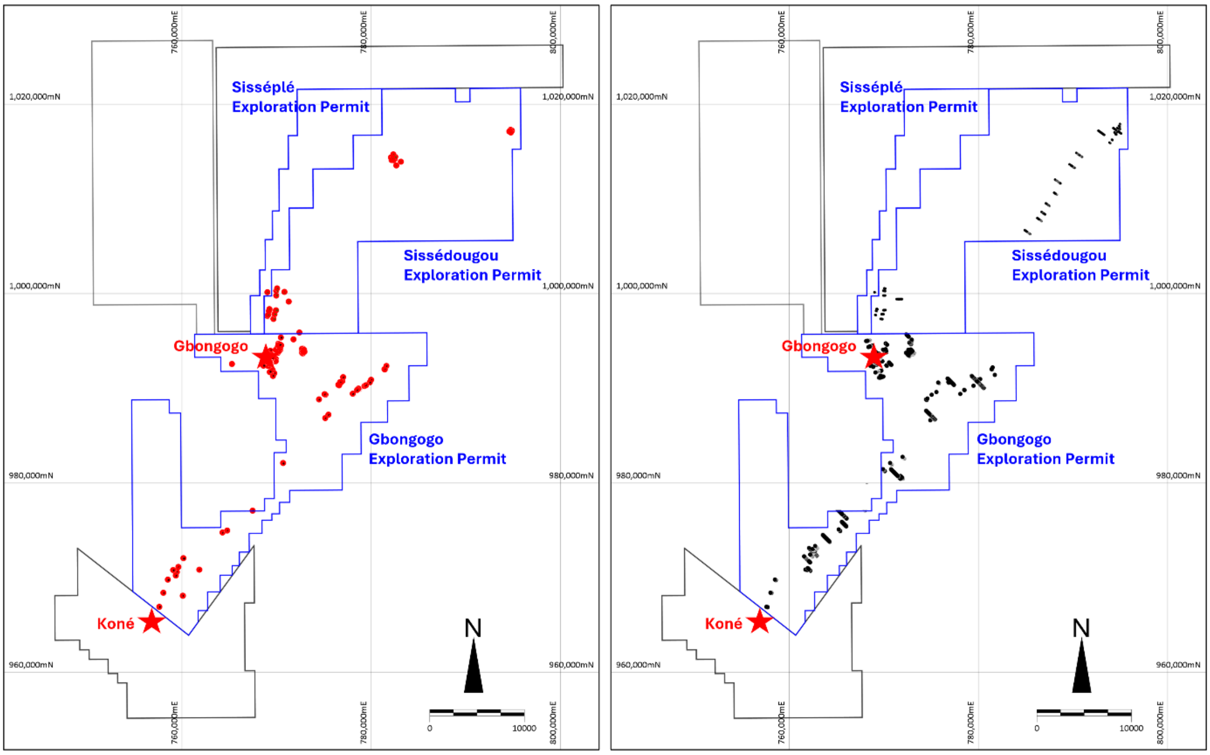


Left: Historic soil. Right: Historic rock chip.
Source: Montage, January 2024.

Trenching was used throughout the Project area (Figure 9.2.2), principally by Randgold / Barrick as follow up to soil sampling surveys. In some areas, trenches were dug by hand to a depth of 3 m to 4 m but often using an excavator to depths of 5 to 6 m. Channel sampling was undertaken close to the base of the trench at 1 m or 2 m intervals and samples generally assayed by fire assay. Randgold / Barrick also employed channel sampling vertically down the trench walls (trench pits).

Hand dug pits were used routinely by Randgold / Barrick as follow up to soil sampling programmes (Figure 9.2.2). Pits were spaced at 50 m intervals along randomly spaced lines and channel sampled down the north side of the pit with samples analysed by fire assay.

Figure 9.2.2 Historic Trenching and Pitting within the Mankono JV Area

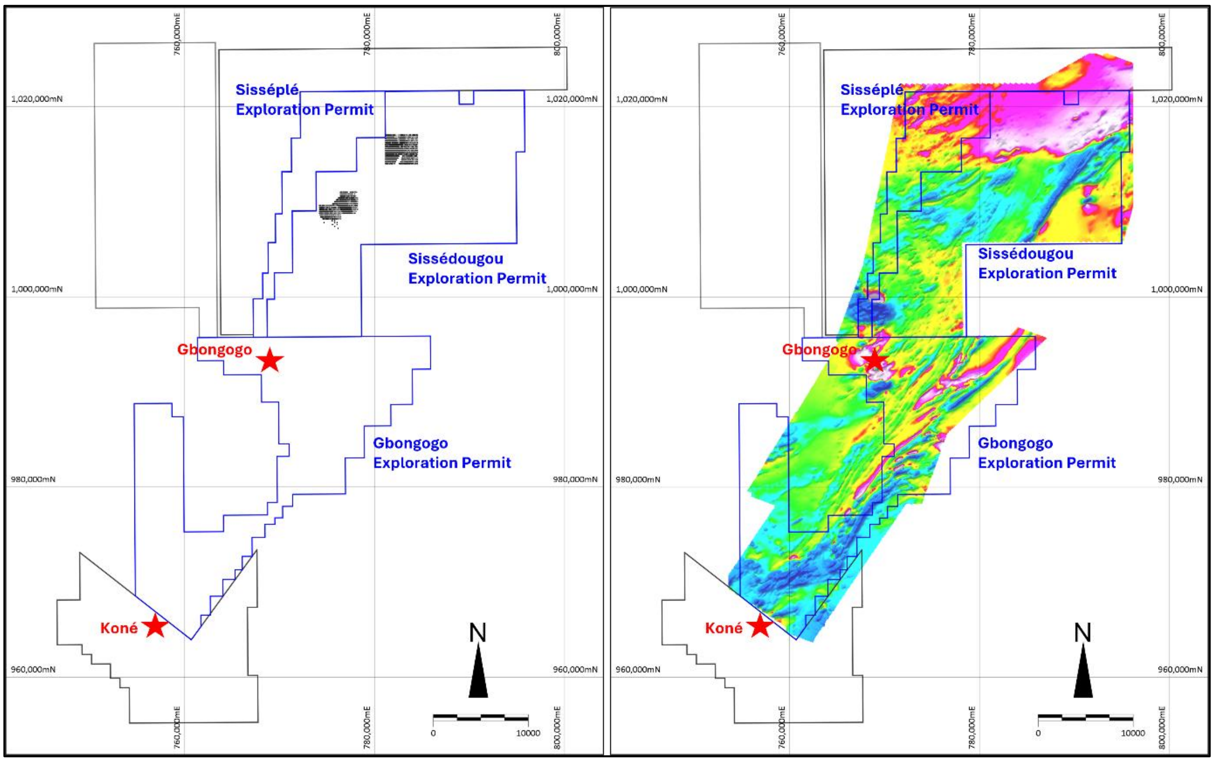


Left: Historic trenching. Right: Historic pitting.
Source: Montage, January 2024.

Two programmes of auger sampling were completed by Barrick over the Kagon soil anomaly and over the northern end of the Sissédougou prospect in 2019 (Figure 9.2.3). Auger holes were drilled to refusal, generally between 3 m and 12 m deep, and sampled at the base of laterite and end of hole and assayed by fire assay.

Barrick contracted two airborne geophysical surveys over the Project area covering the Gbongogo permit area in 2015 to 2016, and the Sissédougou and Sisséplé permit in 2018 (Figure 9.2.3). The surveys used a helicopter borne versatile time domain electromagnetic (VTEM) system with concurrent horizontal magnetic gradiometer and gamma ray spectrometry surveys. Line spacing was 200 m with 2 km tie lines and surveys were flown with a loop clearance of 40 m above ground level.

Figure 9.2.3 Historic Auger Sampling and Geophysical Coverage within the Mankono JV Area

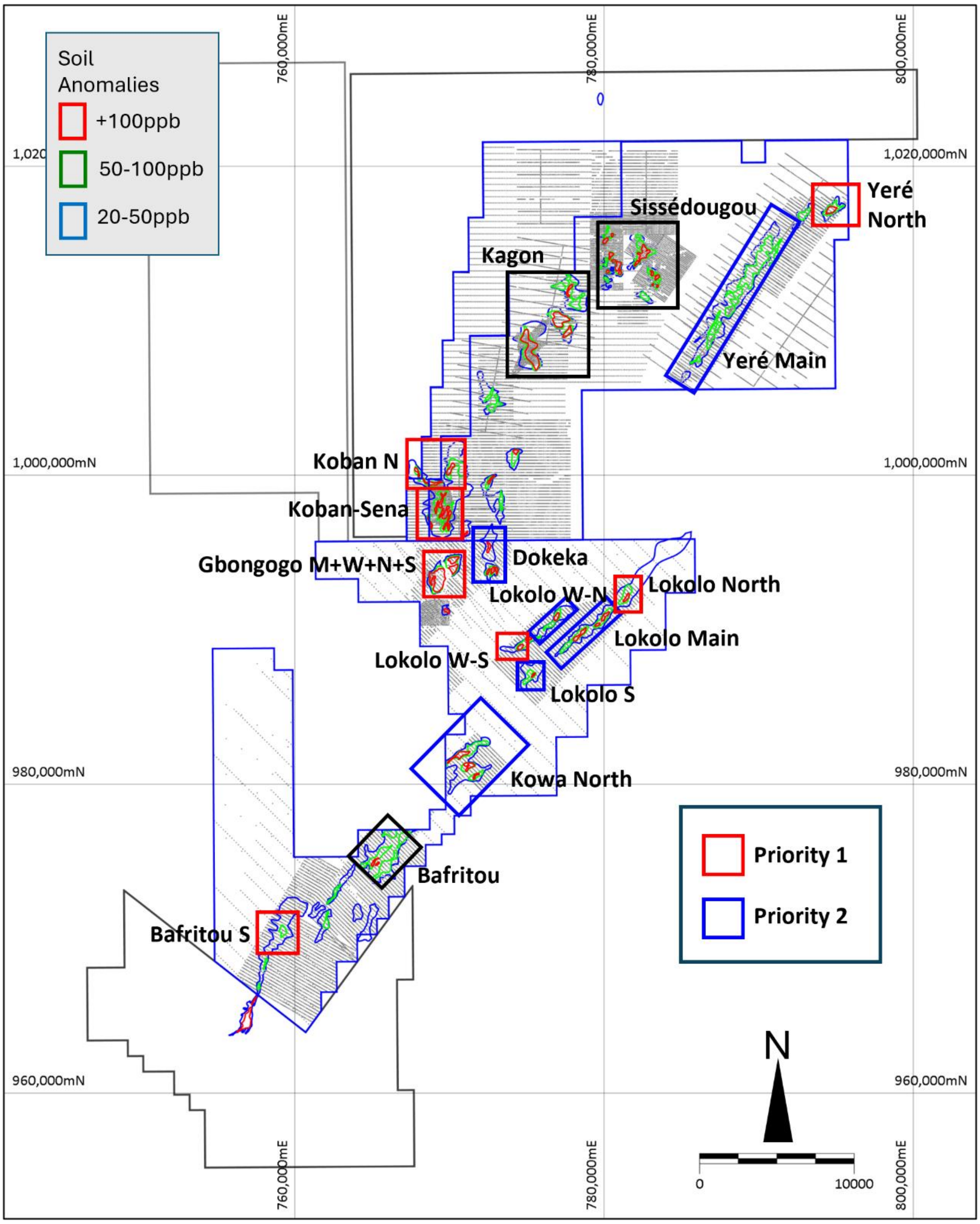


Left: Historic auger sampling. Right: historic geophysical coverage.
Source: Montage, January 2024.

9.3 Exploration Results

A detailed compilation of the historic exploration data by Montage outlined a series of exploration target areas (Figure 9.3.1). The level of follow up exploration completed varies by target with the most detailed work being carried out on the Gbongogo area and at Sissédougou. A number of targets have been explored further by reconnaissance aircore and in some cases RC and core drilling, which is discussed in Section 10.0.

Figure 9.3.1 Exploration Targets Delineated by Historic Exploration on the KGP



Source: Montage, January 2024.

10.0 DRILLING

10.1 Introduction

This section details the drilling completed on the Project. It is divided into three sections:

- Drilling carried out on the Koné Exploration Permit between 2010 and 2021, which includes the drilling completed on the Koné Mineral Resource.
- Drilling carried out by Barrick and Montage at the Gbongogo Mineral Resource and in the immediate area between 2016 and 2023.
- Reconnaissance drilling in the wider project area excluding the Koné Exploration Permit by Endeavour, Barrick and Montage between 2010 and 2023.

10.2 Koné Exploration Permit

As summarised in Table 10.2.1, drilling information available for Koné totals 353 RC and 50 diamond holes for 50,017 m. The RC drill metres shown in Table 10.2.1 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 MRE, Montage's drilling at 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 four holes have had samples submitted for gold analysis. Information from entirely un-assayed drillholes does not inform resource modelling.

Central portions of the currently interpreted Koné mineralisation 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°. These holes are generally spaced at around 50 m 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 KGP 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 mineralisation and returned several low tenor anomalies (<0.2 g/t Au). The 2020 reconnaissance drilling targeted the Petit Yao prospect and intersected narrow mineralised zones. The 2021 reconnaissance drilling tested potential mineralisation 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 mineralised intercepts.

Table 10.2.1 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 10.2.2 presents the number and proportion of assayed mineralised domain estimation dataset composites within the volume of mineralisation classified as Indicated or Inferred in the optimal pit constraining MRE by drilling type and year. This table provides an indication of the relative contribution of assays from each drilling group to MRE.

Table 10.2.2 Koné Mineralised Domain Composite Estimation Dataset within Resource Volume by Drilling Group

Year	Number of Composites			Proportion of Composites		
	RC	Diamond	Total	RC	Diamond	Total
2010	290		290	1%	0	1%
2013	1,231		1,231	4%	0	4%
2017	1,043		1,043	4%	0	4%
2018	3,847	231	4,078	14%	1%	15%
2019	800	754	1,554	3%	3%	6%
2020	1,971	3,689	5,660	7%	13%	21%
2021	3,618	10,051	13,669	13%	37%	50%
Total	**12,800**	**14,725**	**27,525**	**47%**	**53%**	**100%**

10.2.1 Koné RC Drilling

Drilling and Sampling Procedures

The RC drill rigs (Figure 10.2.1) generally utilised 140 mm (5.5 inch (")) face sampling bits. Samples were collected over 1 m downhole 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 3 kg 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 3 kg 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 was 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 1 m RC samples were submitted for analysis, with the exception of selected samples from the 2013 RC drilling, which were composited over 2 m intervals for assaying.

Field geologists geologically logged all RC holes over 1 m 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 10.2.1 Drilling at Koné in 2013

Source: Montage, 2024.

Collar and Downhole Surveying

Drillhole locations prior to 2018 were set out using a handheld global positioning system (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 drillhole 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 downhole surveyed as follows:

- 2010 holes were generally surveyed with a single shot Camteq Pro shot instrument at intervals of around 30 m.
- 2013 holes were generally surveyed at intervals of around 80 m with a Reflex Ez-Trac single-shot survey tool (Reflex).
- 2017 holes were generally surveyed at intervals of around 40 m with a Reflex tool. =
- 2018 RC and diamond holes were generally surveyed at intervals of around 30 m with a Reflex tool.
- 2019 and 2020 Koné holes were generally surveyed with a Reflex tool at intervals of around 10 to 20 m.
- 2021 Koné holes were generally surveyed with a gyro tool at intervals of around 10 to 25 m, and less commonly with Reflex tool at intervals of around 10 to 35 m.

- Reconnaissance holes were generally not downhole surveyed.

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

Sample Representivity

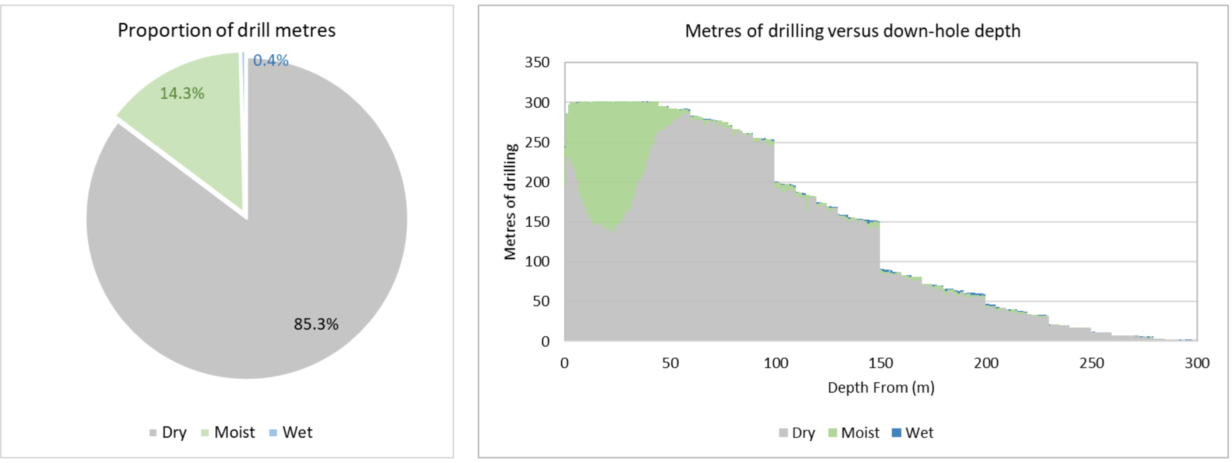
RC Sample Condition

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 10.2.2 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 10.2.2 Sample Condition Logging for Koné RC Drilling



Source: Montage, January 2024.

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 MRE.

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 QPs experience, this is likely to result in some overstatement of average recoveries for oxidised and fresh samples.

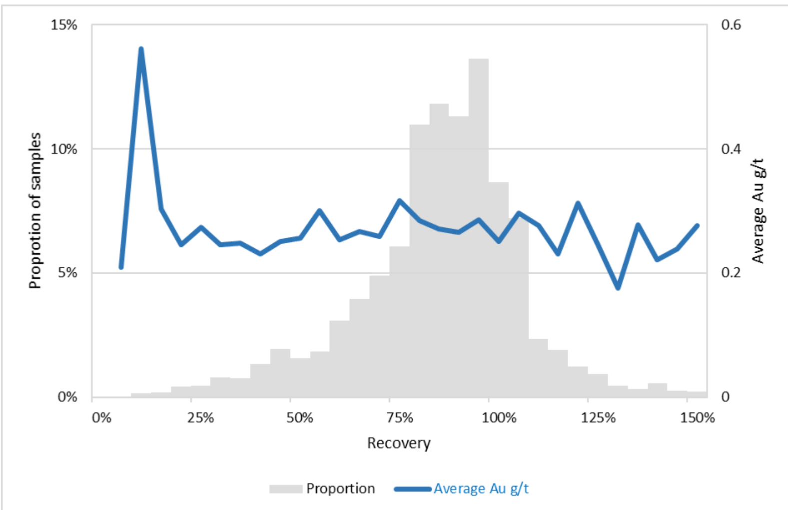
Table 10.2.3 summarises RC sample recovery estimates by logged sample condition, and Figure 10.2.3 shows average gold grade for increments of sample recovery. Notable features of this table and figure include the following:

- 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 sample.
- There is no notable association between estimated recovery and average gold grade.

Table 10.2.3 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 10.2.3 Gold Grade vs. Sample Recovery for RC Drilling



Source: Montage, January 2024.

10.2.2 Diamond Drilling

Drilling and Sampling Procedures

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

All onsite 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 1 m intervals (minimum 0.05 m) 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 mineralisation routinely recorded, along with foliation, cleavage, faulting and veining, including structural measurements of these features.

Collar and Downhole Surveying

Drillhole locations were set out using a handheld GPS and after that by 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 drillhole name, was placed at the collar. After drilling, all diamond hole collars were surveyed using DGPS equipment, with downhole surveying as follows:

- 2018 holes were generally surveyed at intervals of around 30 m with a Reflex tool.

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

The QP considers that hole paths have been located with sufficient accuracy for the MRE and exploration activities.

Sample Representivity

To provide a consistent basis for analysis, measured core recoveries for 0.1 m to 6.0 m core runs available for all resource area diamond holes were composited to 3 m 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 10.2.4) with only approximately 5% of composites showing recoveries of less than 90%. These recoveries are consistent with the QPs experience of high-quality diamond drilling.

Table 10.2.4 Core Recovery for 3 m Composites from Diamond Drilling

Oxidation Zone	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%**

10.3 Reconnaissance RC Drilling

Reconnaissance RC holes were inclined at 50° or 55° at orientations and hole spacings reflecting interpreted local mineralisation trends and previous exploration sampling. Hole spacings vary from rarely around 20 m to around 200 m 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 mineralisation and returned several low tenor anomalies (<0.20 g/t Au). The 2020 reconnaissance drilling targeted the Petit Yao prospect and intersected narrow mineralised zones.

The 2021 reconnaissance RC drilling included 18 holes for 1,823 m targeting potential mineralisation 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 1,832 m of shallow drilling with average hole depths of 39 m and the 1,806 m of deeper RC drilling up to 114 m depth returned several mineralised intercepts.

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

10.4 Gbongogo

The compiled database of Gbongogo drilling by Montage and Barrick totals 91 RC and 62 diamond holes for 26,907.3 m (Figure 10.4.1).

In addition to Gbongogo area drilling which informs MRE, the drilling includes drillhole testing exploration targets in the immediate area of Gbongogo and six Montage diamond holes drilled for geotechnical investigations, for which no samples have been submitted for gold analysis.

Central portions of interpreted Gbongogo mineralisation have been tested by generally 50 m spaced northwest-southeast traverses (140° UTM) of RC and diamond holes generally inclined to the southeast at around 55°. These holes are generally spaced at around 50 m along the traverses with relatively consistent coverage of each traverse extending to vertical depths of generally around 250 to 330 m. For peripheral proportions of the mineralisation intersected by drilling to date, on the margins of the deposit and at depth the drill spacing is notably broader.

For Montage drilling, field procedures and key staff were consistent for all 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 Montage RC and diamond drilling includes RC sample condition logs, recovered RC sample weights and core recovery measurements.

Available information indicates that Barrick RC and diamond drilling employed industry standard methods. Information available to demonstrate the reliability of sampling and assaying for this drilling includes assay results for coarse blanks, field duplicates and certified reference standards. Nearest neighbour comparisons of 2 m downhole composited gold assays from Barrick and Montage drilling show very similar average grades, supporting the general reliability the available for Barrick drill data.

The quality control measures adopted for Gbongogo RC and diamond drilling and reviews described in this report 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 Gbongogo deposit continues, and higher confidence resource estimates are targeted, additional investigations of sample reliability may be warranted.

The sample preparation, security, and analytical procedures adopted for the Gbongogo RC and diamond drilling provide an adequate basis for the MRE and exploration activities.

Table 10.4.2 presents the number and proportion of mineralised domain estimation dataset composites within the optimal pit constraining MRE by drilling group and assay laboratory. This table provides an indication of the relative contribution of information from each drilling and assay group to MRE. It demonstrates that Montage and Barrick drill samples, respectively, provide around three quarters and one quarter of estimation dataset composites for the main mineralised within the resource pit shell.

Table 10.4.1 Gbongogo Drilling Campaigns

Company	Drilling Group	Holes			Metres		
		RC	Diamond	Total	RC	Diamond	Total
Barrick 2016 to 2021	Reconnaissance areas	18	10	28	2,097.0	3,733.1	5,830.1
	Assayed resource drilling	9	18	27	1,195.0	4,827.4	6,022.4
	Subtotal	27	28	55	3,292.0	8,560.4	11,852.4
Montage 2022 to 2023	Reconnaissance areas	14	-	14	1,546.0	-	1,546.0
	Assayed resource drilling	50	28	78	4,924.0	7,329.9	12,253.9
	Geotechnical drilling	-	6	6	-	1,255.0	1,255.0
	Subtotal	64	34	98	6,470.0	8,584.9	15,054.9
Combined	**Reconnaissance areas**	**32**	**10**	**42**	**3,643.0**	**3,733.1**	**7,376.1**
	Assayed resource drilling	**59**	**46**	**105**	**6,119.0**	**12,157.3**	**18,276.3**
	Geotechnical drilling	**-**	**6**	**6**	**-**	**1,255.0**	**1,255.0**
	Total	**91**	**62**	**153**	**9,762.0**	**17,145.3**	**26,907.3**

Table 10.4.2 Gbongogo Resource Pit Mineralised Domain Composites

Phase	Assay Laboratory	Drilling Type	Number of Composites			Proportion of Composites		
			Hanging Wall	Footwall LG & HG	Combined	Hanging Wall	Footwall LG & HG	Combined
Montage	Bureau Veritas	RC	384	805	1,189	19%	32%	26%
		Diamond	1,281	1,107	2,388	62%	44%	52%
		Subtotal	1,665	1,912	3,577	81%	76%	78%
Barrick	Bureau Veritas	RC	60	71	131	3%	3%	3%
		Diamond	-	-	-	-	-	-
		Subtotal	60	71	131	3%	3%	3%
	SGS	RC	115	-	115	6%	-	3%
		Diamond	216	525	741	11%	21%	16%
		Subtotal	331	525	856	16%	21%	19%
	Subtotal	RC	175	71	246	9%	3%	5%
		Diamond	216	525	741	11%	21%	16%
		Subtotal	391	596	987	19%	24%	22%
Combined		**RC**	**559**	**876**	**1,435**	**27%**	**35%**	**31%**
		Diamond	**1,497**	**1,632**	**3,129**	**73%**	**65%**	**69%**
		Total	**2,056**	**2,508**	**4,564**	**100%**	**100%**	**100%**

10.4.1 Montage RC Drilling

Drilling and Field Sampling

RC drill rigs employed for Gbongogo drilling utilised 140 mm (5.5") face sampling bits. Samples were collected over 1 m downhole 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:

- 1 m downhole samples were collected from the cyclone in new 55 cm x 100 cm plastic sample bags labelled with the hole number and interval.
- After weighing, bulk samples were split with a three-tier riffle splitter with approximately 3 kg primary 'original' sub-samples collected in plastic bags which were sealed.
- The remaining bulk sample was riffle split a second time to produce an approximately 3 kg archive sample with the remaining bulk sample stored in the original bag.
- Duplicates were collected at pre-defined intervals at an average frequency of around one duplicate per 21 primary samples by passing the bulk sample through the riffle splitter a third time, producing another approximately 3 kg sub-sample for the interval.
- The riffle splitter was cleaned thoroughly between samples with compressed air.
- Sample 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.
- All sub-samples (original, archive and duplicate) were transported to the field office at the end of each shift, where the archive samples were stored and original and duplicates prepared for dispatch to the analytical laboratory.
- Bulk reject samples were left at the drill site in ordered lines until completion of the program, when the drilling sites were rehabilitated.
- All RC holes were geologically logged over 1 m intervals with logging information recorded on paper drill log sheets by field geologists including rock types, structures, quartz veining type and percentages, sulphide occurrence, and alteration type and intensity. Sieved samples were retained for future reference in plastic chip trays.

Collar and Downhole Surveying

For Gbongogo RC and diamond drilling, drill rigs were set at hole locations set out by DGPS equipment and aligned with designed orientations using compasses corrected for magnetic declination and inclinometers. Upon completion of the drilling, a cement marker, inscribed with the drillhole name, was placed at the collar and all collar location surveyed by DGPS.

RC holes were downhole surveyed at intervals of around 25 m with a Reflex Ez-Trac single-shot survey tool.

Hole paths of Gbongogo RC drillholes have been located with sufficient accuracy for the MRE and exploration activities.

Sample Representivity

Samples from Montage RC drilling contribute around 29% of the composites from the main mineralised domains hosting Mineral Resources (Table 10.4.2) and confidence in the reliability of these data significantly impacts general confidence in estimated resources.

Information available to demonstrate the representivity of Montage RC samples includes sample condition logging and recovered sample weights.

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

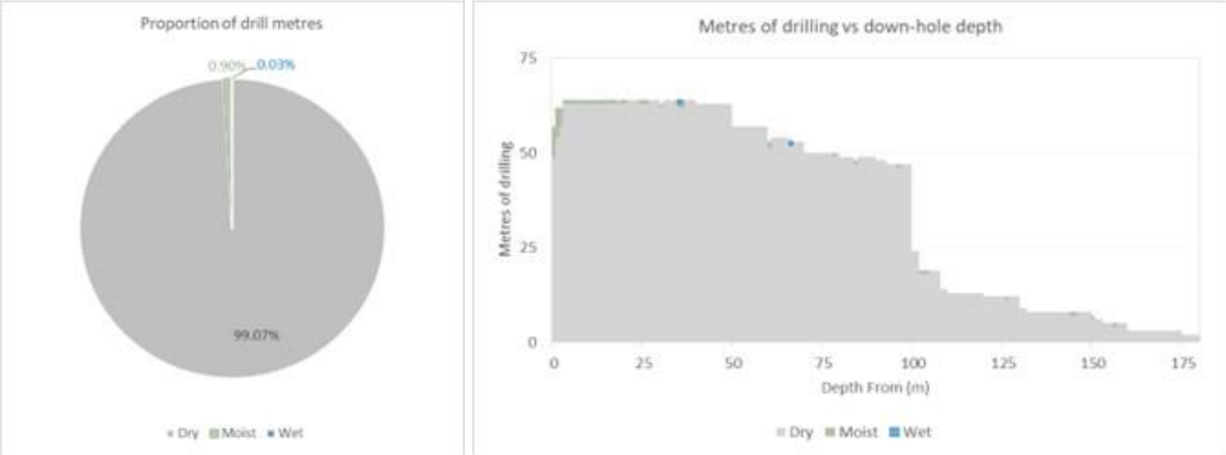
For all Montage RC drilling, field geologists recorded sample condition with samples classified as dry, moist, or wet. The summaries of sample condition logging in Figure 10.4.1 demonstrate that dry and wet samples provide only a small proportion of the RC drilling, and any uncertainty over the reliability of these samples does not significantly affect confidence in resource estimates.

In conjunction with bit diameters, density measurements, moisture content estimates (where available), and 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. The QPs experience indicates that 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.

Montage's Gbongogo RC drilling field procedures included weighing recovered sample material, with weights available for around 94% of this drilling including full coverage of drillholes informing MRE.

Sample recovery was estimated for each weighed sample from bit diameters specified by Montage with densities assigned by oxidation domain using the values used for resource estimates. No moisture content estimates are available for Gbongogo RC samples, and sample recovery estimates make no allowance for moisture.

Figure 10.4.1 Montage RC Drilling Sample Condition Logging



Source: Montage, January 2024.

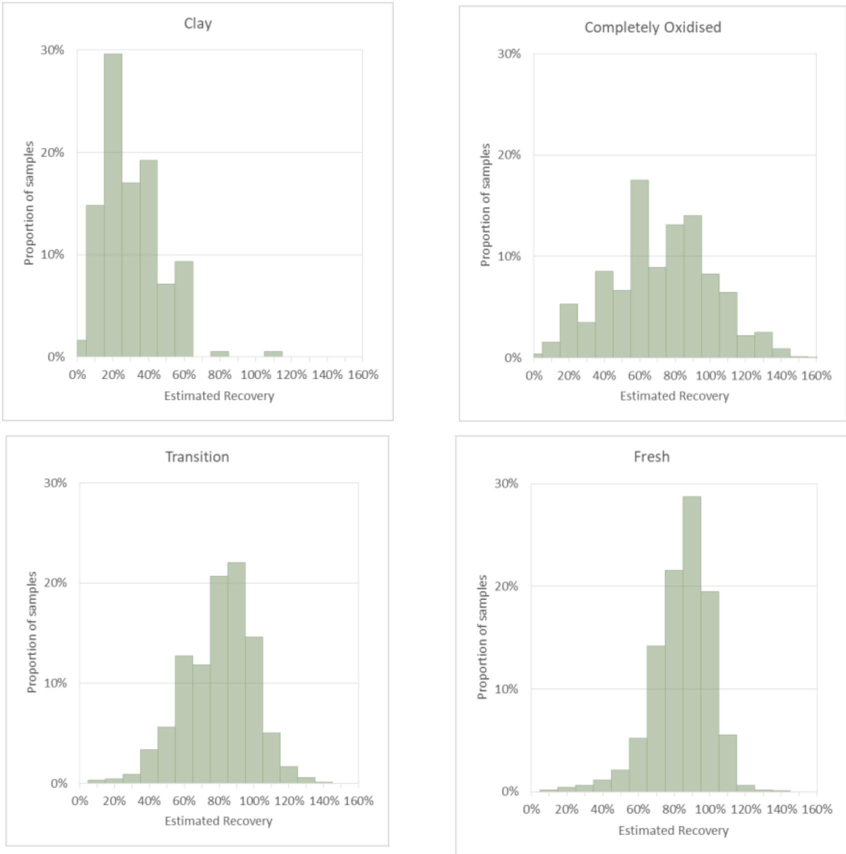
Table 10.4.3 and Figure 10.4.2 summarise estimated RC sample recovery for Montage Gbongogo RC samples by oxidation zone. Notable features of this table and figure include the following:

- At 80%, average estimated RC sample recovery is consistent with the good quality RC drilling.
- There is a general increase in estimated sample recovery with decreasing oxidation zone and average recoveries estimated for surficial clays and completely oxidized material are lower than shown by the QPs experience of high quality RC drilling. These oxidation zones contribute around 2% and 5% of Mineral Resources respectively (Table 10.4.3), and any uncertainty over the reliability of RC sampling for these zones does not significantly impact general confidence in MRE.

Table 10.4.3 Montage RC Drilling Sample Recovery Estimates

Oxidation Zone	Number of Samples	Average Recovery
Clay	182	30%
Completely Oxidised	1,431	71%
Transition	658	80%
Fresh	3,814	85%
Total	**6,085**	**80%**

Figure 10.4.2 Montage RC Drilling Sample Recovery Estimates



Source: Montage, January 2024.

10.4.2 Montage Diamond Drilling

Drilling and Sampling

Montage diamond drilling utilised triple tube core barrels where necessary to achieve high core recovery with generally 3 m drill runs and shorter runs where necessary to maximise core recovery. The drilling was conducted at PQ diameter (122.6 mm hole diameter) to depths of around 23 to 53 m, and HQ diameter (96 mm) for deeper drilling representing around 85% of the combined diamond drilling. Where possible, core was oriented using a Reflex ACT III tool.

All onsite core handling was supervised by Company geologists, with core handling and sampling procedures including the following:

- At the drilling site, core was placed directly in core trays which were 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 1 m intervals (minimum 0.04 m) assigned by logging geologists, respecting changes in rock units.

- Sampled half core was placed in plastic sample bags in sequence and securely stored before batch assignment and submission to the assay laboratory.

- Duplicate half core samples were collected at an average frequency of around one duplicate per 21 primary samples.

- 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, RQD, rock strength, and weathering recorded.

- After transport to the field office, core was geologically logged with rock type, stratigraphic subdivisions, alteration, oxidation, and mineralisation routinely recorded along with foliation, cleavage, faulting, veining including structural measurements of these features.

Collar and Downhole Surveying

For Montage diamond drilling, drill rigs were set at hole locations set out by DGPS and aligned with designed orientations using compasses corrected for magnetic declination and inclinometers. Upon completion of the drilling, a cement marker, inscribed with the drillhole name, was placed at the collar and all collar location surveyed by DGPS.

With the exception of a single down-dip hole drilled primarily for metallurgical sampling, which has only sparse downhole survey coverage, all Montage diamond holes were downhole surveyed with a Reflex gyro tool at 5 m intervals.

Hole paths of Montage diamond holes have been located with sufficient accuracy for the MRE and exploration activities.

Sample Representivity

Montage diamond core samples provide 47% of composites from the main mineralised domains hosting Mineral Resources (Table 10.4.4) and confidence in the reliability of these data significantly impacts general confidence in estimated resources.

Information available to demonstrate the representivity of Montage diamond samples includes measurements of recovered core lengths. To provide a consistent basis for analysis, the supplied measured core recoveries for 0.1 m to 3.8 intervals were composited to 3 m intervals reflecting the dominant length.

Table 10.4.4 and Figure 10.4.3Figure 10.4.3 summarise core recoveries for the 3 m composites, excluding the un-assayed geotechnical diamond holes which do not inform mineral resource modelling.

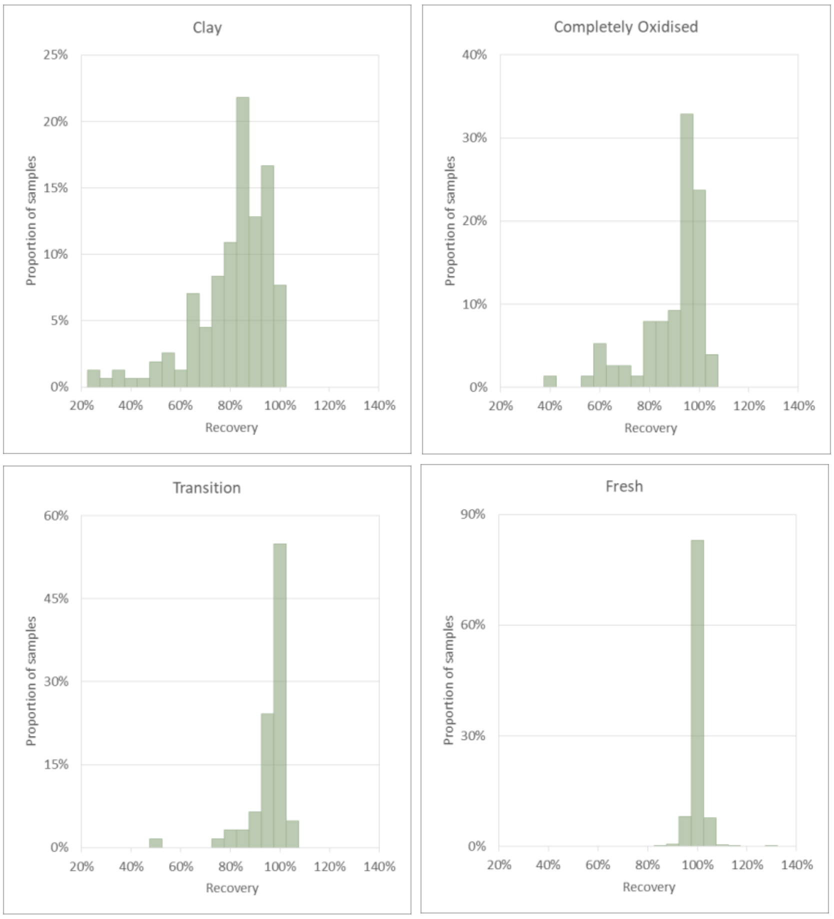
Core recoveries for the 3 m composites average 98.3% with only approximately 4% showing recoveries of less than 80%. These recoveries are consistent with the QPs experience of high-quality diamond drilling, and observations of Gbongogo core which shows generally high recoveries with very little core loss for fresh rock (Figure 10.4.4).

Although lower than for fresh and transitional rock, average core recoveries for the clay and completely oxidised intervals are within the range of reasonable quality diamond drilling. These oxidation zones contribute around 2% and 5% of Mineral Resources respectively, and any uncertainty over the representivity of Montage diamond core from these zones does not significantly impact general confidence in MRE.

Table 10.4.4 Montage Diamond Drilling Core Recovery

Oxidation Zone	Number of Composites	Recovery		
		Minimum	Average	Maximum
Clay	156	22.7%	81.0%	100.7%
Completely oxidised	76	39.0%	89.2%	105.7%
Transition	62	48.4%	95.7%	105.7%
Fresh	2,154	87.2%	100.0%	129.0%
Total	**2,448**	**22.7%**	**98.3%**	**129.0%**

Figure 10.4.3 Montage Diamond Drilling Core Recovery



Source: Montage, January 2024.

Figure 10.4.4 Montage Diamond Drilling Core Photographs

Source: Montage, January 2024.

10.4.3 Barrick RC and Diamond Drilling

Samples from Barrick RC drilling provide only 6% of estimation dataset composites from the main mineralised domains hosting Mineral Resources (Table 10.2.2) and confidence in the reliability of these data does not significantly impact general confidence in estimated resources. Barrick diamond drilling provides 18% of these composites, and confidence in the reliability of these data significantly impacts general confidence in estimated resources.

Available information indicates that Barrick RC and diamond drilling employed industry standard methods. RC holes were samples over 1 m downhole intervals. Diamond drilling was generally conducted at HQ (96 mm hole diameter), with core halved to provide assay samples over intervals ranging from 0.10 to 2.10 m, with a dominant sample length of 1 m.

Montage accurately surveyed the collars for 24 (89%) of Barrick Gbongogo drillholes by DGPS. The remaining three Barrick holes have handheld GPS collar surveys. Barrick RC drillholes were generally downhole surveyed with magnetic multishot tools at intervals of around 50 m. Barrick diamond holes were generally downhole surveyed by magnetic multi-shot tools, or less commonly single shot tools at average intervals of around 40 m.

Hole paths of Barrick Gbongogo drillholes have been located with sufficient accuracy for the MRE and exploration activities.

No core recovery records are available for Barrick diamond drilling. The QPs review of core photographs available for mineralised intervals of these holes shows consistently high recovery, with very little core loss for fresh material, consistent with Montage drill core (Figure 10.4.5).

Table 10.4.5 and Figure 10.4.6 summarise a nearest neighbour comparison of composites from the combined datasets of Montage and Barrick mineralised domain composites with maximum separation distances of 12 m x 25 m x 25 m (cross strike, strike, down-dip). These criteria give 634 composites with an average separation distance of 18 m, and provide a comparison of the general tenor of gold assays from Montage and Barrick drill samples.

Table 10.4.5 and Figure 10.4.6 demonstrate that, although as expected there is considerable variability for individual pairs, excluding rare high grade outliers, the paired composites show very similar average gold grades. Montage drill data are supported by comprehensive quality assurance information, and the nearest neighbour comparison supports the general reliability of information available for Barrick drilling.

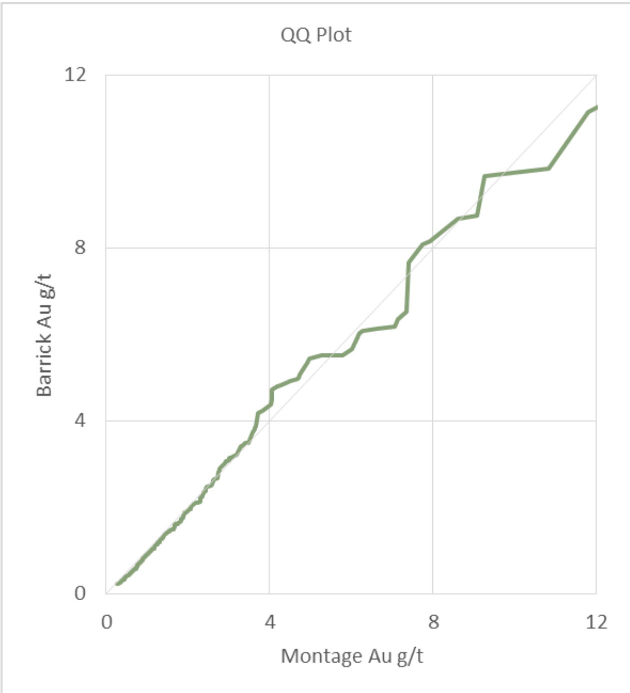
Figure 10.4.5 Barrick Diamond Drilling Core Photographs

Source: Montage, January 2024.

Table 10.4.5 Barrick vs. Montage Nearest Neighbour Comparison

Description	Full Set		< 15g/t	
	Montage Au g/t	Barrick Au g/t	Montage Au g/t	Barrick Au g/t
Number	634		624	
Average	1.36	1.28	1.08	1.08
Difference		-6%		0%
Variance	11.0	10.7	2.4	2.9
Coef. Var	2.44	2.55	1.43	1.59
Minimum	0.005	0.005	0.005	0.005
1st Quartile	0.19	0.16	0.19	0.16
Median	0.62	0.52	0.61	0.51
3rd Quartile	1.32	1.25	1.29	1.22
Maximum	48.1	50.7	12.8	14.7

Figure 10.4.6 Barrick vs. Montage Nearest Neighbour Comparison



10.4.4 Reconnaissance Drilling in the Wider Project Area

Table 10.4.6 below details the reconnaissance drilling completed in the wider Project area excluding the Koné Exploration Permit, the Koné Mineral Resource, and the area of the Gbongogo Mineral Resource.

Table 10.4.6 Drilling Completed in the Wider Project Area

Company	Drilling Group	Holes				Metres			
		AC	RC	Diamond	Total	AC	RC	Diamond	Total
La Mancha / Endeavour	Sissédougou Prospect	0	92	23	115	0	11,071.00	3,439.20	14,510.20
2010 to 2017	Reconnaissance Drilling	0	73	8	81	0	6,866.00	913.20	7,779.20
Barrick 2016 to 2021	Reconnaissance Drilling	1,031	22	0	1,053	31,207	2,778.00	0.00	33,985.00
Montage 2022 to 2023	Reconnaissance Drilling	1,028	100	12	1,140	21,758	10,950.00	2,402.80	42,110.80
Total		**2,059**	**287**	**43**	**2,389**	**62,965**	**31,665**	**6,755**	**101,385**

Endeavour RC and Diamond Drilling

Montage has not, to date, verified the validity of the drilling completed by Endeavour, although the available information indicates that Endeavour RC and diamond drilling employed industry standard methods.

Barrick Aircore and RC Drilling

Available information indicates that Barrick aircore and RC drilling employed industry standard methods. Information available to demonstrate the reliability of sampling and assaying for this drilling includes assay results for coarse blanks, field duplicates, and certified reference standards. Aircore and RC holes were samples over 1 m downhole intervals and assayed by fire assay.

Outside the Gbongogo resource areas, Montage has not verified the collar locations of drillholes which were generally surveyed by handheld GPS. Barrick RC drillholes were generally downhole surveyed with magnetic multishot tools at intervals of around 50 m.

Montage Aircore and RC Drilling

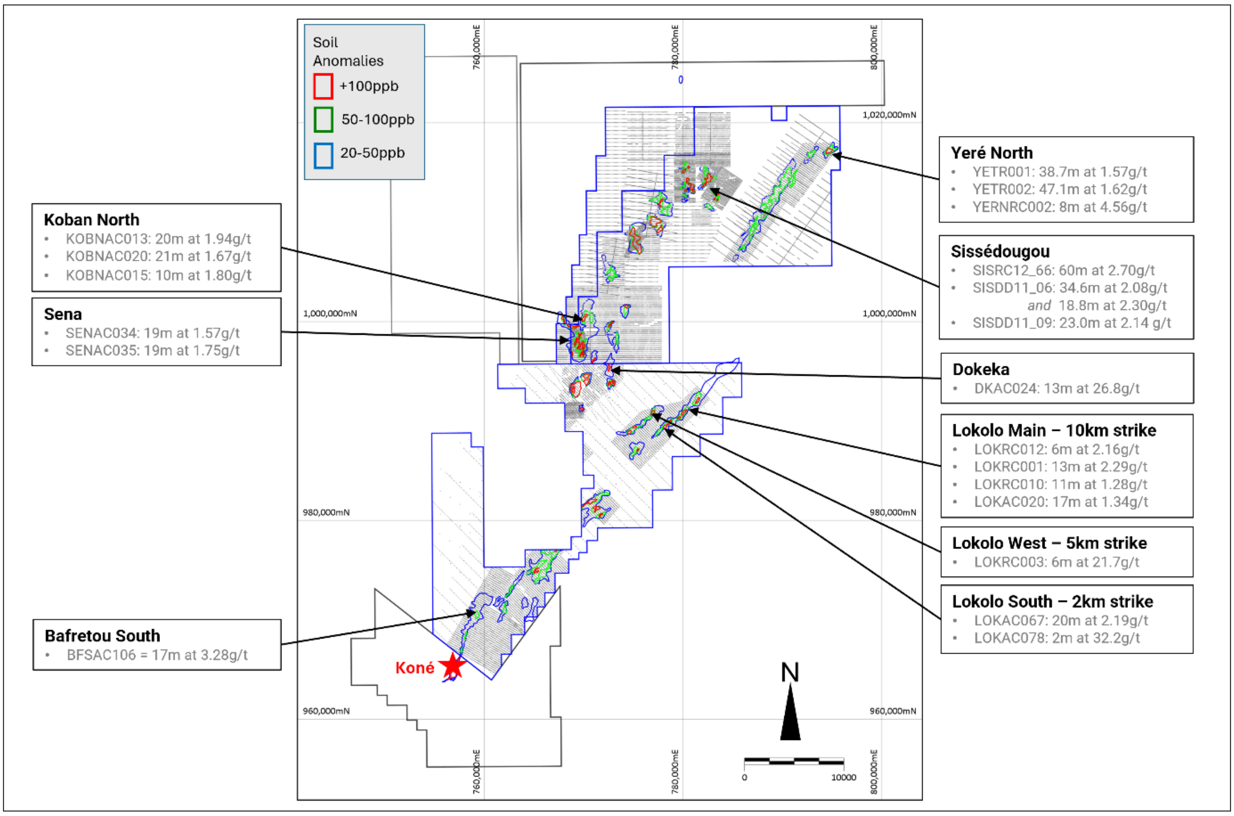
For Montage drilling, field procedures and key staff were consistent for all drilling phases, ensuring consistency in the sampling methodology. All field sampling activities were supervised by field geologists with industry standard methods employed for sampling and geological logging.

Information available to demonstrate the sample representivity for Montage RC drilling includes RC sample condition logs, recovered RC sample weights. All Montage drill collars have been surveyed accurately using DGPS.

Results

Figure 10.4.7 below shows a selection of significant intercepts from aircore and RC drilling within the wider Project area.

Figure 10.4.7 Signifcant RC Driiling Interecepts



Source: Montage, January 2024.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Koné 2010 to 2021

11.1.1 Introduction and Summary

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 (QA/QC) monitoring of the reproducibility and accuracy of sample preparation and assaying, which are consistent with the QPs 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 QPs experience of good, industry standard practise.

The QP 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 QP considers that the sample preparation, security and analytical procedures adopted for the Koné drilling and exploration sampling provide an adequate basis for the MRE and exploration activities.

11.1.2 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 11.1.1) 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-mineralised granite collected from well outside the mineralised 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 11.1.1 Fadiadougou Sample Organisation and Storage Facility

Source: Montage, 2022.

11.1.3 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 11.1.1). 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 recognised 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 recognised 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 11.1.1 Analytical Laboratories by Sampling Phase

Year	Soil / Pit	Trenching	Reconnaissance RC Drilling	Koné Area Drilling	
				RC	Diamond
2009 to 2010	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					

11.2 Koné RC and Diamond Drilling

11.2.1 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 MRE were undertaken by several commercial laboratories (Table 11.1.1). Table 11.2.1 presents the number and proportion of assayed mineralised domain estimation dataset composites within the volume of mineralisation classified as Indicated or Inferred in the optimal pit constraining MRE by laboratory. This table provides an indication of the relative contribution of assays from each laboratory to MRE.

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 20th 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 pulverised in a ring mill to 85% passing 75 µm and a 250 g sub-sample of the pulverised 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 0.01 ppm.

Table 11.2.1 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%**

11.2.2 Routine Monitoring of Sampling and Assay Reliability

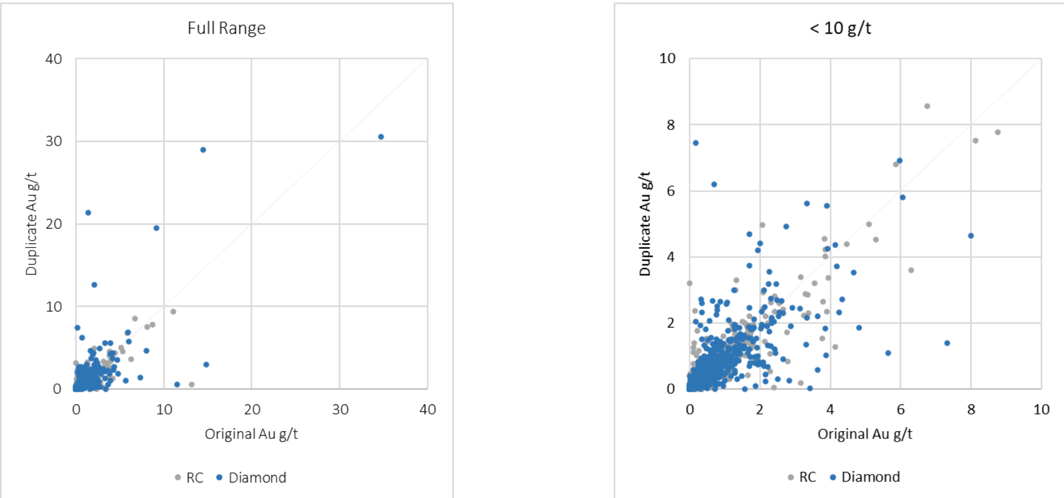
Field Duplicates

Field duplicates were collected for Koné RC and diamond drilling at average frequencies of around one 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 11.2.1 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 11.2.1 Field Duplicates for Koné RC and Diamond Drilling

g/t Au	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	



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 11.2.2 summarises 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 11.2.2 demonstrates that coarse blank assays show very low gold grades relative to typical Koné mineralisation with no indication of significant contamination or sample misallocation.

Table 11.2.2 Coarse Blanks included with Koné Drill Samples

Laboratory	Number Blanks	Gold Assay g/t			Proportion >
		Minimum	Average	Maximum	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%**

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 one 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 drillhole samples.

Table 11.2.3 summarises 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 11.2.3 demonstrates that, although as expected there is some variability for individual samples, average assay results closely match expected values.

Table 11.2.3 Reference Standards Included with Koné Drill Samples

Laboratory	Reference Standard	Number Samples	Gold Grade g/t		Avg. vs. Expected
			Expected	Avg. Assay	
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%

Laboratory	Reference Standard	Number Samples	Gold Grade g/t		Avg. vs. Expected
			Expected	Avg. Assay	
	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%

11.2.3 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 summarised in Table 11.2.4, 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 11.2.4 Alternate Method Duplicate Assays vs. Original Assays for Koné Drill Samples

Description		Original Intertek FA	Duplicate		
			Fire Assay	CN Leach	Screen Fire
Full dataset (59)	Average (g/t Au)	1.42	1.23	1.18	1.10
	vs. Original		-14%	-17%	-23%
	vs. Duplicate FA			-4%	-10%
Exclude anomalous (54)	Average (g/t Au)	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 (g/t Au)	1.05	1.04	1.08	1.01
	vs. Original		-1%	3%	-4%
	vs. Duplicate FA			4%	-3%

11.2.4 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

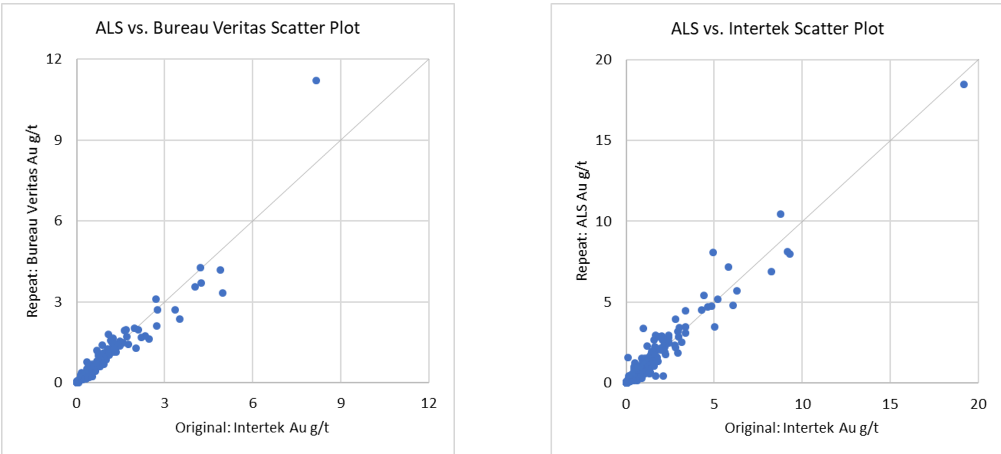
In February 2016, ALS Rosia Montana was accredited by the Standards Council of Canada in accordance with ISO/IEC 17025:2005 (Accredited Laboratory Number 742).

The summary statistics and scatter plots in Figure 11.2.2 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 11.2.2 ALS Interlaboratory Repeat Assays of Koné Drill Samples

Description	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	

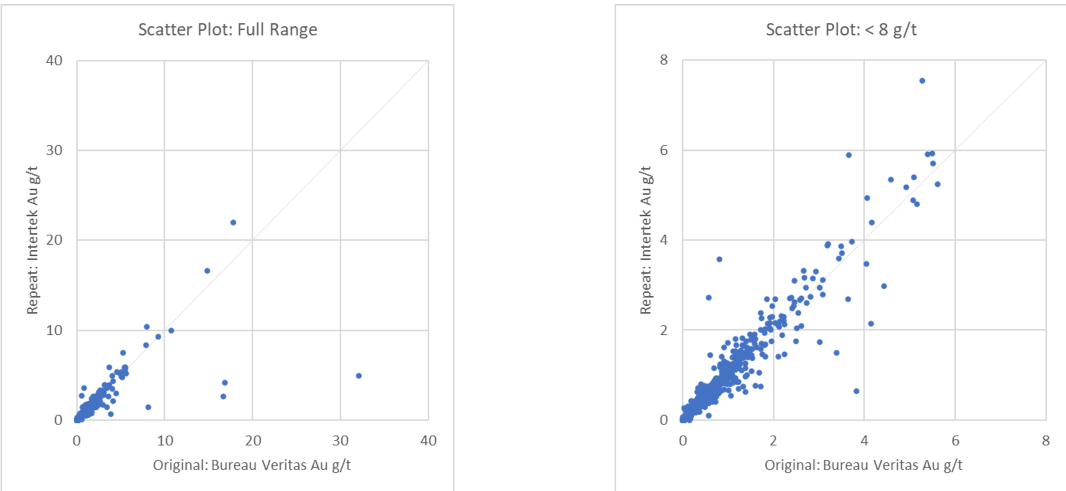


February 2021 Bureau Veritas Repeats

In February 2021, 989 sample pulps with original Intertek gold assays were submitted to Bureau Veritas in Abidjan, Côte d'Ivoire for analysis. As demonstrated by the comparative statistics and scatter plots in Figure 11.2.3, 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 11.2.3 Bureau Veritas Interlaboratory Repeat Assays of Koné Drill Samples

Description	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 11.2.4, 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 11.2.5 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 versus 0.05, and MR457828 0.72 versus 0.08 g/t). Figure 11.2.5 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 QPs opinion this variability is not significant at the current stage of project evaluation.

Figure 11.2.4 SGS Interlaboratory Repeats of Bureau Veritas Assays

Description	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	

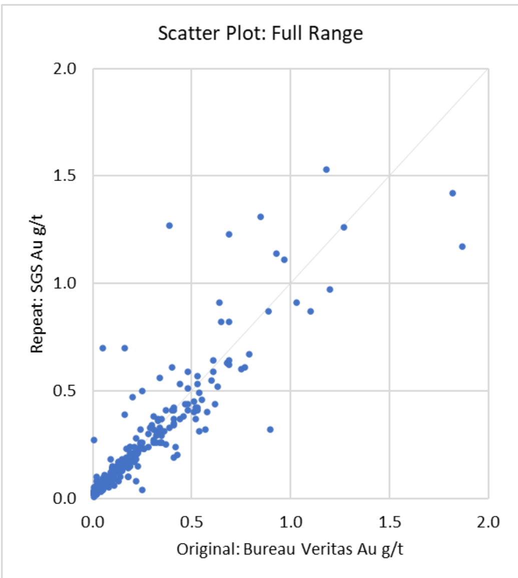
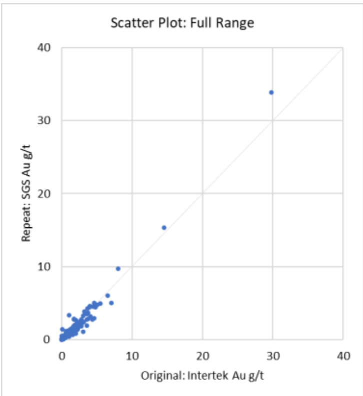
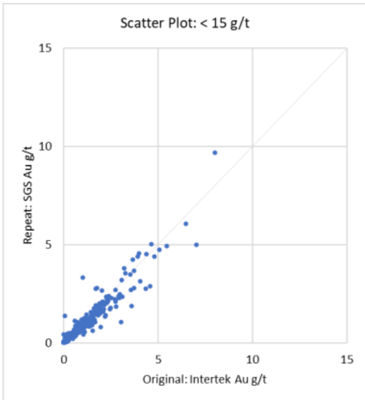


Figure 11.2.5 SGS Interlaboratory Repeats of Intertek Assays

Description	Full Dataset		0.10 to 20 g/t	
	Original Intertek Au /t	Repeat SGS Au g/t	Original Intertek Au /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	





11.3 Reconnaissance RC Drilling

11.3.1 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 drillhole 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 pulverised to nominally to 90% passing 75 µm and a 1 kg sample analysed by 12 h LeachWELL Bulk Leach Extractable Gold (BLEG) and AAS determination with a lower detection limit of 0.01 ppm.

11.3.2 Monitoring of Sampling and Assay Reliability

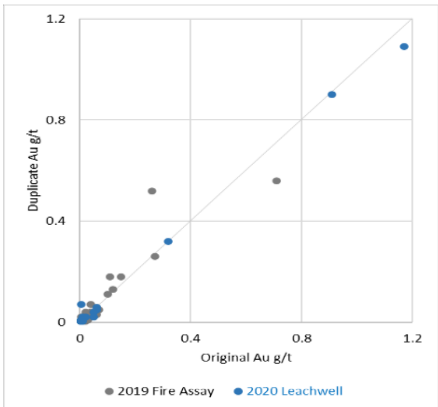
Routine Field Duplicates

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 11.3.1). 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 11.3.1 Field Duplicates for Reconnaissance RC Drilling

Description	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 summarised in Table 11.3.1 with samples assaying at below the detection limit of 0.01 g/t assigned values of half the detection limit.

Reference standards in Table 11.3.1 identified with a prefix of 'G' were produced by Geostats. The 'OREAS' prefixed standard was produced by ORE Research & Exploration Pty.

Table 11.3.1 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 11.3.1 Coarse Blanks and Reference Standards Included with 2019 to 2020 Reconnaissance RC Samples

Coarse Blanks					
Assay Group	**Number Samples**	**Gold Assay (g/t)**			**Proportion >**
		Minimum	**Average**	**Maximum**	**Detection**
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%

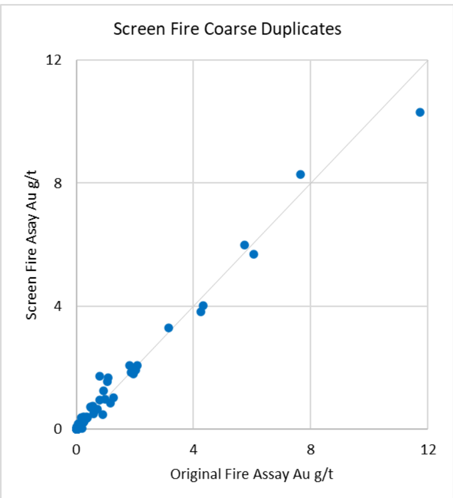
Alternative Method and Inter-Laboratory Duplicate Assays

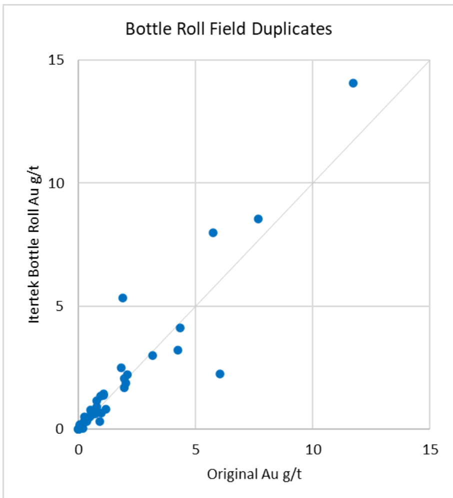
Information available to demonstrate the accuracy of primary Bureau Veritas gold fire assaying for samples from the 2019 reconnaissance RC drilling includes screen fire assays performed by Bureau Veritas on duplicate splits of coarse reject samples and field duplicate bottle roll analyses performed by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

The summary statistics and scatter plot in Figure 11.3.2 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 11.3.2 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	





11.4 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 h at 1,000 degrees Celsius (°C) and wax coated to prevent water absorption. Densities were measured by the Archimedes method with allowance for the wax coating.

Table 11.4.1 summarises the primary oven-dried wax coated immersion density measurements available for Koné coded by combined mineralised domain, oxidation zone and rock type wireframes interpreted by Montage from drillhole 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 mineralised domain.

Information available to demonstrate reliability of the inhouse density measurements includes 50 immersion measurements performed by SGS on core samples collected from around 10 cm deeper downhole than each paired inhouse 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 11.4.1).

The QP considers that the available density measurements provide an adequate basis for the current Indicated and Inferred MRE.

Table 11.4.1 Bulk Density Measurements by Oxidation and Rock Type

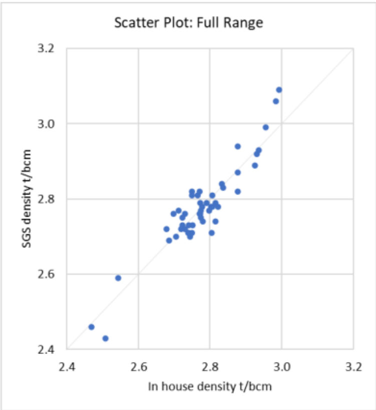
Oxidation Zone	Mineralisation Zone	Rock Code	Density t/bcm			
			Number	Minimum	Average	Maximum
Completely Oxidised	Background	CDI	42	1.19	1.66	2.21
		PGDI	48	1.28	1.63	2.47
		VC	275	1.15	1.67	2.62
		Subtotal	365	1.15	1.66	2.62
	Mineralised Envelope	CDI	110	1.16	1.66	2.56
		PGDI	2	1.56	1.80	2.03
		VC	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
	Mineralised 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
	Mineralised 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 11.4.1 SGS vs. Inhouse Paired Density Measurements

Description	Inhouse 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

Description	Number	Average t/bcm		Difference
		Inhouse	SGS	
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%



11.5 Gbongogo

References to 'inhouse' personnel in this report refer to personnel employed by Barrick or Montage.

All sample preparation and gold assaying samples from the Gbongogo drilling 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 Barrick and Montage personnel.

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

Routine sampling and assaying procedures for Montage drilling included QA/QC monitoring of the reproducibility and accuracy of sample preparation and assaying which are consistent with the QPs experience of good industry standard practices. 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 QPs experience of good, industry standard practise.

Available information indicates that Barrick RC and diamond drilling employed industry standard methods. Information available to demonstrate the reliability of sampling and assaying for this drilling includes assay results for coarse blanks, field duplicates and certified reference standards. Nearest neighbour comparisons of 2 m downhole composited gold assays from Barrick and Montage drilling show very similar average grades, supporting the general reliability the available for Barrick drill data.

The QP considers that quality control measures adopted for sampling and assaying of Gbongogo drillhole samples 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.

Sample preparation, security, and analytical procedures adopted for the Gbongogo drilling provide an adequate basis for the MRE and exploration activities.

Table 10.2.2 presents the number and proportion of mineralised domain estimation dataset composites within the optimal pit constraining MRE by drilling group and assay laboratory. This table provides an indication of the relative contribution of information from each drilling and assay group to MRE. It demonstrates that Montage and Barrick drill samples respectively provide around three quarters and one quarter of estimation dataset composites for the main mineralised within the resource pit shell.

11.5.1 Montage RC and Diamond Drilling

Field Sampling and Sample Security

Montage RC and diamond drilling respectively provide 24% and 53%, for a combined 77% of the composites from the main mineralised domains hosting Mineral Resources (Table 10.2.2) and confidence in the reliability of assaying for samples from this drilling significantly impacts general confidence in estimated resources.

All sample handling and field sub-sampling for Montage's Gbongogo RC and diamond drilling was supervised by inhouse geologists. Prior to collection by laboratory staff, all sample collection and transportation including delivery of samples to assay laboratories were undertaken or supervised by inhouse personnel. No other personnel were permitted unsupervised access to samples before delivery the assay laboratories.

For RC drilling, samples were collected over 1 m downhole 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:

- Samples collected from the cyclone in new 55 by 100 cm plastic sample bags labelled with the hole number and interval.
- After weighing, bulk samples were split with a three-tier riffle splitter with approximately 3 kg primary 'original' sub-samples collected in plastic bags which were sealed.
- The remaining bulk sample was riffle split a second time to produce an approximately 3 kg archive sample with the remaining bulk sample stored in the original bag.
- Duplicates were collected at pre-defined intervals at an average frequency of around one duplicate per 21 primary samples by passing the bulk sample through the riffle splitter a third time producing another approximately 3 kg sub-sample for the interval.
- The riffle splitter was cleaned thoroughly with compressed air between samples.
- Sample 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.
- All sub-samples (original, archive and duplicate) were transported to the field office at the end of each shift, where the archive samples were stored and original and duplicates prepared for dispatch to the analytical laboratory.
- Bulk reject sample were left at the drill site in ordered lines until completion of the program, when the drilling sites were rehabilitated.
- Company personnel transported RC drill samples directly to the sample storage facility at Montage's Gbongogo camp where the samples were arranged in order and archive samples separated and stored.

All onsite handling of diamond drill core was supervised by Company geologists with core handling and sampling procedures including 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 1 m intervals (minimum 0.04 m) assigned by logging geologists, respecting changes in rock units.
- Sampled half core was placed in plastic sample bags in sequence and securely stored before batch assignment and submission to the assay laboratory.
- Duplicate half core samples were collected an average frequency of around one duplicate per 21 primary samples.

- All core was digitally photographed prior to cutting in a wet and dry state and stored in plastic core trays at the field office.

Montage's routine monitoring of the reliability of sampling and assaying included the following:

- Field duplicates of RC and diamond samples 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-mineralised granite collected from well outside the mineralised area were inserted into sample sequences at pre-defined intervals. These blanks, which were blind to the assay laboratory 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 prepared by Ore Research & Exploration P/L in Perth (OREAS) Western Australia were inserted into sample sequences at pre-defined intervals. Assay results for these standards provide an indication of assaying accuracy.

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 for each batch. A hardcopy submission form was included with the submitted samples and an electronic copy emailed to the laboratory.

All assay pulps were returned to the field office from the laboratory and stored for future reference.

Sample Preparation and Analysis

Primary samples from Montage's Gbongogo RC and diamond drilling were submitted to Bureau Veritas in Abidjan, Côte d'Ivoire for preparation and analysis. Bureau Veritas sample preparation and analytical procedures comprised the following:

- Each batch of samples 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.

- After oven drying, the nominally 3 kg samples were jaw crushed to >80% passing 2 mm and riffle split to produce two 1.5 kg sub-samples. After every 20th sample and at the end of each assay batch, barren vein quartz was crushed to clean the crusher plates.

- 1.5 kg sub-samples of crushed sample were pulverised in a ring mill to 85% passing 75 µm and a 250 g sub-sample of the pulverised material collected as the primary sample pulp.

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

Interlaboratory repeats were performed by SGS in Yamoussoukro Côte d'Ivoire utilizing analytical procedures consistent with those employed by Bureau Veritas.

TÜV NORD CERT GmbH certified that Bureau Veritas operations, including the Abidjan laboratory, operate management systems that comply with ISO 9001:2015, ISO 14001: 2015 and ISO 45001:2018. The current certification certificates are valid from February 2022 to the end of January 2025.

SGS Yamoussoukro has not accredited by any recognised accreditation authority. SGS services include quality assurance protocols in line with ISO 17025.

11.5.2 Monitoring of Sampling and Assay Reliability

Field Duplicates

Field duplicates were collected for Montage Gbongogo RC and diamond drilling at average frequencies of around one duplicate per 21 primary samples for both drill types. The field duplicates were collected consistently with and assayed in the same batch as original samples. RC field duplicates were collected by passing the bulk sample through the riffle splitter a third time (after collection of archive samples), and diamond core duplicates represent the second half of core remaining after collection of primary samples.

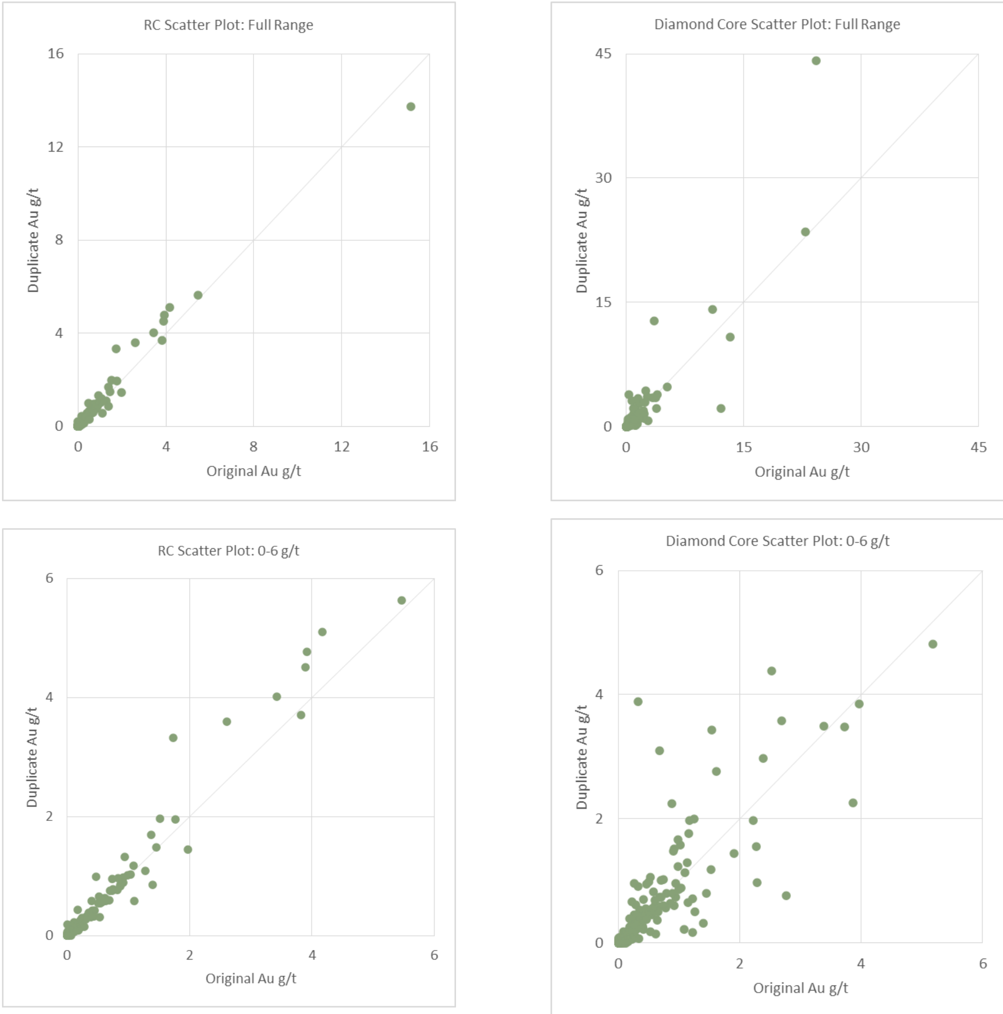
The summary statistics in Table 11.5.1 and scatter plots in Figure 11.5.1 demonstrate that although there is some scatter for individual pairs, duplicate assay results generally correlate reasonably well with original results demonstrating the adequacy of field sub-sampling procedures.

Rather than a consistent, systematic feature of the duplicates, the differences in average gold grades between duplicates and original RC samples reflects a small number of anomalously poorly correlating pairs. For diamond duplicates, the difference in mean grade for the combined dataset largely reflects a single high grade outlier pair.

Table 11.5.1 Montage Field Duplicates

Au g/t	RC				Diamond Core			
	Full Set		Less Anomalous		Full Set		<30 g/t	
	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate
Number	310		304		303		302	
Average	0.28	0.30	0.22	0.22	0.62	0.72	0.55	0.57
Difference		5%		-1%		15%		5%
Variance	1.1	1.1	1.0	0.8	5.4	10.1	3.5	3.8
Coef. Var	3.76	3.56	4.37	4.13	3.71	4.42	3.45	3.41
Minimum	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005
1st Quartile	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Median	0.02	0.02	0.02	0.02	0.03	0.03	0.03	0.03
3rd Quartile	0.12	0.12	0.11	0.11	0.42	0.45	0.41	0.44
Maximum	15.2	13.7	15.2	13.7	24.2	44.1	22.9	23.5
Correl Coef.	0.986		0.995		0.905		0.890	

Figure 11.5.1 Montage Field Duplicates



Coarse Blanks

Montage routinely included coarse blanks with assay batches of Gbongogo RC and diamond core samples submitted to Bureau Veritas at an average frequency of around one blank per 20 primary samples for both drilling types. In addition to checking for contamination during sample preparation, these coarse blank samples provide a check of sample misallocation by field staff, the laboratory and during database compilation. Table 11.5.2 summarises gold assays for these blanks with samples assaying at below the detection limit of 0.01 g/t assigned values of half the detection limit. This table excludes a single anomalous result of 0.98 g/t which appears to reflect misallocation.

Table 11.5.2 demonstrates that coarse blank assays show very low gold grades relative to typical Gbongogo mineralisation, suggesting that the sample preparation for RC and diamond drilling is generally free from significant contamination or sample misallocation.

Table 11.5.2 Montage coarse blanks

Sample Type	Number of Blanks	Assay: Au g/t			Proportion
		Minimum	Average	Maximum	>Detection Limit
RC	330	0.005	0.007	0.03	15%
Diamond	319	0.005	0.006	0.02	5%
Total	649	0.005	0.006	0.03	10%

Reference Standards for Primary Assaying

Samples of OREAS certified reference standards were inserted with Montage's primary Gbongogo RC and diamond drill samples at an average rate of around one standard per 21 primary samples. Expected gold grades for the standards range from 0.332 to 2.53 g/t covering the range of common gold grades shown by Gbongogo drillhole samples.

The summary of Bureau Veritas gold assay results for reference standards in Table 11.5.3 excludes five values which match expected values so poorly they are suggestive of sample misallocation. This table demonstrates that although, as expected there is some variability for individual samples, average assay results closely match expected values, supporting the accuracy of Bureau Veritas assaying.

Table 11.5.3 Montage Primary Assaying Reference Standards

Standard	Expected Au g/t	Assays Au g/t				Avg. vs. Expected
		Number	Minimum	Average	Maximum	
OREAS-250b	0.332	128	0.30	0.32	0.35	-3%
OREAS-252b	0.837	154	0.78	0.84	0.88	0%
OREAS-253b	1.24	163	1.17	1.25	1.32	1%
OREAS-254B	2.53	168	2.39	2.52	2.67	-1%

SGS Interlaboratory Repeats

Information available to demonstrate the accuracy of primary Bureau Veritas gold assaying of Montage's Gbongogo drill samples includes interlaboratory repeat assays of selected pulp samples by SGS in Yamoussoukro. SGS assays of OREAS standards included with the interlaboratory repeats support the accuracy of SGS assaying (Table 11.5.4).

The summary statistics in Table 11.5.5 and scatter plots in Figure 11.5.2 demonstrate that although, as expected for repeat assaying of pulp samples, there is some scatter for individual pairs, the SGS repeat generally correlate reasonably well with original results providing additional confidence in the accuracy of the primary Bureau Veritas assaying.

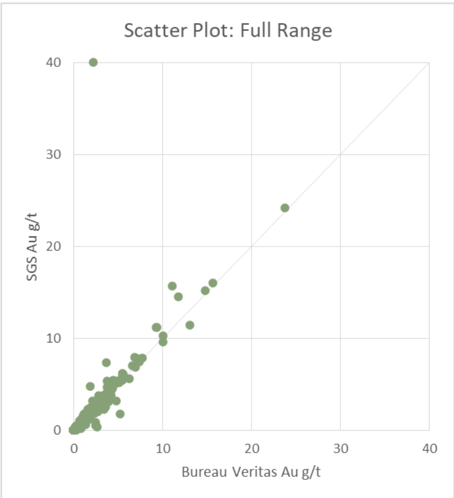
Table 11.5.4 Reference Standards Included with SGS Interlaboratory Repeats

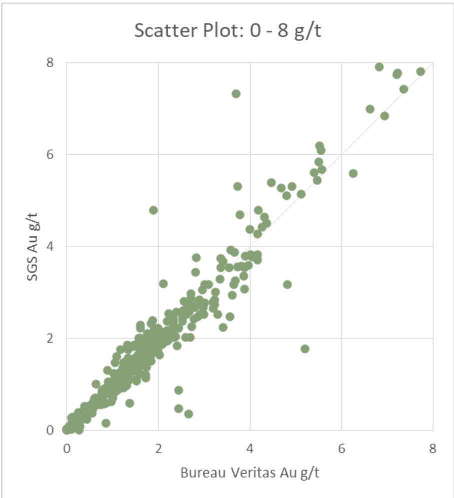
Standard	Expected Au g/t	Assays (Au g/t)				Avg. vs. Expected
		Number	Minimum	Average	Maximum	
OREAS-250b	0.332	7	0.32	0.33	0.34	0%
OREAS-252b	0.837	7	0.83	0.84	0.85	0%
OREAS-253b	1.24	8	1.21	1.23	1.25	-1%
OREAS-254B	2.53	7	2.49	2.52	2.54	-1%

Table 11.5.5 SGS Interlaboratory Repeats of Montage Drill Samples

Description	Full Set		< 40 g/t	
	Bureau Veritas Au g/t	SGS Au g/t	Bureau Veritas Au g/t	SGS Au g/t
Number	564		563	
Average	**1.56**	**1.62**	**1.56**	**1.55**
Difference		**4%**		**0%**
Variance	4.48	7.68	4.49	5.07
Coef. Var	1.35	1.71	1.36	1.45
Minimum	0.005	0.005	0.005	0.005
1st Quartile	0.33	0.30	0.33	0.30
Median	1.08	0.99	1.07	0.98
3rd Quartile	1.92	1.90	1.92	1.90
Maximum	23.8	40.0	23.8	24.2
Correl Coef.	0.803		0.980	

Figure 11.5.2 SGS Interlaboratory Repeats of Montage Drill Samples





11.5.3 Barrick RC and Diamond Drilling

Field Sampling and Sample Security

Samples from Barrick RC drilling provide only 5% of estimation dataset composites from the main mineralised domains hosting Mineral Resources (Table 10.2.2) and confidence in the reliability of these data does not significantly impact general confidence in estimated resources. Barrick diamond drilling provides 18% of these composites, and confidence in the reliability of these data significantly impacts general confidence in estimated resources.

Available information indicates that sampling of Barrick RC and diamond drilling employed industry standard methods. RC holes were sampled over 1 m downhole intervals. Diamond core was generally halved for analysis over intervals ranging from 0.10 m to 2.10 m, with a dominant sample length of 1 m.

Although no details of the specific sample security measures adopted by Barrick are available, the available information suggests that Barrick employed industry standard methods for sample handling. Nearest neighbour comparisons of 2 m downhole composited gold assays from Barrick and Montage drilling show very similar average grades, supporting the general reliability of the information available for Barrick drilling.

Information available to demonstrate the reliability of sampling and assaying for Barrick drilling includes assay results for coarse blanks, field duplicates and certified reference standards. The certified reference standards were prepared by OREAS and less commonly Rocklabs Pty Ltd (Rocklabs) in Perth Western Australia.

Sample Preparation and Analysis

Samples from Barrick's Gbongogo RC and diamond drilling were submitted to SGS Tongon, Côte d'Ivoire or less commonly Bureau Veritas in Abidjan for analysis. Available information indicates that SGS and Bureau Veritas employed industry standard methods for sample preparation and analysis, broadly comparable with those employed for analysis of Montage Gbongogo drill samples.

SGS analysed pulverised sub-samples for gold by method FA505, comprising fire assay of 50 g samples with AAS determination. Bureau Veritas utilised the same method (FA450) employed for their assaying of Montage Gbongogo drill samples.

Available information indicates that SGS Tongon is not accredited by any recognised accreditation authority, and SGS services include quality assurance protocols in line with ISO 17025.

During the period of their assaying of Barrick samples, Bureau Veritas Abidjan was not accredited by any recognised accreditation authority. The laboratory operated under the ISO 17025 accreditation of the Bureau Veritas Vancouver, as endorsed by the Standards Council of Canada.

Monitoring of Sampling and Assay Reliability

Field Duplicates

Field duplicate results available for Barrick Gbongogo RC and diamond drill samples assayed by SGS and Bureau Veritas represent average frequencies of around one duplicate per 24 primary samples for each dataset. These duplicate assays were assayed in the same batch as original samples.

Table 11.5.6 and Figure 11.5.3 summarise gold assays for Barrick Gbongogo RC and diamond field duplicates by assay laboratory and sample type. Notable features shown by this table and figure include the following:

- For SGS duplicate assays of RC samples, excluding three very poorly correlating pairs that are suggestive of sample misallocation significantly improves correlation. The duplicate samples have generally low gold grades, with only nine pairs assaying at greater than 0.10 g/t.
- SGS diamond core duplicate assays show notably poorer correlation than demonstrated by other sets of duplicates, including Montage Gbongogo core duplicates. Excluding a single high grade outlier pair (15.6, 6.57 g/t) reduces the difference in mean grades shown by the full dataset.
- The 50 Bureau Veritas RC duplicates have generally low gold grades, with only nine pairs assaying at greater than 0.10 g/t.
- Bureau Veritas diamond core duplicates are generally low grade, with only ten pairs having gold grades of greater than 0.1 g/t. Excluding one high grade outlier pair reduces the difference in mean grades shown by the full dataset.

The generally low gold grades of duplicated samples from Barrick drilling reduces usefulness of the duplicates in demonstrating the repeatability of field sampling. With the exception of the SGS diamond core duplicates, the duplicates show repeatability consistent with the QPs experience of similar mineralisation styles.

Reasons for the comparatively poor repeatability shown by SGS diamond core duplicates are unclear. Possible reasons include these samples representing quarter core, rather than half core, which in the QPs general experience is common for diamond drilling, with the smaller samples giving greater variability.

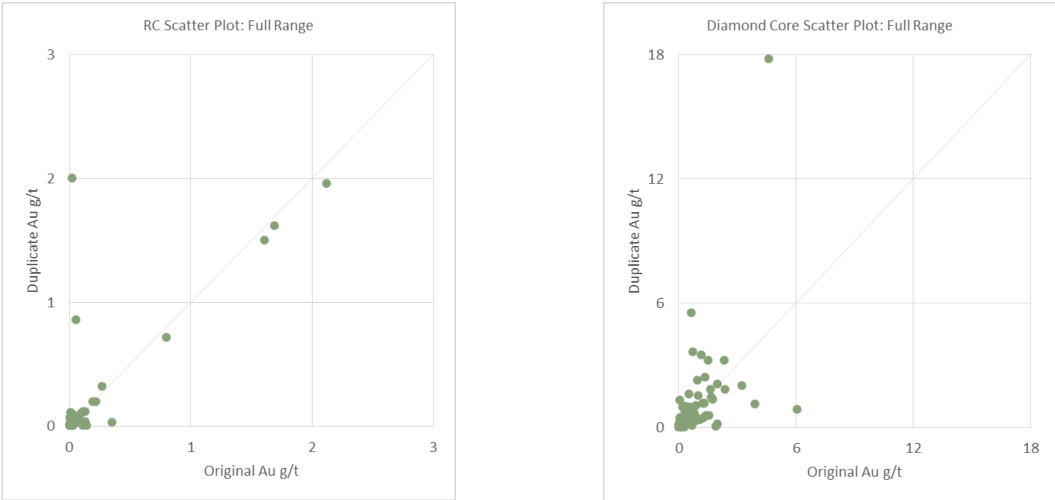
Combined with the other datasets available to demonstrate the reliability of Barrick drill data, the duplicate assays support the repeatability of these data with sufficient confidence for MRE and exploration activities.

Table 11.5.6 Barrick Field Duplicates

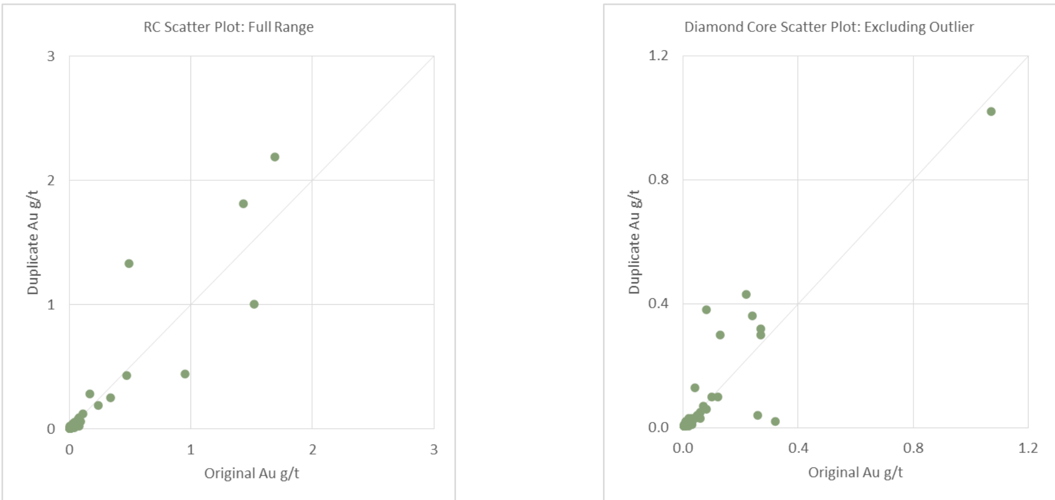
SGS Assays								
Au g/t	**RC**				**Diamond Core**			
	Full Set		**Less Anomalous**		**Full Set**		**Excluding Outlier**	
	Original	**Duplicate**	**Original**	**Duplicate**	**Original**	**Duplicate**	**Original**	**Duplicate**
Number	90		87		305		304	
Average	0.11	0.13	0.10	0.10	0.25	0.31	0.24	0.25
Difference		20%		-6%		21%		4%
Variance	0.1	0.1	0.1	0.1	0.41	1.38	0.35	0.38
Coef. Var	3.17	2.99	3.26	3.25	2.51	3.82	2.45	2.45
Minimum	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005
1st Quartile	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Median	0.02	0.02	0.02	0.02	0.03	0.03	0.03	0.03
3rd Quartile	0.04	0.04	0.04	0.04	0.18	0.17	0.18	0.17
Maximum	2.12	2.00	2.12	1.96	6.03	17.8	6.03	5.53
Correl Coef.	0.798		0.996		0.598		0.552	
Bureau Veritas Assays								
Au g/t	**RC**				**Diamond Core**			
	Full Set		**Excluding outlier**		**Full Set**		**Excluding outlier**	
	Original	**Duplicate**	**Original**	**Duplicate**	**Original**	**Duplicate**	**Original**	**Duplicate**
Number	50		49		46		45	
Average	0.17	0.18	0.14	0.14	0.42	0.23	0.08	0.09
Difference		6%		0%		-45%		6%
Variance	0.1	0.2	0.1	0.1	5.2	0.9	0.0	0.0
Coef. Var	2.31	2.51	2.37	2.52	5.38	4.17	2.02	2.02
Minimum	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005
1st Quartile	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Median	0.02	0.02	0.02	0.02	0.03	0.02	0.03	0.02
3rd Quartile	0.08	0.08	0.07	0.06	0.08	0.07	0.08	0.06
Maximum	1.69	2.19	1.52	1.81	15.6	6.57	1.07	1.02
Correl Coef.	0.913		0.871		0.992		0.883	

Figure 11.5.3 Barrick Field Duplicates

SGS Assays



Bureau Veritas Assays



Coarse Blanks

Coarse blanks assays are available for Barrick Gbongogo drilling at an average frequency of around one blank per 49 primary samples for both RC and diamond drilling. In addition to checking for contamination during sample preparation, these coarse blank samples provide a check of sample misallocation by field staff, the laboratory and during database compilation. Table 11.5.7 summarises 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.

Table 11.5.7 demonstrates that assays of coarse blanks included in batches of Barrick drilling show very low gold grades relative to typical Gbongogo mineralisation, suggesting that the Bureau Veritas and SGS sample preparation was generally free from significant contamination or sample misallocation.

Table 11.5.7 Barrick Coarse Blanks

Sample Type	Number of Blanks	Assay: Au g/t			Proportion
		Minimum	Average	Maximum	>Detection Limit
Bureau Veritas	52	0.005	0.006	0.02	12%
SGS	186	0.005	0.007	0.05	36%
Total	**238**	**0.005**	**0.007**	**0.05**	**31%**

Reference Standards

Assay results of certified reference standards available submitted in batches of Gbongogo drill samples assayed by Bureau Veritas and SGS represent an average frequency of around one blank per 54 primary samples.

A total of 162 SGS assays are available for 19 standards with certified gold grades of 0.33 to 4.08 g/t and between one and 27 assays available for each standard. A total of 48 Bureau Veritas assays are available for 10 standards with certified gold grades of 0.31 to 3.54 g/t, and between one and 16 assays available for each standard.

As shown by the summary of assays for standards included in batches of Barrick drill samples in Table 11.5.8, although as expected there is some variability for individual assays, for the combined datasets, the average grades reported by Bureau Veritas and SGS closely match average expected values supporting the accuracy of primary assaying for Barrick drill core.

Table 11.5.8 Barrick Reference Standards

SGS Assays						
Standard	**Expected Au g/t**	**Assays Au g/t**				**Avg. vs. Expected**
		Number	**Minimum**	**Average**	**Maximum**	
OREAS-200	0.340	24	0.30	0.33	0.36	-4%
OREAS-203	0.871	1	0.81	0.81	0.81	-7%
OREAS-209	1.580	4	1.47	1.54	1.58	-3%
OREAS-214	3.030	2	2.57	2.64	2.70	-13%
OREAS-215	3.540	23	3.28	3.51	3.71	-1%
OREAS-217	0.338	8	0.31	0.33	0.34	-3%
OREAS-220	0.866	27	0.70	0.85	0.92	-2%
OREAS-222	1.220	5	1.13	1.18	1.24	-3%
OREAS-238	3.030	1	2.21	2.21	2.21	-27%
OREAS-250	0.332	9	0.29	0.31	0.34	-6%
OREAS-251	0.504	1	0.56	0.56	0.56	11%
OREAS-252	0.674	14	0.62	0.66	0.72	-2%
OREAS-253	1.220	1	1.14	1.14	1.14	-7%
OREAS-254	2.550	6	2.42	2.54	2.59	0%
OREAS-255	4.080	1	3.92	3.92	3.92	-4%
OREAS-620	0.685	5	0.65	0.66	0.68	-4%
OXG104[R]	0.925	1	0.91	0.91	0.91	-2%
SF85[R]	0.848	13	0.78	0.85	0.92	1%
SI81[R]	1.790	16	1.67	1.81	1.96	1%
Total	**1.331**	**162**	**0.29**	**1.31**	**3.92**	**-2%**
Bureau Veritas Assays						
Standard	**Expected Au g/t**	**Assays Au g/t**				**Avg. vs. Expected**
		Number	**Minimum**	**Average**	**Maximum**	
OREAS-200	0.340	16	0.31	0.34	0.36	-1%
OREAS-209	1.580	1	1.55	1.55	1.55	-2%
OREAS-215	3.540	3	3.49	3.64	3.72	3%
OREAS-217	0.338	6	0.32	0.33	0.35	-1%
OREAS-220	0.866	8	0.82	0.84	0.87	-3%
OREAS-222	1.220	1	1.19	1.19	1.19	-2%
OREAS-250	0.309	3	0.33	0.34	0.35	10%
OREAS-252	0.674	5	0.63	0.66	0.67	-2%
OREAS-254	2.550	4	2.48	2.58	2.67	1%
OREAS-620	0.685	1	0.63	0.63	0.63	-8%
Total	**0.896**	**48**	**0.31**	**0.90**	**3.72**	**0%**
R: Rocklabs standard. All others OREAS						

11.6 Bulk Density Measurement

11.6.1 Available Information

Bulk density measurements available for Gbongogo comprise 2,637 immersion measurements performed on diamond drill core by inhouse personnel, including the following:

- 430 measurements of core from the first six Barrick Gbongogo diamond holes (GBDDH001 to GBDDH006), for which core was wrapped in plastic cling film with the apparent goal of preventing water absorption. The QPs experience of numerous projects indicates that it very difficult to exclude all air from plastic wrapped core samples, and that this approach tends to understate average densities.

- 1,717 measurements of generally 10 cm samples from latter Barrick diamond holes (GBDDH007 to GBDDH028), for which core was not sealed or coated prior to measurement. The QPs experience indicates that, although water absorption can lead to significant overstatement of densities for porous material, for competent fresh rock such as fresh Gbongogo mineralisation, measurements of uncoated samples can be sufficiently reliable for high confidence resource estimation.

- 488 measurements of oven dried, wax coated core from Montage drilling, with specified lengths ranging from 3 to 40 cm and averaging 11 cm. For 407 of these intervals, the supplied information also includes density measurements of uncoated core.

Table 11.6.1 compares the uncoated and coated density measurements of Montage diamond core coded by the oxidation zone wireframes used for resource modelling and excluding three anomalous pairs. The close agreement in average uncoated and coated density measurements for each oxidation zone shown by this table confirms that Gbongogo mineralisation is not significantly porous and suggests that the lack of wax coating for later Barrick measurements may not have introduced significant biases to those data.

The available density measurements provide an adequate basis for the current MRE.

Table 11.6.1 Montage Coated vs. Uncoated Density Measurements

Density t/m^3	Oxide		Transition		Fresh		Combined	
	Uncoated	Coated	Uncoated	Coated	Uncoated	Coated	Uncoated	Coated
Number	4		20		380		404	
Average	2.65	2.64	2.67	2.67	2.76	2.77	2.76	2.77
Difference		-0.5%		0.0%		0.3%		0.3%
Variance	0.014	0.022	0.006	0.007	0.005	0.005	0.006	0.006
Coef. Var	0.04	0.06	0.03	0.03	0.03	0.02	0.03	0.03
Minimum	2.53	2.44	2.48	2.46	2.54	2.63	2.48	2.44
1st Quartile	2.57	2.55	2.63	2.64	2.72	2.72	2.71	2.72
Median	2.62	2.63	2.66	2.66	2.75	2.76	2.75	2.76
3rd Quartile	2.70	2.72	2.72	2.71	2.80	2.81	2.80	2.81
Maximum	2.84	2.85	2.83	2.84	3.07	3.05	3.07	3.05

11.6.2 Summary of Results

Table 11.6.2 summarises Gbongogo bulk density measurements by oxidation zone, rock type and drilling phase. For preparation of this table, the measurements were coded by oxidation and rock type wireframes supplied by Montage, excluding data from reconnaissance drilling outside the resource area. With the exception of three intervals with anomalous coated results, for which the uncoated measurement was prioritized, Montage coated measurements were selected in preference to the uncoated measurements. Three anomalously high or low results were excluded.

Notable features demonstrated by Table 11.6.2 include the following:

- Measurements from the initial phase of Barrick drillholes, which included plastic wrap coating gave lower average values than later Barrick and Montage data. This trend is consistent with the author's general experience, which demonstrates that measurements of plastic wrapped samples tend to understate average densities.

- There is comparatively little variability between average measurements from later Barrick holes and Montage measurements.

- For the combined dataset of later Barrick holes and Montage measurements, there is little variability in average densities by rock type within each oxidation zone.

Table 11.6.2 Density Measurements by Oxidation Zone and Rock Group

Phase	Rock Unit	Clay t/m³				Oxide t/m³			
		No.	Min.	Avg.	Max	No.	Min.	Avg.	Max
Barrick Phase 1 (GBDDH001-06)	HW sediment	2	1.29	1.31	1.33	8	1.04	1.20	1.36
	FW Sediment	-	-	-	-	-	-	-	-
	Quartz diorite	-	-	-	-	-	-	-	-
	Combined	**2**	**1.29**	**1.31**	**1.33**	**8**	**1.04**	**1.20**	**1.36**
Barrick Phase 2 (GBDDH007-28)	HW sediment	7	1.19	1.42	1.62	34	1.11	1.52	2.32
	FW sediment	-	-	-	-	3	1.29	1.74	2.41
	Quartz diorite	1	1.25	1.25	1.25	5	1.41	1.67	2.41
	Combined	**8**	**1.19**	**1.40**	**1.62**	**42**	**1.11**	**1.56**	**2.41**
Montage	HW sediment	68	1.03	1.65	2.02	13	1.64	1.98	2.68
	FW sediment	-	-	-	-	-	-	-	-
	Quartz diorite	2	1.50	1.58	1.66	1	2.44	2.44	2.44
	Combined	**70**	**1.03**	**1.65**	**2.02**	**14**	**1.64**	**2.02**	**2.68**
Combined Barrick Phase 2 And Montage	HW sediment	75	1.03	1.63	2.02	47	1.11	1.65	2.68
	FW sediment	-	-	-	-	3	1.29	1.74	2.41
	Quartz diorite	3	1.25	1.47	1.66	6	1.41	1.80	2.44
	Combined	**78**	**1.03**	**1.63**	**2.02**	**56**	**1.11**	**1.67**	**2.68**

Phase	Rock Unit	Transition t/m³				Fresh t/m³			
		No.	Min.	Avg.	Max	No.	Min.	Avg.	Max
Barrick Phase 1 (GBDDH001-06)	HW sediment	17	1.54	2.46	2.66	227	2.22	2.63	2.90
	FW sediment	-	-	-	-	79	2.42	2.63	2.86
	Quartz diorite	3	2.53	2.60	2.69	94	2.17	2.55	2.82
	Combined	**20**	**1.54**	**2.48**	**2.69**	**400**	**2.17**	**2.61**	**2.90**
Barrick Phase 2 (GBDDH007-28)	HW sediment	36	1.73	2.55	2.77	221	2.18	2.72	2.99
	FW sediment	8	2.06	2.44	2.74	167	2.26	2.75	2.88
	Quartz diorite	3	2.47	2.56	2.70	117	2.56	2.70	2.83
	Combined	**47**	**1.73**	**2.53**	**2.77**	**505**	**2.18**	**2.72**	**2.99**
Montage	HW sediment	18	2.46	2.68	2.84	119	2.64	2.79	3.05
	FW sediment	-	-	-	-	140	2.63	2.78	2.96
	Quartz diorite	3	2.58	2.67	2.78	123	2.66	2.75	2.90
	Combined	**21**	**2.46**	**2.68**	**2.84**	**382**	**2.63**	**2.77**	**3.05**
Combined Barrick Phase 2 And Montage	HW sediment	54	1.73	2.59	2.84	340	2.18	2.74	3.05
	FW sediment	8	2.06	2.44	2.74	307	2.26	2.77	2.96
	Quartz diorite	6	2.47	2.61	2.78	240	2.56	2.73	2.90
	Combined	**68**	**1.73**	**2.58**	**2.84**	**887**	**2.18**	**2.75**	**3.05**

12.0 DATA VERIFICATION

12.1 Drilling Information

12.1.1 Introduction

This section describes data verification checks for Koné and Gbongogo drilling information.

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

Verification checks undertaken by the QP to confirm the validity of information for the working databases compiled for Koné and Gbongogo RC and diamond drilling 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 databases compiled by the QP and check both the validity of Montage's master database and potential data transfer errors in compilation of the working databases.

The consistency checks showed no significant inconsistencies for either deposit.

12.1.2 Gbongogo Drill Data

While visiting Montage's Gbongogo field office in September 2023, the QP compared original field records with database entries for Montage drillholes included in the composite estimation dataset informing Mineral Resources. These checks included collar entries for 20 drillholes, 72 downhole survey table records and downhole depths and sample identifiers for 2,379 assay intervals representing approximately 29%, 4% and 21% of database entries respectively for Montage drillholes contributing to the estimation dataset informing mineral resources (Table 12.1.1).

The collar table checks included dates, depths, and original handheld GPS coordinates, which were checked for reasonable consistency with the DGPS surveys. Downhole survey table spot checks comprised comparing original hardcopies with database entries for Reflex survey measurements. Gyro measurements were recorded electronically.

For all 17,457 primary assay table entries from Barrick (6,002) and Montage (11,455) drillholes included in the estimation dataset, the QP compared database assay entries with gold grades in laboratory source files supplied by Montage. These checks showed no inconsistencies.

Nearest neighbour comparisons of 2 m downhole composited gold assays from Barrick and Montage drilling show very similar average grades. This comparison supports the general reliability of the information available for Barrick drilling.

Verification checks undertaken by the QP to confirm the validity of information for the RC and diamond drilling compiled Gbongogo drillhole database 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.

Table 12.1.1 Gbongogo Drillhole Database Spot Checks

Description	Collar Table	Surve Table	Assay Table
Number: Total / Montage Holes	105 / 69	1,831 / 1,671	17,457 / 11,455
Number Checked	20	72	2,379
Proportion Checked: Total / Montage Holes	19% / 29%	4% / 4%	14% / 21%

12.1.3 Koné Drill Data

While visiting Montage's field office in Fadiadougou in August 2018, the QP compared original field records with entries in the compiled Koné drillhole database. These checks included 180 downhole survey table records, and downhole depths and sample identifiers for 5,523 assay intervals representing approximately 25% and 33% of database entries respectively at that time.

Additional comparisons of original records with database entries for post August 2018 Koné drilling included 156 downhole survey table records and downhole depths and sample identifiers for 4,124 assay intervals.

Relative to the drillholes informing the current estimates, the combined assay and downhole survey table spot checks represent around 12% and 3% of database records respectively.

The spot checks showed no significant inconsistencies.

For 99.5% of routine assays from RC and diamond drilling, and all 956 SGS 2021 interlaboratory repeat assays, the QP compared database assay entries with gold grades in laboratory source files supplied by Montage (Table 12.1.2). These checks showed no inconsistencies.

Table 12.1.2 Koné Laboratory Source File Checks

Period	Number of Assays		Proportion
	In Database	Checked	Checked
2010	925	925	100.0%
2013	1,766	1,766	100.0%
2017 to 2018	13,878	13,878	100.0%
2019 to 2020	22,812	22,812	100.0%
2021	49,765	49,307	99.1%
Combined	**89,146**	**88,688**	**99.5%**

12.2 Metallurgical

12.2.1 Metallurgical Sample Selection

The metallurgical samples were selected by the Montage geologists to reflect:

- Spatial distribution through the mineralised zones.
- Oxidation domain.
- Grade distribution.
- Lithology distribution.

The samples were selected to approximate the tonnage proportions by domain within the mineralised zone.

Table 12.2.1 Comminution Samples by Domain

		JK Tech SMC		Ball Mill Work Index		Abrasion Index		Crusher Work Index	
Ore Domain	Deposit %	No. of Samples	Sample %	No. of Samples	Sample %	No. of Samples	Sample %	No. of Samples	Sample %
Fresh	83%	52	76%	52	76%	11	65%	13	93%
FW Fresh	7%	3	4%	3	4%				
Trans	4%	9	13%	9	13%	4	24%	1	7%
Oxide	6%	4	6%	4	6%	2	12%		
Total	**100%**	**68**	**100%**	**68**	**100%**	**17**	**100%**	**14**	**100%**

Table 12.2.2 Leach Samples by Domain

Deposit	Domain	Deposit %	Leach Samples #	Leach Samples %
South	Fresh	74.3%	53	36%
South	FW Fresh	9.0%	13	9%
North	Fresh	0.2%	12	8%
GB	Fresh	5.4%	8	5%
South	Transition	4.0%	17	12%
North	Transition	0.2%	5	3%
GB	Transition	0.3%	4	3%
South	Oxide	5.5%	21	14%
North	Oxide	0.5%	9	6%
GB	Oxide	0.4%	4	3%
Kone	**LOM**	**100.0%**	**146**	**100%**

Table 12.2.3 Leach Samples by Grade Bin

Koné

Grade Bin Au g/t	Fresh			FW Fresh			Trans			Oxide			Total		
	Deposit t %	Leach Samples #	Leach Samples %	Deposit t %	Leach Samples #	Leach Samples %	Deposit t %	Leach Samples #	Leach Samples %	Deposit t %	Leach Samples #	Leach Samples %	Deposit t %	Leach Samples #	Leach Samples %
<0.50	28%	17	12%	4%	3	2%	2%	7	5%	3%	7	5%	37%	34	23%
0.50 – 0.75	25%	15	10%	4%	3	2%	1%	4	3%	2%	8	5%	32%	30	21%
0.75 – 1.00	14%	22	15%	1%	3	2%	1%	4	3%	1%	9	6%	16%	38	26%
1.00 – 1.25	7%	11	8%	0%	4	3%	0%	6	4%	0%	4	3%	8%	25	17%
>1.25	6%	0	0%	0%	0	0%	0%	1	1%	0%	2	1%	6%	3	2%
	79%	**65**	**45%**	**10%**	**13**	**9%**	**5%**	**22**	**15%**	**6%**	**30**	**21%**	**100%**	**130**	**89%**

Gbongogo

Grade Bin Au g/t	Fresh			FW Fresh			Trans			Oxide			Total		
	Deposit t %	Leach Samples #	Leach Samples %	Deposit t %	Leach Samples #	Leach Samples %	Deposit t %	Leach Samples #	Leach Samples %	Deposit t %	Leach Samples #	Leach Samples %	Deposit t %	Leach Samples #	Leach Samples %
<0.50	0%	0	0%	0%	0	0%	1%	0	0%	0%	0	0%	1%	0	0%
0.50 – 0.75	8%	0	0%	2%	0	0%	1%	0	0%	2%	0	0%	14%	0	0%
0.75 – 1.00	19%	0	0%	3%	0	0%	1%	0	0%	1%	1	1%	23%	1	1%
1.00 – 1.25	13%	2	1%	1%	0	0%	0%	0	0%	1%	0	0%	15%	2	1%
>1.25	37%	6	4%	5%	0	0%	2%	4	3%	3%	3	2%	47%	13	9%
	77%	**8**	**5%**	**11%**	**0**	**0%**	**5%**	**4**	**3%**	**7%**	**4**	**3%**	**100%**	**16**	**11%**

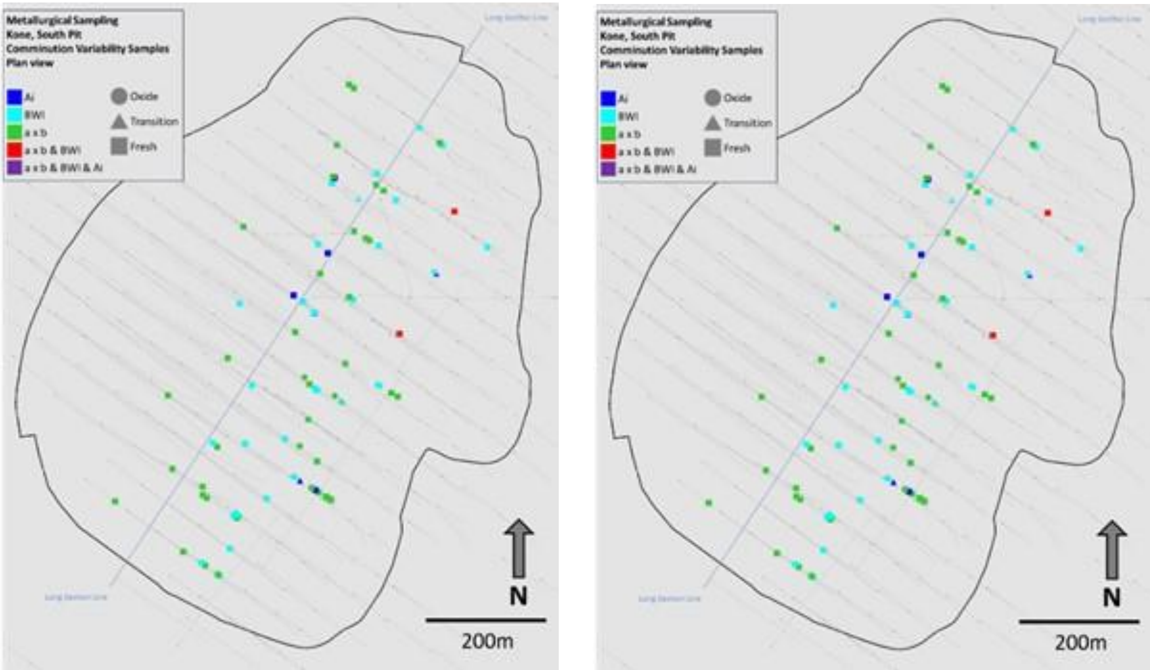
Grade Bin Au g/t	Fresh			FW Fresh			Trans			Oxide			Total		
	Deposit t %	Leach Samples #	Leach Samples %	Deposit t %	Leach Samples #	Leach Samples %	Deposit t %	Leach Samples #	Leach Samples %	Deposit t %	Leach Samples #	Leach Samples %	Deposit t %	Leach Samples #	Leach Samples %
<0.50	26%	17	12%	4%	3	2%	2%	7	5%	3%	7	5%	35%	34	23%
0.50 – 0.75	24%	15	10%	4%	3	2%	1%	4	3%	2%	8	5%	31%	30	21%
0.75 – 1.00	14%	22	15%	1%	3	2%	1%	4	3%	1%	10	7%	17%	39	27%
1.00 – 1.25	8%	13	9%	0%	4	3%	0%	6	4%	0%	4	3%	9%	27	18%
>1.25	8%	6	4%	0%	0	0%	0%	5	3%	1%	5	3%	9%	16	11%
	79%	**73**	**50%**	**10%**	**13**	**9%**	**5%**	**26**	**18%**	**6%**	**34**	**23%**	**100%**	**146**	**100%**

The QP considers that the sample selection provides an adequate basis for the determination of gold recoveries and reagent consumption by domain for the Koné and Gbongogo deposits.

12.2.2 Metallurgical Sample Locations

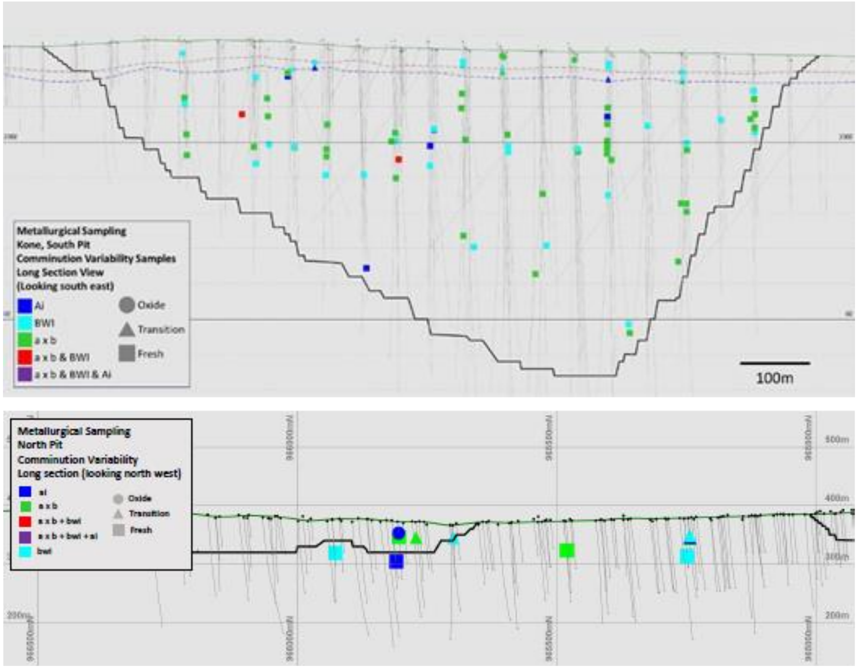
The location of the comminution samples from all testwork campaigns are shown in Figure 12.2.1, Figure 12.2.2, Figure 12.2.5 and Figure 12.2.6.

Figure 12.2.1 Koné Comminution Sample Locations - Plan



Source: Montage, January 2024.

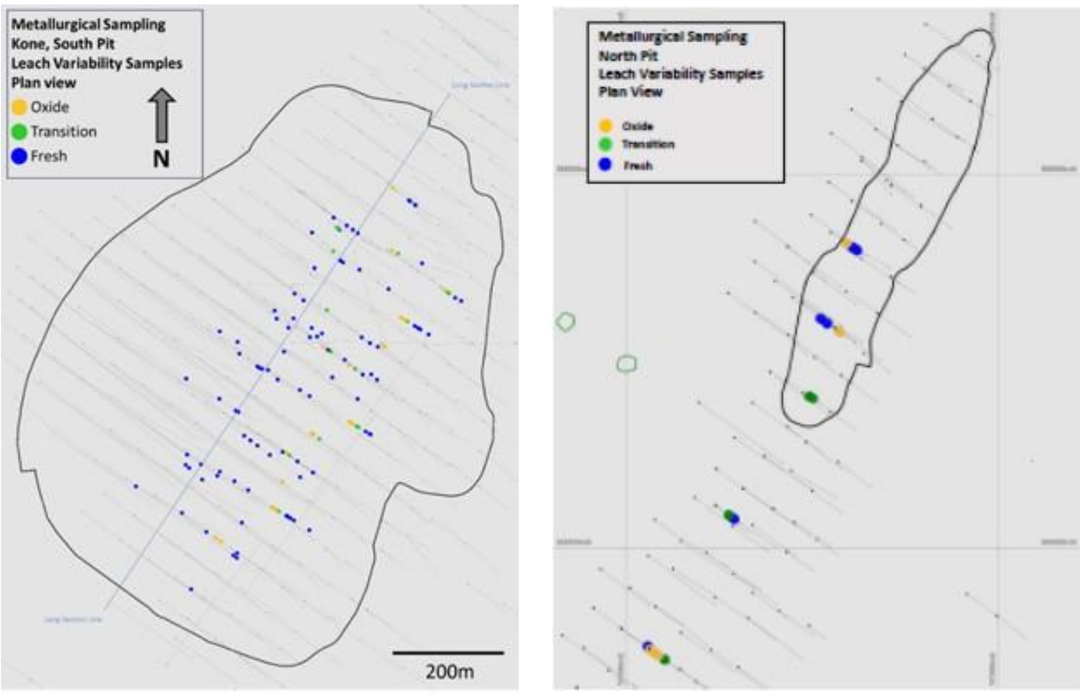
Figure 12.2.2 Koné Comminution Sample Locations -Section



Source: Montage, January 2024.

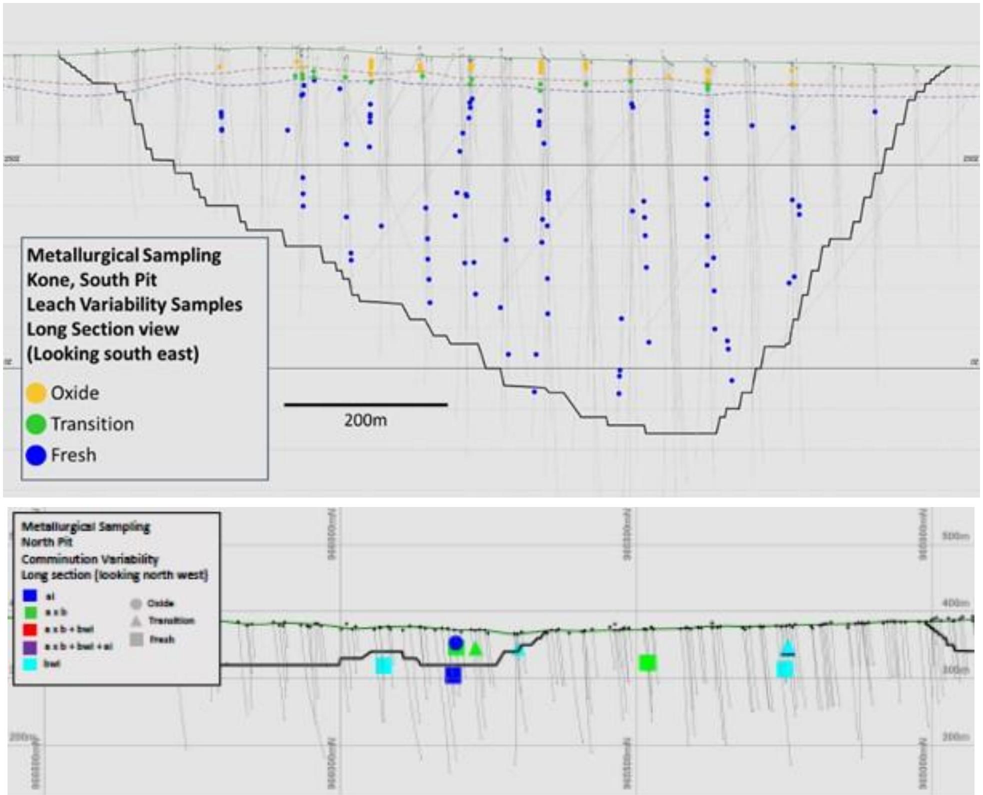
The location of the leach variability samples from all testwork campaigns are shown in Figure 12.2.3 and Figure 12.2.4.

Figure 12.2.3 Koné Leach Variability Sample Locations - Plan



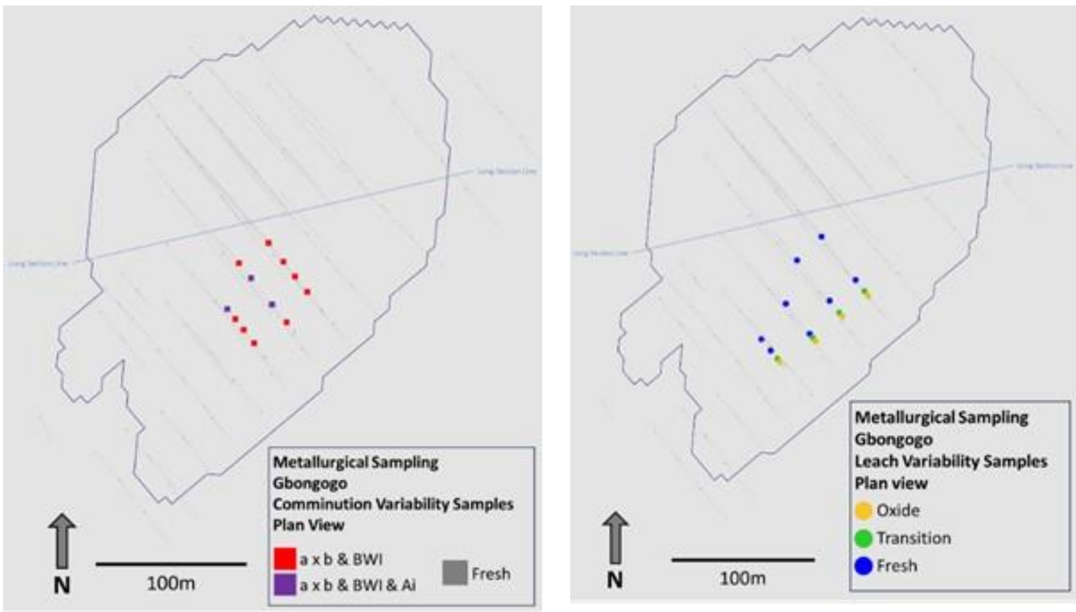
Source: Montage, January 2024.

Figure 12.2.4 Koné Leach Variability Sample Locations - Section



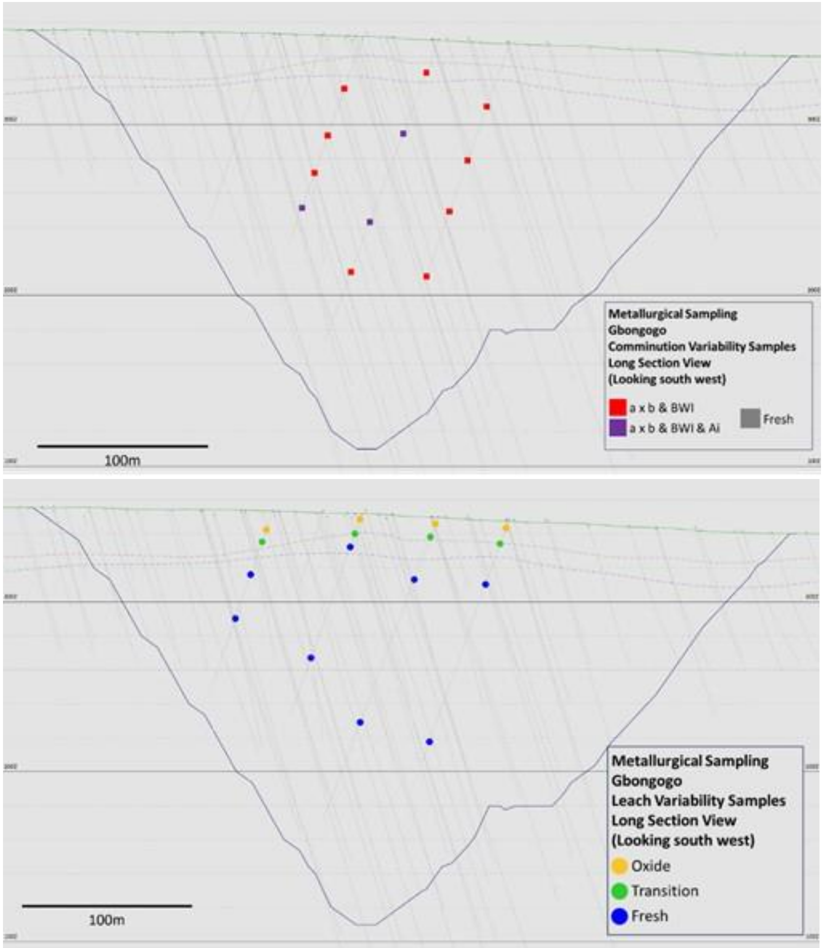
Source: Montage, January 2024.

Figure 12.2.5 Gbongogo Comminution and Leach Sample Locations - Plan



Source: Montage, January 2024.

Figure 12.2.6 Gbongogo Comminution and Leach Variability Sample Locations - Section



Source: Montage, January 2024.

12.3 Mining

Verification checks undertaken by the QP to confirm the validity of information for the block models compiled for Koné and Gbongogo include the following:

- Density is consistent with material types.
- High grade areas are consistent with drillhole trace grades.
- Domains in the model match the oxidation surfaces.
- Deswik model validation routine to ensure models are valid (i.e. no overlapping blocks, no double counted material, densities within a set range, grades within a set range).

In the opinion of the QP, the parameters used to produce the pit shells and define cut-off grades are appropriate for providing MRE consistent with CIM guidelines (CIM 2014).

12.4 Hydrogeology

Verification checks undertaken by the QP to confirm the validity of information for the groundwater monitoring, and testing for Koné and Gbongogo include the following:

- Groundwater quality analysis was checked for ion-balance (target is ±10%) and anomalies when compared with historical datasets. Any anomalies were cross-checked to determine evaluated and, when necessary, re-sampling was undertaken.

- Groundwater level monitoring datasets were checked for anomalies and, where necessary, boreholes re-measured.

- Aquifer test data was reviewed by qualified Senior Hydrogeologist for typing errors or where pump tests procedures were not in line with international best practice.

- Groundwater models were calibrated against groundwater monitoring datasets.

In the opinion of the QP, the groundwater data used to produce the hydrological models is appropriate and adequate.

13.0 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 0.49 kg/t CaO.

13.2 Metallurgical Testing 2018

In September 2018, ALS Global (ALS), Report No. A18882, 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 h residence time.

The results are summarised in Table 13.2.1.

Table 13.2.1 Summary of 2018 Cyanide Leach Testwork Results

Leaching Testwork: Summary of Results						
Comp. ID	**Crush / Grind Size**	**Leach Duration h**	**Leach Type**	**Au Grades g/t**		**Au Extraction %**
				Head	**Tail**	
Oxide	P_{100} 20 mm	504	Coarse-crush IBR	1.49	0.06	96.4
	P_{100} 10 mm			1.43	0.05	96.5
	P_{100} 5 mm			1.10	0.05	95.5
	P_{100} 1 mm			1.20	0.06	95.2
	P_{100} 75 µm	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} 20 mm	504	Coarse-crush IBR	0.94	0.19	80.7
	P_{100} 10 mm			1.28	0.31	76.1
	P_{100} 5 mm			0.98	0.21	79.2
	P_{100} 1 mm			0.98	0.11	88.9
	P_{100} 75 µm	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} 20 mm	504	Coarse-crush IBR	1.20	0.75	37.1
	P_{100} 10 mm			1.06	0.52	51.2
	P_{100} 5 mm			1.24	0.53	57.4
	P_{100} 1 mm			0.87	0.19	78.7
	P_{100} 75 µm	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} 10 mm	504	Coarse-crush IBR	1.85	1.16	37.3
	P_{100} 5 mm			1.89	0.89	51.9
	P_{100} 75 µm	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 to 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.

13.3 PFS 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 (SGS Lakefield Report 17236-1).

13.3.1 Variability Comminution and 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 13.3.1 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.

13.3.2 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 optimised leach conditions were as follows:

- Grind size P_{80} target of 75 µm.

- Pulp density 50% solids (w/w).
- Pulp pH 10.5 to 10.7 (maintained with lime).
- 36 h leach retention time.
- Cyanide concentration of 0.5 g/L NaCN (maintained for 8 h).
- 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 to 0.07 g/t Au) in all tests performed excluding test 1, 2, 3 and 4 at coarser grind sizes, and test 29 at shorter (24 h) residence time.

Cyanide and lime consumptions were also consistently low at 0.18 kg/t and 0.25 kg/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 percent (%) solids confirmed that it is possible to increase pulp density on the fresh mineralisation, to 55% solids. A pulp density of 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.

13.3.3 Leach Variability Testing

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

The average back calculated head assay was 1.10 g/t Au.

The leach kinetic results indicated that the samples continued to leach over the entire 36 h leach retention time for the majority of the samples. The average cyanide and lime consumptions were all low for fresh mineralisation 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 13.3.1 shows the relationship between Au g/t 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 13.3.1 Relationship Between Grade in Feed and Grade in Residues by Domain

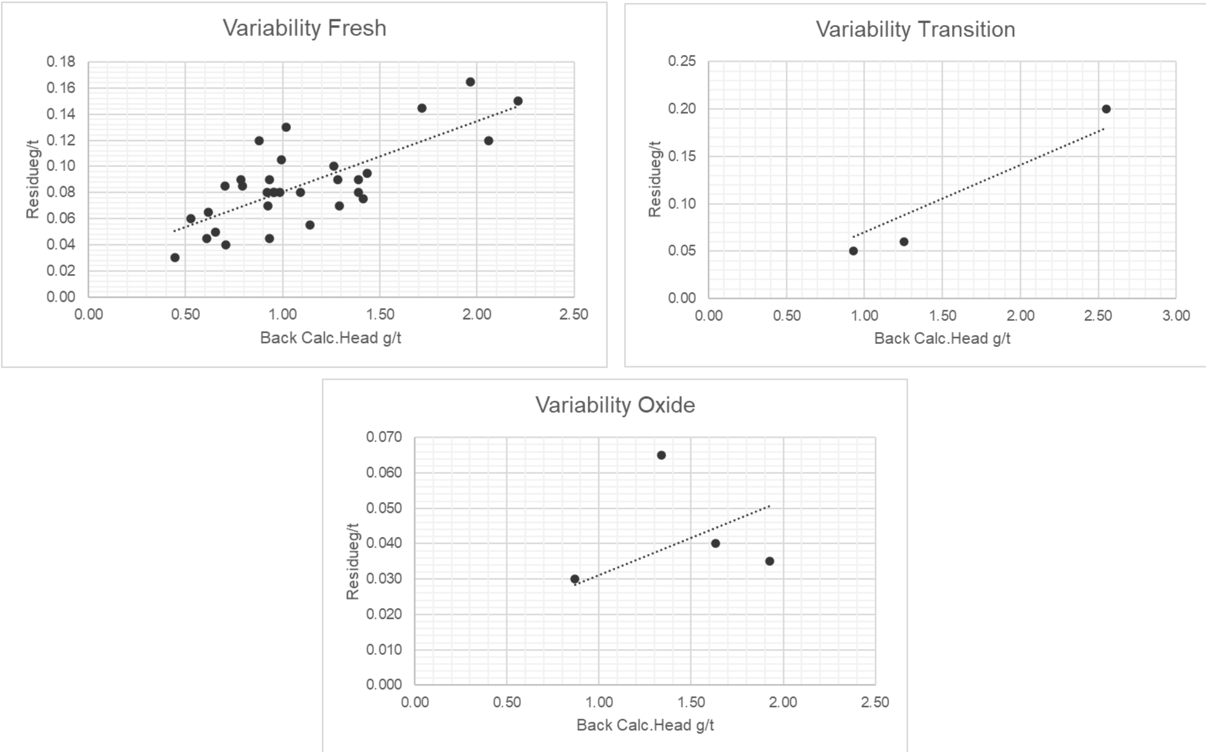


Figure 13.3.2 indicates that there is a linear relationship between feed sulphur and residue grade. This was consistent across all samples tested.

Figure 13.3.2 Fresh Samples Head Sulphur and Residue Gold Grade

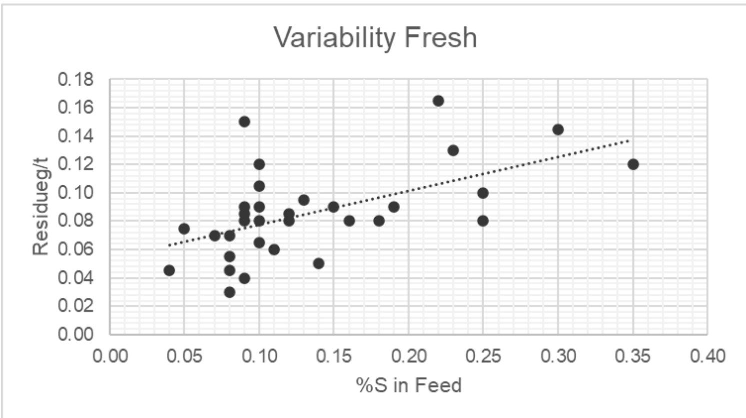
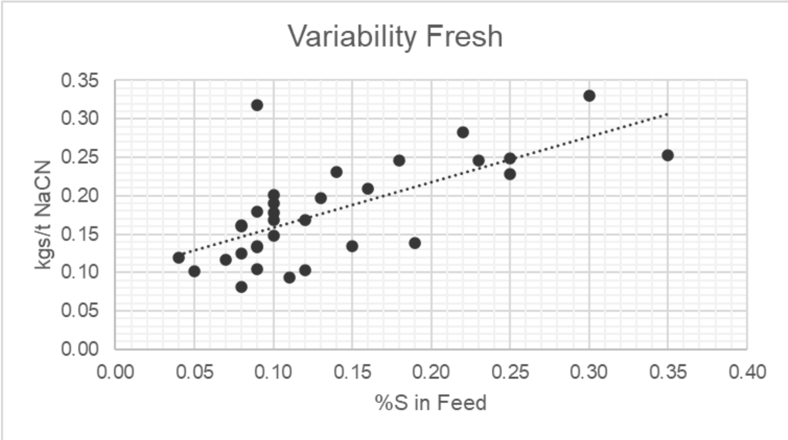


Figure 13.3.3 indicates that there is a linear relationship between feed sulphur and cyanide consumption. This was consistent across all samples tested.

Figure 13.3.3 Fresh Samples Head Sulphur and Cyanide Consumption



13.3.4 Carbon Modelling

Full carbon modelling testwork was performed on the fresh composite sample and a 60% fresh to 40% oxide blend sample to determine whether the oxide impeded carbon loadings. Leach and adsorption kinetics were also conducted on an 80% fresh : 20% oxide blend sample.

The kK constants are summarised below in Table 13.3.2.

Both mineralisation types show moderate adsorption kinetics.

Table 13.3.2 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.

13.3.5 Tailings Sample Generation

Three samples were generated for geotechnical and environmental testing by Knight Piésold ('KP'). The samples were generated using the fresh, transition and oxide composites.

13.3.6 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 solid / liquid (S/L) separation tests. The samples were ground to the target grind size P_{80}, discharged from the mills and provided as a slurry.

The results are summarised in Table 13.3.3.

Table 13.3.3 Dynamic Thickening Testwork Results

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

The testwork shows that the fresh and 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 35 g/t coagulant to control thickener overflow clarity.

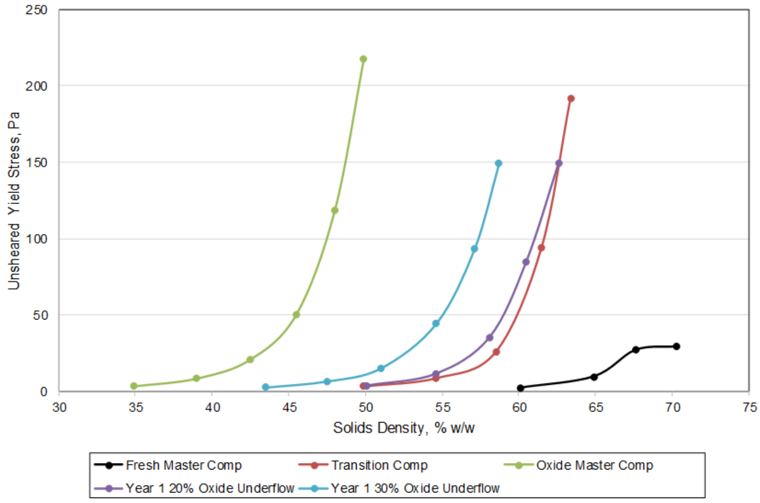
13.3.7 Rheological Tests

Five samples were submitted for rheology testwork. The three mineralisation 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 P_{80} 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 13.3.4 summarises 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 13.3.4 Thickener Test Yield Stress Curve



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

13.4 DFS Metallurgical Testing 2021 and 2023

The 2021 scope of work on samples from Koné 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 mineralisation.
- Extend variability testing, specifically on low grade samples to verify metallurgical response at the lower grades at site ambient temperature and design dissolve oxygen (DO) conditions.
- Further metallurgical testing to re-evaluate whether introduction of a gravity stage has a beneficial effect on metallurgy.

The 2023 scope of work on samples from Gbongogo satellite deposit was designed to determine the response of Gbongogo metallurgical response under the Koné parameters.

13.4.1 Variability Comminution and Physical Testing

The sample selection and testwork 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.

A total of 72 SMC tests were conducted over all studies (2018, 2020, 2021, and 2023) and the total data base was used to support the DFS comminution circuit design.

The SMC results of all studies are summarised in Table 13.4.1.

Table 13.4.1 Summary of All SMC Test Results

Domain	No. 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	53	87.7	0.37	32	81	0.3	8.85	77	23.9	18.7	9.7	11.24	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	21.3	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 characterised as moderately hard with respect to resistance to impact breakage. The three oxide saprolite samples tested resulted in extremely soft A x bs, and were not used in the comminution design. The four oxide saprock samples tested were classified as very soft, with A x bs 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 13.4.2. The samples fell in the hard to very hard range of hardness of the SGS database.

Table 13.4.2 Summary of All Bond Low Energy Impact Test Results

Domain	No. Sample	Average kWh/t	Min. kWh/t	Max. kWh/t	Std Dev. kWh/t	Relative Density	Hardness Percentile
Fresh	13	16.4	7.5	30.7	6.0	2.75	84
Trans	1	8.5	3.3	15.2	2.6	2.67	37

A total of 68 Bond BWi tests were performed using a closed screen size of 106 µm and these results for all studies are summarised in Table 13.4.3.

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 13.4.3 Summary of All Bond Ball Mill Work Index Test Results

Domain	No. Sample	Mesh of Grind	F_{80} mm	P_{80} mm	Gram per Revolution	Work Index kWh/t	Hardness Percentile	Feed Passing %	Bulk Density	Closing Screen Size mm	Mib kWh/t
Fresh	52	150	2,381	86	1.9	11.8	25.6	13.4	1,859	106	15.0
FW Fresh	3	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 18 samples were subjected to the Bond Abrasion Index (Ai) during all studies. The test results are summarised in Table 13.4.4. The Ai values ranged from 0.120 g to 1.040 g, meaning that the samples ranged from very mild to very abrasive when placed in the SGS database. Gbongogo fresh was found to be the most abrasive domain with an average Ai value of 0.78 g.

Table 13.4.4 Summary of All Abrasion Work Index

Domain	# Sample	Ai	Percentile of Abrasivity
Fresh	11	0.497	73
FW Fresh	0		
Trans	4	0.152	34
Oxide	3	0.115	21

A Koné fresh ore sample was sent to SGS Lakefield for a full suite of HPGR testing, with two Koné footwall fresh ore samples for SPT. The full HPGR investigation, included batch testing and a locked-cycle test, the Bond ball mill grindability test on both the HPGR feed and HPGR product and the SPT test.

The summary of these results is shown in Table 13.4.5.

Table 13.4.5 Grindability Summary – BWi and STP Tests

Sample Name	BWi kWh/t	HPi kWh/t
HW Fresh HPGR Feed	12.2	13.2
HW Fresh HPGR Product	10.0	-
HW Fresh (A)	-	13.4
HW Fresh (B)	-	14.5

The bulk sample and the SPT test data (SGS 17236-04) gave very similar high pressure grinding index (HPi) of 13.2 kWh/t for the bulk sample and 13.4 kWh/t and 14.5 kWh/t for the two samples. All three HPi values are categorised as 'medium' in terms of the materials amenability to HPGR crushing.

The HPGR grinding data is summarised in Table 13.4.6.

Table 13.4.6 HPGR Grindability Data

Test / Sample Name	Operating Cond.	t/h	Net kWh/t	N/mm²	m_f	CL[1] %	P_{80}	BWi kWh/t[2]	Reduction kWh/t[2]	Reduction kWh/t[3]
HPGR Batch Test	45 bars / 2% moist.	3.9	1.93	3.80	349	-	4,324	-	-	-
HPGR Locked-Cycle Test	45 bars / 2% moist.	2.5	2.60	3.50	336	51	1,961	-	-	-
HW Fresh HPGR Feed	-	-	-	-	-	-		12.2	-	-
HW Fresh HPGR Product	-	-	-	-	-	-		10.0	18	28

[1]Circulating load.
[2]kWh/t reduction based on direct work indices.
[3]kWh/t reduction based on [gross gram per revolution]$^{-0.82}$

The locked cycle test was performed at 2% moisture and initial nitrogen and hydraulic pressures of 30 and 45 bars respectively. The bulk sample was characterised as very soft in terms of specific throughput (Mf) and the specific energy requirement (kWh/t) was medium hard.

Furthermore, the Bond ball mill grindability test, performed on the HPGR product, measured a reduction of 18% in the BWi, from 12.2 kWh/t to 10.0 kWh/t. Due to the additional fines created by the HPGR (similar to SAG mills), an overall reduction of 28.0% in the power requirement would be expected (based on equivalent preparation to passing six mesh).

13.4.2 XRD Clay Speciation

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

13.4.3 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:

- Four E-GRG tests.
- Four bulk gravity separation and cyanidation tests (concentrate and tail leaches).
- Variability gravity separation and cyanidation tests.

Table 13.4.7 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 13.4.7 Summary of E-GRG Testwork

Summary of E-GRG Testwork				
Sample No.	2	3	4	5
Au g/t	0.89	1.76	0.64	1.00
E-GRG	50	6	42	49
Av GRG Size, µm	52	70	50	46
FLS GRG	17	25	14	16

13.4.4 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. From this, 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 to 46 µm, whereas the chalcopyrite was more consistently finer ~20 µm.

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

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

13.4.5 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 and 99 DFS) were tested using the optimised conditions as follows:

- 1 kg standard bottle roll tests.
- 35° to 40°C pulp temperature.
- Grind size P_{80} target of 75 µm.
- Pulp density 50% solids (w/w).

- Pulp pH 10.5 to 10.7 (maintained with lime).
- 36 h 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 h of the test. After 8 h, 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 g 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 h).

Table 13.4.8 summarise the 146 cyanide leach variability test results conducted to support the DFS study.

Table 13.4.8 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 NaCN	CaO	Au Extraction, % 2 h	4 h	8 h	12 h	24 h	32 h	36 h	Avg. g/t Au Residue	Back Calc Au , g/t	Direct g/t Au	Cu Head g/t Meas	%S Head Meas
SOUTH HW FRESH	min	59	17	0.08	0.30	22.3	27.3	43.5	55.1	76.4	79.2	79.8	0.03	0.26	0.24	13	0.01
SAMPLES	max	78	27	0.39	0.81	77.6	86.9	87.8	91.0	96.0	97.9	98.1	0.19	2.41	5.61	1490	0.35
53	avg.	71	21	0.22	0.47	48.4	61.0	73.6	80.1	88.1	89.3	90.1	0.08	0.86	0.86	237	0.15
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	0.25	0.23	75	0.38
SAMPLES	max	76	26	0.61	0.52	75.0	78.6	81.4	82.5	87.2	87.8	87.8	0.23	1.84	1.76	2100	1.96
12	avg.	70	20	0.37	0.43	59.6	66.2	71.8	74.2	77.1	78.3	79.2	0.13	0.71	0.64	509	0.90
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	0.40	0.41	132	0.03
SAMPLES	max	78	23	0.59	1.10	64.7	72.3	79.8	85.6	94.2	97.7	95.6	0.30	1.62	2.08	949	0.44
13	avg.	72	19	0.23	0.45	44.4	54.6	69.1	75.8	85.2	87.7	88.7	0.10	0.88	0.83	393	0.12
GB FRESH	min	71	15	0.27	0.41	32.7	61.8	71.2	76.4	80.2	80.1	83.6	0.16	1.02	0.87	7	0.79
SAMPLES	max	81	18	0.57	0.92	67.2	72.0	84.2	86.0	88.5	88.1	91.6	0.25	2.93	2.69	49	1.05
8	avg.	75	16	0.42	0.55	46.3	66.9	76.9	79.7	83.3	83.4	86.3	0.20	1.54	1.45	16	0.91
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	0.26	0.25	16	0.01
SAMPLES	max	78	24	0.92	2.18	86.5	92.6	95.9	94.7	98.4	98.3	97.9	0.29	2.73	1.40	1050	0.17
17	avg.	69	21	0.18	0.99	40.7	54.4	71.2	79.4	89.2	91.5	92.4	0.06	0.90	0.76	246	0.02
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	0.30	0.33	180	0.02
SAMPLES	max	69	25	0.76	1.34	64.0	73.1	84.1	87.3	90.5	91.5	91.7	0.14	1.25	1.10	1070	0.43
5	avg.	68	22	0.35	0.75	47.6	57.8	68.7	76.1	87.0	88.8	89.1	0.08	0.77	0.68	604	0.14
GB TRANS	min	66	18	0.11	0.86	44.8	59.9	71.7	74.7	80.7	81.1	84.7	0.08	0.95	0.79	8	0.05
SAMPLES	max	71	19	0.32	1.34	72.4	81.4	88.3	88.7	90.8	90.8	95.6	0.23	2.07	2.16	23	0.53
4	avg.	68	18	0.21	1.06	63.5	73.7	81.8	83.4	86.0	86.6	90.8	0.14	1.61	1.49	15	0.29
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	0.25	0.25	45	0.01
SAMPLES	max	87	22	0.63	4.48	93.4	97.2	99.0	99.0	99.2	99.8	98.2	0.07	1.99	1.18	1360	0.01
21	avg.	69	19	0.18	2.50	69.4	81.9	90.8	92.2	95.0	95.1	95.6	0.03	0.88	0.65	285	0.01
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	0.32	0.38	191	0.01
SAMPLES	max	71	22	0.18	6.16	94.3	96.1	97.0	99.0	99.0	99.9	97.5	0.06	0.80	0.72	591	0.02
9	avg.	66	20	0.13	2.79	87.8	92.4	93.2	95.2	96.1	95.5	95.1	0.03	0.59	0.54	367	0.01
GB OXIDE	min	71	17	0.12	1.17	50.7	67.6	82.6	88.8	90.9	92.3	92.4	0.08	1.57	1.35	14	0.01
SAMPLES	max	83	18	0.42	3.66	65.7	78.0	86.5	90.2	94.2	96.6	96.3	0.18	3.73	3.72	43	0.08
4	avg.	77		0.29	2.60	55.5	72.9	84.7	89.5	92.3	94.2	94.5	0.13	2.40	2.15	27	0.03

The average gold concentration in leach residues achieved was good averaging 0.08 g/t Au on the predominant Koné South fresh ore domain which represents 71.8% of gold ounces and 0.13 g/t Au on the South FW fresh domain which represents 7.3% of gold ounces. The smaller Koné South transition and oxide ore domains representing 3.4% and 4.5% of gold ounces respectively have better leach extraction efficiencies and the average leach residue grades were 0.06 g/t Au and 0.03 g/t Au respectively. The Gbongogo fresh (residue 0.20 g/t and 11.0% gold ounces) represents 11.0% of the gold ounces, whilst the transition (residue .09 g/t) and oxide (residue .09 g/t), which represent 0.3% and 0.4% respectively, domain assays were typically higher due to the higher feed assays.

Figure 13.4.1, Figure 13.4.2 and Figure 13.4.3 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 13.4.1 Relationship Between Grade in Residues and Grade in Feed for Fresh Domains

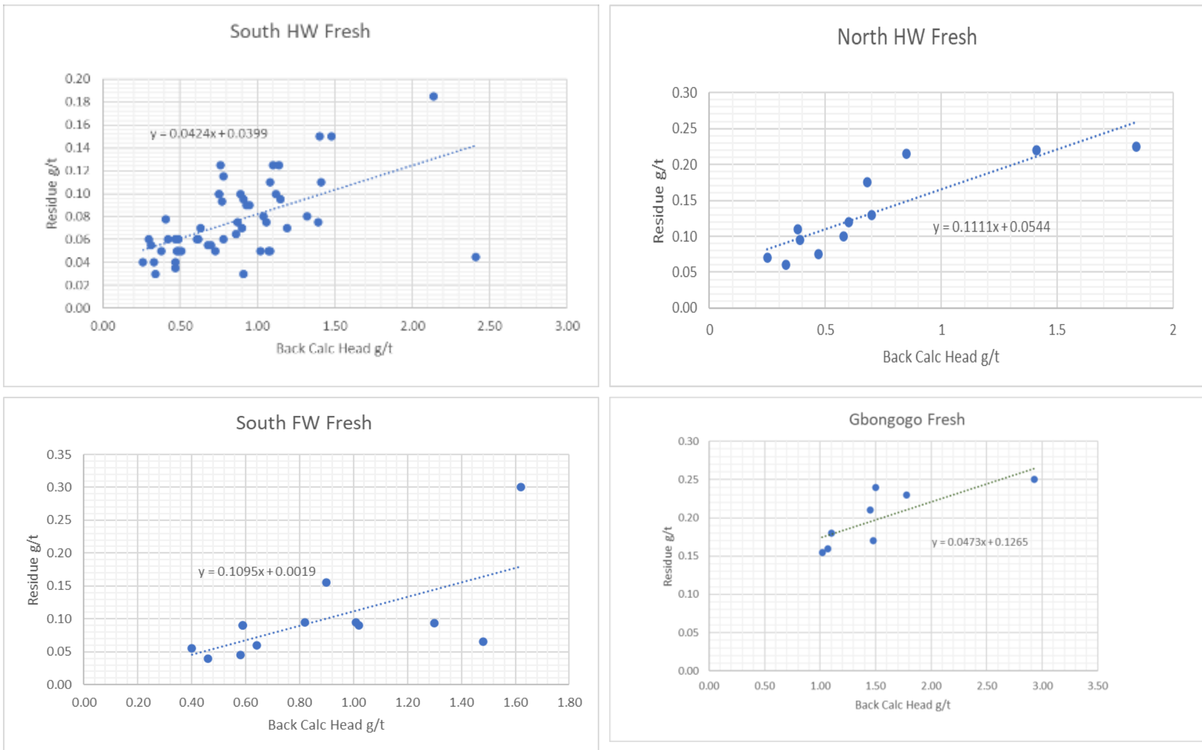


Figure 13.4.2 Relationship Between Grade in Residues and Grade in Feed for Transition Domains

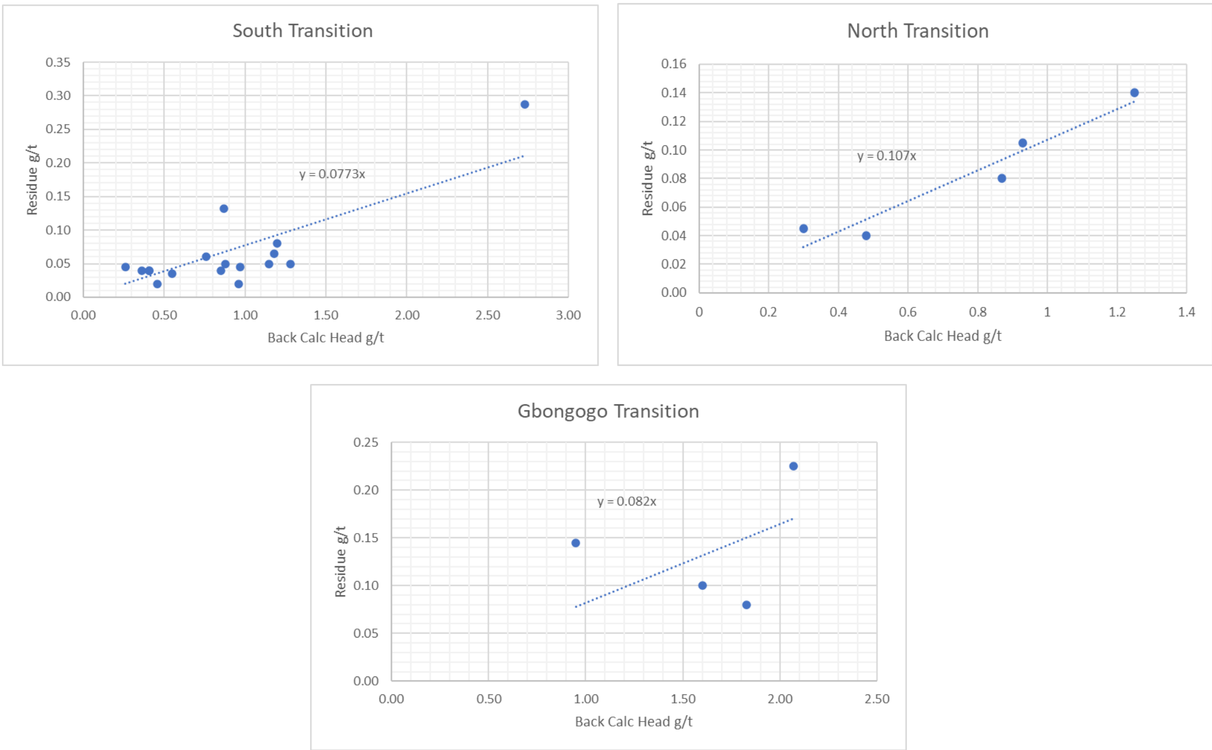


Figure 13.4.3 Relationship Between Grade in Residues and Grade in Feed for Oxide Domains

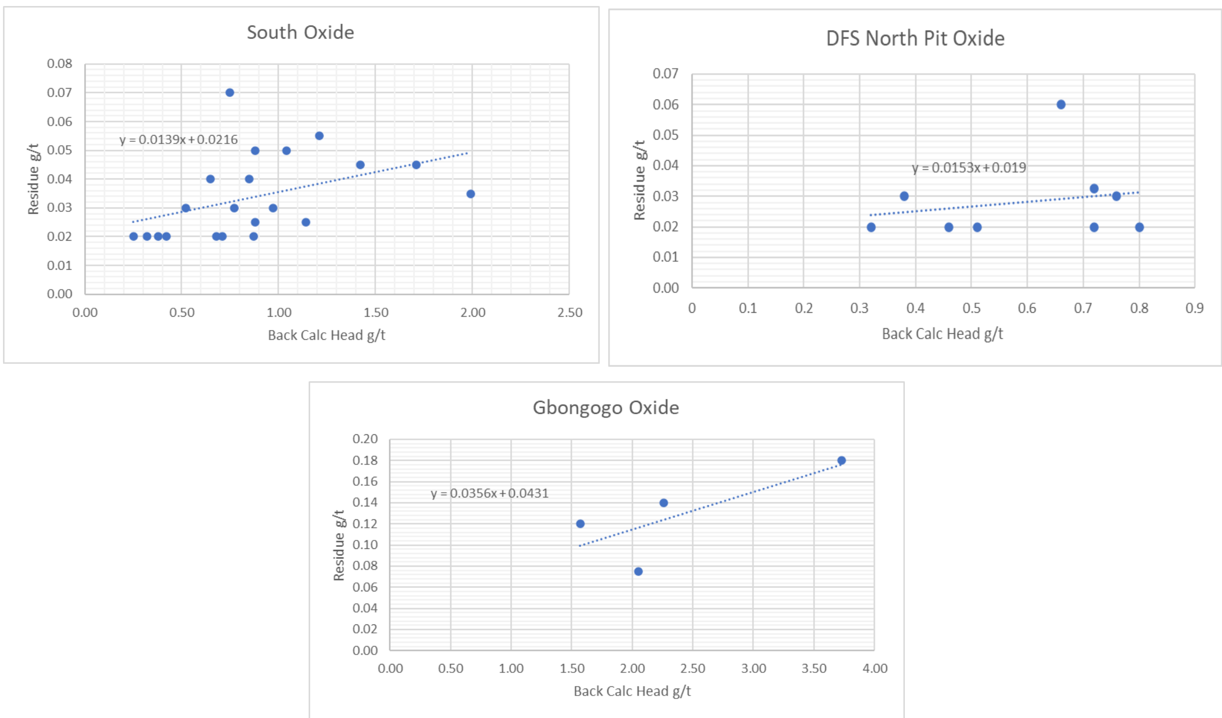


Figure 13.4.4 shows the gold leach extraction kinetics for each of the ore domains. The results indicated that oxide samples had the fastest leach kinetics and leaching was essentially complete after ~24 h. The fresh, fresh FW and transition test results indicated that the samples continued to leach over the entire 36 h leach retention time. North HW fresh is the worst performer due to the elevated sulphides which were found to encapsulate the gold

Figure 13.4.4 Gold Leach Kinetics

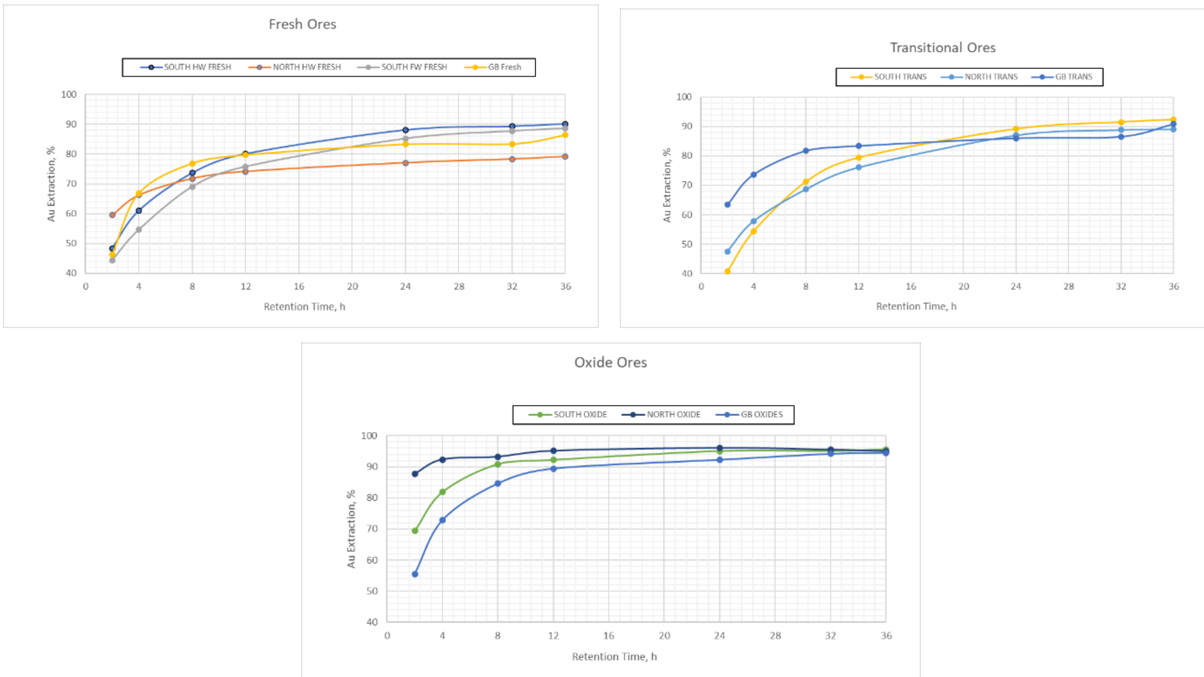


Figure 13.4.5 show cyanide consumption rates were generally all low averaging 0.25 kg/t based upon domain tonnage weighted average for all samples tested.

Gbongogo fresh domain was found to consume more cyanide due to the elevated %S and north fresh and transition domains were found to consume more cyanide due to the localised higher copper levels

Figure 13.4.5 Summary of Sodium Cyanide Consumptions

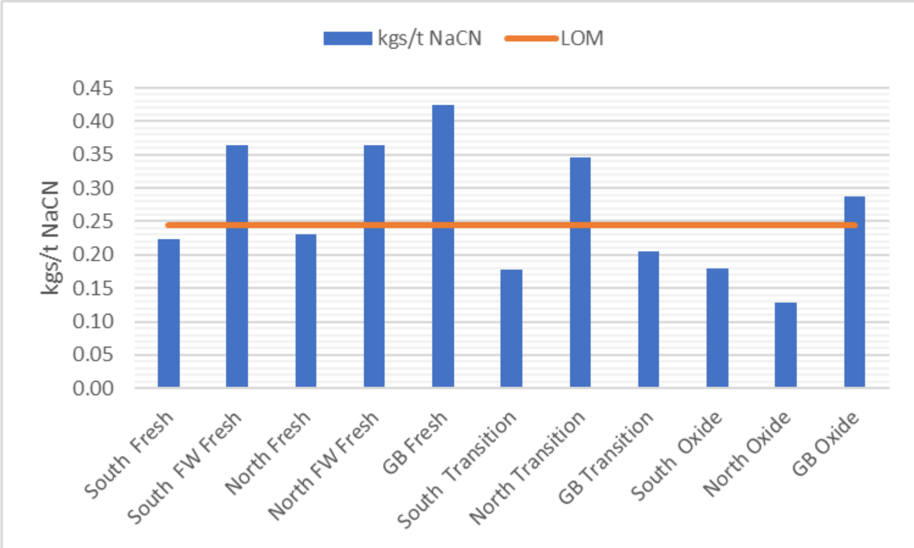
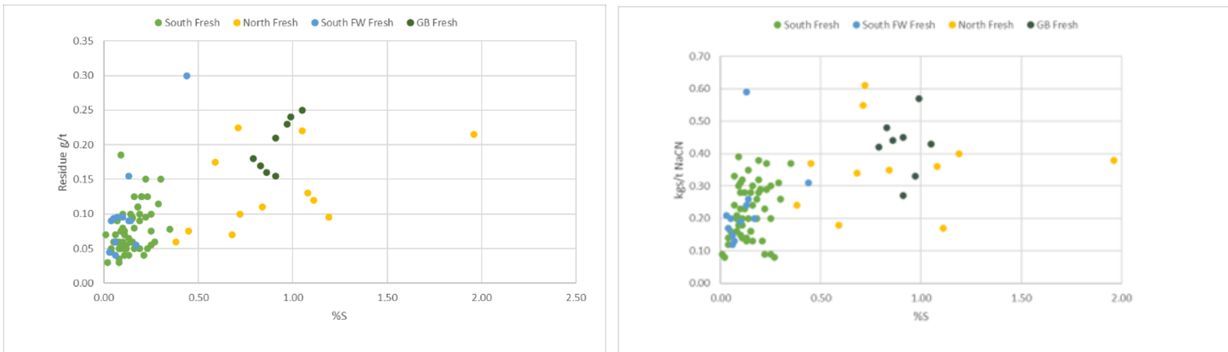


Figure 13.4.6 shows the relationship between %sulphur and cyanide leach residue and sodium cyanide consumption for all fresh samples tested.

Figure 13.4.6 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 13.4.7 provides a breakdown of the reported sodium cyanide consumption by species.

For the predominant Koné 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 Koné transition ore domains appear to be much more associated with copper in solution, whereas for Koné 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.

The Gbongogo domains cyanide consumption are predominantly linked to iron levels. The absence of copper sulphides in Gbongogo ore is apparent compared to the Konè domains.

Figure 13.4.7 Cyanide Consumption Breakdown by Species

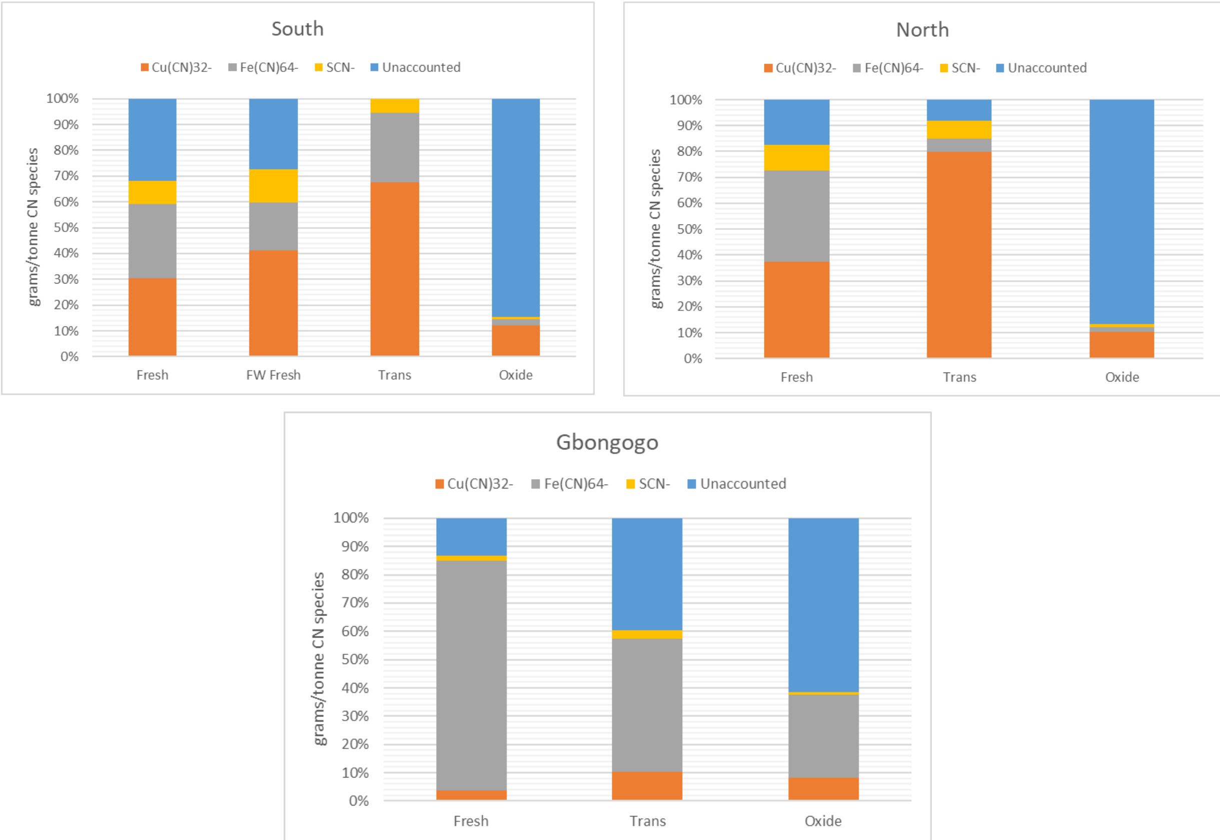
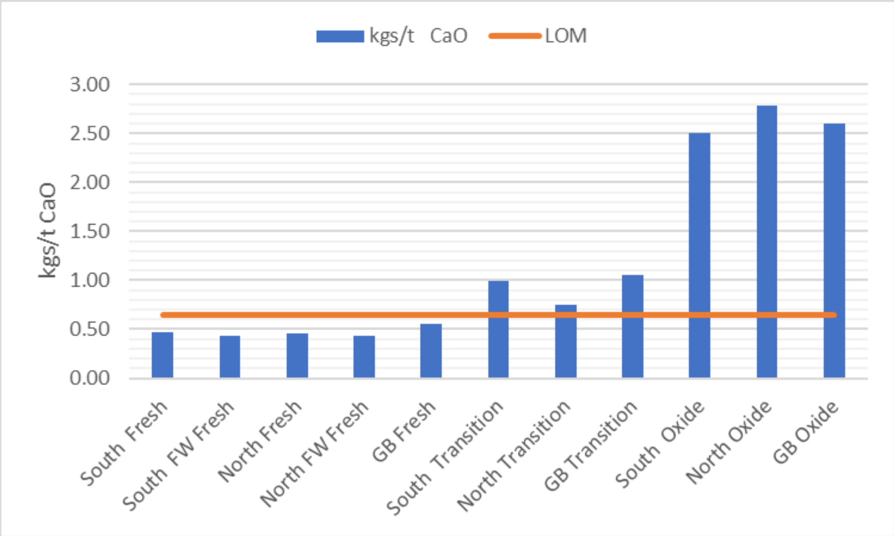


Figure 13.4.8 summarises lime consumption by domain. The three oxide domains were found to consume significantly more lime than the transition and fresh domains, but their impact on overall LOM consumptions is low because of the relatively tonnages in these domains.

Figure 13.4.8 Summary of Lime Consumptions



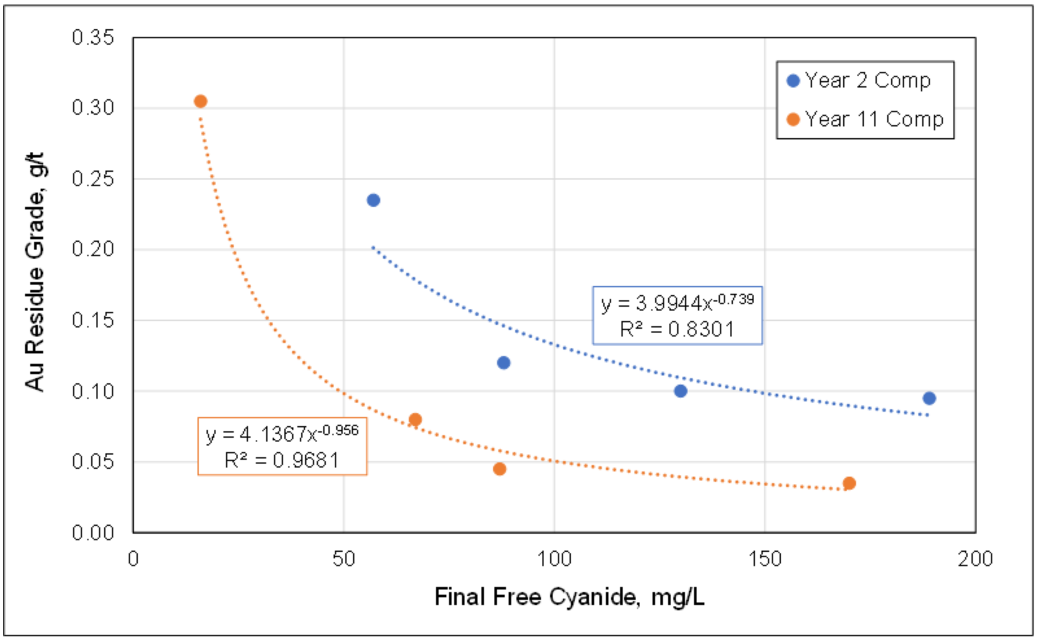
13.4.6 Sodium Cyanide Decay Testing

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 h (0.25 to 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 h 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 13.4.9 shows that ~130 ppm 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 13.4.9 Relationship Between Free Cyanide and Grade in Residues



13.4.7 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, SGS 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 and Year 11 composite respectively.

13.4.8 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.

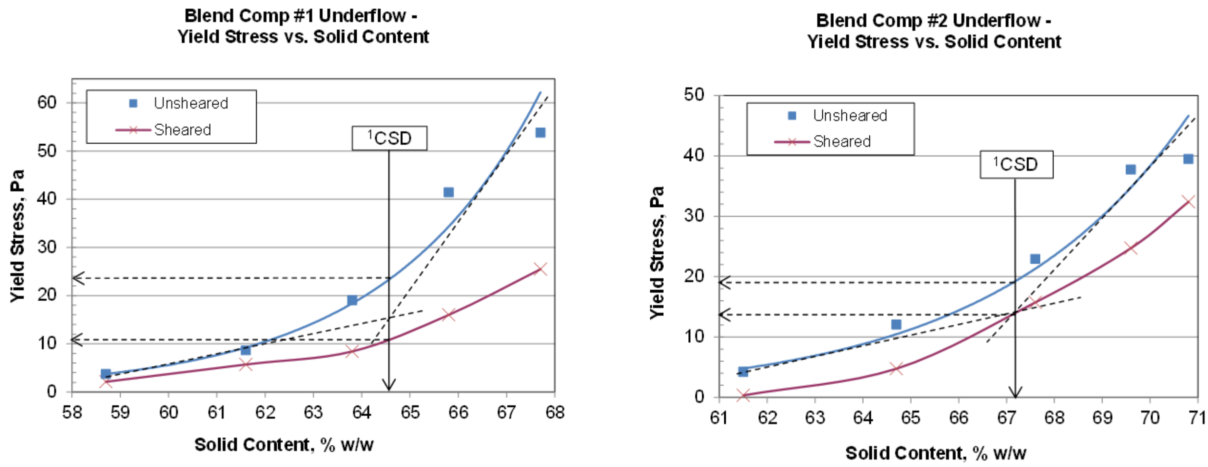
The objective was to confirm if coagulant is required in the design with the lower quantities (10% maximum) of oxide in the typical plant feed blend.

The results show that adding 5 g/t coagulant produced less than 200 ppm 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.

13.4.9 Rheology

Figure 13.4.10 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 13.4.10 Yield Stress vs. Solids Content 10% Oxides



13.5 Metallurgical Results Summary

13.5.1 Comminution

Table 13.5.1 summarises the comminution testwork that has been conducted in all (2018, 2020, 2021 and 2023) studies.

A total of 68 JK Tech SMC, 68 Bond BWi, 17 Abrasion Work Index and 14 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 mineralisation 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.

A 400 kg fresh ore sample was used for a full suite of HPGR testing, with a further two, 20 kg footwall fresh ore samples submitted for SPT. The results (400 kg bulk sample = 13.2 kWh/t, 20 kg SPT samples = 13.4 and 14.5 kWh/t) show that the fresh ore types tested are 'medium' in terms of their HPi values.

Table 13.5.1 Comminution Testwork

Ore Type	Deposit %	JK Tech SMC A x b			Ball Mill Work Index		Abrasion Index		Crusher Work Index	
		No. of Samples	Relative Density	JK SMC Axb	No. of Samples	Bond BWi kWh/t	No. of Samples	Bond Ai g	No. of Samples	Bond CWi kWh/t
Fresh	83%	52	2.74	31.9	52	11.8	11	0.5	13	16.4
FW Fresh	7%	3	2.77	31.1	3	9.7	-	-	-	-
Trans	4%	9	2.69	76.5	9	7.8	4	0.2	1	8.5
Oxide	6%	4	2.54	488.9	4	3.9	3	0.1	-	-
Total	**100%**	**68**	**2.68**	**59.6**	**68**	**33.6**	**18**	**0.4**	**14**	**11.9**

Figure 13.5.1 shows the variability in the JK Tech SMC A x b value for 55 fresh and nine transition samples tested during all (2018, 2020, 2021, and 2023) studies.

The predominant fresh and FW fresh domain, accounting for 89% 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% of planned processing tonnage and was found to be soft, and ranged from medium to very soft.

The oxide domain accounts for 6% 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 13.5.2 shows the variability of the Bond Low Energy Work Index 'Crusher Work Index' and Bond Abrasion Work Index tested and the total number of samples tested is shown (in brackets) on the y axis.

Figure 13.5.1 SMC and Bond Ballmill Work Index Variability by Domain

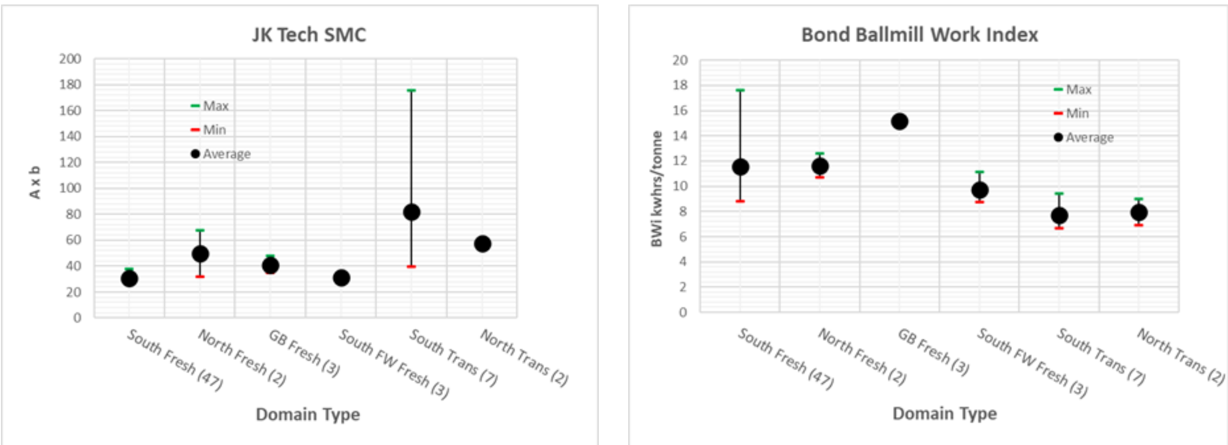
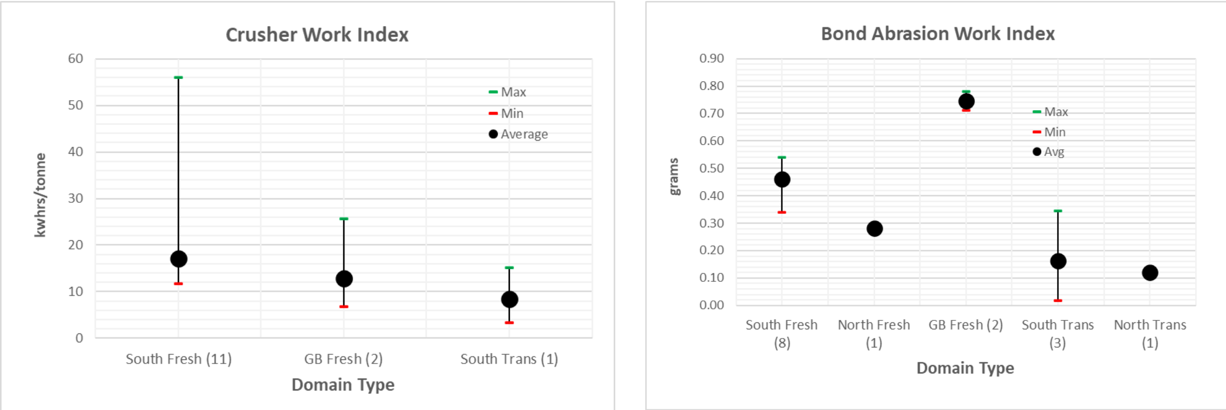


Figure 13.5.2 Bond Abrasion and Low Energy Impact (Crusher Work Index) Variability by Domain



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 Gbongogo fresh domain is the highest of all domains tested and the fresh domains are consistently more abrasive than the transition and oxide 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 observed higher resistance to SAG milling, 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. Labwal HPGR testing performed on a sample of Koné fresh and two SPT tests on samples of Koné FW fresh ore showed that the HPi values are rated as 'medium'. Further pilot scale HPGR testing is planned for 1Q24 to confirm equipment operating parameters and provide performance guarantees.

13.5.2 Metallurgical Data

Table 13.5.2 shows the summary of metallurgical testwork (SGS Lakefield Report 17236-04 and SGS Lakefield Report 17236-05), with recoveries reported on 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.

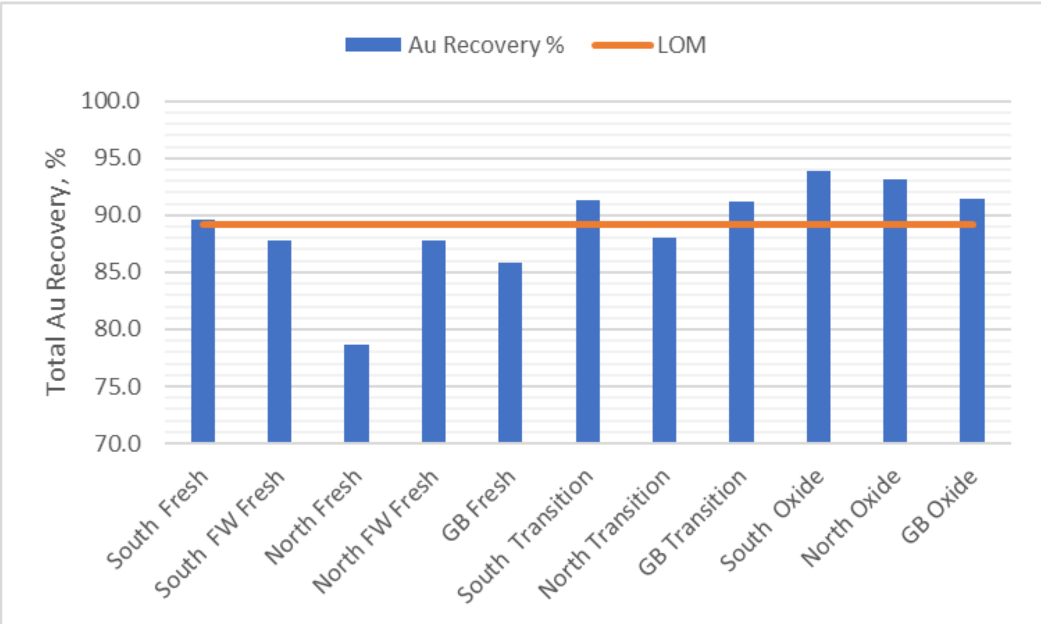
A solution assay of 0.005 mg/L has been used to calculate soluble loss of gold, the actual modelled mg /l Au were lower. Gold loss to carbon fines of 0.15% has been assumed.

Table 13.5.2 Summary of Metallurgical Data Produced

# Samples	Deposit	Domain	Processed '000 t	Processed Au g/t	Calc Residue	Au Leach Extraction %	Soluble Au Losses		Carbon Fines Loss %	Au Recovery %	kg/t NaCN	kg/t CaO
53	South	Fresh	129,510	0.69	0.07	89.99	0.005	0.72	0.15	89.1	0.22	0.47
13	South	FW Fresh	15,776	0.58	0.07	88.72	0.005	0.86	0.15	87.7	0.37	0.43
12	North	Fresh	416	0.51	0.11	78.25	0.005	0.98	0.15	77.1	0.23	0.45
8	GB	Fresh	9,427	1.46	0.20	86.60	0.005	0.34	0.15	86.1	0.42	0.55
17	South	Transition	6,957	0.60	0.05	92.27	0.005	0.83	0.15	91.3	0.18	0.99
5	North	Transition	425	0.44	0.05	89.30	0.005	1.15	0.15	88.0	0.35	0.75
4	GB	Transition	523	1.09	0.09	91.80	0.005	0.46	0.15	91.2	0.21	1.06
21	South	Oxide	9,628	0.59	0.3	94.93	0.005	0.85	0.15	93.9	0.18	2.50
9	North	Oxide	943	0.47	0.03	94.38	0.005	1.08	0.15	93.2	0.13	2.79
4	GB	Oxide	742	1.36	0.09	93.28	0.005	0.37	0.15	92.8	0.29	2.60
146	**Koné**	**LOM**	**174,345**	**0.72**	**0.07**	**89.86**	**0.005**	**0.70**	**0.15**	**89.0**	**0.24**	**0.62**

Figure 13.5.3 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 13.5.3 Summary of Total Gold Recovery



13.5.3 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.

13.5.4 Gravity Concentration

The testwork (SGS 17236-02) 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.

13.5.5 Leach Residue Diagnostic Testwork

The testwork (SGS Lakefield Report 17236-1 supplementary) 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.

13.5.6 Cyanide Leach Variability Testwork

Figure 13.5.4 shows the maximum, average and minimum values for the grade, %S and ppm Cu in all samples tested.

The average grade of the Koné South and North samples tested was 0.83 g/t Au compared to the Resource grade of 0.68 g/t. The average grade of the Gbongogo samples tested was higher at 1.77 g/t Au compared to the Resource grade of 1.46 g/t. The range of grade studies contained sufficient data to provide relationships between the feed grade and residue grade as shown above. The sulphur and copper in feed was found to be higher in the North HW fresh domain, however this domain only contains 0.2% of total process tonnage and 0.2% of the contained ounces. Some copper spikes were seen in north transition domain that results in higher cyanide consumption. The Gbongogo fresh domain contains similar elevated sulphur as North HW fresh, but differs in that there are virtually no copper sulphides present.

Figure 13.5.4 Grade, %S and ppm Cu in Feed Variability by Domain

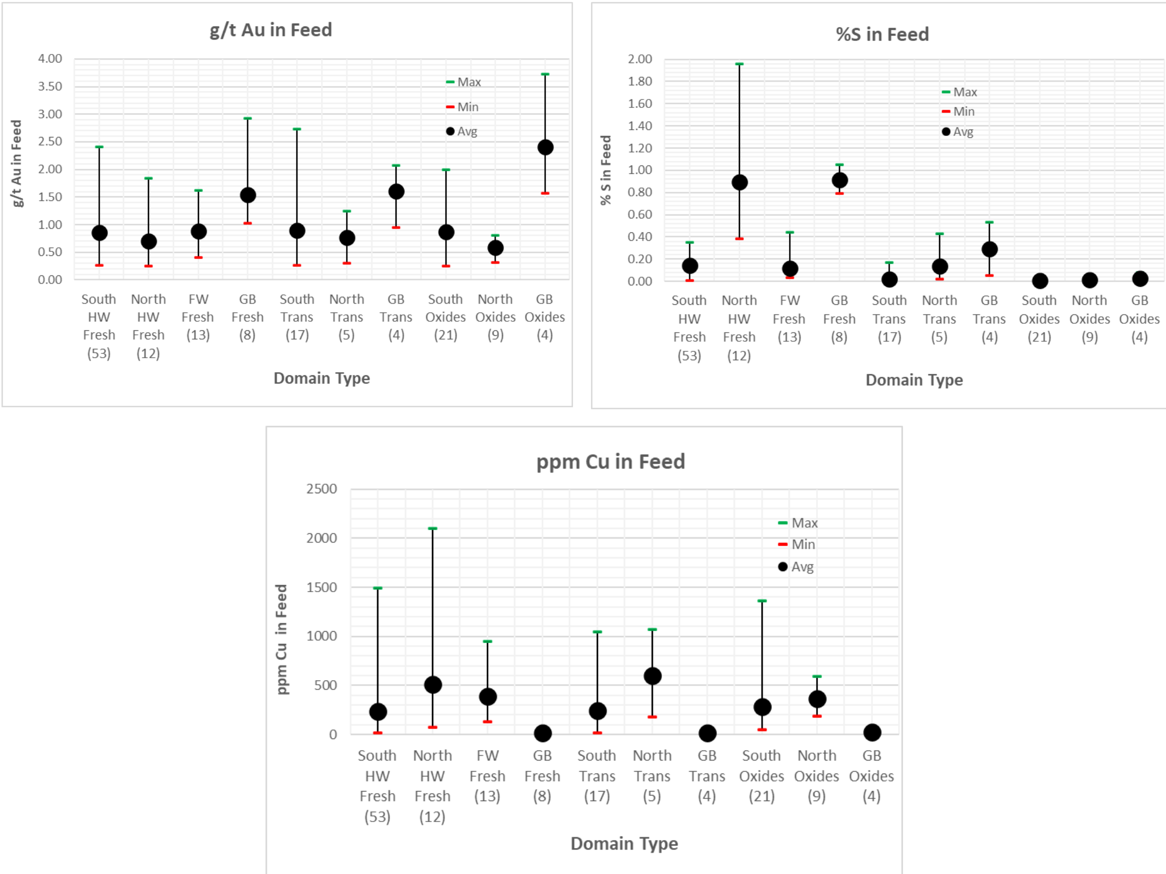


Figure 13.5.5 shows the variability in leach extraction efficiency for each ore domain. The low proportion North HW fresh ore domain was seen to vary the most, between 70.8% and 87.8%, averaging 79.2% and is due to the increased levels of sulphides and specifically copper sulphides. The Gbongogo fresh ore domain gave lower gold leach extractions despite the higher grade in the samples tested, this is considered to be due to the higher sulphur in this domain. The South HW and FW fresh, transition and oxide ore domains have higher gold leach extractions due to these ores containing less sulphides.

Figure 13.5.5 % Au Leach Extraction Variability by Domain

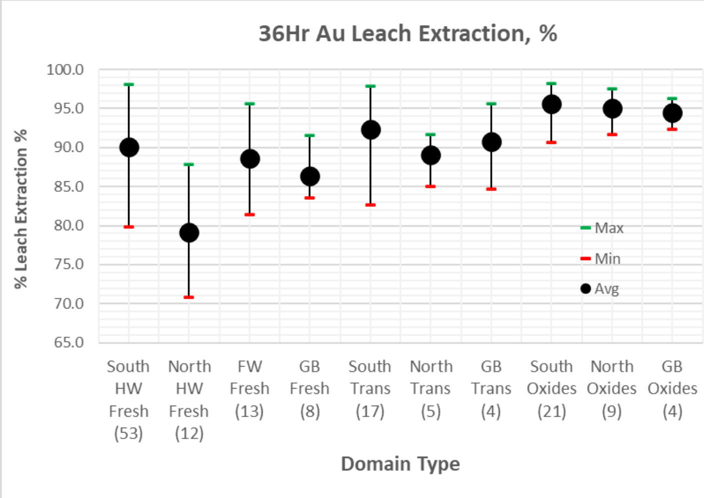


Figure 13.5.6 shows the variability observed with sodium cyanide consumption by ore domain. The variability in the sodium cyanide consumptions is highest in the transition ore domain due to localised higher copper bearing zones. Small localised higher copper zones in the small North HW fresh and trans ore domains, and higher sulphur in Gbongogo fresh ore domain also results in elevated cyanide consumptions.

Figure 13.5.6 kg/t NaCN Consumption by Domain

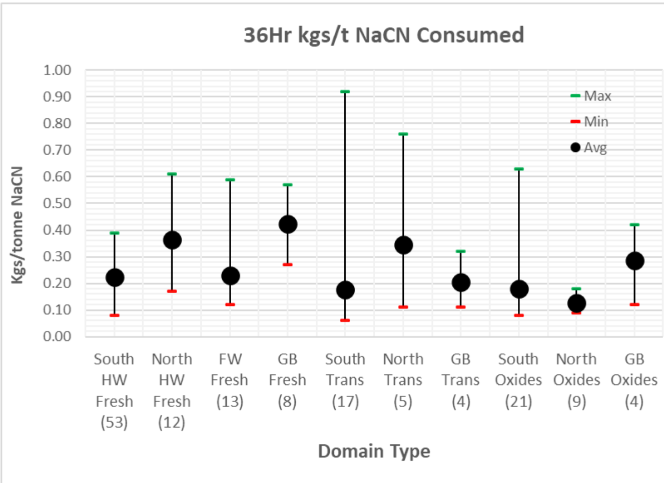
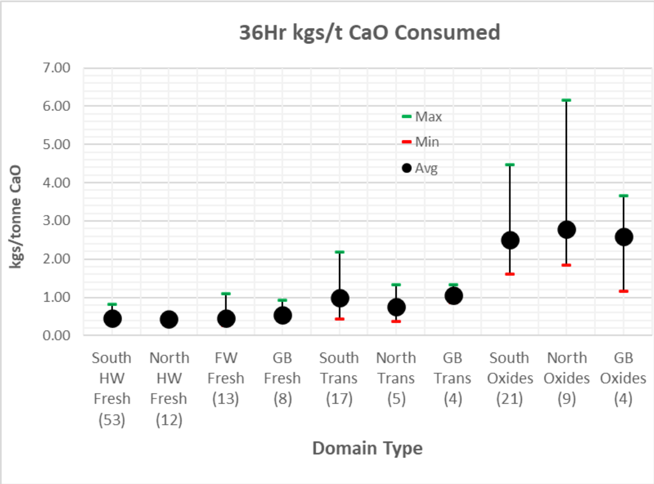


Figure 13.5.7 shows the variability in hydrated lime consumptions by domain. The consumption is consistently low for the HW fresh, FW fresh, Gbongogo fresh and transitional ore domains, but is much higher and varies more in the oxide ore domains.

Figure 13.5.7 kg/t CaO Consumption by Domain



13.5.7 Cyanide Leach Decay Testwork

Sodium cyanide decay testing has shown that typical anticipated free sodium cyanide levels in final leach tailings will be ~130 ppm for the predominant fresh ore type to ensure adequate free cyanide is present to leach the gold within 36 h. 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.

13.5.8 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 eight stages of Kemix adsorption.

13.5.9 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 13.5.3 shows the silver stage adsorption efficiencies obtained from these tests.

Table 13.5.3 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 36 h 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 doré.

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

- Years 1 to 9 inclusive treating high grade ore only.
- Years 10 to 15 inclusive treating low grade stockpiled ore.

The estimated doré fineness are shown in Table 13.5.4.

Table 13.5.4 Silver Stage Adsorption Efficiencies

Description	Distribution	g/t Au	Au Rec, %	%Au in Doré
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

13.5.10 Thickener Testing

The additional DFS thickener testing at the fixed flux rate of 1.09 tph/m^2 has been shown that small (5 g/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.

13.5.11 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.

14.0 MINERAL RESOURCE ESTIMATE

14.1 Introduction

Recoverable resources were estimated for the Koné and Gbongogo deposits by MIK with block support correction to reflect open pit mining selectivity, a method that has been demonstrated to provide reliable estimates of resources recoverable by open pit mining for a wide range of mineralisation styles.

The Koné and Gbongogo estimates are based on RC and diamond drilling data supplied by Montage in August 2021 and August 2023 respectively. Details of this sampling and assaying 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 MRE have been classified and reported in accordance with NI 43 101 and the classifications adopted by CIM Council in May 2014.

Estimates for the Koné deposit tested by drilling spaced at around 50 m x 50 m are classified as Indicated, with Inferred estimates based on generally 100 m spaced drilling. All Mineral Resources reported for Gbongogo, which are tested by generally 50 m spaced drilling are classified as Indicated.

The estimates are constrained within optimal pit shells generated at a gold price of US$1,800 /oz below topographic wire frames produced by Montage from DGPS surveys.

Resource modelling was undertaken in a local grids defined by Montage, which comprise a rotation of and plan view offset from WGS84 and an elevation increase of 1,000 m (Table 14.1.1). For both deposits, these transformations align the RC and diamond drilling traverses with local grid east-west section lines. All figures, coordinate and direction references in this chapter reflect the respective local grids.

Table 14.1.1 WGS84 to Local Grid Transformations

Description		WGS84	Local Grid
Koné	Easting	756,452.21 mE	5,000.00 mE
	Northing	964,427.14 mN	24,600.00 mN
	Rotation	-35°	
	Elevation change	+1,000 m	
Gbongogo	Easting	769,239.48 mE	5,000.00 mE
	Northing	993,671.10 mN	25,000.00 mN
	Rotation	-50°	
	Elevation change	+1,000 m	

14.2 Koné Resource Modelling

14.2.1 Mineralisation Interpretation and Domaining

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

The Koné resource modelling incorporates a mineralised envelope interpreted by the QP, 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 mineralised envelope, show generally low gold grades and rare, generally discontinuous zones of elevated gold grades. Domain boundaries were digitised on cross sections, snapped to drillhole traces where appropriate, then wire framed into a 3D solid.

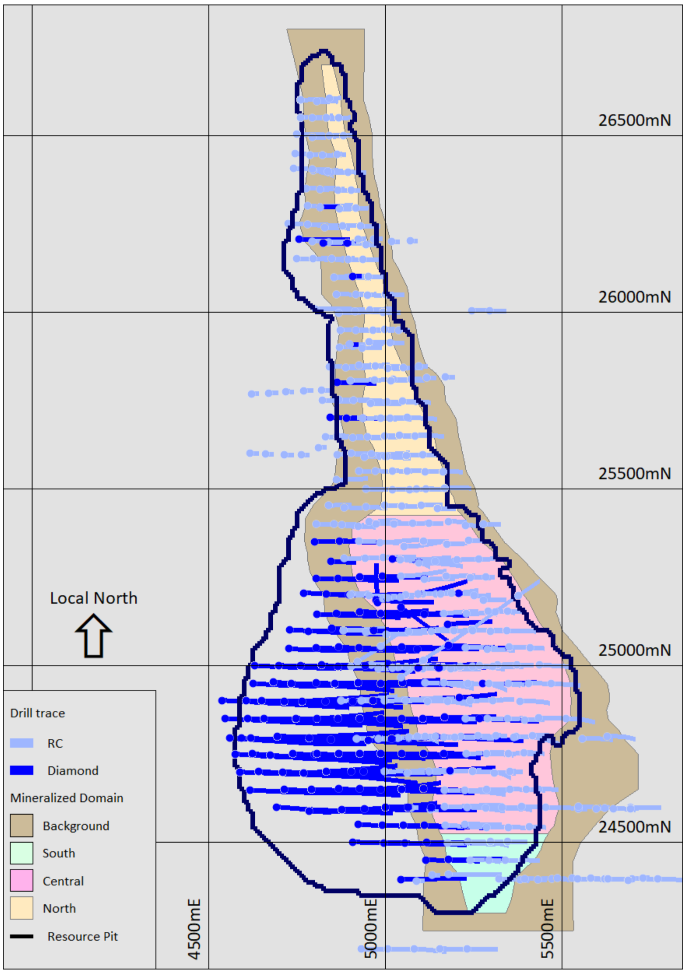
The mineralised envelope, which strikes north-north east (350°) and dips to the west at an average of around 50°, is interpreted over 2.4 km of strike with horizontal widths ranging from around 35 m to 450 m and averaging around 215 m. True widths are up to 350 m.

The mineralised envelope is subdivided into three mineralised domains comprising lower grade southern and northern domains, and a main higher grade central zone. For each mineralised domain, average drillhole 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 drillhole 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 mineralised envelope area, the depth to the base of complete oxidation ranges from around 8 m to 45 m and averages 24 m with fresh rock occurring at of around 15 m to 56 m, averaging 35 m.

Figure 14.2.1 shows the surface expression of the Koné mineralised domains relative to traces of the RC and diamond drilling utilised for Resource estimation. Figure 14.4.1 shows an example cross sections of the estimation domains relative to drillhole traces coloured by composited gold grades and block model estimates. These plots demonstrate that for some cross sections, the resource drillholes do not all penetrate the full width of the mineralised envelope. For these sections, drilling preferentially tests the western generally higher average gold grade portions of the mineralised envelope.

Figure 14.2.1 Koné Mineralised Domains and Drill Traces



Source: Matrix, December 2023. Koné Local Grid.

14.2.2 Bulk Density Measurements

Table 14.2.1 summarises the primary immersion density measurements available for Koné coded by combined mineralised domain, oxidation zone, and rock type wireframes interpreted by Montage from drillhole logs. This table, which excludes three anomalously low values, shows that for each oxidation zone, average density measurements show little variability with rock type and mineralised domain.

Table 14.2.1 Koné Bulk Density Measurements

Oxidation Zone	Mineralisation Zone	Rock Code	Density (t/m3)			
			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
	Mineralised 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
	Mineralised 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
	Mineralised 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
	Combined		3,431	1.73	2.81	3.64

14.2.3 Estimation Dataset

The Koné estimates are based on 2 m downhole 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 mineralised domain composites.

Table 14.2.2 presents univariate statistics of composite gold grades for the estimation dataset by mineralised 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 mineralised domains demonstrating that the domaining has been effective in assigning most mineralised composites into the mineralised domains.

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

- With the exception of the South mineralised domain, which represents around 4% of the combined mineralised domain dataset and contains comparatively few completely oxidised and transition zone composites, average composite grades for each mineralised 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 14.2.2 Koné Estimation Dataset Statistics

Au g/t	Background Domain				South Mineralised 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 Mineralised Domain**				**North Mineralised 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

14.2.4 Estimation Parameters

The block model frame work used for the Koné MIK modelling covers the full extents of the informing composites and mineralised 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 mineralised 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 QPs experience, the approach adopted for determination of upper bin grades is appropriate for MIK modelling of highly variable mineralisation such as Koné.

Table 14.2.3 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 mineralised 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 14.2.2 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 14.2.3) used for MIK estimation were aligned with dominant domain mineralisation orientation and inclined towards the west at 50°. Search pass four informs a small number of panels in broadly sampled areas. Panels informed by this search pass represent only small proportions of Indicated and Inferred 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 5 m x 10 m x 5 m (east, north, vertical). The variance adjustments were applied using the direct lognormal method and the adjustment factors listed in Table 14.2.3.

Bulk densities of 1.65, 2.55 and 2.80 t/bcm assigned to completely oxidised, transitional and fresh material respectively. These values reflect the average of the available measurements for the deposit.

Figure 14.2.2 Koné 3D Variogram Plot

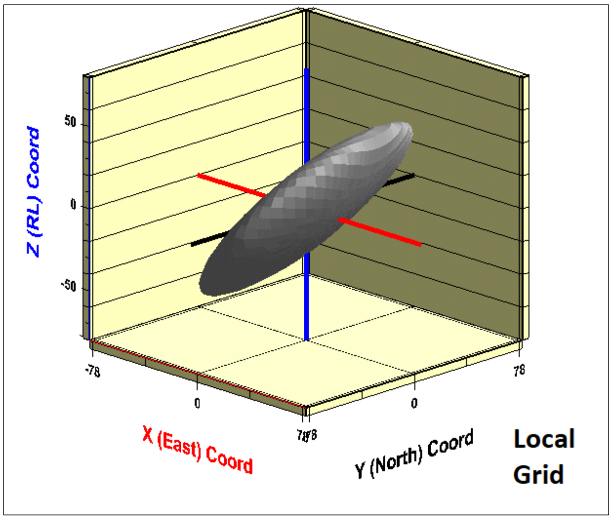


Table 14.2.3 Koné Key Estimation Parameters

Indicator Thresholds and Bin Mean Grades								
Percentile	Background Domain		South Mineralised Domain		Central Mineralised Domain		North Mineralised 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	

Search Criteria				
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
Ellipsoid Rotation: Z+10,Y-50 (Local Grid)				

Variance Adjustment Factors		
Block/ Panel	Information Effect	Total Adjustment
0.165	0.842	0.139

14.2.5 Classification of the Estimates

In the QPs opinion, the available sampling does not define the Koné mineralisation with sufficient confidence for estimation of the Measured resources. Estimates tested by drilling spaced at around 50 m x 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 mineralised envelope tested by drilling generally spaced at closer than 100 m x 100 m. More broadly sampled and peripheral mineralisation 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 m x 50 m spaced drilling are classified as Indicated. Remaining panels within polygons defining the limits of generally 100 m x 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 m x 50 m drilling were reclassified 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 upper plot in Figure 14.4.1 shows the polygons used for assignment of confidence categories in red and green respectively.

14.3 Gbongogo Resource Modelling

14.3.1 Mineralisation Interpretation and Domaining

Gbongogo deposit drilling has delineated a northerly trending mineralised zone interpreted to dip to the west at around 55°. The transition from gold mineralisation to barren host rock is generally characterised by diffuse grade boundaries.

Mineralised domain interpretation included reference to a set of 3D wireframes representing key rock units, comprising the hanging wall volcaniclastic sediments footwall sediments and quartz diorite intrusive interpreted by Montage from drillhole logging.

The resource modelling incorporates a mineralised envelope interpreted by the QP on the basis of composited gold grades capturing continuous intervals of greater than 0.1 g/t, and a background domain. Mineralised envelope boundaries were digitised on cross sections, snapped to drillhole traces where appropriate, then wireframed into a 3D solid. Background domain composites were selected from a plan view polygon.

Drill samples within the background domain, which are within around 80 m of the mineralised envelope show generally low gold grades, and rare, generally discontinuous zones of elevated gold grades. The background domain does not contribute to MRE.

The mineralised envelope, strikes north-south over around 700 m of strike and dips to the west at around 55°, with horizontal widths ranging from around 35 m to 450 m and averaging around 215 m. True widths are up to around 270 m.

The mineralised envelope was subdivided into a hanging wall domain of generally low gold grades, and a higher grade footwall domain. The footwall domain was further subdivided into northern and southern zones, which are comparatively higher and lower grade respectively, giving three mineralised domains designated as the hanging wall, low grade footwall, and high grade footwall domains respectively.

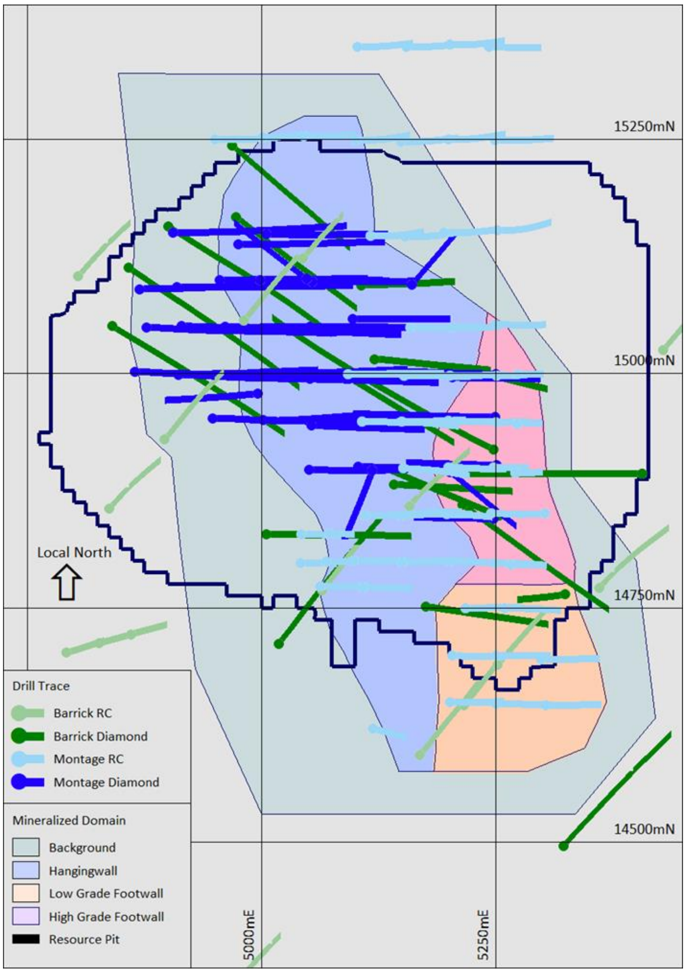
The hanging wall domain, which contains only comparatively rare drillhole composites of elevated ore grades and represents only a small proportion of Mineral Resources, averages around 100 m thick.

The contact between the low grade and high grade footwall domains plunges towards the north at around 55˚. The low grade footwall domain which hosts a small proportion of estimated Mineral Resources averages around 45 m thick. The high grade footwall domain, which encompasses and is generally slightly more extensive than the interpreted quartz dolerite hosts the majority of higher gold grade composites. It represents the majority of Mineral Resources averages around 80 m thick.

Montage supplied surfaces representing the base of surficial clays, base of oxidation and the top of fresh rock interpreted from drillhole geological logging. These surfaces were used for flagging of estimation dataset composites by oxidation zones, density assignment and partitioning resources by oxidation type. Within the mineralised envelope area, the depth to the base of complete oxidation, inclusive of surficial clays ranges from around 9 m to 45 m and averages 24 m. The transition zone ranges in thickness from around 1 m to 18 m, averaging around 8 m thick with fresh rock occurring at depths ranging from around 12 m to 50 m and averaging 50 m.

Figure 14.3.1 shows the surface expression of the mineralised domains relative to of RC and diamond drillhole traces. Figure 14.4.1 shows an example cross section of the estimation domains relative to drillhole traces coloured by composited gold grades.

Figure 14.3.1 Gbongogo Mineralised Domains and Drill Traces



Produced by Matrix in December 2023. Gbongogo Local Grid.

14.3.2 Bulk Density Measurements

Bulk density measurements available for Gbongogo include 1,717 measurements of generally 10 cm core intervals from Barrick diamond holes for which core was not sealed prior to measurement and 488 measurements of oven dried, wax coated core from Montage drilling, with specified lengths ranging averaging 11 cm. Uncoated measurements are also available for 407 of the Montage core intervals.

Subdivided by oxidation zone, the paired uncoated and coated density measurements of Montage diamond core show very similar averages, indicating that Gbongogo mineralisation is not significantly porous, and suggests that the lack of wax coating for the compiled Barrick measurements may not have introduced significant biases to those data.

Table 14.3.1 summarises the compiled density measurements coded by oxidation and rock type wireframes supplied by Montage, excluding data from reconnaissance drilling outside the resource area. Montage coated measurements were selected in preference to the uncoated measurements with the exception of three intervals with anomalous coated results, for which the uncoated measurement was prioritised. Three anomalously high or low results were excluded.

Table 14.3.1 shows that there is little variability in average density measurements by rock type within each oxidation zone.

Table 14.3.1 Gbongogo Density Measurements

Oxidation Zone	Rock Type	Number of Measurements	Density (t/m^3)		
			Minimum	Average	Maximum
Clay	HW sediment	75	1.03	1.63	2.02
	FW sediment	-	-	-	-
	Quartz diorite	3	1.25	1.47	1.66
	Combined	78	1.03	1.63	2.02
Completely Oxidised	HW sediment	47	1.11	1.65	2.68
	FW sediment	3	1.29	1.74	2.41
	Quartz diorite	6	1.41	1.80	2.44
	Combined	56	1.11	1.67	2.68
Transition	HW sediment	54	1.73	2.59	2.84
	FW sediment	8	2.06	2.44	2.74
	Quartz diorite	6	2.47	2.61	2.78
	Combined	68	1.73	2.58	2.84
Fresh	HW sediment	340	2.18	2.74	3.05
	FW sediment	307	2.26	2.77	2.96
	Quartz diorite	240	2.56	2.73	2.90
	Combined	887	2.18	2.75	3.05

14.3.3 Estimation Dataset

The Gbongogo estimates are based on 2 m downhole composited gold grades from RC and diamond drilling with the estimation dataset comprising 8,789 composites with gold grades ranging from 0.000 to 50.7 g/t and averaging 0.41 g/t. Samples from RC and diamond drilling provide around one third and two thirds of the estimation dataset respectively.

Around 18% composites from the high grade footwall domain are from holes inclined towards local grid west, sub-parallel to the mineralised domain trends. These 'down dip' holes, which rarely intersect the other mineralised domains, include Barrick drilling and Montage holes primarily drilled to provide metallurgical test samples.

Table 14.3.2 presents univariate statistics of the Gbongogo estimation dataset gold grades by modelling domain subdivided by general drilling orientation. Figure 14.3.2 shows cumulative histograms of high grade footwall domain composite gold grades by drilling orientation. Notable features shown by this table and figure include the following:

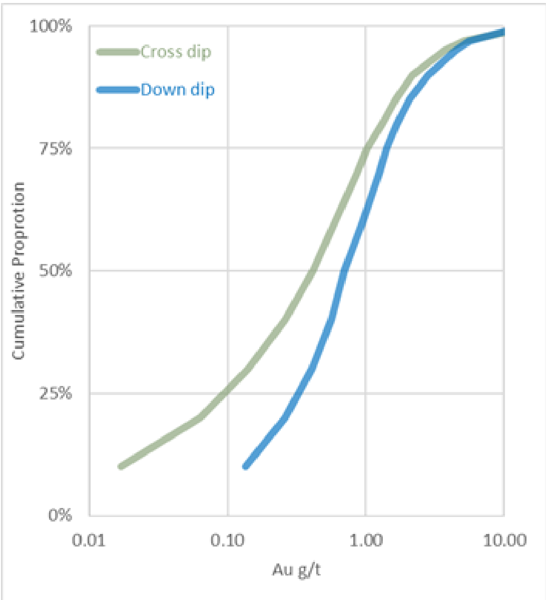
- At 0.034 g/t, the mean gold grade for the background domain composites is notably lower than for the mineralised domain, demonstrating that the domaining has effectively assigned most mineralised composites into the mineralised domains.

- For the high grade footwall domain composites from down dip holes show proportionally more high grades and a higher average grades than cross dip holes.
- Gold grades show strong positive skewness with a coefficient of variation of around two indicating that MIK is an appropriate estimation technique.

Table 14.3.2 Gbongogo Estimation Dataset Statistics

Au g/t	Background Domain Cross Dip	Hanging Wall Domain Cross Dip	Low Grade Footwall Domain		High Grade Footwall Domain	
			Cross Dip	Down Dip	Cross Dip	Down Dip
Number	1,881	3,034	740	7	2,559	568
Mean	0.034	0.098	0.22	0.008	1.13	1.40
Variance	0.021	0.13	0.12	0.000	10.5	8.01
Coef. Var.	4.36	3.68	1.61	0.81	2.88	2.03
Minimum	0.000	0.000	0.000	0.004	0.000	0.000
1st Quartile	0.005	0.005	0.035	0.005	0.095	0.35
Median	0.010	0.013	0.093	0.005	0.42	0.70
3rd Quartile	0.030	0.055	0.28	0.009	1.03	1.43
Maximum	5.31	8.40	3.29	0.023	54.6	42.0

Figure 14.3.2 Gbongogo High Grade Footwall Cumulative Histograms



14.3.4 Estimation Parameters

The block model framework used for Gbongogo modelling covers the full extents of the informing composites and mineralised domains. It comprises panels with dimensions of 25 m east-west by 25 m north-south and 10 m vertical defined in local grid coordinates. These dimensions were selected on the basis of sample spacing in central portions of the deposit.

Indicator grade thresholds were defined using a consistent set of percentiles for data in each domain. For the high grade footwall domain, composites from down-dip drillholes, which appear less representative of the general composite grade distribution than composites from cross dip holes were excluded from the dataset used to determine indicator thresholds and class grades. No composites were excluded from the dataset used for the MIK modelling.

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. In the QPs experience, the approach adopted for determination of upper bin grades is appropriate for MIK modelling of highly variable mineralisation such as Gbongogo. Table 14.3.3 presents the indicator thresholds and bin grades used for modelling with the 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 footwall 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 55˚. As an example of the variogram models, Figure 14.3.3 presents a 3D variogram surface map of the median indicator variogram model at variogram value of 0.50.

The four progressively more relaxed search criteria (Table 14.3.3) were aligned with dominant domain mineralisation orientation and inclined towards the west at 55˚. MRE are primarily informed by search passes 1 and 2 with search pass 3 panels contributing only a small proportion. No panels informed by search pass 4 lie within the pit shell constraining Mineral Resources and estimates from this search pass do not inform Mineral 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 5 m x 10 m x 5 m (east, north, vertical). The variance adjustments were applied using the direct lognormal method and the adjustment factors listed in Table 14.3.3.

Bulk densities were assigned to the Gbongogo block model by oxidation zone with densities of 1.65, 2.55 and 2.70 t/bcm assigned to the combined clay and completely oxidised, transitional and fresh material respectively. These values reflect the average of the compiled measurements for Montage and Barrick diamond core. The QP considers that these density measurements provide an adequate basis for the Gbongogo and MRE.

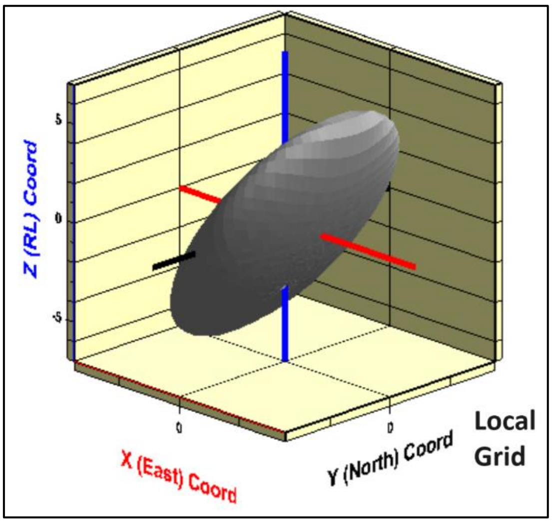
Table 14.3.3 Gbongogo Key Estimation Parameters

Indicator Thresholds and Bin Mean Grades								
Percentile	Background Au g/t		Hanging Wall Au g/t		Low Grade Footwall Au g/t		High Grade Footwall Au g/t	
	T'hold	Mean	T'hold	Mean	T'hold	Mean	T'hold	Mean
10%	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.01
20%	0.01	0.01	0.01	0.00	0.02	0.02	0.06	0.04
30%	0.01	0.01	0.01	0.01	0.04	0.03	0.14	0.10
40%	0.01	0.01	0.01	0.01	0.06	0.05	0.26	0.20
50%	0.01	0.01	0.01	0.01	0.09	0.07	0.42	0.34
60%	0.02	0.01	0.02	0.02	0.13	0.11	0.60	0.50
70%	0.02	0.02	0.04	0.03	0.20	0.16	0.86	0.71
75%	0.03	0.03	0.06	0.05	0.28	0.23	1.03	0.95
80%	0.04	0.03	0.08	0.06	0.33	0.30	1.30	1.16
85%	0.04	0.04	0.12	0.10	0.42	0.37	1.66	1.47
90%	0.06	0.05	0.20	0.15	0.56	0.48	2.19	1.90
95%	0.11	0.08	0.41	0.29	0.80	0.66	3.73	2.84
97%	0.17	0.13	0.67	0.54	1.06	0.92	5.17	4.45
99%	0.42	0.25	1.44	0.97	1.66	1.39	11.64	8.03
100%	5.31	0.17	8.40	2.19	3.29	2.20	54.56	21.41
	97^{th} %ile		Median		Median		Mean < 40 g/t	

Search Criteria				
Search	Radii (m)	Minimum Data	Minimum Octants	Maximum Data
1	30,30,12	16	4	48
2	60,60,24	16	4	48
3	60,60,24	8	2	48
4	120,120,30	8	2	48
Ellipsoid Rotation: Z+0,Y-55 (Local Grid)				

Variance adjustment factors		
Block/ Panel	Information Effect	Total Adjustment
0.18	0.55	0.10

Figure 14.3.3 Gbongogo 3D Variogram Plot



14.3.5 Classification of the Estimates

In the QPs opinion, the available sampling information does not define Gbongogo mineralisation with sufficient confidence for estimation of Measured resources.

Estimates tested by drilling spaced at around 50 m x 50 m are classified as Indicated, with Inferred estimates generally based on spaced drilling spaced at around 100 m x 100 m.

Model estimates were classified as Indicated and Inferred by estimation search pass and a set of cross sectional polygons defining areas of consistently spaced drilling for each model row. Panels within the mineralised domains informed by search pass 1 and 2 within polygons defining the outer limits of consistently 50 m x 50 m spaced drilling were classified as Indicated. Remaining panels, including all search pass 3 and 4 panels, were initially classified as Inferred. Comparatively rare, isolated Search Pass 3 panels within areas of generally 50 m x 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.

The lower plot in Figure 14.4.1 shows the polygons used for assignment of confidence categories within the mineralised envelope as thick red lines.

The pit shell constraining Mineral Resources captures very few panels classified as Inferred, and estimates from these panels were excluded from Mineral Resources. All Gbongogo MRE are classified as Indicated.

14.4 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.

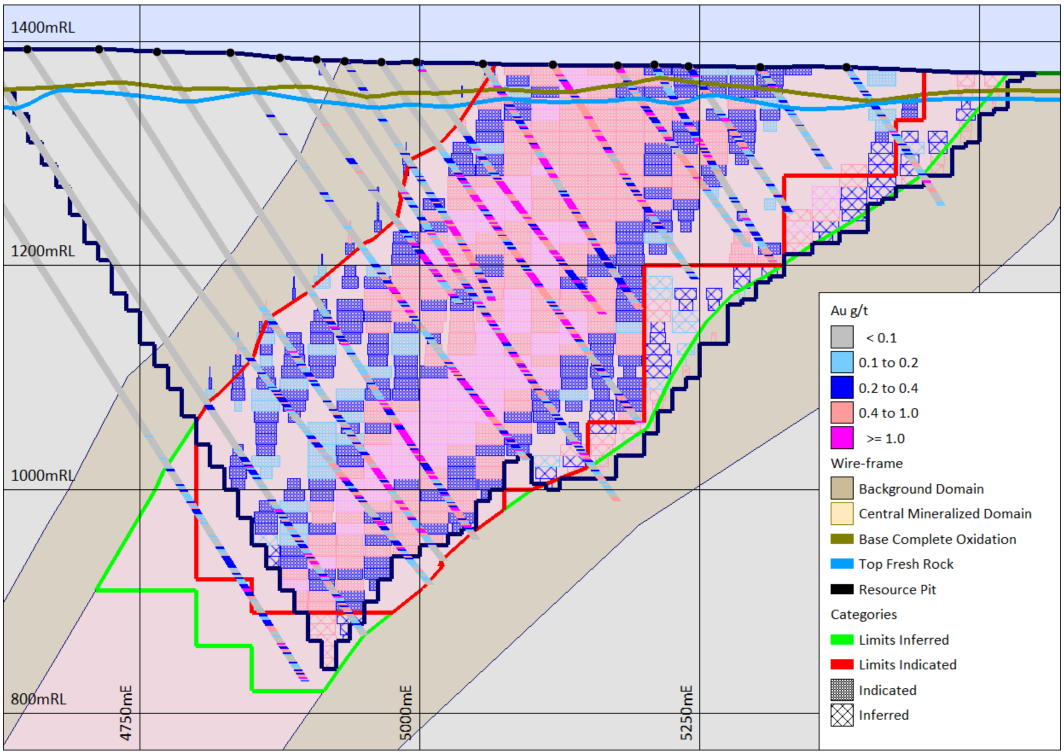
Figure 14.4.1 shows representative cross sections of the Koné and Gbongogo block models at 0.20 and 0.50 g/t cut-off respectively. This figure shows model panels scaled by the estimated proportion above the relevant cut-off grade relative to the estimation domains and drillhole traces coloured by 2 m composited gold grades. Model panels shown in this figure are restricted to those within the optimal pits used for constraining MRE.

The upper swath plot in Figure 14.4.2 compares average mineralised domain composite grades and average MIK panel grades within the volume of Koné model blocks classified as Indicated. For preparation of this plot, average composite gold grades include uppercuts representing the 99.75th percentile of each domain reducing the impact of a small number of outlier composite grades.

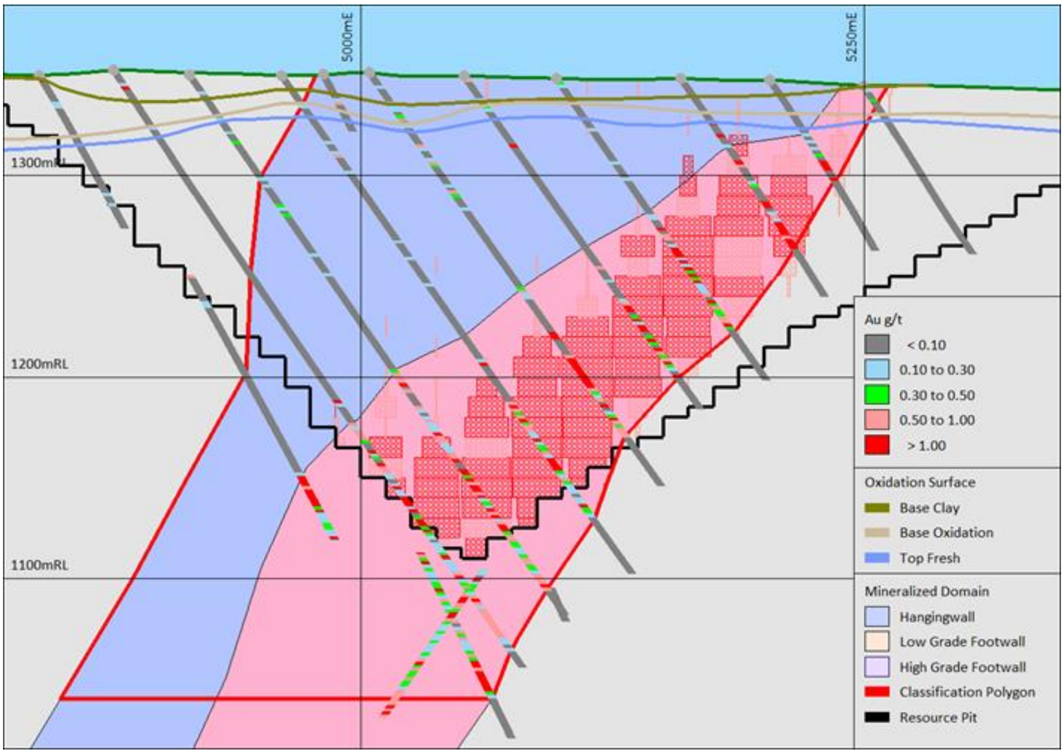
The lower plot in Figure 14.4.2 compares average composite grades and average MIK panel grades for the combined Gbongogo footwall mineralised domains for the volume of Indicated model panels within the pit shell constraining Mineral Resources. For preparation of this plot, average composite gold grades include uppercuts representing the 99th percentile of the combined dataset reducing the impact of a small number of outlier composite grades.

Figure 14.4.2 shows that although, as expected average MIK panel grades are smoothed relative to average composite grades they generally closely follow the trends shown by the composite mean grades. The figure shows local apparent deviations between model and composite trends which are influenced by the variability in drillhole spacing, including clustering of drilling in areas of higher average grade mineralisation. These features reflect the distribution of drilling and do not represent biases in the model estimates.

Figure 14.4.1 Cross Section Plots of Models and Informing Data

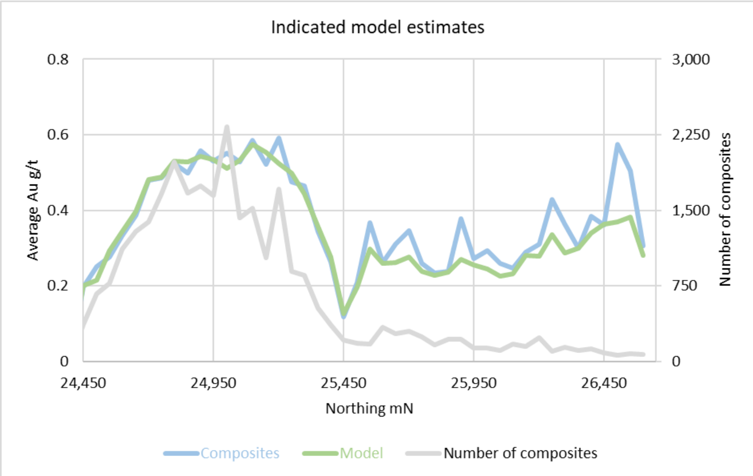


Koné 24,700 mN 0.20 g/t cut-off, Koné Local Grid. (Matrix December 2023)

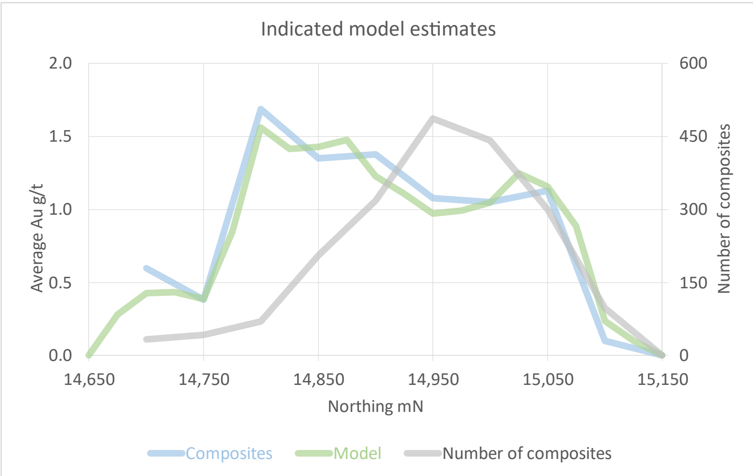


Gbongogo 24,900 mN 0.50 g/t cut-off, Gbongogo Local Grid. (Matrix December 2023)

Figure 14.4.2 Panel Grades vs. Composite Grades



Koné (Local Grid)



Gbongogo (Local Grid)

14.5 Criteria for Economic Extraction

To provide estimates with reasonable prospects for economic extraction, Koné and Gbongogo Mineral Resources are reported within optimised pit shells generated from the models using parameters supplied by Montage. The optimisation parameters reflect a large scale conventional open pit operation with the cost and revenue parameters detailed in Table 14.5.1. Wall angles assigned for Gbongogo fresh material vary with wall azimuth and elevation, ranging from 35 to 59° and averaging around 48°.

The selected gold price of US$1,800 /oz reflects the three year trailing average of monthly average gold prices reported by the World Gold Council to the end of November 2023 (World Gold Council, 2023) of US$1,843 /oz with rounding (Figure 14.5.1).

Koné and Gbongogo Mineral Resources are reported at cut-off grades of 0.5 g/t and 0.20 g/t respectively. These cut-offs reflect the cost and revenue parameters use for pit optimisation.

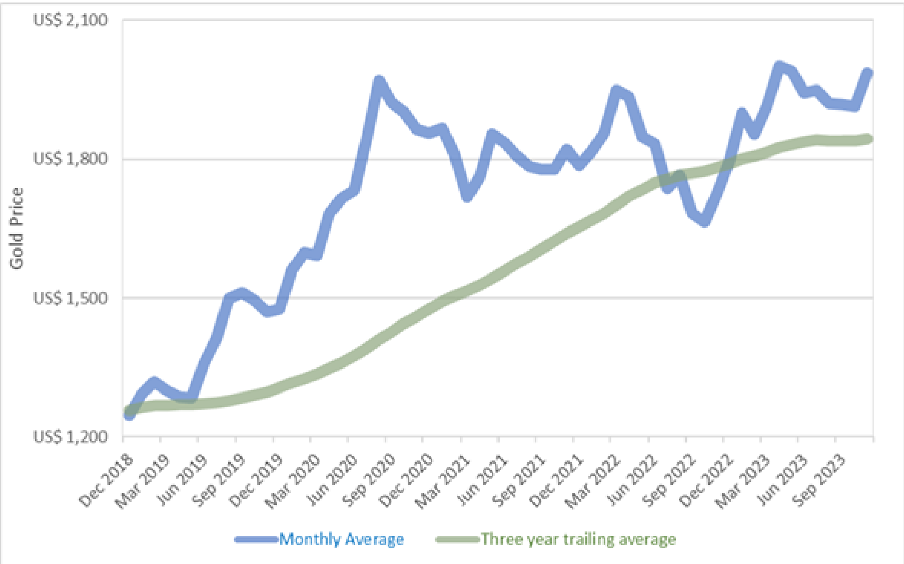
In the opinion of the QP, the parameters used to produce the pit shells constraining Mineral Resources and define cut-off grades are appropriate for providing estimates with reasonable prospects of eventual economic extraction consistent with CIM guidelines (CIM, 2019; CIM 2020).

The pit shell constraining the Koné estimates extends over 2.4 km of strike to a maximum depth of around 570 m (Figure 14.2.1 and Figure 14.4.1). The pit shell constraining Gbongogo estimates extends over approximately 620 m of strike to a maximum depth of around 250 m (Figure 14.3.1 and Figure 14.4.1).

Table 14.5.1 Resource Pit Shell Parameters

General	Gold price (US$)	$1,800 /oz			
	Government royalty	5.00%			
	Community development	0.50%			
	Selling costs (US$)	$4.71 /oz			
Description		**Oxide**	**Transition**	**Fresh**	**Total**
Koné	Wall angle	39º	58º	60º	
	Average mining cost (US)	$3.02 /t	$3.02 /t	$3.27 /t	$3.23 /t
	Mill processing cost (US)	$7.96 /t	$8.20 /t	$9.41 /t	$9.27 /t
	Mill recovery	93.0%	91.0%	89.0%	89.3%
	Triple Flag royalty	2.00%	2.00%	2.00%	2.00%
Gbongogo	Wall angle	30º	30º	35-59º	
	Average mining cost (US)	$2.13 /t	$2.89 /t	$3.13 /t	$2.95 /t
	Mill processing cost (US)	$10.11 /t	$8.19 /t	$9.80 /t	$9.76 /t
	Haulage cost (US)	$7.53 /t	$7.53 /t	$7.53 /t	$7.53 /t
	Mill recovery	93.0%	91.0%	86.0%	86.7%
	Barrick / Endeavour royalty	2.00%	2.00%	2.00%	2.00%

Figure 14.5.1 Monthly Average Gold Price December 2018 to November 2023



14.6 Mineral Resource Estimates

Table 14.6.1 shows the Koné Indicated and Inferred Mineral Resources and Gbongogo Indicated Mineral Resources. Table 14.6.2 shows the MRE for each deposit subdivided by oxidation type.

Table 14.6.3 shows estimates from the Koné and Gbongogo resource models within the pit shells constraining Mineral Resources for ranges of cut-off grades. The QP considers the estimates at 0.20 g/t and 0.50 g/t cut-off represent the base case or preferred scenario for the Koné and Gbongogo, respectively. The other estimates are included only to demonstrate the sensitivity of the MRE to changes in cut-off grade and are not the QPs estimate of the Mineral Resources for Koné and Gbongogo. All estimates resulting from each of the cut-off grade scenarios meet the test of reasonable prospect of economic extraction.

The figures in Table 14.6.1, Table 14.6.2 and Table 14.6.3 are rounded to reflect the precision of the estimates and include rounding errors.

The MREs have an effective date of 19 December 2023.

The MREs have been classified and reported in accordance with the classifications adopted by CIM Council in May 2014 (CIM, 2014).

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

The Indicated Mineral Resources are inclusive of Mineral Reserves. The Inferred Mineral Resources are additional to Mineral Reserves.

Table 14.6.1 Indicated and Inferred Mineral Resource Estimates

Deposit and Cut-Off Au	Indicated			Inferred		
	Tonnes Million	Au g/t	Au Moz	Tonnes Million	Au g/t	Au Moz
Koné 0.20 g/t	229	0.59	4.34	25	0.5	0.40
Gbongogo 0.50 g/t	11	1.48	0.52	-	-	-
Total	**240**	**0.63**	**4.87**	**25**	**0.5**	**0.40**

Table 14.6.2 Indicated and Inferred Mineral Resource Estimates by Oxidation Type

Koné 0.20 g/t cut-off						
Oxidation	**Indicated**			**Inferred**		
	Tonnes Million	**Au g/t**	**Au Moz**	**Tonnes Million**	**Au g/t**	**Au Moz**
Completely oxidised	14	0.54	0.24	0.3	0.5	0.005
Transition	9	0.55	0.16	0.1	0.6	0.002
Fresh	207	0.59	3.93	24	0.5	0.39
Total	229	0.59	4.34	25	0.5	0.40

Gbongogo at 0.50 g/t cut-off			
Oxidation	**Indicated**		
	Tonnes Million	**Au g/t**	**Au Moz**
Clay	0.2	1.64	0.01
Completely oxidised	0.5	1.47	0.02
Transition	0.4	1.34	0.02
Fresh	10	1.48	0.47
Total	11	1.48	0.52

Table 14.6.3 Estimates by Cut-Off Grade

Koné						
Cut-Off Au g/t	**Indicated**			**Inferred**		
	Mt	**Au g/t**	**Au Moz**	**Mt**	**Au g/t**	**Au Moz**
0.1	286	0.50	4.60	37	0.3	0.36
0.2	**229**	**0.59**	**4.34**	**25**	**0.5**	**0.40**
0.3	170	0.70	3.83	16	0.6	0.31
0.4	130	0.81	3.39	10	0.7	0.23
0.5	100	0.92	2.96	7.0	0.8	0.18
0.6	78	1.03	2.58	5.0	0.9	0.14
0.7	61	1.14	2.24	3.0	1.1	0.11
0.8	47	1.25	1.89	2.0	1.2	0.08

Gbongogo			
Cut off	**Indicated**		
Au g/t	**Mt**	**Au g/t**	**Au Moz**
0.2	15	1.16	0.56
0.3	14	1.26	0.57
0.4	12	1.37	0.53
0.5	**11**	**1.48**	**0.52**
0.6	9.9	1.59	0.51
0.7	8.8	1.71	0.48
0.8	7.8	1.83	0.46
0.9	6.9	1.96	0.43
1.0	6.1	2.09	0.41

15.0 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.

The estimated Mineral Reserves are shown in Table 15.1.1. The deposit is divided into the South, North and Gbongogo Pits. There are no existing stockpiles on the property.

[1]https://mrmr.cim.org/media/1128/cim-definition-standards_2014.pdf

Table 15.1.1 Summary of Mineral Reserves for the Koné and Gbongogo Deposits

	Classification	Oxide			Transitional			Fresh			Total		
		Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz	Mt	Au g/t	Au Moz
South Pit	Probable	9.6	0.59	0.2	7.0	0.60	0.1	145.3	0.68	3.2	161.9	0.67	3.5
North Pit	Probable	0.9	0.47	0.0	0.4	0.44	0.0	0.4	0.47	0.0	1.8	0.46	0.0
Gbongogo	Probable	0.7	1.36	0.0	0.5	1.09	0.0	9.4	1.91	0.6	10.7	1.43	0.5
Total	**Probable**	**11.3**	**0.54**	**0.2**	**7.9**	**0.59**	**0.2**	**155.1**	**0.68**	**3.8**	**174.3**	**0.72**	**4.0**

In the opinion of the QP, the parameters and modifying factors used to produce the pit shells constraining Mineral Reserves and define cut-off grades are appropriate for providing estimates of the economically mineable Probable Reserve consistent with CIM guidelines (CIM 2014).

15.2 Basis of Estimate

The Mineral Reserve estimate was undertaken by Carci Mining Consultants Ltd ('Carci') using Deswik mine planning software (Version 2023.1) 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 Koné site in November 2021. 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.
- Joeline McGrath visited the Koné and Gbongogo sites in July 2023 which included the newly acquired Gbongogo deposit and assessing the routes between the two deposits.
- The geological model was provided by Montage as an MIK model in Datamine format.
- The geotechnical review of the Koné and Gbongogo Pit designs was undertaken by SRK Consulting (UK) Ltd and the reports were provided to Carci by Montage. Slope parameters for the optimisation and design were obtained from these reports.
- Hydrogeological investigation results for the Koné and Gbongogo Pits included groundwater 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 generated by Lycopodium and provided to Carci by Montage.
- The mining productivity and cost estimates are based on information received from major equipment suppliers and a mining contract tender budget exercise conducted in 2023 involving six 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 (UK23-0010)' 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 December 2023 resource model, as discussed in Section 14.0. The resource model was provided by Montage and was classified by Indicated and Inferred Resources. Only the Indicated Resources were used in the mining study and resultant reserve estimates.

The initial December 2023 geological block models were provided as a MIK 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.

Both the Gbongogo and Koné models were re-supplied to Carci in UTM coordinates and re-estimated with a block size of 12.5 m x 12.5 m x 5 m to allow better accuracy within the upper oxidation benches for the saprolite and saprock.

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

15.3.2 Dilution and Ore Recovery

The MIK resource models 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 assessed the geotechnical conditions of the Koné and Gbongogo Pits. The following information in this section has been extracted from the SRK report entitled 'Geotechnical Study on the Koné Project, Côte d'Ivoire (UK31101)', December 2021, and the 'Geotechnical Study for the Gbongogo Open Pit, Côte d'Ivoire (UK32030)', October 2023.

15.3.4 North and South Koné Pits

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 15.3.1. 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.

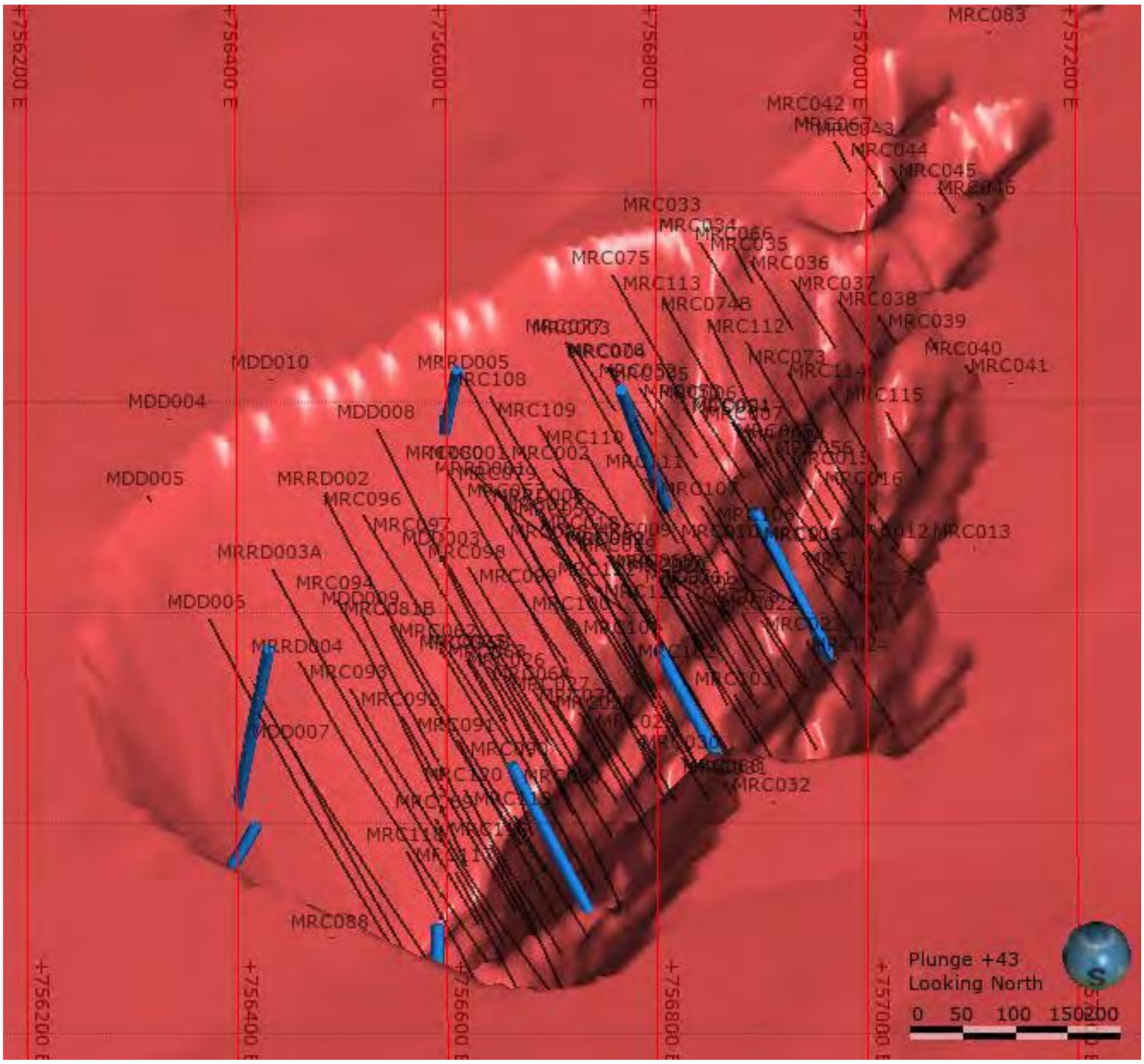
Logging analysis yielded three main geotechnical units: surficial soil and highly weathered rock (Soil), slightly to moderately weathered rock (MW/SW) and unweathered rock (UW), with thicknesses of 15 m to 25 m, 14 m to 25 m, and >350 m respectively. By far, UW is the predominant unit, making up 84% 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 Volcaniclastic (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 2 m 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° to 70° and the second is sub-horizontal.

Figure 15.3.1 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 to 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. Early green dykes (EGD), while being deformed, chlorite-altered and strongly foliated, have a high RMR'89 (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 30 m high with an 80° bench face angle and a berm of 8 m, yielding a 66° inter ramp angle (IRA). With this configuration and using pre-split blasting, stacks of up to seven benches can be achieved for a maximum UW slope height of 210 m. Geotechnical berms have been removed, although ramps are in place in the final pit design such that there is a maximum of seven benches per stack.

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

Table 15.3.1 Koné Bench Design Summary

Unit	Wall	Bench Height m	Batter Angle °	Berm Width m	IRA °
Soil	All	6	60	4	39
MW/SW	All	20	80	9	58
UW	All	30	80	8	66

Table 15.3.2 Koné IR and OS Design Summary

Unit	Wall	Benches per Stack	Inter Ramp 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	

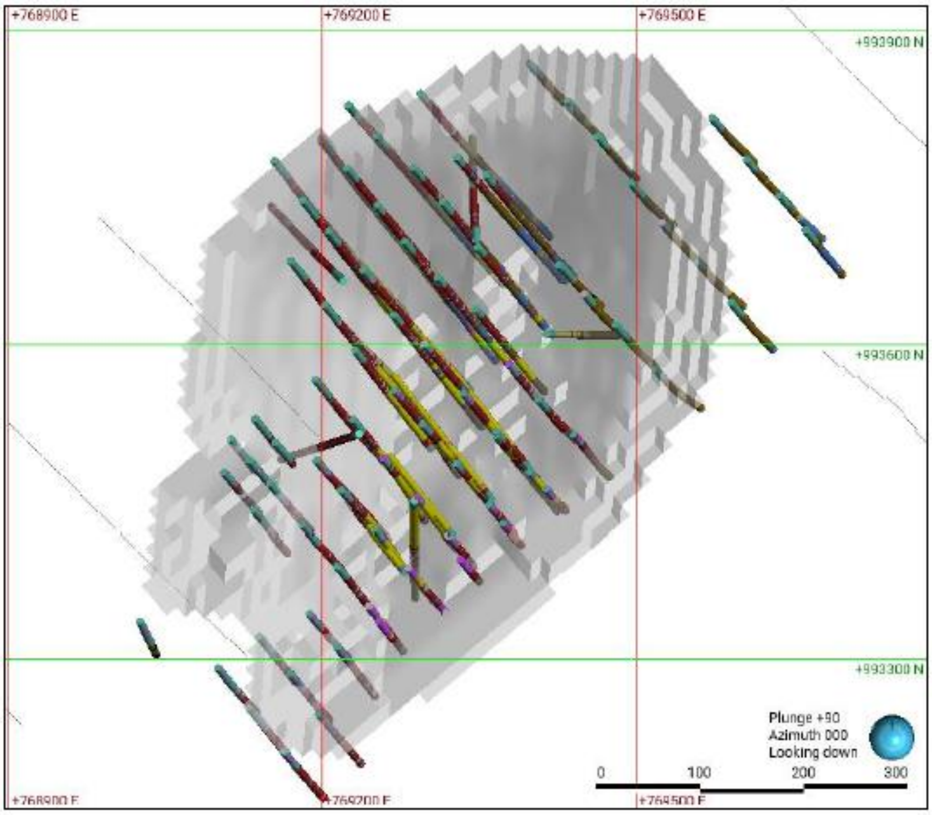
SRK considers the geotechnical investigations and subsequent analyses to date to have been developed to a level appropriate for a FS level open pit slope design. SRK makes one recommendation that should be included in any future geotechnical study:

> "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."

Gbongogo Pit

SRK personnel visited the site in June 2023 and logged six dedicated oriented geotechnical drillholes of nearly 1,260 m total length. There are also approximately 13,000 m of resource drillholes in or around the Gbongogo Pit shell, all of which contain lithology information. A total of 7,300 m of said resource boreholes was oriented, from which 5,338 joints have been recorded.

Figure 15.3.2 Gbongogo Pit Shell as of August 2023 and Six Cored Geotechnical Boreholes



Source: SRK, August 2023.

SRK personnel selected samples for laboratory testing while on site:

- 40 samples for rock mass testing: uniaxial (UCS, with or without Young's modulus measurement) and triaxial (TX).
- 144 samples for shear strength testing: 27 on saw-cut discontinuities (Direct Shear (DS)), 117 on natural discontinuities (Natural Shear (NS)).

Montage personnel under SRK's guidance collected 10 soil samples, on which classification and Soil Consolidated Undrained Triaxial Compression tests (CIUC) were carried out.

The Gbongogo deposit is hosted within a quartz-diorite intrusion located within the north-south- striking Gbongogo shear zone. On the western side of the shear zone the geology is characterised by fine-grained metasedimentary rocks, and on the eastern side of the shear zone the geology is characterised by coarse-grained siliciclastic sediments (conglomerates and sandstones). The boundary between the eastern and western domains is marked by a rhyolitic volcanic unit interpreted to be structurally emplaced, and the Gbongogo quartz-diorite is intruded into this rhyolite volcanic unit. The surrounding felsic volcanic rocks have deformed plastically and developed a strong foliation, while the quartz diorite has deformed in brittle manner, fracturing, and creating the space for gold-bearing quartz veins.

The hot, humid climate and intense rainfall within the region has resulted in a weathering profile of variable depth and a generation of residual soils . In general, the degree of weathering decreases with depth resulting in different tropical soil horizons. While the 'contacts' between each horizon are typically gradational, it is necessary to differentiate them for engineering purposes.

Three geotechnical domains have been defined: saprolite, weathered rock, and fresh rock, based on the degree of weathering and strength estimated logged in geotechnical drillholes. One domain and several minor structural sets (joints, foliation) have been defined for Gbongogo.

The foliation is by far the most pervasive set identified, with over 1,500 structures intersected across the deposit.

Feasibility-level hydrogeological investigation analysis were completed by Australasian Groundwater and Environmental Consultants Pty Ltd (AGE) in 2023.

Open pit stability will be controlled by shear strength within the weathered horizons (Table 15.3.3) and structurally controlled within the fresh rock. The fresh rock above +220 m RL (Table 15.3.4) is impacted by the foliation in the footwall zone between 80° and 150° azimuth. Within the fresh rock below +220 m RL (Table 15.3.5) the impact of the foliation is reduced, allowing for steeper wall angles. Fresh rock lithologies were grouped into two based on their strength: Group 1 (SSC, VRH, IQ and IDI).and Group 2 (SCO and IMD).

Table 15.3.3 Gbongogo Bench Design Criteria for Saprolite and Weathered Domains

Geotechnical Zone	Wall	Inter-Ramp Slope Angle °	Berm Width m	Batter Angle °	Bench Height - Designed m	Benches per Stack	Stack Height m
Saprolite	FW	31°	4	50	5	5	25
Weathered Rock	FW	41°	6	50	10	2	20
Saprolite	Other Walls	36°	4	60	5	4	25
Weathered Rock	Other Walls	40°	6	60	10	2	20

Table 15.3.4 Gbongogo Bench Design Criteria for Fresh Domains Above +220 m RL

Geotechnical Zone	Wall	Inter-ramp Slope Angle °	Berm Width m	Batter Angle °	Bench Height-Designed m	Benches per Stack	Geotechnical Berm m	Stack Height m
180	Southwest	53	8	70	20	4	30 m every 80 m	80
240	Southwest	64	8	85	20	4	30 m every 80 m	80
270	HW	49	6	75	10	8	25 m every 80 m	80
330	HW	64	8	85	20	4	30 m every 80 m	80
10	Northeast	56	8	75	20	4	30 m every 80 m	80
80	FW	35	6	50	10	8	20 m every 80 m	80
150	FW	53	8	70	20	4	30 m every 80 m	80

Table 15.3.5 Gbongogo Bench Design Criteria for Fresh Domains Below +220 m RL

Geotechnical Zone	Wall	Inter-ramp Slope Angle °	Berm Width m	Batter Angle °	Bench Height-Designed m	Benches per Stack	Geotechnical Berm m	Stack Height m
180	Southwest	53	8	70	20	4	30 m every 80 m	80
240	Southwest	64	8	85	20	4	30 m every 80 m	80
270	HW	49	6	75	10	8	25 m every 80 m	80
330	HW	64	8	85	20	4	30 m every 80 m	80
10	Northeast	56	8	75	20	4	30 m every 80 m	80
80	FW	40	6	60	10	8	20 m every 80 m	80
150	FW	53	8	70	20	4	30 m every 80 m	80

15.3.5 Optimisation Constraints

There are several surface features that require consideration for the optimisation. The Toudian Forest Reserve covers part of the northern Koné concession area, although this is not considered to be a hard constraint for mining activities the incursion has been minimised. 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 (TOS) 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.

A new haul road (38.1 km) from Gbongogo to the Koné plant site has been located outside of the Kowa Forest Reserve.

15.3.6 Base Mining Cost Estimate

In 3Q23, a budget tender process was conducted to ascertain market rates for contract mining operations. Six contractors provided a tender response. The tender evaluation was by formal tender. An average fixed cost per tonne of US$2.79 /t and US$2.80 /t (excluding rehandle costs) was developed based on the chosen bid for the Koné and Gbongogo deposits respectively.

The mining costs also include a grade control assay cost of US$0.26 /t for oxide ore and US$0.20 /t for fresh and transition ore. This cost has not been applied to the waste.

Table 15.3.6 shows the fixed mining costs derived from the contractor pricing study for the Koné and Gbongogo deposits.

Table 15.3.6 Summary of Fixed Mining Costs

Material	Koné	Gbongogo
	Fixed Unit Mining Cost US$ /t	Fixed Unit Mining Cost US$ /t
Oxide Ore	3.65	2.27
Transitional Ore	3.28	3.25
Fresh Ore	2.95	3.18
Oxide Waste	2.64	2.07
Transitional Waste	2.71	2.73
Fresh Waste	2.52	2.74

Incremental haulage costs were applied during the optimization process to account for vertical haulage. An incremental hauling cost of US$0.030 /t per 10 m of vertical haul was calculated and applied to all blocks within both deposits. Koné was applied from a reference RL of 375 m UTM (1,375 m RL local grid) and Gbongogo from a reference RL of 350 m UTM (1,350 m RL local grid).

A US$5.56 /t cost applied to Gbongogo ore hauled to the Koné processing plant. A rehandle cost of US$0.93 /t was applied to low grade ore rehandled from the stockpiles to the Koné ROM.

15.3.7 Processing Costs

Processing costs were developed by Lycopodium in consultation with Montage. Processing costs based on the deposit and oxidation category were provided for the Koné North and South Pits and the Gbongogo Pit. Table 15.3.7 shows the fixed and variable processing costs. Fixed costs were applied to the cost per tonne based on an 11 Mtpa processing rate.

Table 15.3.7 Processing costs (US$ /t Processed)

Material	Fixed Costs annual	Variable Cost US$ /t	Total Unit Cost US$ /t
South Oxide	$26,758k	$5.53	$7.96
South Transitional	$28,905k	$5.58	$8.20
South Fresh (Hanging Wall)	$28,910k	$6.78	$9.41
South Fresh (Footwall)	$28,910k	$6.79	$9.42
North Oxide	$26,758k	$5.53	$7.96
North Transitional	$28,905k	$5.90	$8.53
North Fresh	$28,910k	$6.74	$9.37
Gbongogo Oxide	$26,758k	$7.67	$10.11
Gbongogo Transitional	$28,905k	$5.56	$8.19
Gbongogo Fresh	$28,495k	$7.12	$9.80

15.3.8 Gold Price, Royalties and Selling Costs

A gold price of US$1,550 /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 4.0%, the Third Party Royalty of 2.0% (Triple Flag at Koné and Barrick / Endeavour at Gbongogo), and the Community Royalty of 0.5% (Table 15.3.8).

Table 15.3.8 Revenue and Selling Parameters

Item	Value
Gold Price (US$)	$1,550 /oz
Refining and Selling Cost (US$)	$5 /oz
Government Royalty	4.0%
Third Party Royalty	2.0%
Community Royalty	0.5%

15.3.9 Processing Recovery

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

The processing recoveries shown in Table 15.3.9 have been calculated for the different metallurgical zones at the predicted break-even cut-off grades.

Table 15.3.9 Processing Recoveries at Breakeven Cut-Off Grade and Average Pit Inventory Grade

Description	Processing Recovery at Cut-Off Grade	Processing Recovery at Average LOM Grade
South Oxide	88.2%	93.9%
South Transitional	90.2%	91.3%
South Fresh (Hanging Wall)	86.7%	87.7%
South Fresh (Footwall)	90.2%	89.1%
North Oxide	88.7%	93.2%
North Transitional	87.4%	88.0%
North Fresh	73.1%	77.1%
Gbongogo Oxide	83.6%	95.0%
Gbongogo Transitional	90.3%	91.2%
Gbongogo Fresh	70.8%	86.1%

15.3.10 Cut-Off Grade Determination

Cut-off grades were calculated for oxide, transitional and fresh material within each deposit, with the fresh material for the South Pit being further divided into footwall and hanging wall material. Table 15.3.10 shows the calculated cut-off grades at various gold prices.

Table 15.3.10 Cut-Off Grade Calculations

Description	US$1,300	US$1,400	US$1,550	US$1,600	US$1,700	US$1,800
South Oxide	0.23	0.22	**0.19**	0.19	0.18	0.17
South Transitional	0.23	0.22	**0.20**	0.19	0.18	0.17
South Fresh (Hanging Wall)	0.29	0.27	**0.25**	0.24	0.23	0.21
South Fresh (Footwall)	0.28	0.26	**0.23**	0.23	0.22	0.20
North Oxide	0.23	0.21	**0.19**	0.19	0.18	0.17
North Transitional	0.25	0.23	**0.21**	0.20	0.19	0.18
North Fresh	0.33	0.31	**0.28**	0.27	0.25	0.24
Gbongogo Oxide	0.51	0.47	**0.42**	0.41	0.39	0.37
Gbongogo Transitional	0.42	0.38	**0.34**	0.33	0.32	0.30
Gbongogo Fresh	0.59	0.54	**0.49**	0.47	0.45	0.42

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 15.3.11 shows the calculated cut-off grades at US$1,550 /oz and the associated grade bins.

Table 15.3.11 Cut-Off Grade Bins

	Cut-Off Grade	Model Grade Bin
South Oxide	0.19	0.20
South Transitional	0.20	0.20
South Fresh (Hanging Wall)	0.25	0.25
South Fresh (Footwall)	0.23	0.20
North Oxide	0.19	0.20
North Transitional	0.21	0.20
North Fresh	0.28	0.25
Gbongogo Oxide	0.42	0.40
Gbongogo Transitional	0.34	0.30
Gbongogo Fresh	0.49	0.45

15.4 Pit Optimisation Results

15.4.1 Methodology and Software

Pit optimisations were undertaken in the Deswik software (V2023.1) 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.

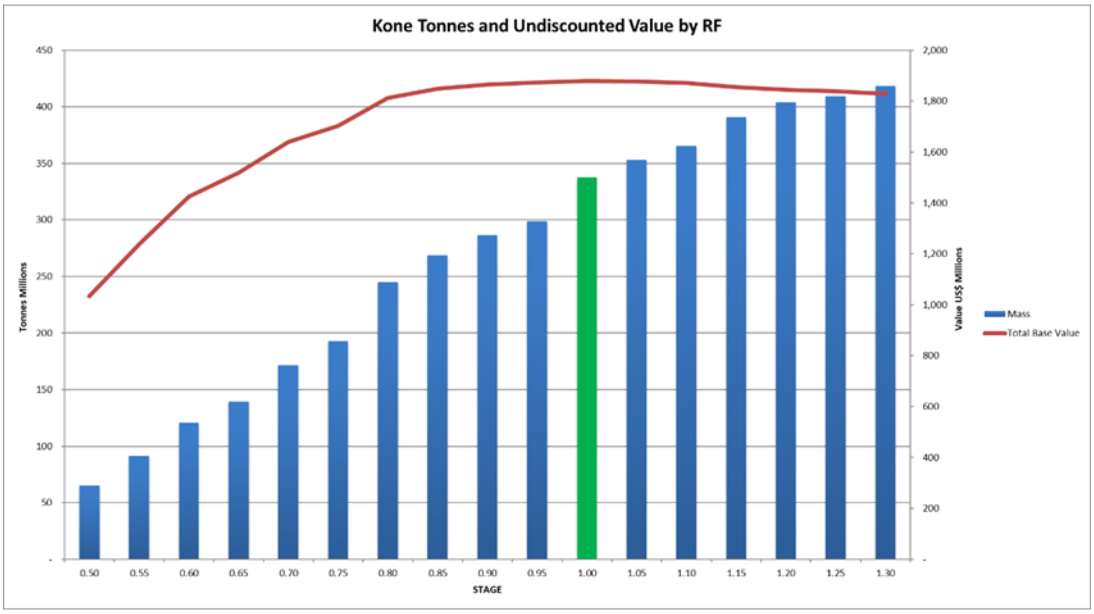
15.4.2 Optimisation Results and Pit Shell Selection

The optimisation process for the Koné deposit results for the in a series of nested shells, representing the economic material at each revenue factor. Given the reasonably standard results (shown in Figure 15.4.1), the Revenue Factor 1 shell was chosen for the basis of further work.

The chosen pit shell contained 161.4 Mt of ore at an average gold grade of 0.67 g/t, with 163.0 Mt of waste, producing a strip ratio of 1.01:1 (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 159.6 Mt of ore at a grade of 0.67 g/t while the North Pit shell contained 1.8 Mt of ore at a grade of 0.49 g/t.

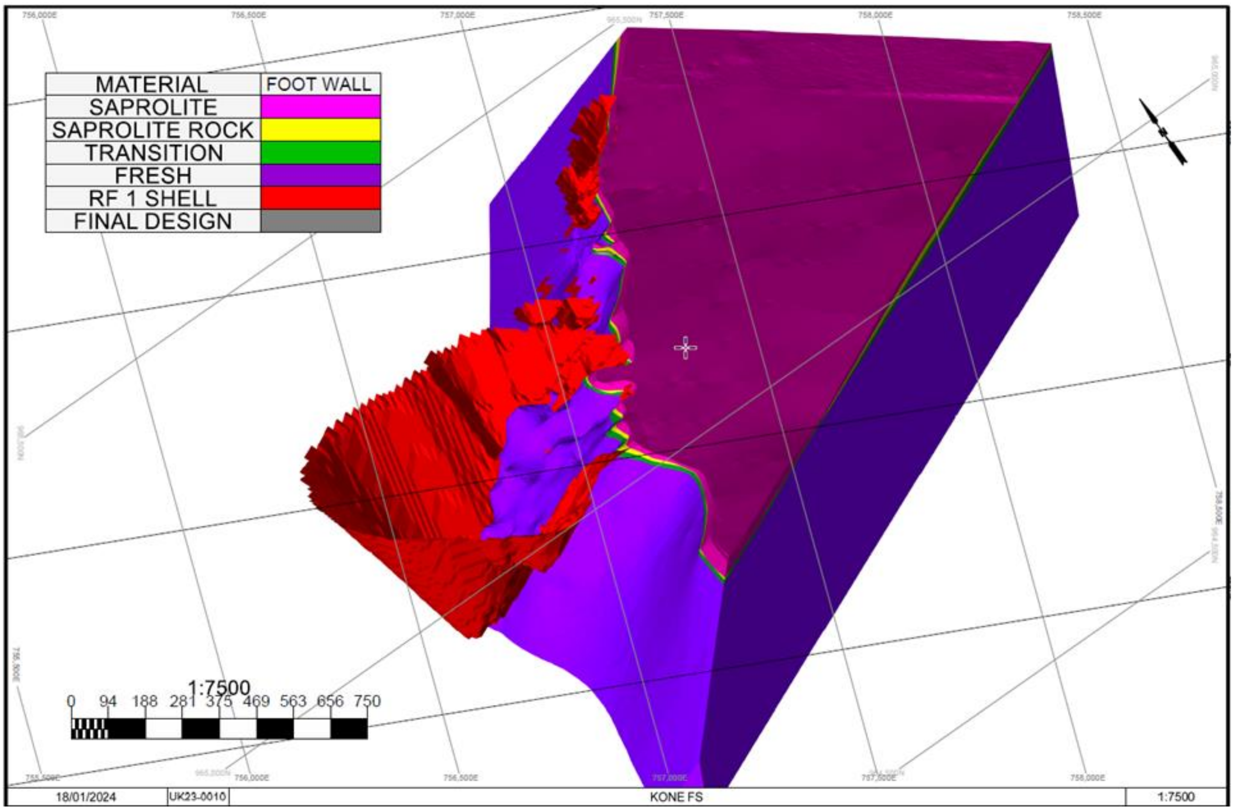
Figure 15.4.2 shows the chosen optimisation shell (RF=1), along with the solid identifying the footwall of the deposit. The east wall follows this contact and is considerably shallower than the west wall. This is due to the shell following the dip of the orebody and recovering ore in the east wall, while the west wall is contained entirely within waste rock.

Figure 15.4.1 Results of Koné Optimisation Study



Source: Carci, October 2023.

Figure 15.4.2 Koné RF 1 Optimisation Shell and Footwall Oxidation State Solid



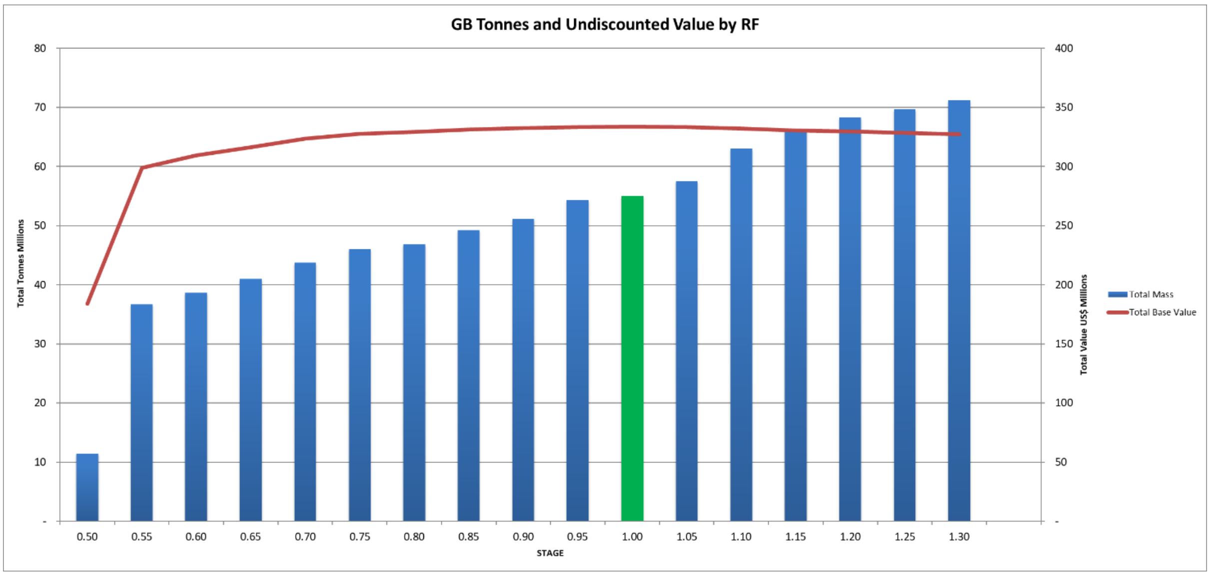
Source: Carci, January 2024.

The optimisation process for Gbongogo deposit results in a series of nested shells, representing the economic material at each revenue factor. Given the reasonably standard results (Figure 15.4.3), the Revenue Factor 1 shell was chosen for the basis of further work.

The chosen pit shell for the Gbongogo deposit contained 10.58 Mt of ore at an average gold grade of 1.45 g/t, with 29.8 Mt of waste, producing a strip ratio of 2.82:1 (waste:ore). There is a single pit for Gbongogo.

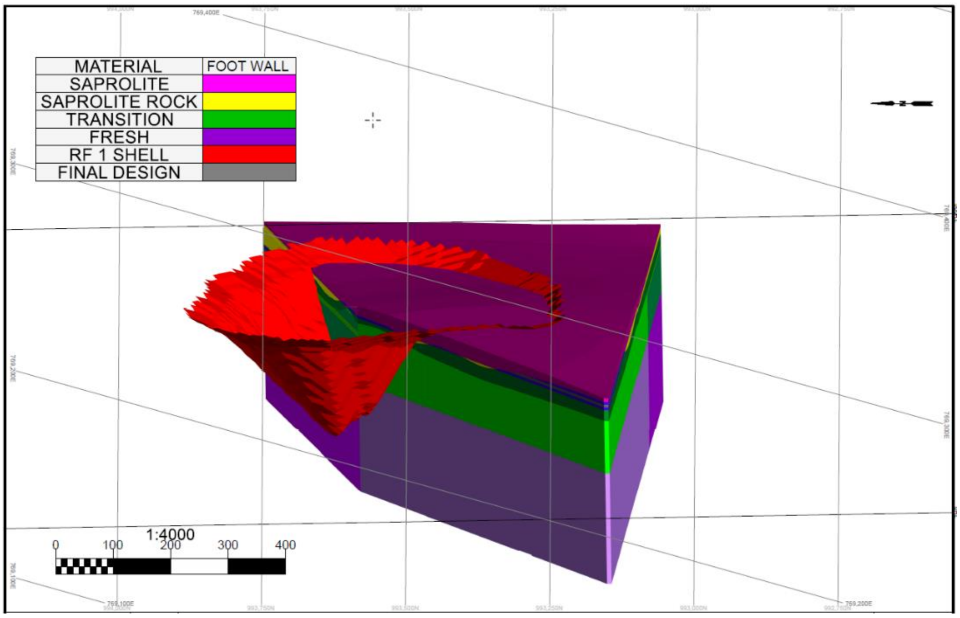
Figure 15.4.4 shows the chosen optimisation shell (RF=1) along with the solid identifying the footwall of the deposit. It can be seen that the east wall follows this contact and is considerably shallower than the west wall. This is due to the shell following the dip of the orebody and recovering ore in the east wall, while the west wall is contained entirely within waste rock.

Figure 15.4.3 Results of Gbongogo Optimisation Study



Source: Carci, October 2023.

Figure 15.4.4 Gbongogo RF 1 Optimisation Shell and Footwall Oxidation State Solid



Source: Carci, January 2024.

15.5 Mine Design

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 FS. 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 from the Koné and Gbongogo Pits, and is processed once mining is completed. A cut-off grade of 0.65 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, mining rates of 39 Mtpa and 15.5 Mtpa were used for Koné and Gbongogo respectively while the processing rate was capped at 11 Mtpa. This is discussed further in Section 16.0.

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.

Overall, 85% of the total material mined comes from the Koné pit due to the lower strip ratio. Koné contributes 94% of the ore within the Reserve. However, due to a higher grade in Gbongogo (1.43 g/t) compared to Koné (0.65 g/t), the Koné deposit contributes 88% of the contained ounces (3,518 koz) compared to 12% contained ounces from the Gbongogo Pit (493 koz).

The Gbongogo Pit is mined over a period of four years, including the pre-strip year, which allows for the highest grade ore feed to the mill during the initial mining phase, as well as bringing fresh rock online earlier to balance the feed to the plant.

15.5.2 Pit Design Criteria

The geotechnical constraints for the Koné design were based upon the main oxidation state of the material. Table 15.5.1 shows the geotechnical inputs for the Koné final pit design.

Table 15.5.1 Modified Geotechnical Inputs to Koné Pit Design

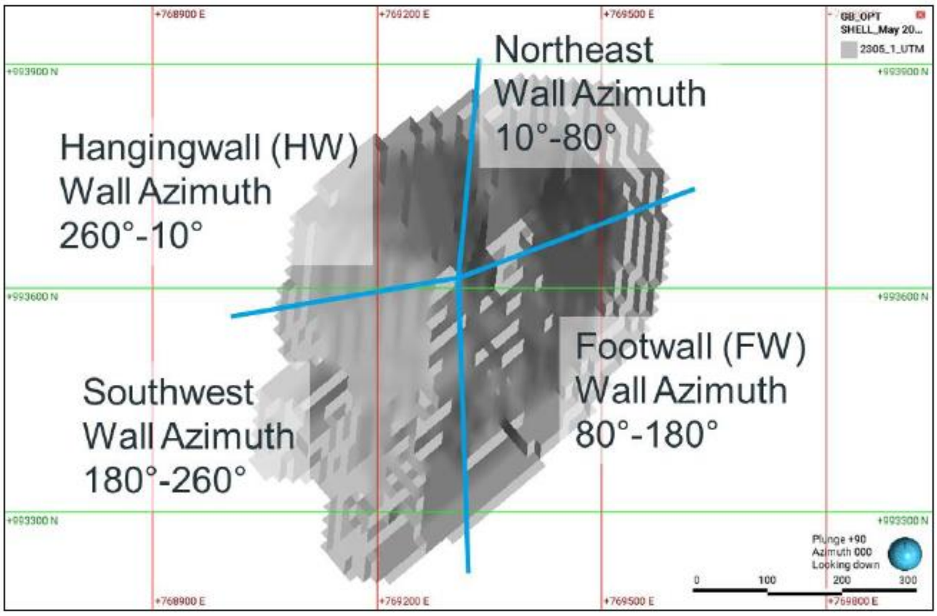
Geotechnical Zone	RL's	Inter-ramp Slope Angle °	Berm Width m	Batter Angle °	Bench Height-Designed m	Benches per Stack	Geotechnical Berm m	Stack Height m	Overall Angle with Ramps & Geotechnical Berms °
Oxide	Above 360	39°	3.5	60	5	1	6	10	24.6
Transition	Between 340-360	58°	9	80	10	2	9	40	33.3
Fresh Rock 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 m was maintained in all cutbacks within the Koné deposit, except for the 'goodbye cuts' at the base of the cutbacks, which were reduced to 30 m.

The geotechnical constraints for the Gbongogo pit were based upon the oxidation state as well as the foliation within the footwall limits. The follow pit sectors have been defined (Figure 15.5.1):

- Northeast Wall (NE): azimuth between 10° and 80°.
- Hanging Wall (HW): azimuth between 260° and 10°.
- Footwall (FW): azimuth between 80° and 180°.
- Southwest Wall (SE): azimuth between 180° and 260°.

Figure 15.5.1 Gbongogo Pit Design Sectors and Corresponding Aziumths



Source: SRK, November 2023.

The pit wall dip directions, pit design sectors and corresponding azimuths are clarified in Table 15.5.2 for the saprolite and transition within Gbongogo Pit. The fresh rock above the 220 RL (Table 15.5.3) is impacted in the foliation zone and has shallower IRA (35°) within the footwall zone between azimuths of 80° to 180°, this is steepened to 40° below the 220 RL (Table 15.5.4).

Table 15.5.2 Geotechnical Inputs for Saprolite and Transition within Gbongogo Pit Design

Geotechnical Zone	Wall	Inter-ramp Slope Angle	Berm Width	Batter Angle	Bench Height-Designed	Benches per Stack	Stack Height
		(°)	(m)	(°)	(m)		(m)
Saprolite	FW	31°	4	50	5	5	25
Weathered Rock	FW	41°	6	50	10	2	20
Saprolite	Other Walls	36°	4	60	5	4	25
Weathered Rock	Other Walls	40°	6	60	10	2	20

Geotechnical berms would be added as required after ramps had been taken into account within the overall slope angle.

Table 15.5.3 Bench and Inter-Ramp Design Criteria for GB Fresh Domain (Above +220 mRL)

Geotechnical Zone	Wall	Inter-ramp Slope Angle °	Berm Width m	Batter Angle °	Bench Height – Designed m	Benches per Stack	Geotechnical Berm m	Stack Height m
180	Southwest	53	8	70	20	4	30 m every 80 m	80
240	Southwest	64	8	85	20	4	30 m every 80 m	80
270	HW	49	6	75	10	8	25 m every 80 m	80
330	HW	64	8	85	20	4	30 m every 80 m	80
10	Northeast	56	8	75	20	4	30 m every 80 m	80
80	FW	40	6	60	10	8	20 m every 80 m	80
150	FW	53	8	70	20	4	30 m every 80 m	80

Table 15.5.4 Bench and Inter-Ramp Design Criteria for GB Fresh Domain (Below +220 m RL)

Geotechnical Zone	Wall	Inter-ramp Slope Angle °	Berm Width m	Batter Angle °	Bench Height – Designed m	Benches per Stack	Geotechnical Berm m	Stack Height m
180	Southwest	53	8	70	20	4	30 m every 80 m	80
240	Southwest	64	8	85	20	4	30 m every 80 m	80
270	HW	49	6	75	10	8	25 m every 80 m	80
330	HW	64	8	85	20	4	30 m every 80 m	80
10	Northeast	56	8	75	20	4	30 m every 80 m	80
80	FW	35	6	50	10	8	20 m every 80 m	80
150	FW	53	8	70	20	4	30 m every 80 m	80

15.5.3 Haul Road Design

The haul road design parameters used in the pit designs for all deposits are summarised in Table 15.5.5 and are based upon the manufacture's standards and best practice operating procedures. The equipment selection process nominated CAT 777 trucks, with an operating width of 6.5 m, as the preferred unit and this was used for all the productivity and haulage calculation.

Table 15.5.5 Haul Road Design

Parameter	Units	Value
Ramp Gradient	%	10%
Gradient style		Constant
Ramp Width (double lane)	m	25
Ramp Width (single lane)	m	15
Turning Radius	m	12.5

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 1.45 m in height.

A distance of 0.5 m 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 15.5.6. The design for the dumps in South of Koné allows for the contractor to use larger equipment if more economic, the North Pit dumps and stockpiles are limited to a width of 32 m to minimise the footprint within the forest reserve.

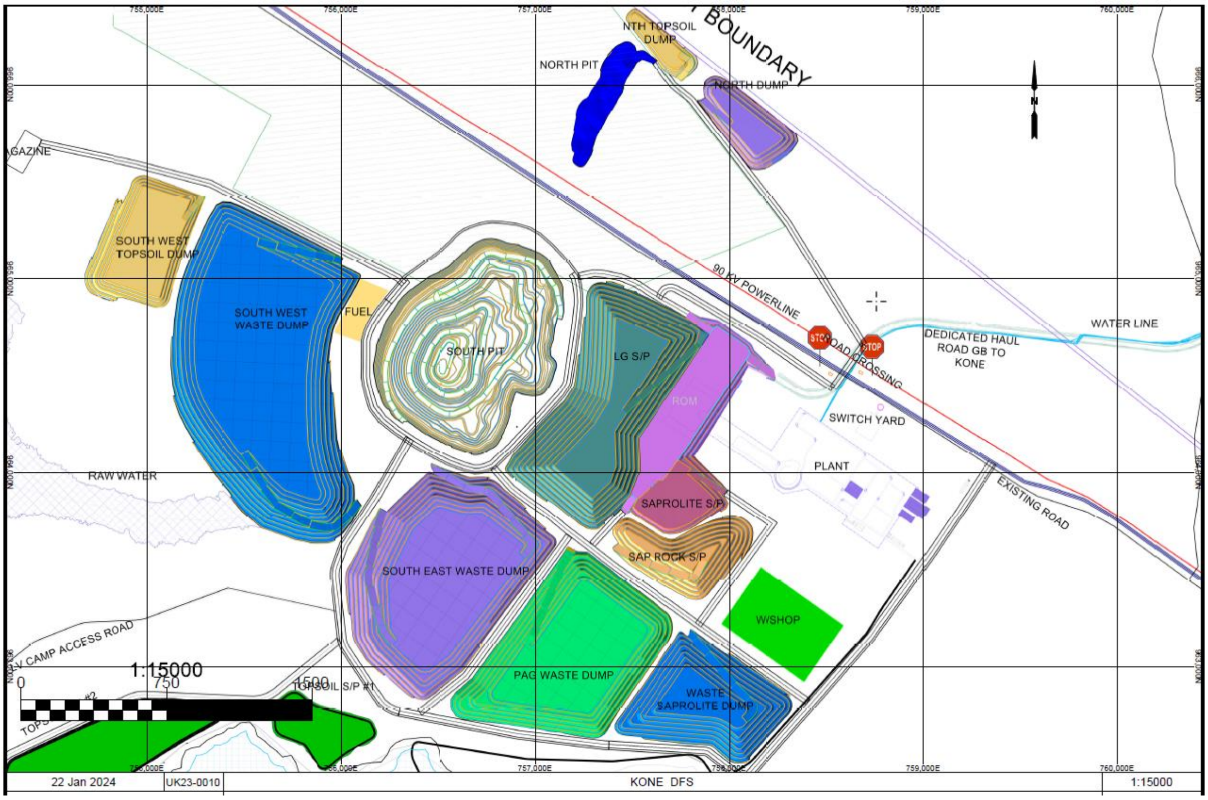
Table 15.5.6 Waste Dump and Stockpile Design Parameters

Parameter	Koné	Gbongogo
Lift height (m)	10	10
Lift angle (°)	37	37
Berm width (m)	10	10
Ramp width (m)	40	25
Ramp gradient (%)	10	10

There are seven areas that are to be used to stockpile Koné waste material: the construction of the ROM pad, the Southwest Dump, Southeast Dump, Potentially Acid Generating (PAG) dump, Saprolite Dump, North Dump, and backfilling of the of the North Pit in the latter years (Figure 15.5.2). The PAG Dump will be lined with waste saprolite material prior to any PAG material being deposited. The Saprolite Dump will contain additional saprolite material that will be used to cap the exposed sides of the PAG Dump when completed.

The North and South Pits contain a total waste tonnage of 165.0 Mt. Once mining of the South Pit is completed, tailings will be discharged into the pit void and the South 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.

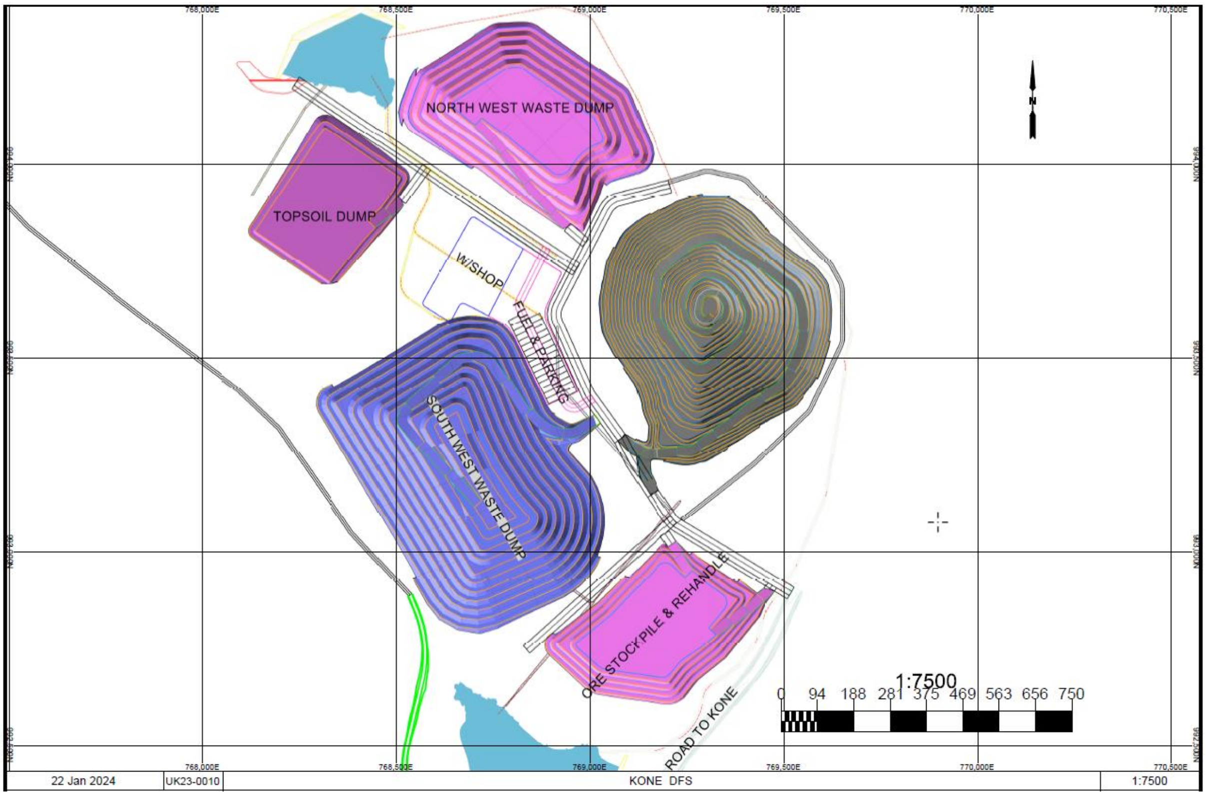
Figure 15.5.2 Koné Waste Dump Locations



Source: Carci, January 2024.

The Gbongogo Pit contain a total waste tonnage of 40.3 M (Figure 15.5.3). The stockpiles and topsoil areas are only active during the extraction period and will be rehabilitated along with the South West and North West waste dumps at the completion of mining.

Figure 15.5.3 Gbongogo Waste Dump Locations



Source: Carci, January 2024.

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 163.7 Mt of mill feed at a grade of 0.67 g/t Au. This is associated with 165.0 Mt of waste rock, providing a strip ratio of 1.01:1 (waste:ore).

The pit inventory by deposit is summarised in Table 15.6.1. Nearly 90% of the material is fresh rock. To best accommodate this, the excavators and trucks will be purchased and configured for fresh rock density and hard rock characteristics. 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 (Figure 15.6.1). 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.

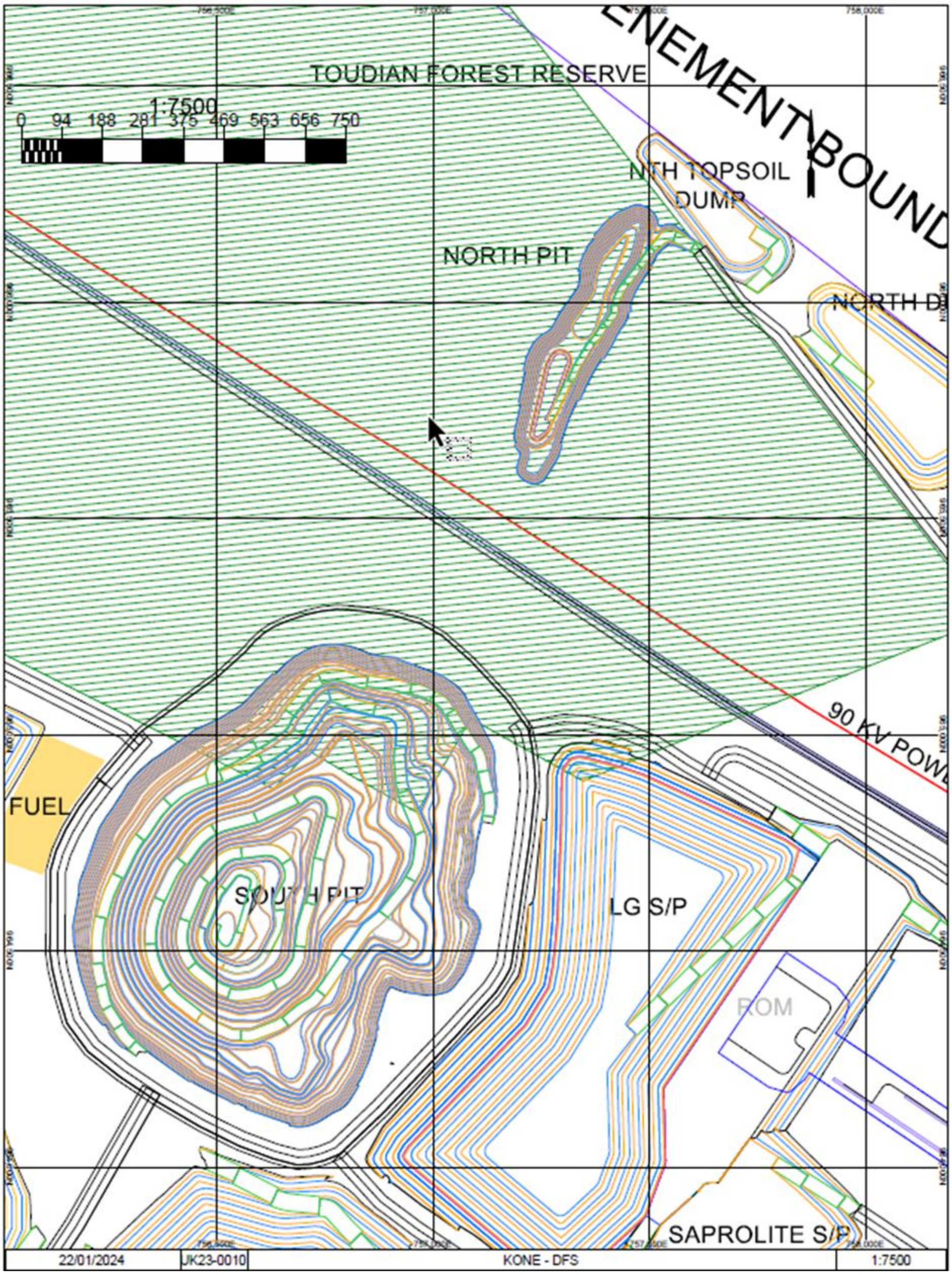
Table 15.6.1 Pit Inventory

Material	Units	South	North	Gbongogo	Total
Saprolite Oxide Mined Tonnes	Mt	6.6	0.9	0.2	7.6
Saprolite Oxide Mined Grade	g/t	0.60	0.47	1.42	0.60
Saprolite Rock Mined Tonnes	Mt	3.1	0.1	0.5	3.7
Saprolite Rock Mined Grade	g/t	0.56	0.37	1.34	0.67
Transitional Mined Tonnes	Mt	7.0	0.4	0.5	7.9
Transitional Mined Grade	g/t	0.60	0.4	1.09	0.63
Fresh Footwall Mined Tonnes	Mt	15.8	0.0	1.1	16.9
Fresh Footwall Mined Grade	g/t	0.58	0.28	1.46	0.64
Fresh Haning Wall Mined Tonnes	Mt	129.5	0.4	8.3	138.2
Fresh Hanging Wall Mined Grade	g/t	0.69	0.51	1.46	0.74
Total Mined Tonnes	**Mt**	**161.9**	**1.8**	**10.7**	**174.3**
Total Mined Grade	**g/t**	**0.67**	**0.47**	**1.43**	**0.72**
Oxide Waste Tonnes	Mt	18.7	1.5	7.6	27.8
Transitional Waste Tonnes	Mt	11.6	0.2	3.7	15.5
Fresh Footwall Waste Tonnes	Mt	6.2	0.0	13.0	19.2
Fresh Hanging Wall Waste Tonnes	Mt	126.4	0.4	16.0	142.7
Total Wast Tonnes	**Mt**	**162.8**	**2.1**	**40.3**	**205.3**
Total Tonnes	**Mt**	**324.7**	**4.2**	**51.4**	**380.3**
Strip Ratio	t waste : t ore	1.01	1.19	3.77	1.18

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 m (excluding the final goodbye cuts at 30 m), the double width ramp width of 25 m, 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. The South Pit also has an initial waste rock starter pit to provide material suitable for the initial TSF and WSF construction at Koné. This starter pit provides sufficient material during the first period of pre-mining to satisfy all waste construction including the ROM pad.

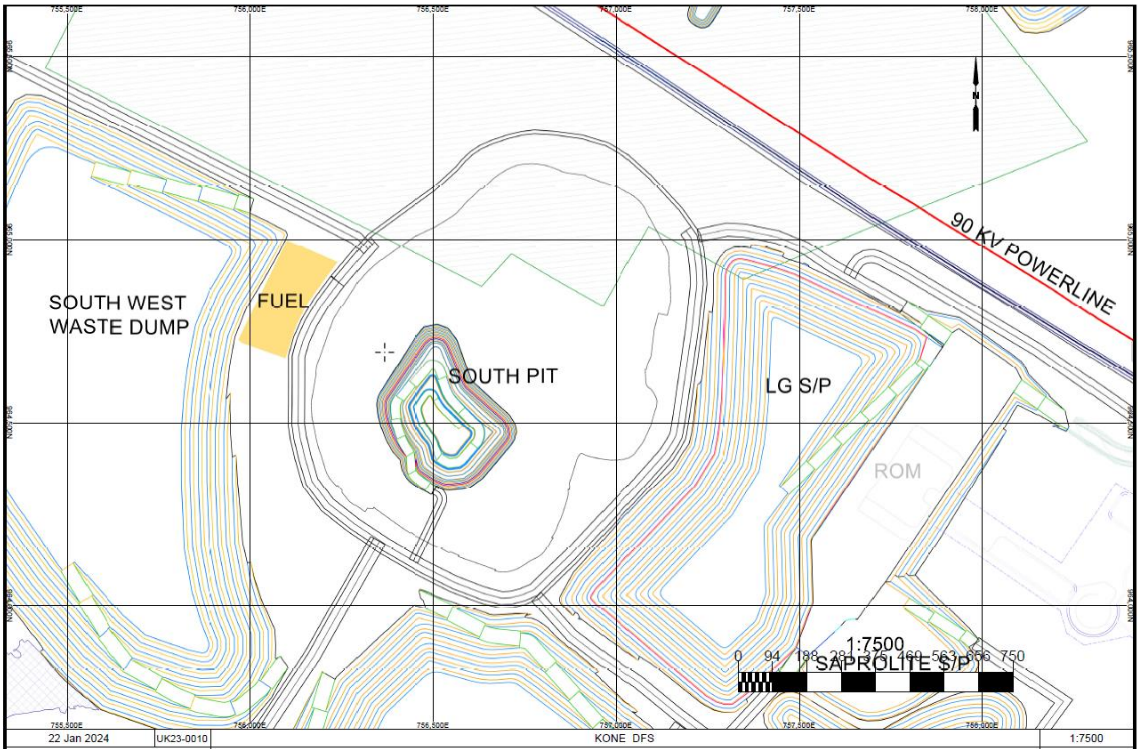
As part of the elevated cut-off grade methodology, the requirement of the initial two cutbacks is to provide sufficient feed of greater than 0.65 g/t gold to the processing plant within the early years of the Project. The designs for the southern cutbacks (Figure 15.6.2) show the initial waste starter pit required for pre-construction activities this has a total tonnage of 10.2 Mt with 0.75 Mt ore at 0.50 g/t. Figure 15.6.3 for Cutback 1 and Figure 15.6.4 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 two to three years of the Project.

Figure 15.6.1 North and South Pit Final Designs



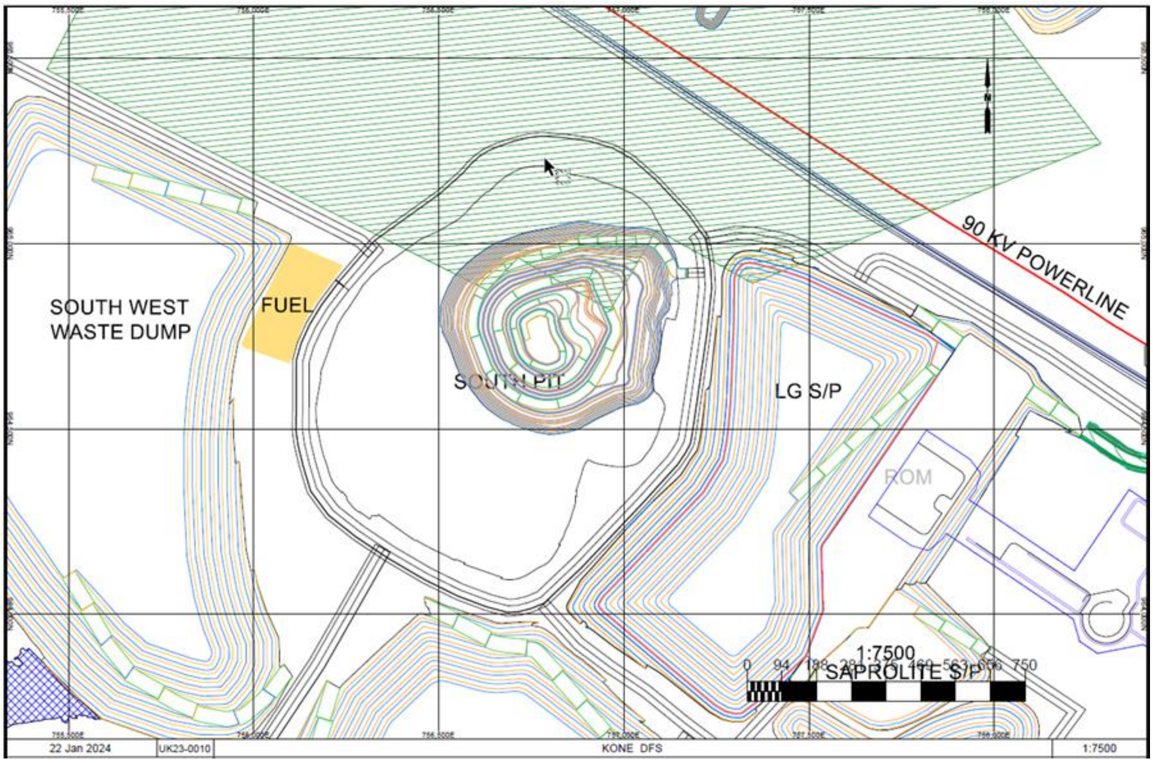
Source: Carci, January 2024.

Figure 15.6.2 South Pit Waste Starter Pit Engineered Design



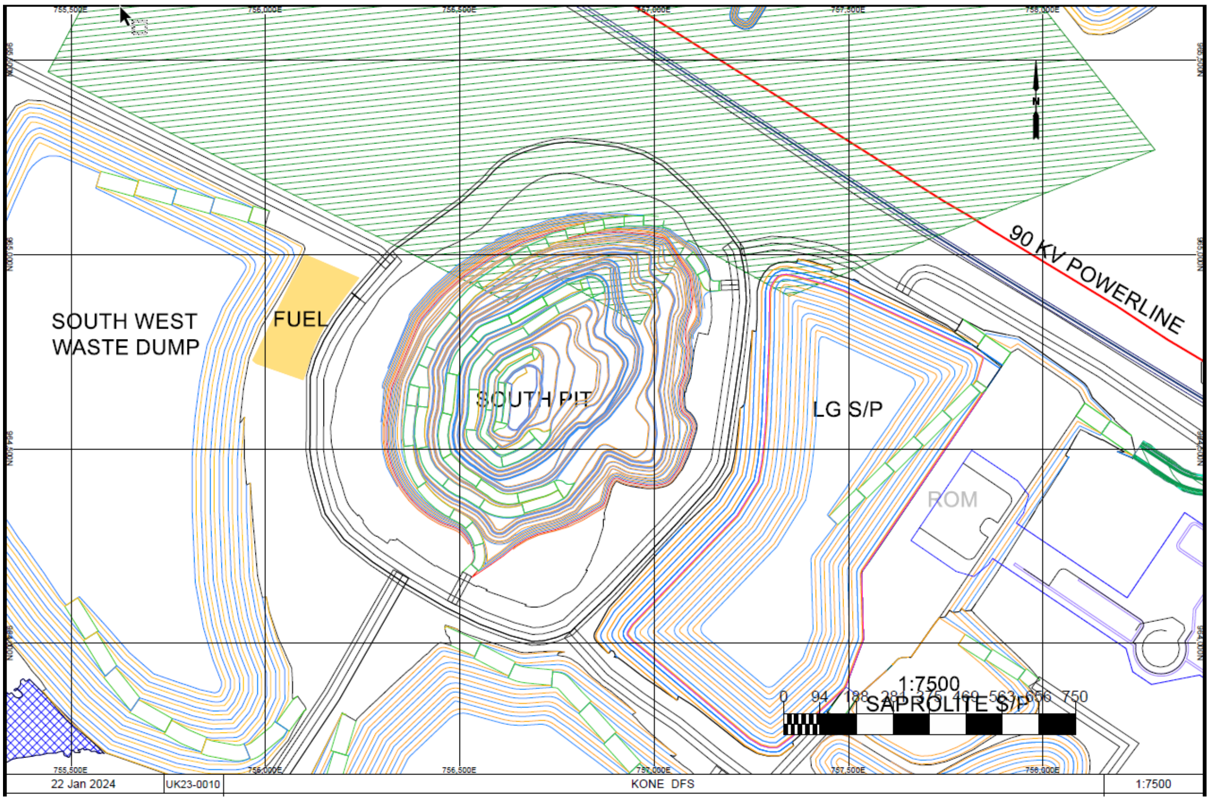
Source: Carci, January 2024.

Figure 15.6.3 South Pit Cutback 1 Engineered Design



Source: Carci, January 2024.

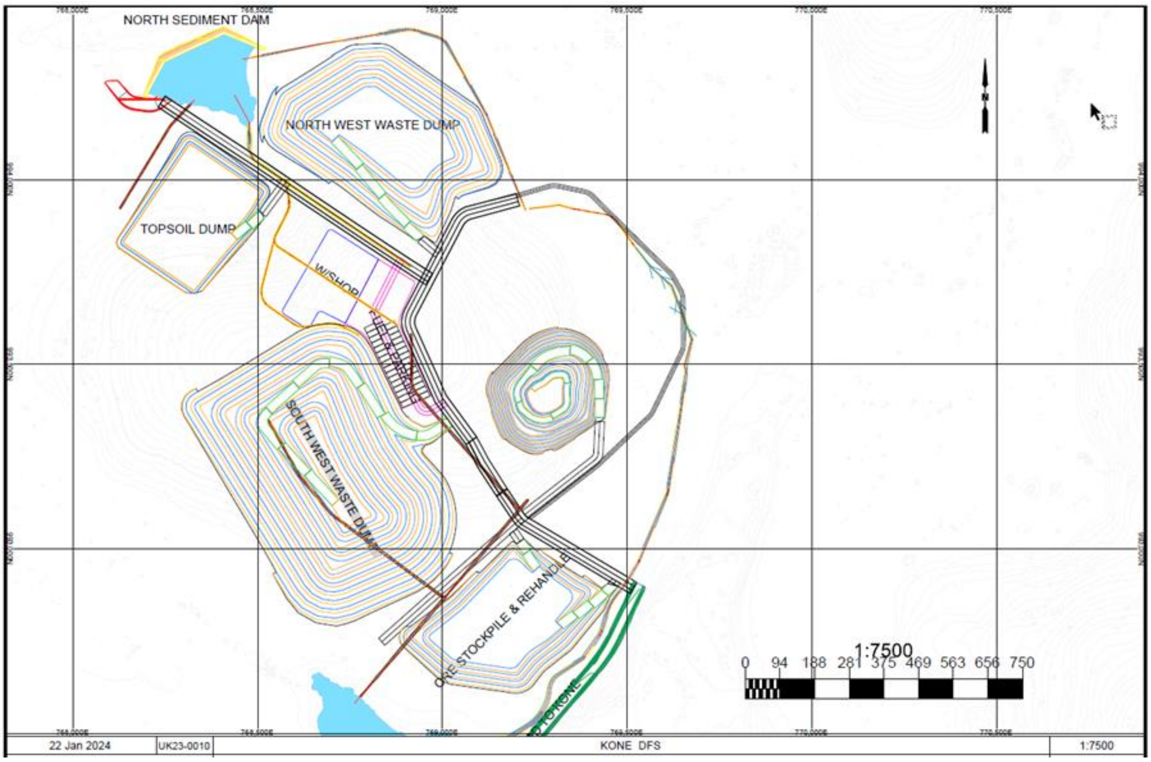
Figure 15.6.4 South Pit Cutback 2 Engineered Design



Source: Carci, January 2024.

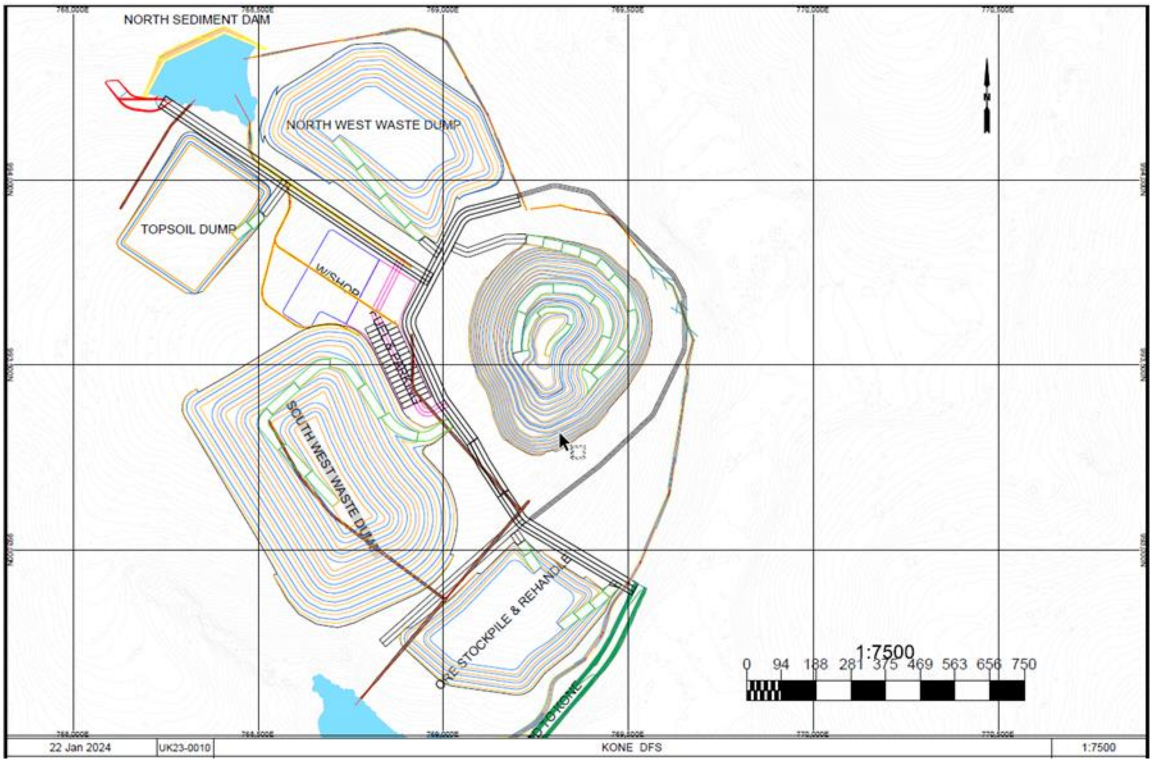
The satellite asset Gbongogo has also been split into three cutbacks, this is to provide an even strip ratio as well as access the high grade fresh rock ore within the first pre-strip period. The initial Cutback 1 (Figure 15.6.5) has a strip ratio of 0.86:1 with an average grade of 1.66 g/t. Cutback 2 (Figure 15.6.6) has a strip ratio of 5.6:1 primarily due to the shallow wall angles required by the foliation, this has an average grade of 1.2 g/t. The final Cutback 3 for Gbongogo (Figure 15.6.7) has a strip ratio of 5.5:1 with an average grade of 1.49. All of the material within Gbongogo is direct high grade feed to the plant, although some saprolite and saprolite rock is stockpiled for blending purposes.

Figure 15.6.5 Gbongogo Cutback 1 Engineered Design



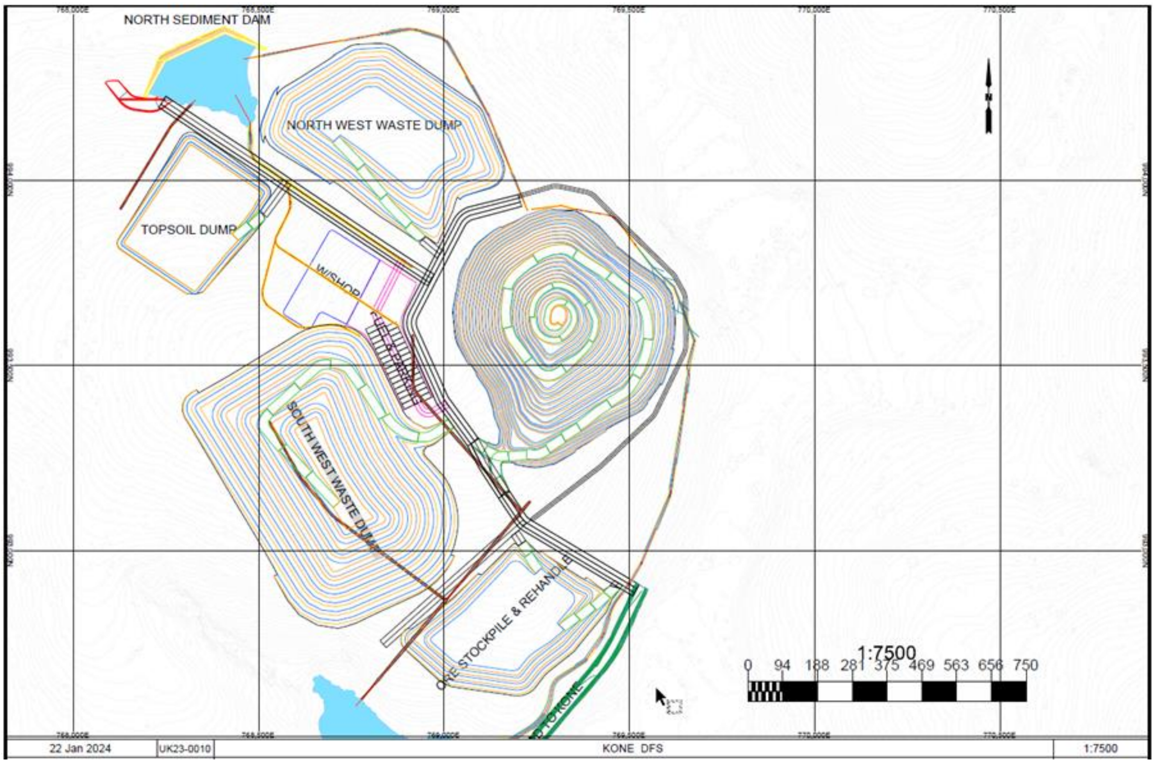
Source: Carci, January 2024.

Figure 15.6.6 Gbongogo Cutback 2 Engineered Design



Source: Carci, January 2024.

Figure 15.6.7 Gbongogo Cutback 3 Engineered Design



Source; Carci, January 2024.

15.7 Risk Assessment

A risk assessment (Table 15.7.1) for the development stage of the Project identified risks related to the investigations, data quality, estimations, planning, and decision making.

Table 15.7.1 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
Haul road to satellite causing seasonal issues	Possible	Minor	Medium

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 2023 levels while 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.

16.0 MINING

16.1 Mining Method

16.1.1 General Approach

The report 'Koné Gold Project Feasibility Study Mining Report (UK23-0010)' produced by Carci contains a detailed discussion on the mining methods anticipated for this project.

Based on the geometry and physical properties of the deposits, the proximity of the deposits to surface and the gold grades contained within the deposits, 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 mine life for the South, North and Gbongogo is 9.0, 4.0 and 3.75 years respectively, including nine months of preproduction mining in the South and Gbongogo Pits. The mining strategy utilises an elevated cut-off grade approach for the South and North Pits, where ore with a gold grade over 0.65 g/t is preferentially processed while ore with a gold grade over economic cut-off but under 0.65 g/t is considered low grade material and stockpiled or used as supplementary feed. As Gbongogo has higher grades, it is processed directly. This strategy is in place for the first eight 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 eight years.

The optimal mining rates for Koné and Gbongogo are 39 Mtpa and 15.5 Mtpa respectively and processing at 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 Koné South and Gbongogo Pits 24 h a day utilising several shifts. The number of shifts per 24-h 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, which minimises the interaction of haul trucks crossing the public road. Rehandle at the North Pit will be minimised as much as possible.

Gbongogo will run a separate crew 24 h a day utilising several shifts. The purpose built haul road will allow for the ore to be trucked 24 h a day from Gbongogo to Koné processing plant (38.1 km).

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

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 FS 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 (200 t + capacity trucks, etc.) is limited in West Africa.

A Caterpillar 777 (91 t) 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 6030 excavator was matched to the CAT 777 haul trucks for the purposes of this study.

At Koné South, two CAT 6030 excavators (30 t bucket capacity) are required in the pre-production year, with the third and fourth coming online in Year 1. This gives an overall capacity of 40.8 Mtpa in hard rock material (which accounts for 85% of the material within the schedule). On average that the number of excavators utilised during a year is 3.9.

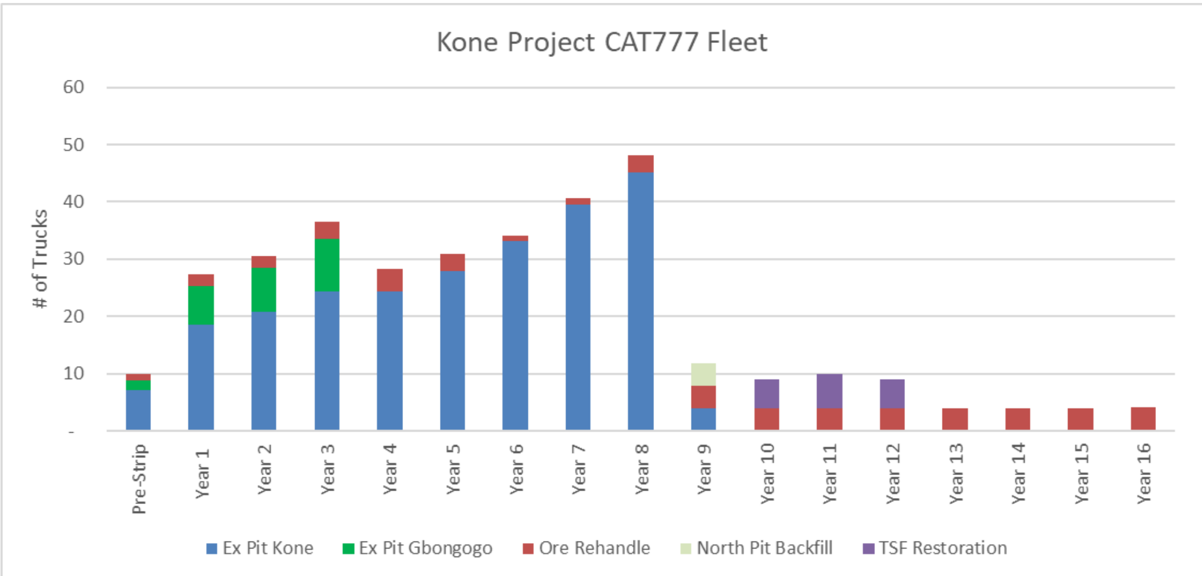
The same equipment types are utilised at Gbongogo with a combination of 90 t haul trucks matched with a 30 t bucket capacity excavator. The initial preproduction at the Gbongogo Pit requires one shovel with an additional shovel in Year 1. The average number of shovels employed is 1.5. The Gbongogo Pit is completed by the end of Year 3, when one of the excavators will become available for North Pit, which will be campaigned on night shift only. The North 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 48 trucks in Year 9 (Table 16.1.1). The open pit mining finishes in July of Year 9. The waste rehandle for the tailings closure and the North Pit backfill utilises the retired mining fleet from Year 9 onwards (Figure 16.1.1). Following the completion of mining in Year 9, tailings will be deposited into the South Pit.

Table 16.1.1 Total CAT 777 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	Yr 15	Yr 16
Ex Pit Koné	7	19	21	24	24	28	33	40	45	4		-	-	-	-	-	-
Ex Pit Gbongogo	2	7	8	9	-	-	-	-	-	-		-	-	-	0	0	0
Rehandle																	
Ore Rehandle	1	2	2	3	4	3	1	1	3	4	4	4	4	4	4	4	4
North Pit Backfill	0	0	0	0	0	0	0	0	0	4	0	0	0	0	0	0	-
TSF Restoration	0	0	0	0	0	0	0	0	0	0	5	6	5	0	0	0	-
Total CAT777	**10**	**27**	**31**	**37**	**28**	**31**	**34**	**41**	**48**	**12**	**9**	**10**	**9**	**4**	**4**	**4**	**4**

Figure 16.1.1 Total CAT 777 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 16.1.2).

Table 16.1.2 Ancillary Equipment Requirements

Description	Type	Koné No. of Units	Gbongogo No. of Units
Tyre Handler	MANITOU MHT	1	1
Forklift	JCB 940	1	
Boom Lift	JLG 600	1	1
Munk Truck Crane 8 x 4	PK56502 (20T)	1	
Truck	9Ton	1	
High Capacity Low Bed +Tractor	100T	1	
Diesel Truck Diesel	35 m3	2	1
Service / Lub Truck	MAN 6x6	2	1
Lighting Towers	Atlas Copco	10	10
Generator	100 kVA	1	1
Mobile Crane	90 Ton	1	
Pickups	Toyota	24	4
Stemming Loader		1	1
Water Pumps		1	1

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.

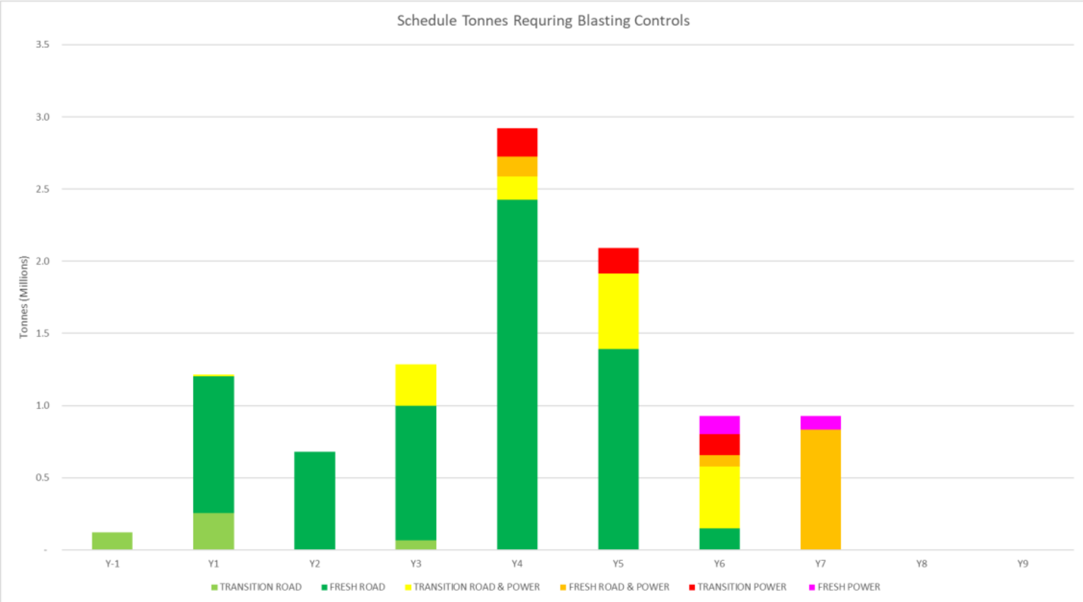
16.2 Drilling and Blasting

The production drilling and blasting will be carried out by the mining contractor on 10 m benches in most situations. It is expected that most drillholes will be 165 mm 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 16.2.1 shows the mined tonnes by year contained within the buffer zone. This represents less than 3% of the total tonnes within the Koné Pit. The Gbongogo Pit doesn't require any blasting controls as there is no interaction with the national road or powerlines. A set 500 m standoff with blast guards will be in place during all blasting.

Figure 16.2.1 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 will be reduced to within acceptable levels by using programmable electronic detonators to control the output vibrations.

A study will be undertaken prior to blasting within the 500 m 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

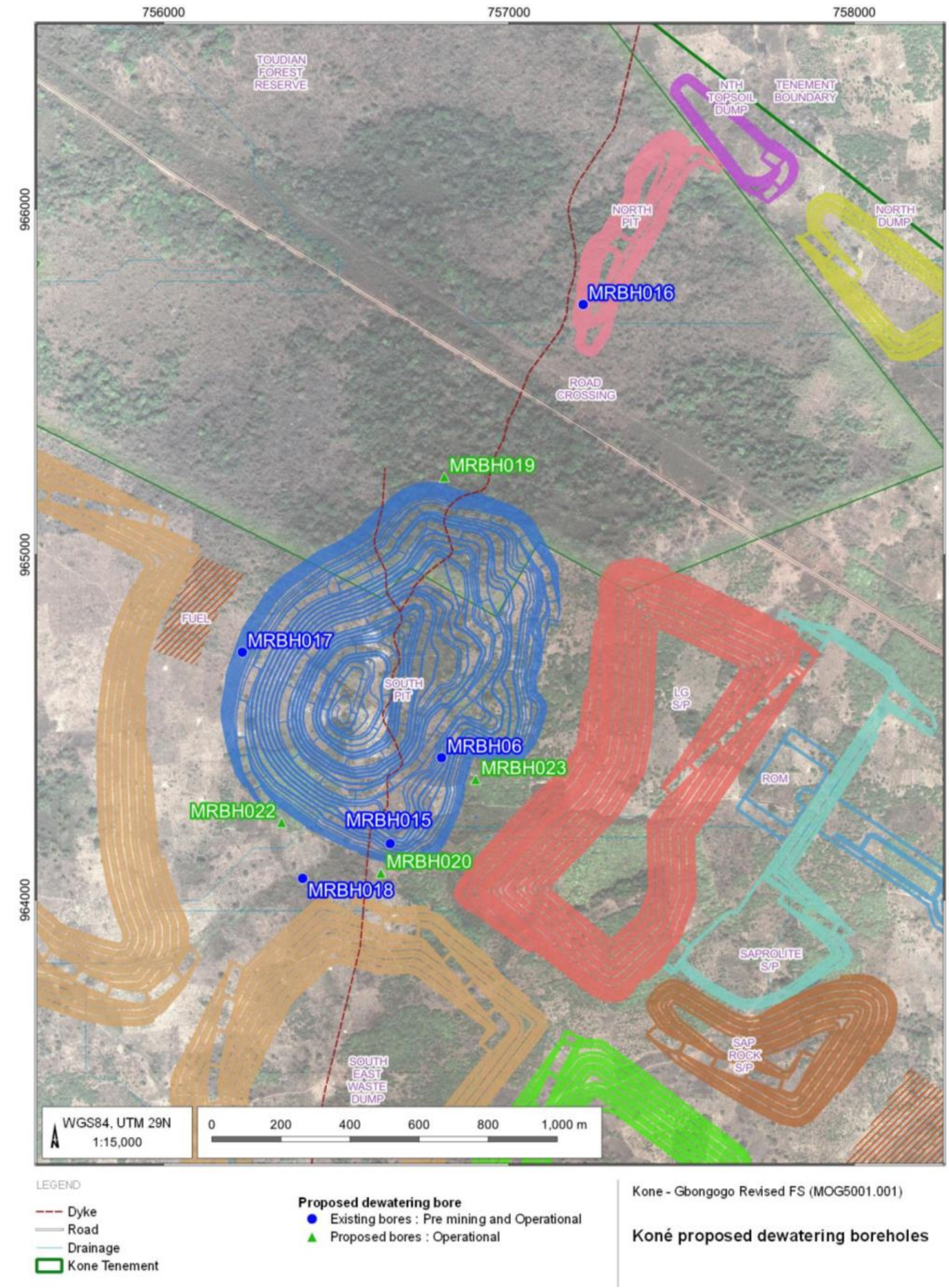
AGE conducted a hydrogeological and dewatering study on the Koné deposits. As part of this study, AGE proposed a pit dewatering concept that consisted of two stages. The first was a pre-mining stage, using in-pit and out of pit dewatering boreholes. These boreholes would target the high yielding areas 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 WSF for later use. AGE has recommended a minimum in-pit pump capacity of approximately 8,000 m^3/day for the Koné Pit and approximately 3,500 m^3/day for the Gbongogo Pit.

Maximum daily flood events were considered (120 mm in 24 h) to estimate the potential delays from storm events in mining and subsequent potential dewatering periods. It is estimated that the 1:100 year flood event will take between five and 10 days to dewater. The plant would switch to processing the stockpiled ore during these periods. Dewatering requirements will peak during the wet season, allowing for rainfall flood events, to approximately 13,000 m^3/day for Koné and 5,000 m^3/day for Gbongogo.

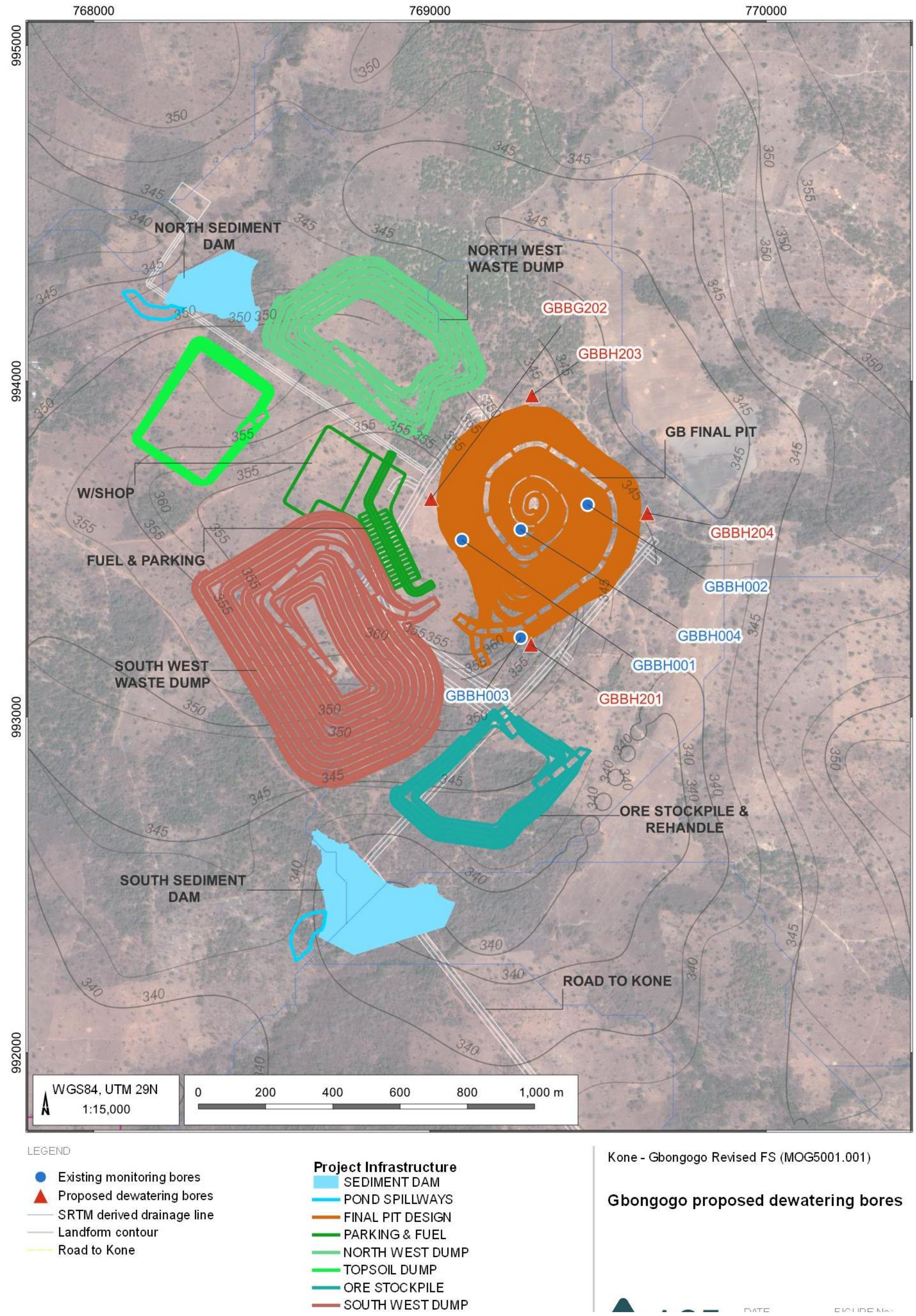
Dewatering boreholes will also be established around both the Koné and Gbongogo Pits limits to be used before and during operations (Figure 16.3.1 and Figure 16.3.2). Nine dewatering bores will be operated during different stages for the Koné pit and will extract between 1,700 and 3,200 m^3/day. Similarly, four dewatering bores will be used at the Gbongogo Pit to extract in the order of 1,200 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 16.3.1 Koné Dewatering Borehole Plan



Source: AGE, January 2024.

Figure 16.3.2 Gbongogo Dewatering Borehole Plan



Source: AGE, January 2024.

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 has been obtained through a combination of a 39 Mtpa and 15.5 Mtpa mining rate at Koné and Gbongogo respectively with an 11 Mtpa mill throughput.

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 16.4.1 shows the mining schedule while Table 16.4.2 shows the processing schedule.

Table 16.4.1 Koné Mining Schedule

Description	Unit	LOM Total	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9
Mining												
GB Tonnes	Mt	10.7	1.8	2.4	2.7	3.7						
GB Grade	g/t Au	1.43	1.47	1.46	1.28	1.52						
North Pit Tonnes	Mt	1.8					0.2	0.4	0.6	0.6		
North Pit Grade	g/t Au	0.47					0.52	0.47	0.44	0.48		
South Pit Tonnes	Mt	161.9	6.9	15.0	17.4	20.0	18.2	14.2	16.3	20.6	29.6	3.6
South Pit Grade	g/t Au	0.67	0.67	0.63	0.65	0.69	0.68	0.61	0.61	0.66	0.73	0.82
Total Tonnes	Mt	174.3	8.7	17.5	20.1	23.8	18.4	14.6	16.8	21.2	29.6	3.6
Total Grade	g/t Au	0.72	0.84	0.75	0.74	0.82	0.68	0.60	0.60	0.65	0.73	0.82
Total Contained	'000 oz	4,011										
GB Waste Tonnes	Mt	40.3	2.9	13.1	12.8	11.6						
North Pit Waste Tonnes	Mt	2.1					0.5	0.6	0.4	0.6		
South Pit Waste Tonnes	Mt	162.8	6.4	23.3	21.8	19.0	19.8	23.8	21.8	17.2	9.4	0.5
Total Waste Tonnes	Mt	205.3	9.3	36.4	34.5	30.6	20.3	24.4	22.2	17.8	9.4	0.5
Strip Ratio	W:O	1.18	1.06	2.08	1.72	1.29	1.10	1.66	1.32	0.84	0.32	0.13

Table 16.4.2 Koné Processing Schedule

Description	Unit	LOM Total	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	Yr 15	Yr 16
Processing																			
Stockpile Rehandle	Mt	106.4		3.3	2.9	1.1	4.4	3.1	1.1	0.9	2.8	10.0	11.0	11.0	11.0	11.0	11.0	11.0	10.9
Oxide Tonnes	Mt	11.3		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	0.7	0.7
Oxide Grade	g/t Au	0.63		1.33	1.17	0.87	0.96	0.96	0.78	0.57	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39
Transition Tonnes	Mt	7.9		2.8	0.3	0.0			0.2	0.0			4.4						0.3
Transition Grade	g/t Au	0.63		1.01	0.57	0.45			0.47	0.61			0.41						0.39
Fresh Tonnes	Mt	139.4		5.8	9.7	10.1	9.6	7.5	8.1	9.5	10.2	5.5	2.1	10.3	10.3	10.3	10.3	10.3	9.9
Fresh Grade	g/t Au	0.74		1.26	1.05	1.18	0.94	0.76	0.74	0.88	1.10	0.93	0.46	0.46	0.46	0.46	0.46	0.46	0.46
FW Fresh Tonnes	Mt	15.8		0.3	0.4	0.2	0.7	2.8	2.0	0.7	0.0	4.8	3.8						0.0
FW Fresh Grade	g/t Au	0.58		1.04	0.82	0.90	0.89	0.66	0.64	0.67	0.86	0.48	0.45						0.28
Total Processed Time	Mt	174.3		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	11.0	10.9
Total Processed Grade	g/t Au	0.72		1.18	1.03	1.15	0.94	0.75	0.72	0.85	1.05	0.70	0.43	0.45	0.45	0.45	0.45	0.45	0.45
Total Process Recoveries	%	89.0%		89.8%	89.0%	89.3%	90.6%	89.8%	89.7%	90.5%	91.5%	89.8%	88.6%	86.1%	86.1%	86.1%	86.1%	86.1%	85.8%
Total Recovered	000 oz	3,570		323	326	364	300	237	229	271	340	222	135	138	138	138	138	138	136

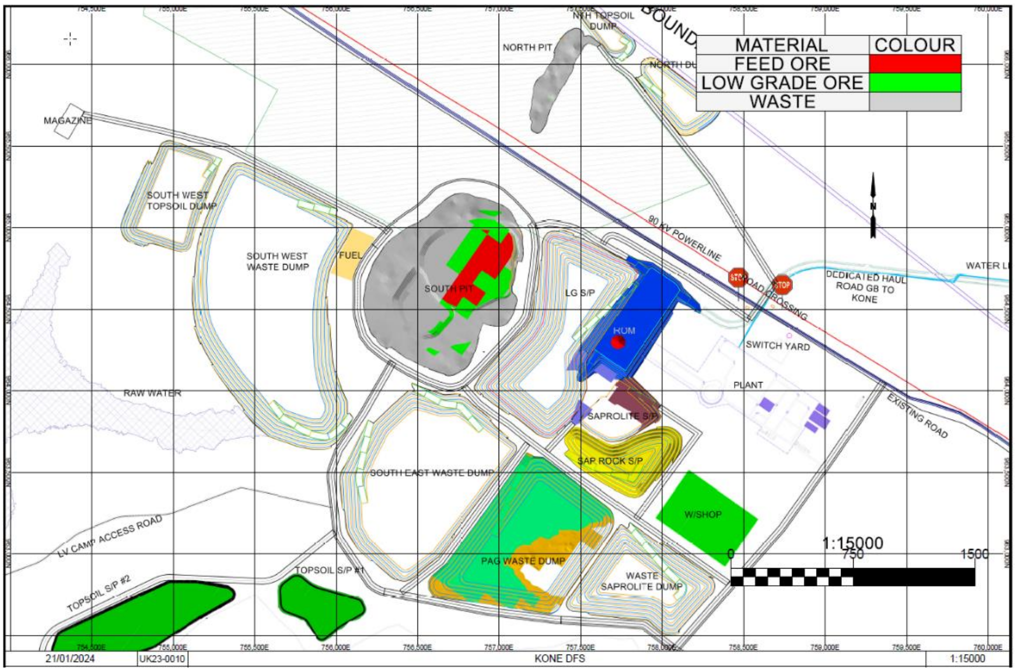
The Koné Southern Pit pre-strip year starts 12 months prior to processing and waste is removed from the starter pit and Cutback 1. This allows for sufficient waste to be mined for the initial tailing storage facility civil requirements and for the construction of the ROM pad and skyway. The skyway is 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 nine years of mining is via direct tip from the mine haul trucks. Roads and infrastructure pads for the workshop, plant and fuel farm are also constructed during this period.

As a majority of the saprock and saprolite material is mined within the first few years, this accumulates on the oxide 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 4.5% of saprolite feed and 2.1% of saprock feed. These stockpiles are built within the first few years and reclaimed over eight years following the completion of the South Pit mining.

At Koné, there are two pits mined during the pre-strip year; a waste starter pit in a Non-Potentially Acid Generating (NPAG) waste rock area. This pit has 9 Mt of waste rock which is used for the construction projects. The Cutback 1 is the optimised ore pit for producing the highest grade ore at the start of production processing in Year 1. The production rate for the starter pit and cutback one in the initial pre-strip year is 13.6 Mt over 12 months.

Figure 16.4.10 shows the Koné pit surfaces at the end of the initial pre-strip year (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 (>0.65 g/t Au) or feed ore, green represents the low grade stockpile ore (<0.65 g/t Au) while the grey colour is waste.

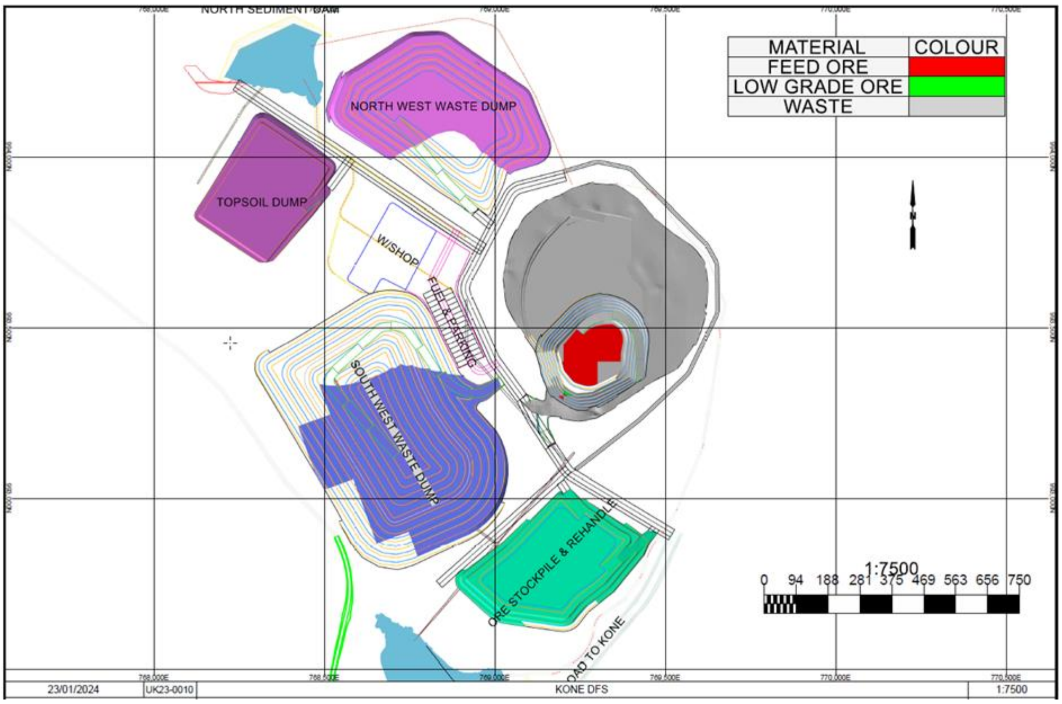
Figure 16.4.1 Koné Starter Pit Surfaces Prior to Mining Pre-Strip (Year -1)



Source: Carci, January 2024.

The Gbongogo Pit (Figure 16.4.2) commences pre-strip in 2Q of Year -1. The initial production of 4.9 Mt requires one excavator and two trucks. There is sufficient stockpile capacity to stockpile the 1.79 Mt of ore at 1.47 g/t within the Gbongogo ore stockpile. The haulage of ore from Gbongogo to the Koné plant commences in Year 1 at a rate of 3.2 Mt, increasing to a maximum rate of 3.5 Mtpa from Year 2.

Figure 16.4.2 Gbongogo Starter Pit Surfaces Prior to Mining Pre-Strip (Year -1)



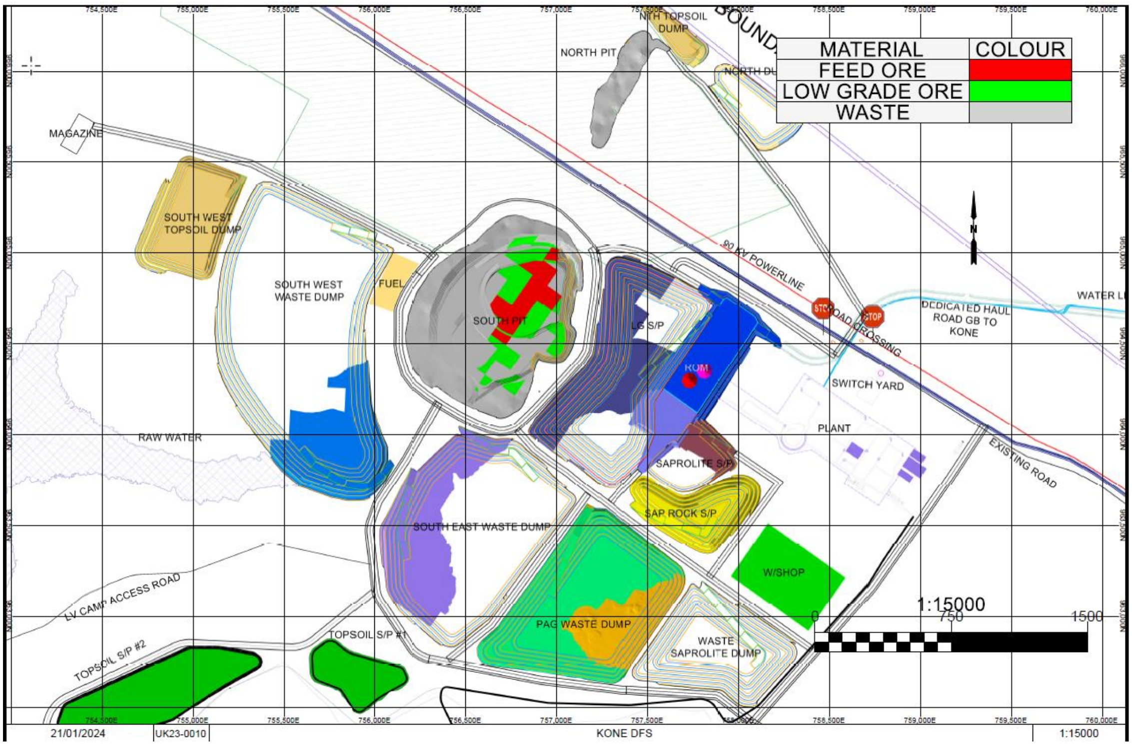
Source: Carci, January 2024.

Year 1 is the first year at the full production rate of 39 Mt in the Koné pit and requires four excavators and a fleet of 19 trucks (Figure 16.4.3). The Gbongogo Pit is mined at a rate of 15.6 Mt in Year 1, of which 2.4 Mt is high grade rom feed average grade of 1.46 g/t. Gbongogo requires two excavators and seven trucks in Year 1 (Figure 16.4.4).

The processing plant is commissioned at the start of the Year 1 with a processing rate of 9.9 Mt, which incorporates an incremental ramp up during commissioning. 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.

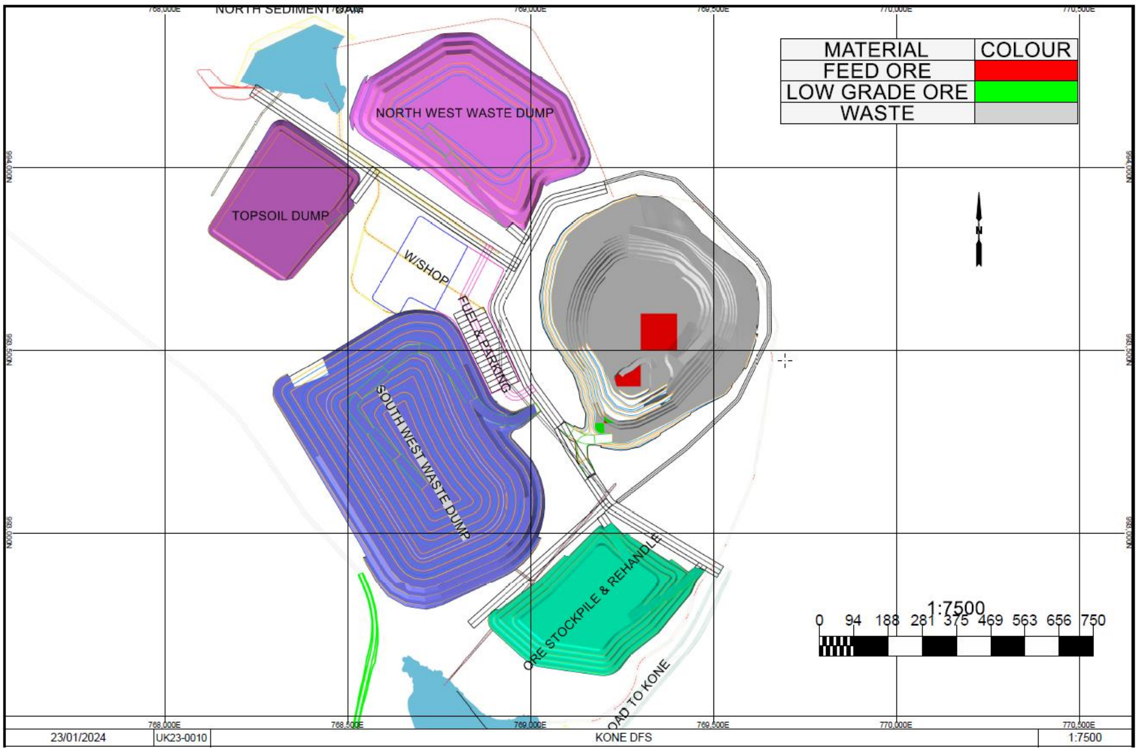
The total ore mined during this period is 21.7 Mt, with 9.8 Mt of high grade ore at average grade of 0.95 g/t with the remaining 11.9 Mt being low grade ore with an average grade of 0.44 g/t.

Figure 16.4.3 Koné Pit Surface Year 1



Source: Carci, January 2024.

Figure 16.4.4 Gbongogo Pit Surface Year 1

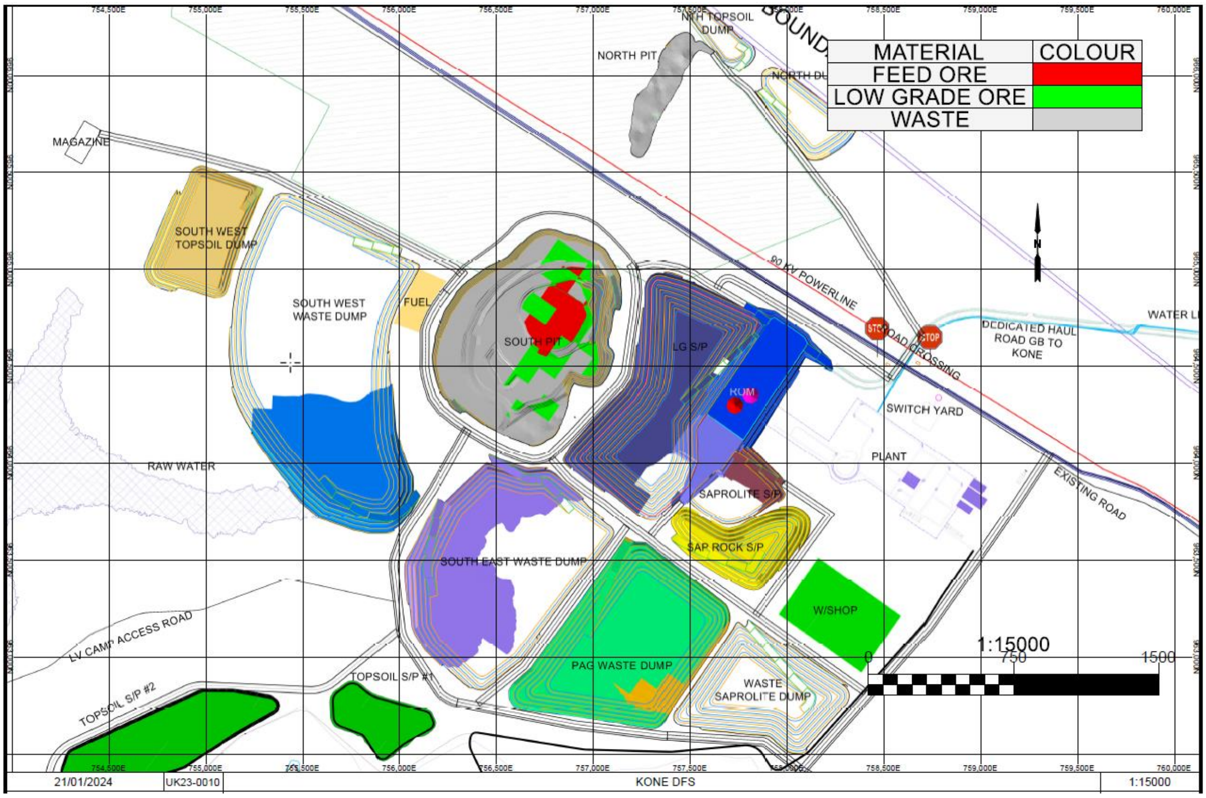


Source: Carci, January 2024.

During Year 2, material is excavated from the Koné southern Cutbacks 1 and 2 with 4.8 Mt of waste pre-strip in the final southern cutback (Figure 16.4.5). The ore is mined from Cutback 1 and Cutback 2, producing a total of 18.7 Mt with a average grade of 0.68 g/t Au. The high grade component of this is 8.2 Mt at 0.95 g/t Au. The remaining 10.5 Mt ore is low grade with an average grade of 0.46 g/t Au and is sent to the low grade stockpile. The total amount of material mined for the year is 38.6 Mt. The Southwest and Southeast waste rock dumps are partially active with approximately 40% of the surface area being used.

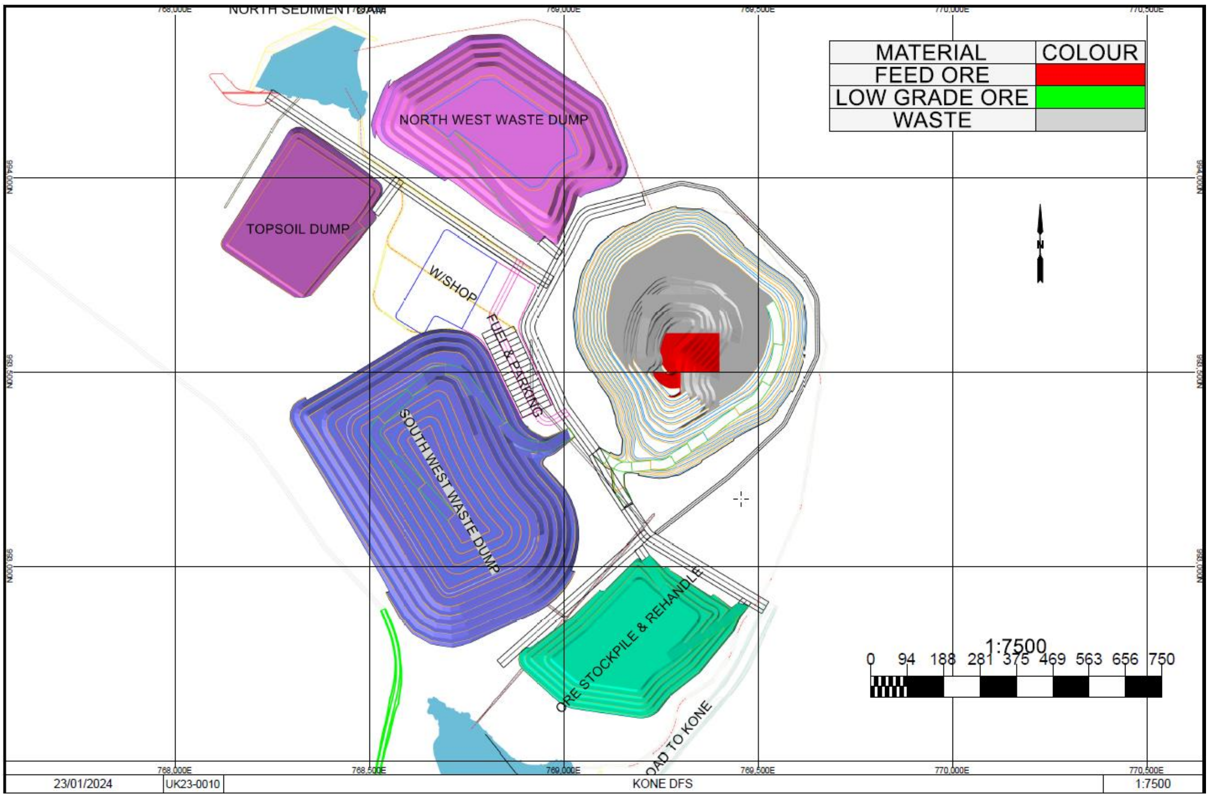
During Year 2, Gbongogo has mining from Cutback 2 and Cutback 3, producing a total of 2.69 Mt of ore with an average grade of 1.28 g/t. The waste total is 12.8 Mt giving an overall strip ratio of 4.73 (Figure 16.4.6).

Figure 16.4.5 Koné Pit Surface Year 2



Source: Carci, January 2024.

Figure 16.4.6 Gbongogo Pit Surface Year 2

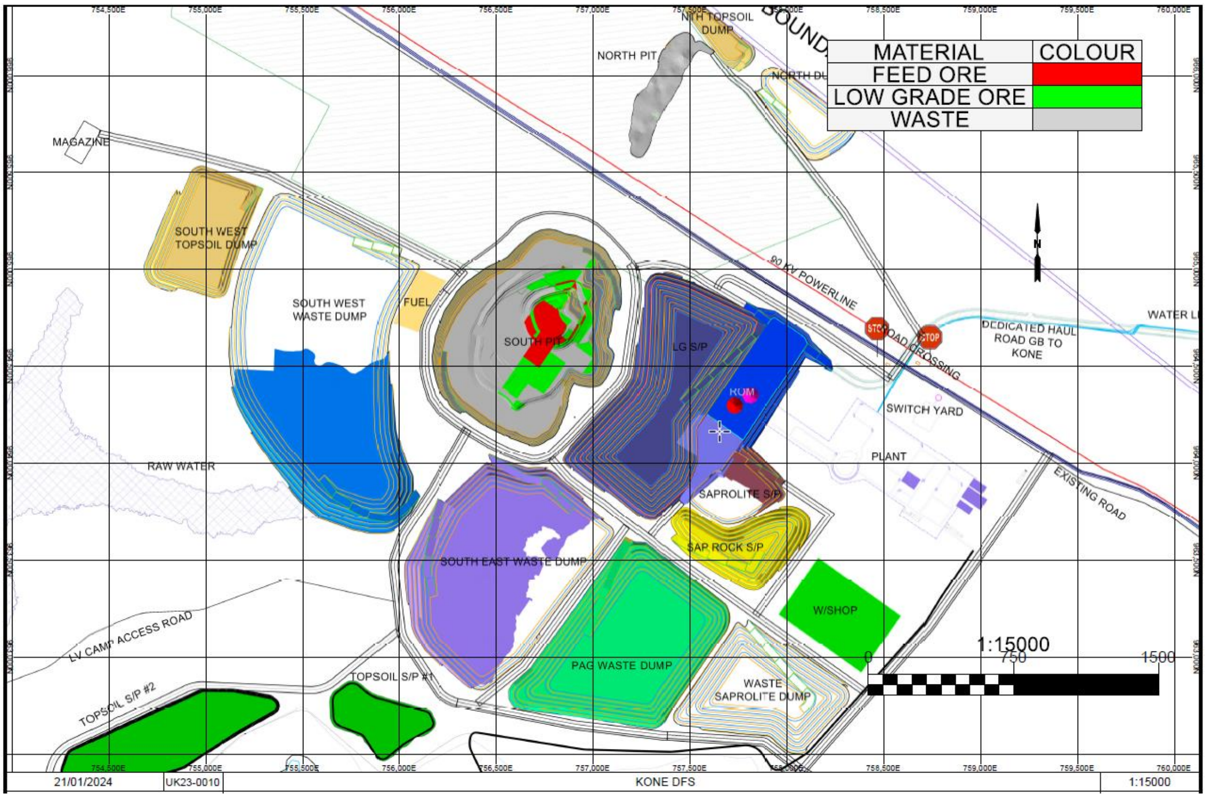


Source: Carci, January 2024.

During Year 3, a majority of the Koné ore is mined from Cutback 1 and Cutback 2 (Figure 16.4.7) and waste stripping in the final cutback, with 9.8 Mt of high-grade ore at an average grade of 1.00 g/t, supplemented with 10.9 Mt of low-grade ore with an average grade of 0.44 g/t being delivered to the processing plant and stockpiles. The Southwest and Southeast waste dumps are active.

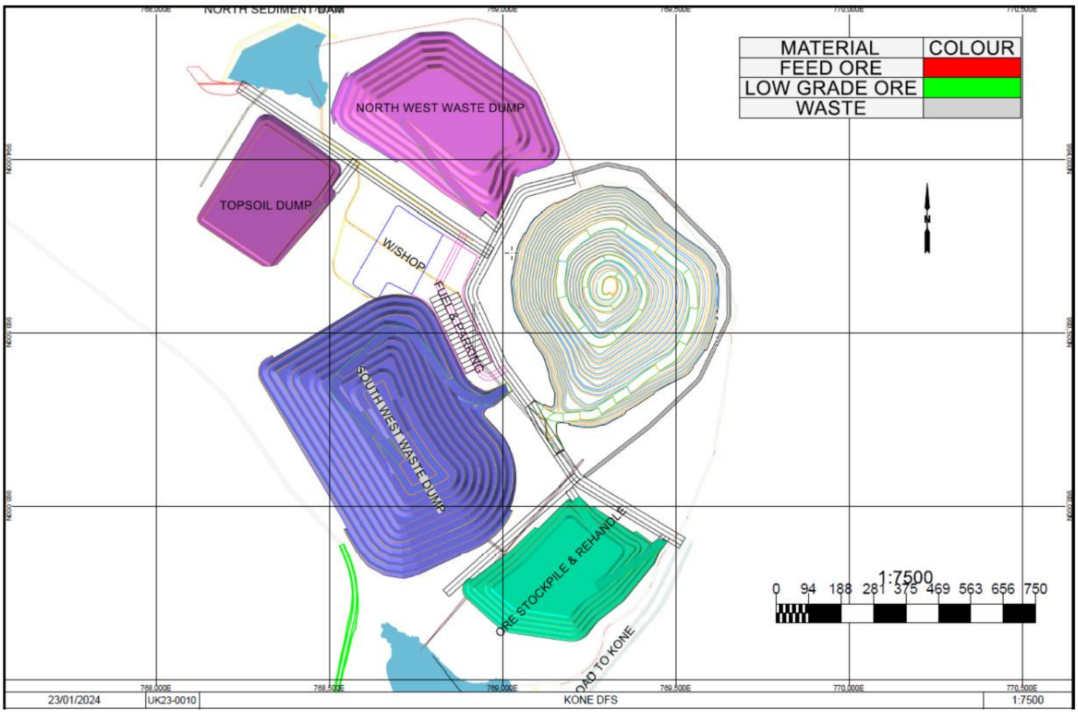
Year 3 is the final year for mining in Gbongogo (Figure 16.4.8) with mining only from Cutback 3, 3.77 Mt ore with an average grade of 1.51 g/t are mined, the remaining 11.5 Mt of material mined during the third year is waste.

Figure 16.4.7 Koné Pit Surface Year 3



Source: Carci, January 2024.

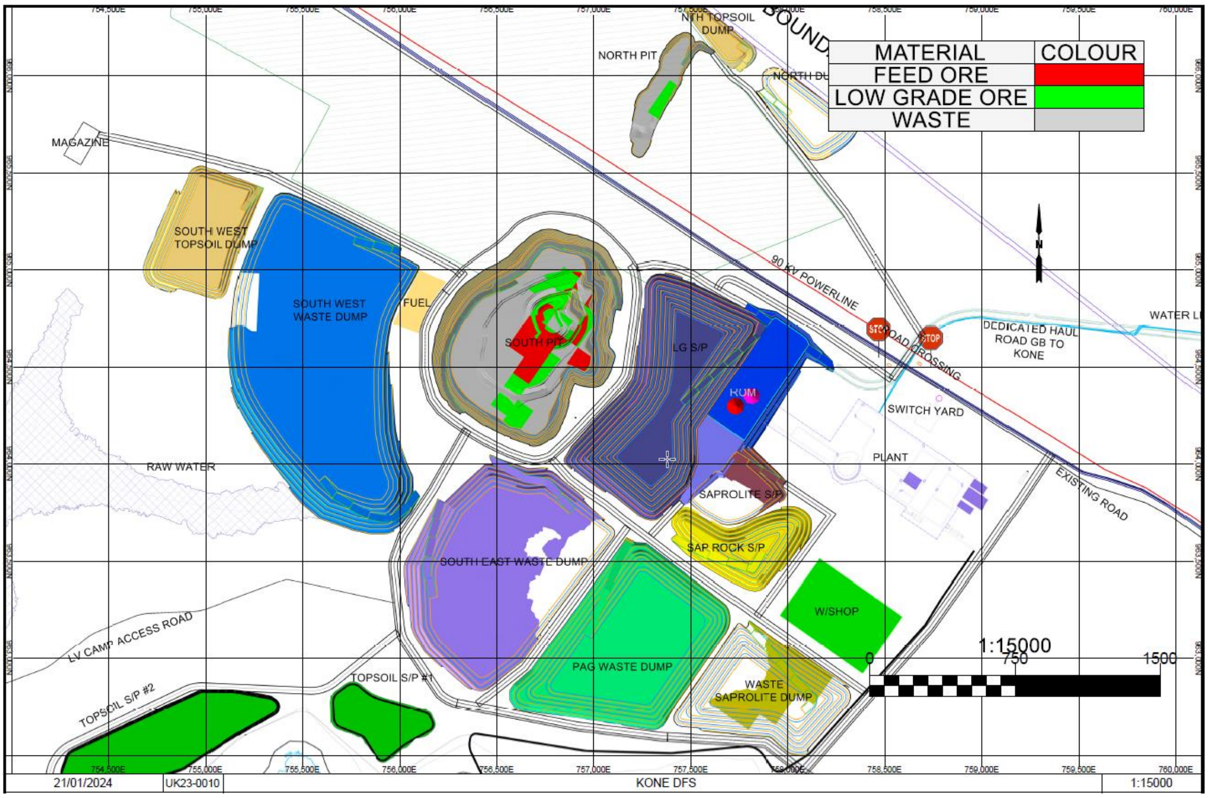
Figure 16.4.8 Gbongogo Pit Surface Year 3



Source: Carci, January 2024.

During Year 4, Koné material is mined from all cutbacks. The mining of the North Pit commences in Year 4 on a 3-month night shift campaign during the dry season with a total of 0.8 Mt moved. The topsoil will also need to be stripped from the North Pit and dump and haul road locations. The ore within this pit is mostly low grade with 0.4 Mt at an average head grade of 0.46 g/t being mined. The South Pit has high grade tonnes being mined from Cutback 1, Cutback 2 and Cutback 4, generating 4.3 Mt of high grade ore at an average grade of 0.83 g/t, which will be supplemented with some of the 8.5 Mt of low grade feed with an average grade of 0.46 g/t. The remaining low grade material is stockpiled (Figure 16.4.9).

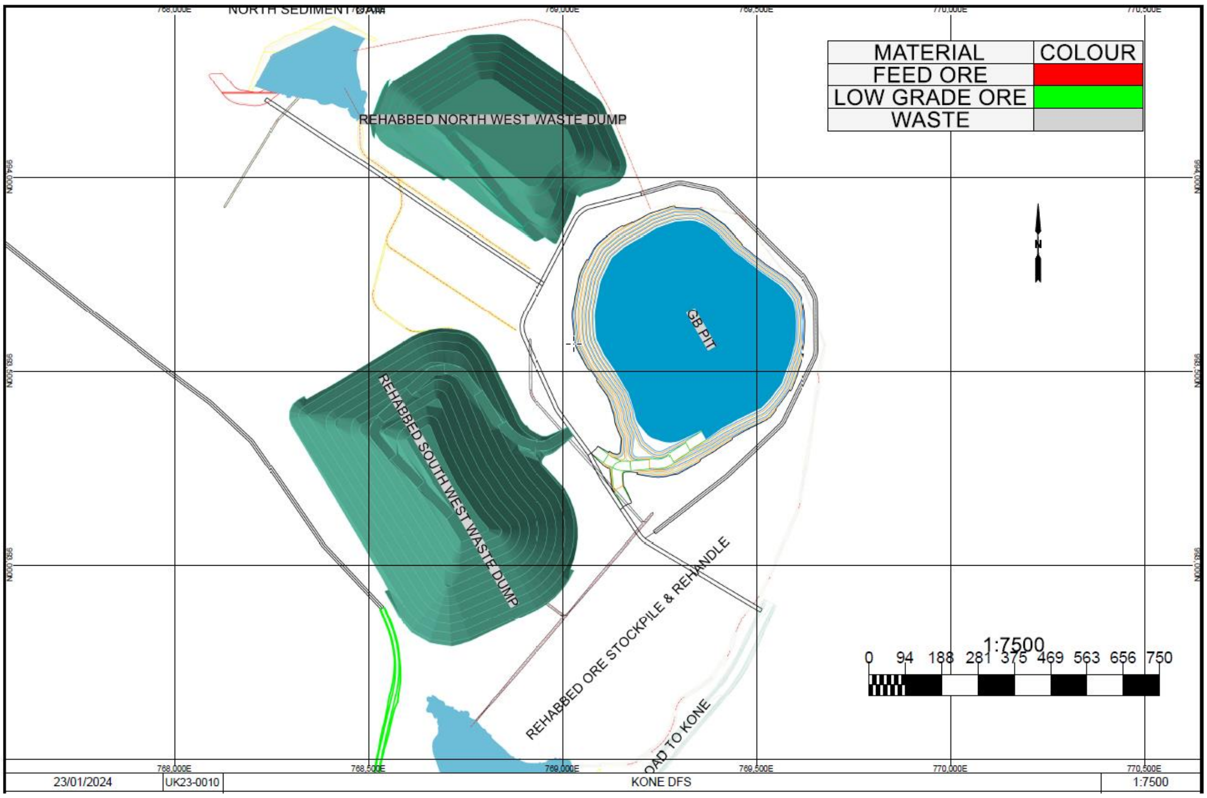
Figure 16.4.9 Koné Pit Surface Year 4



Source: Carci, January 2024.

During Year 4, Gbongogo is rehabilitated (Figure 16.4.10), the dumps are reprofiled and then topsoil is replaced. The stockpile area is cleared and rehabilitated as well as areas used for workshops and truck parking. The pit will be bunded and allowed to fill with water.

Figure 16.4.10 Gbongogo Pit Rehabilitated Surface Year 4



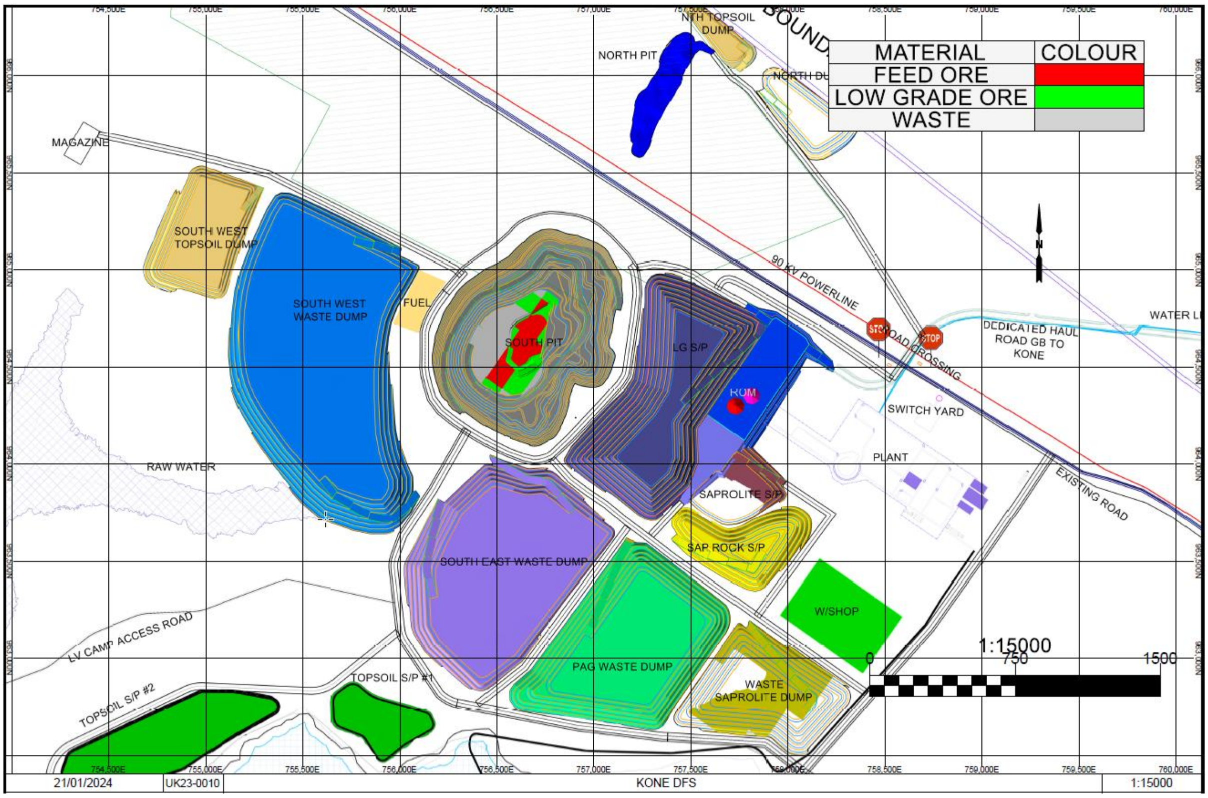
Source: Carci, January 2024.

All Koné cutbacks are active during Year 5, with the high grade plant feed coming predominantly from Cutback 1 and Cutback 2. Cutback 1 is completed in the early part of the fifth year. The final pit cutback is mostly lower grade material and waste. 14.1 Mt ore is mined during this period with an average grade of 0.61 g/t Au. Waste material is placed on both the Southwest and Southeast waste dumps depending upon the closest available waste dump space. 1 Mt is mined from the North Pit during a three-month night shift campaign. 0.39 Mt is stockpiled with an average gold grade of 0.47 g/t.

During Year 6, the North Pit is being mined and is again mined over a three-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. 17.9 Mt ore are mined during Year 6 with an average grade of 0.61 g/t Au. 11 Mt ore are direct plant feed with the remaining 6.9 Mt going to the low grade stockpile.

By Year 7, mining in the North Pit is in its final year 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 placed by dozer if required. Year 7 has 9.3 Mt of high grade ore at 0.96 g/t mined from Cutback 2 and the final pit, supplemented by 11.5 Mt of low grade ore at 0.46 g/t Au. During this period, the plant will have to process 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 16.4.11).

Figure 16.4.11 Koné Mine Pit Surface Year 7

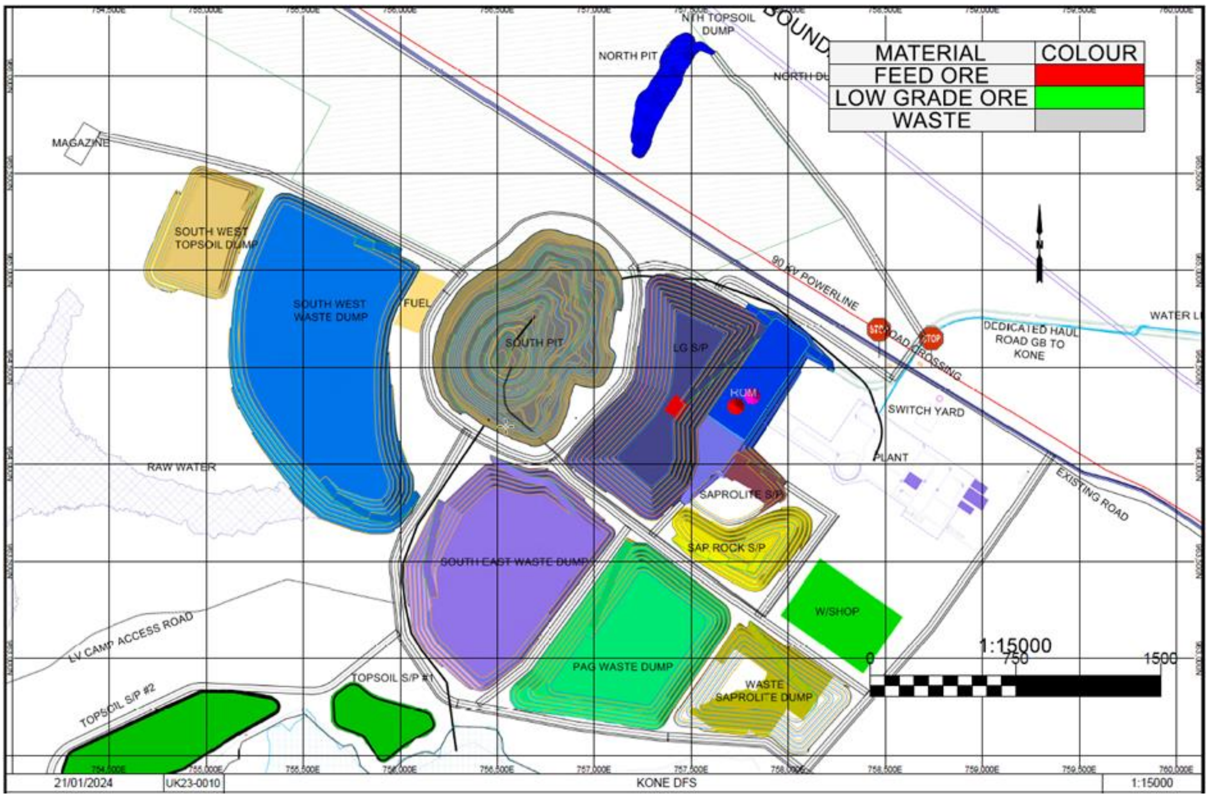


Source: Carci, January 2024.

During Year 8 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. The remainder of the backfill for the North Pit comes from the North Dump. The material mined from the pit in Year 8 is all fresh hard rock material with an average grade of 0.71 g/t Au. The total ore tonnes mined are 27.7 Mt with 12.3 Mt of that classified as high grade ore at 1.01 g/t Au and the remaining 15.4 Mt as low grade ore at 0.46 g/t Au.

Mining operations are completed early in Year 9 (Figure 16.4.12). After the mining has completed, the fleet is used to backfill the North Pit with material from the Northern Waste Dump and to 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 a total sum of 3.6 Mt ore at an average grade of 0.82 g/t Au.

Figure 16.4.12 Koné Mine Pit Surface Year 9

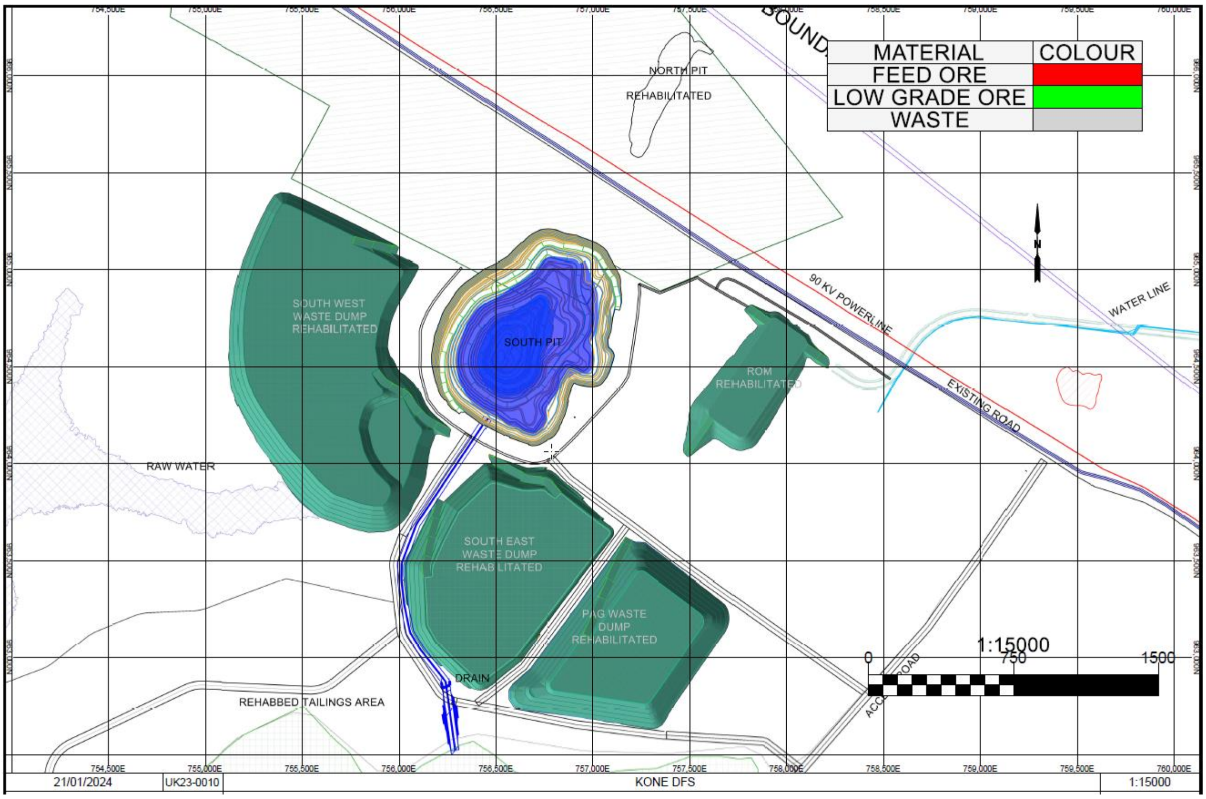


Source: Carci, January 2024.

From mid-Year 9, the mining from the Koné pits is completed and the processing of the low grade stockpiles continues. The material from the South East Waste Dump will be hauled in Year 10 to commence covering the TSF for rehabilitation. The covering of the TSF will take 24 months. Following the completion of the South Pit, it will be used for tailings deposition, allowing the TSF to dry and rehabilitated while the processing plant is still in operation.

Figure 16.4.13 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°. It is then covered in topsoil and seeded. The plant and other mining infrastructure will be dismantled and removed.

Figure 16.4.13 Koné Rehabilitated Site



Source: Carci, January 2024.

16.5 Mining Risk Assessment

Table 16.5.1 outlines the risk assessment for the operations stage of the Project.

Table 16.5.1 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 (P_{80}) 75 µm.
- Process plant availability of 91.3% supported by the selection of standby equipment in critical areas and reputable western vendor supplied equipment and connection to the National Grid.
- Sufficient automated plant control to minimise 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 P_{80} of approximately 31 mm. 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 wt, 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 P_{80} 75 µm.
- Pre-leach thickening to increase the slurry density feeding the leach and CIP circuit to minimise tankage and reduce overall reagent consumption.
- Leach circuit incorporating 14 leach tanks, arranged in two parallel trains of seven in series, to provide 36 h leach residence time.

- A Kemix Pumpcell CIP circuit consisting of eight CIP tanks for recovery of gold onto carbon, to minimise carbon inventory, gold in circuit and operating costs. The CIP and elution circuit design is based on daily carbon harvesting.

- 20 t 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 TSF.

A simplified overall flow diagram depicting the sequence of the unit operations incorporated in the selected process flowsheet is shown in Figure 17.1.1. The site plan and plant layout are included in Figure 17.1.2 and Figure 17.1.3.

Figure 17.1.1 Overall Schematic Flow Diagram

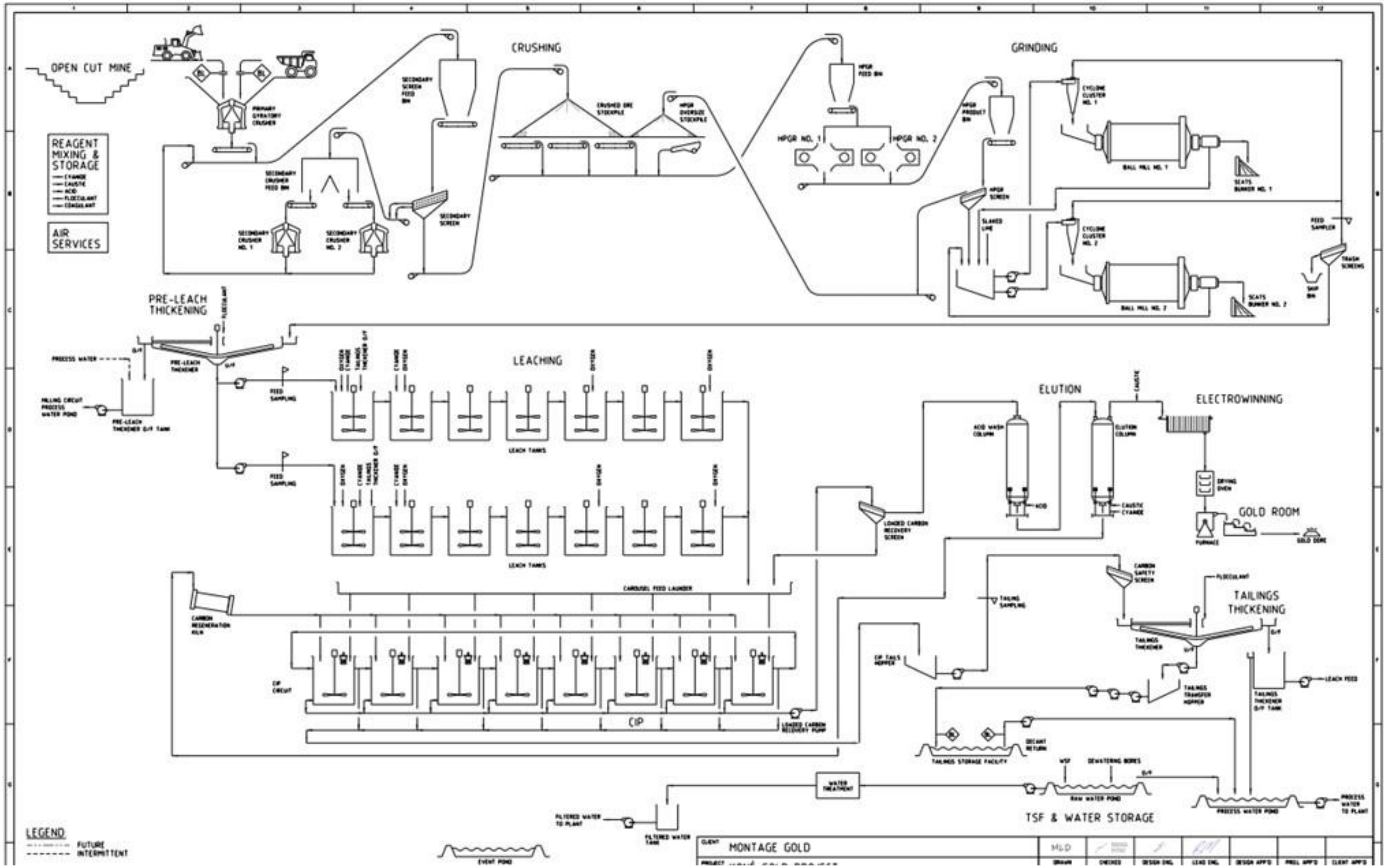


Figure 17.1.2 Site Plan

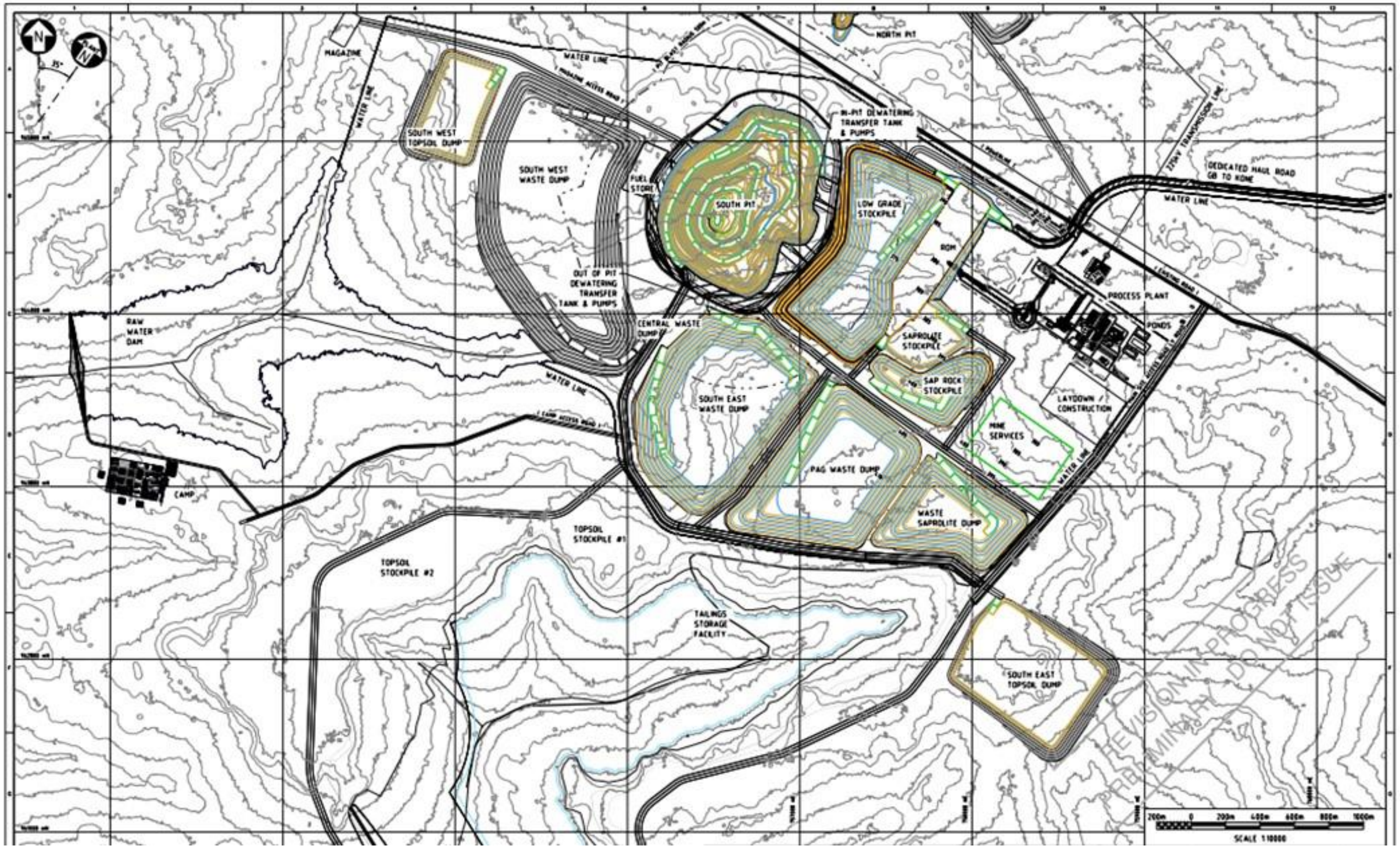
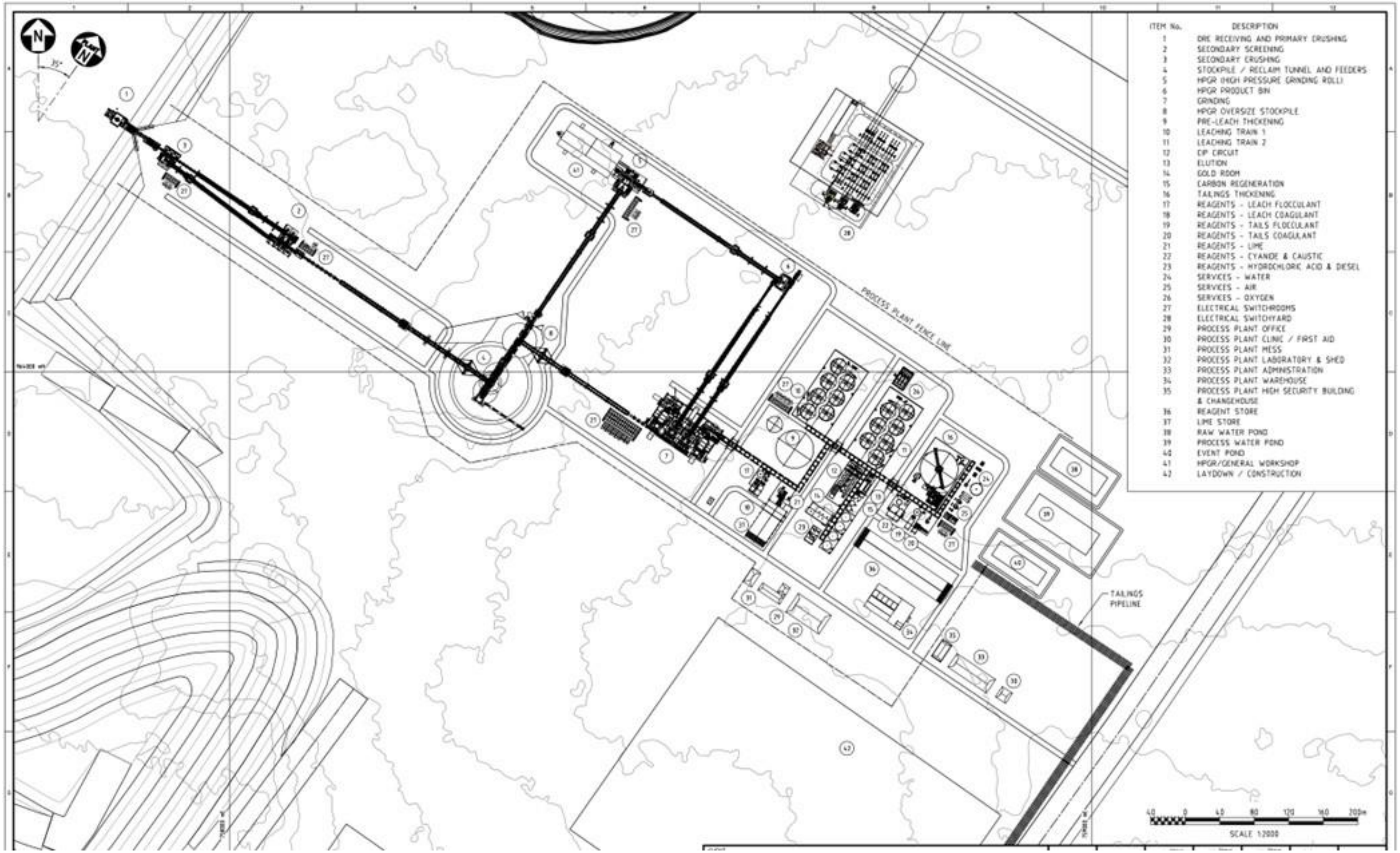


Figure 17.1.3 Plant Layout



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 design head grade of 1.51 g/t Au with design gold recovery 92.1%. 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 (saprock) ore of 7.2% and transition ore of 5.0% 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 75% circuit availability. 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 88% availability. 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 roller assembly 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).

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, HPGR grinding, 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 nominal 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 a vibrating feeder.

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 ball mill size selection has been based upon the current mine plan fresh blend.

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. The wet screens will be fed at 70% solids w/w, reaching less than 50% solids w/w with the inclusion of spray water to ensure high screening efficiency and minimum undersize recycle.

17.2.7 Classification

Cyclones of 500 mm diameter have been selected for the classification duty to provide operational flexibility. Design has been based on up to 262% recirculating load to the ball mills.

The inclusion of a pre-leach thickener will allow operation of the grinding and classification circuit to be optimised by operating cyclones at feed densities that maximise classification efficiency, thereby reducing circulating load and overall circuit power consumption.

17.2.8 Trash Screening

Linear trash screens followed by vibrating dewatering 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 h was selected requiring 14 leach tanks each with a 5,000 m^3 capacity 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 minimise 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 8 h. 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 t 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 goldroom 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 maximise 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 TSF to be recovered for re-use in the leaching circuit.

The tailings thickener underflow will be pumped to the TSF 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 17.3.1 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 OMC, and Lycopodium calculations and modelling and vendor advice.

Table 17.3.1 Key Process Design Criteria

Parameter	Units	Oxide Blend*	Fresh Blend**	Source
Plant Capacity	tpa	11,000,000	11,000,000	Montage
Design Gold Head Grade	g Au/t	1.23	1.51	Montage
Design Gold Recovery	%	92.5	92.1	Testwork
Crushing Circuit Utilization	%	75.0	75.0	OMC
HPGR Circuit Utilisation	%	88.0	88.0	OMC
Grinding Circuit and Plant Utilisation	%	91.3	91.3	Lyco
Crushing Work Index (CWi) 85th %	kWh/t	N.A.	16.9	Testwork
A*b (SMC Test) 15th %		451	34.0	OMC / Testwork
Bond Ball Mill Work Index (BWi) 85th %	kWh/t	4.7	13.1	OMC / Testwork
Abrasion Index (Ai) 15th %		0.241	0.473	OMC / Testwork
Grind Size (P80)	µm	75	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	0.72	0.72	Testwork
Quicklime Addition – Plant	kg/t	2.87	0.50	Testwork
Elution Circuit Type		Split AARL	Split AARL	Lyco
Elution Circuit Size	t	20	20	Lyco
Frequency of Elution	strips / wk	7	7	Lyco

***Oxide Blend is based on up to 10% oxide processed in plant feed. Comminution parameters listed are based on 100% oxide ore**

****Fresh Blend comminution parameters listed are based on the current mine plan fresh blend for ball mill equipment sizing.**

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 minimise 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 75 mm and 40 mm 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 enable operators 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 MP800 or equivalent units operating with a closed side setting of 32 mm. 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 P_{80} of 31 mm, 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 nominal live capacity of approximately 22,000 t, which is 15 h 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 a vibrating feeder 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 two 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 HPGR 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 8 mm 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, each with a cluster of classification hydrocyclones.

Two ball mills will be installed, each with its own dedicated cyclone pack. The ball mills will be 7.0 m x 10.8 m EGL fixed speed mills, each fitted with two 5.0 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 online. 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 h 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.

Onsite 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 tpd 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 online monitoring of the free cyanide concentration will allow the sodium cyanide dose rate to be optimised. 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 Goldroom Operations

The following operations will be carried out in the elution and goldroom 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 goldroom 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 t batch of carbon.

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.

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 5 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.

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 goldroom vent fans will be provided to ensure there is adequate ventilation inside the goldroom.

Goldroom

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 goldroom and electrowinning area will be serviced by a gold trap and dedicated goldroom area sump pump. Any spillage within this area will be pumped back to the leach circuit.

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°C to 750°C and held at this temperature for 15 min 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

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 minimise 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.

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.

Caustic

Caustic (sodium hydroxide) will be delivered to site in 1.2 t 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.

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.

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.

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.

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.

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.

Plant Diesel

Diesel will be delivered to the 17 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.

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.

Reagents Storage

A minimum of 30 days stock of quicklime and 90 days stock of the remaining 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

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 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 WSF by a series of river water extraction transfer and booster pumps, with intermediate break tanks as required. At the WSF, 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 9 h 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.

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 6 h.

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 minimised 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.

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, goldroom, and some hose points.

Waste reject from the water treatment plant will report to the process water pond.

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.

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.

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.

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.

Oxygen

Oxygen gas, for use within the leach circuit, will be supplied by two duty oxygen gas generation plants. A standby plant will also be provided. Oxygen gas will be generated to a supply pressure of 500 kPa(g). Oxygen gas 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 minimise operator intervention in standard start-up functions and to provide key monitoring and control to minimise process excursions and maintain steady operation.

18.0 PROJECT INFRASTRUCTURE

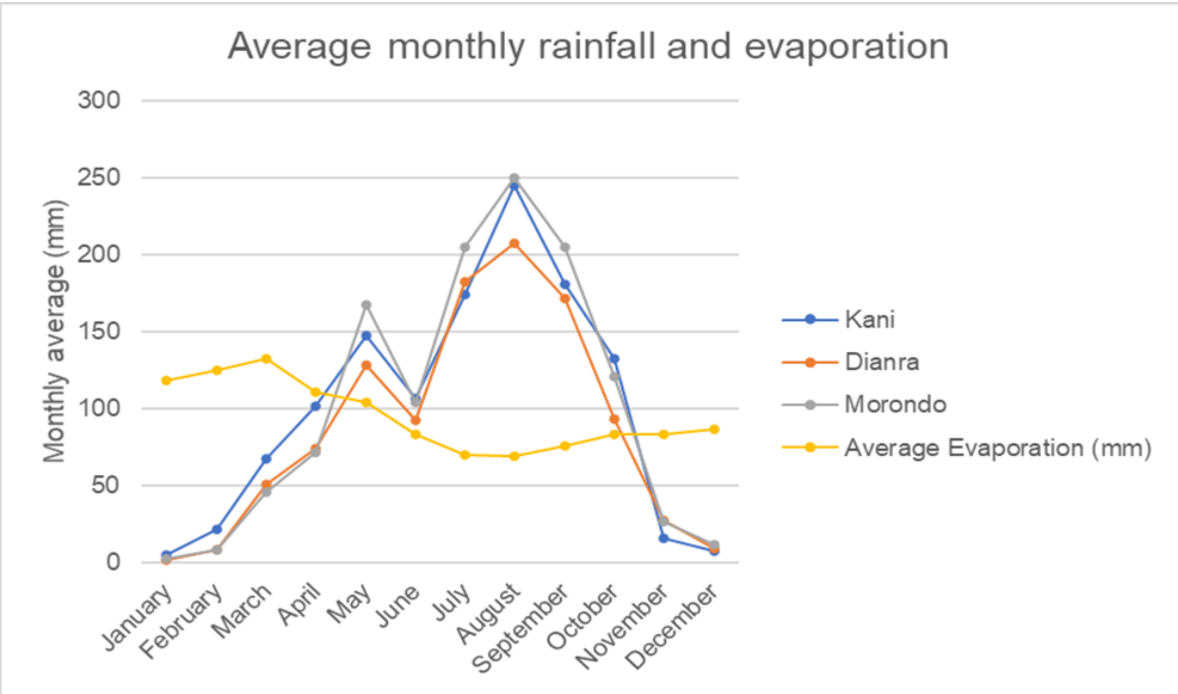
18.1 Water Supply

18.1.1 Preliminary Surface Water Assessment

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 stations located in the Koné region and are depicted in Figure 18.1.1. 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 18.1.1 Monthly Rainfall and Evaporation for Koné



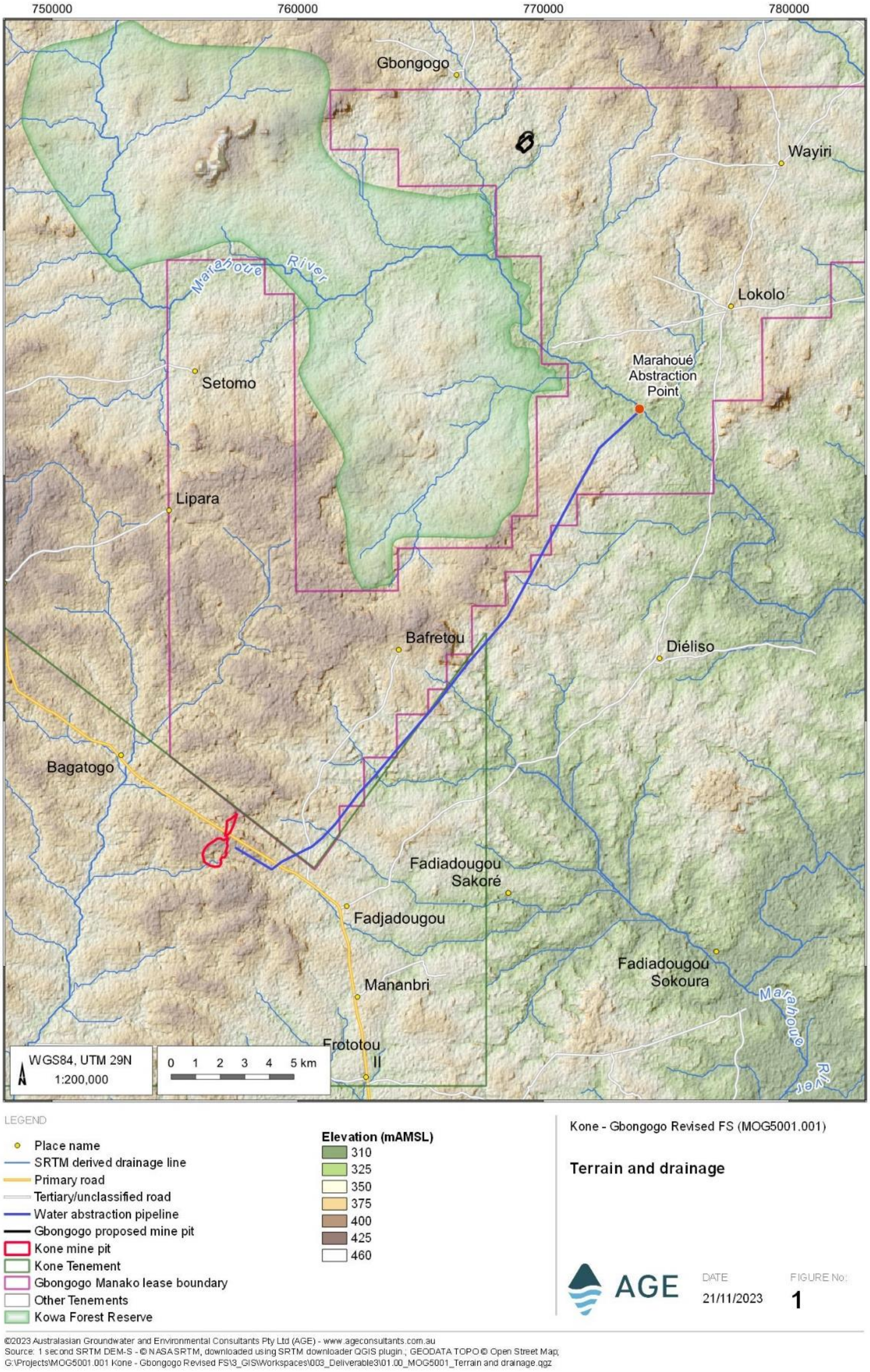
Terrain, Drainage and Land Use

The Project area is characterised by moderate relief between 300 and 400 metres above mean sea level (mAMSL) (Figure 18.1.2). 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 nearby communities of Fadiadougou and Batogo, situated approximately 7 km to the south-east and north-west respectively of the Koné site, are shown in Figure 18.1.2. As shown in Figure 18.1.2, the Gbongogo village is about 4 km northwest of the Gbongogo site area.

Figure 18.1.2 Koné – Gbongogogo Terrain and Drainage



Source: AGE, November 2023

18.1.2 Groundwater Assessment

Dykes and Structural Geology

Multiple sets of dykes are observed through the Koné 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:

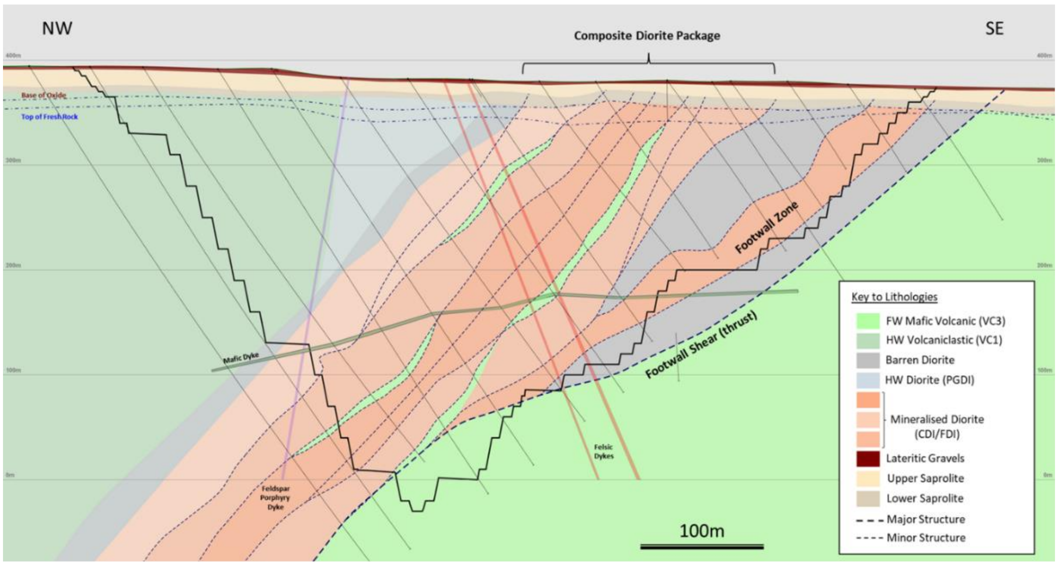
- The first dyke variation displays weak banding and deformation, with residual calcite. Further data needs to be collected to define the orientation, continuity and frequency of these early dykes.

- The second dyke variation is a late sub-horizontal version (Figure 18.1.4) 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 ~015° through the deposit and dipping ~75 to 80°. Two main felsic dykes have been logged in the main pit, with increased frequency seen to the north (Figure 18.1.3 and Figure 18.1.4).

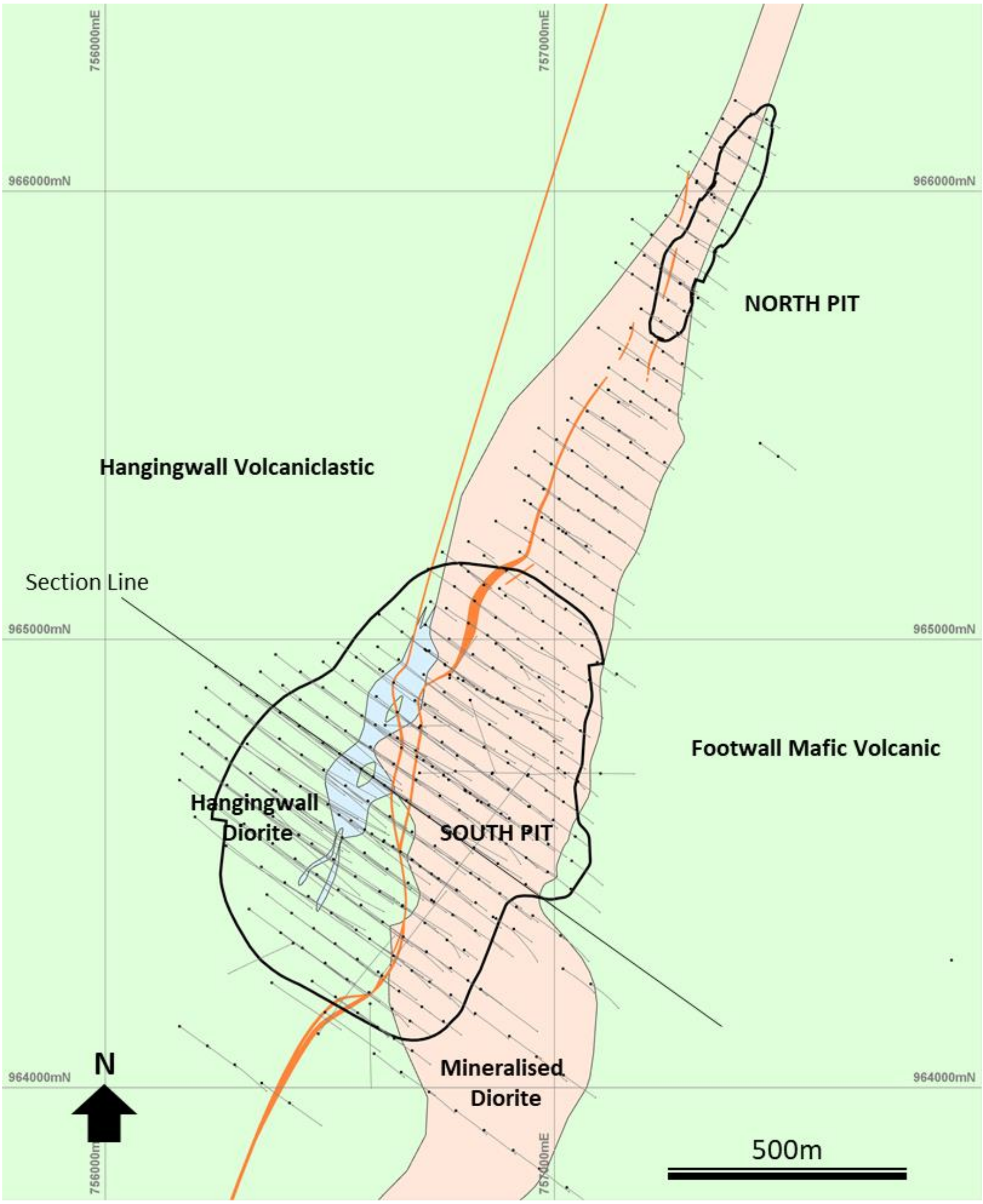
Koné is interpreted to have formed as part of a stacked thrust-shear under a compressional tectonic regime. The major thrust faulting is located at the footwall of the diorite. Very few more significant faults, clays / breccias, are observed within the deposit, with the majority of faulting observed being minor faults displaying both normal and reverse movement.

Figure 18.1.3 Koné Major Geological Units – Section



Source: Montage, 2022.

Figure 18.1.4 Koné Major Geological Units – Plan



Source: Montage, 2022.

Hydrocensus

A hydrocensus was completed in 2020 within a 15 km 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 mbgl to 2 mbgl.
- 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.

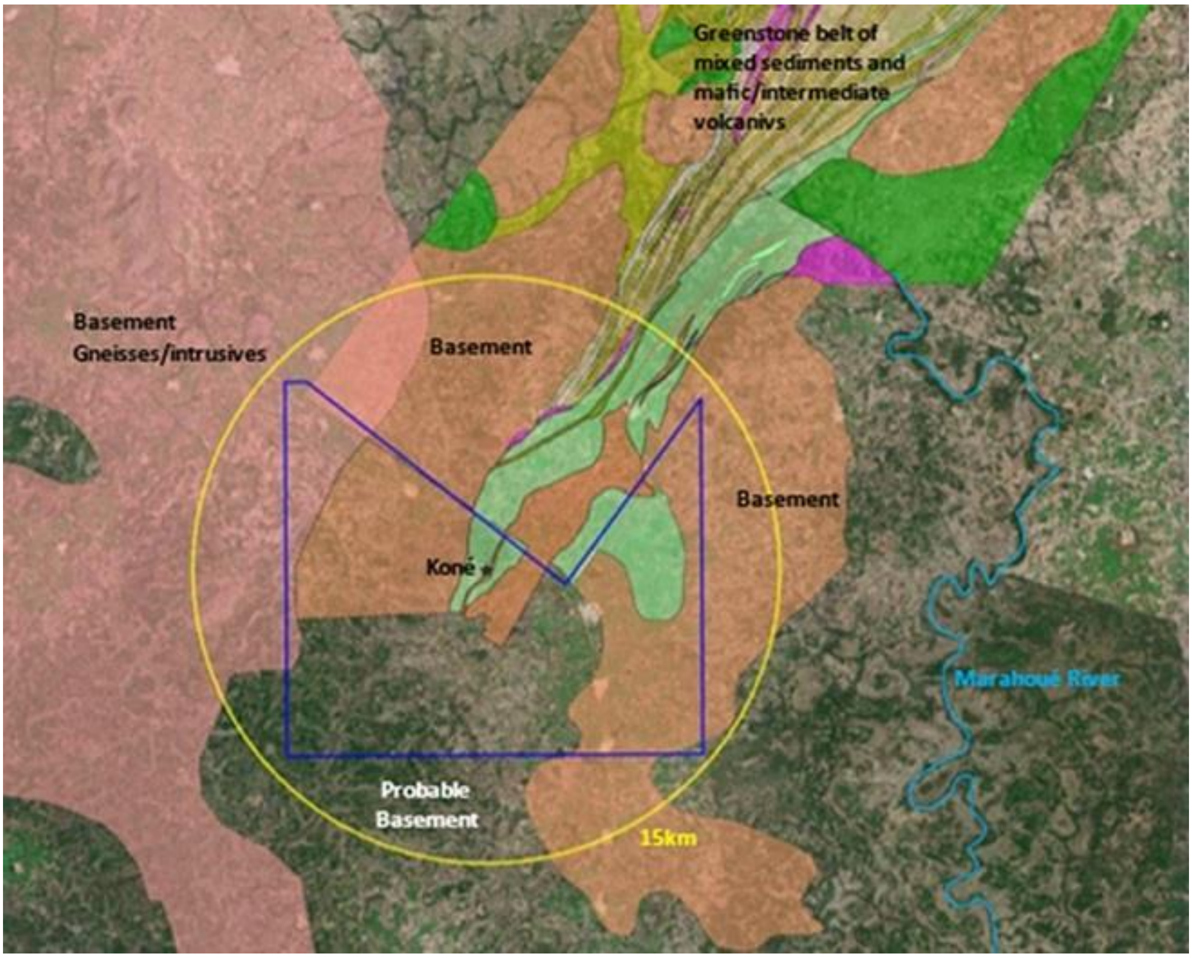
A hydrocensus was completed in the Gbongogo area in early 2023 and data obtained from the three closest villages; Gbongogo (4 to 5 km northwest), Ouyeré (Wayiri, approximately 10 km east), and Lokoloo (approximately 12 km southeast).

Geology and Proposed Mining

The Koné prospect is located 5 to 10 km south of the southern end of the Boundiali Basin, probably on the southern continuation of a merged Fonondara / Gbéou Shear Zone (Figure 18.1.5).

The weathering profile observed at Koné and Gbongogo is transitional, from surface laterite, through saprolite and saprock down to fresh un-weathered material. The upper saprolite or weathering profile is, on average, 20 m thick and the transition zone is approximately 10 m thick. The mineralisation runs almost vertically from south to north and is generally associated with the diorite intrusion and quartzite veins. It is also expected that this narrow N-S zone will have preferential groundwater flow properties, especially where it is consistent with shallow sub-outcrop levels.

Figure 18.1.5 Simplified Geological Setting



Source: Montage, June 2020.

Drilling indicates that the mineralised zones strike NNE and dip approximately 40° towards the west. Drillholes plunge 50° towards an azimuth of 120°, perpendicular to the orientation of mineralised zones in the core of the prospect.

The Koné pit depth will be in the order of 500 m (Figure 18.1.4). The surface area covered by both Koné mine pits is approximately 90 ha. The Gbongogo Pit layout covers an area of approximately 25 ha and will extend to a depth of approximately 200 m.

Both the surface areas at Koné and Gbongogo are 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 18.1.6 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 m and 25 m below surface.

Aquifer Characteristics

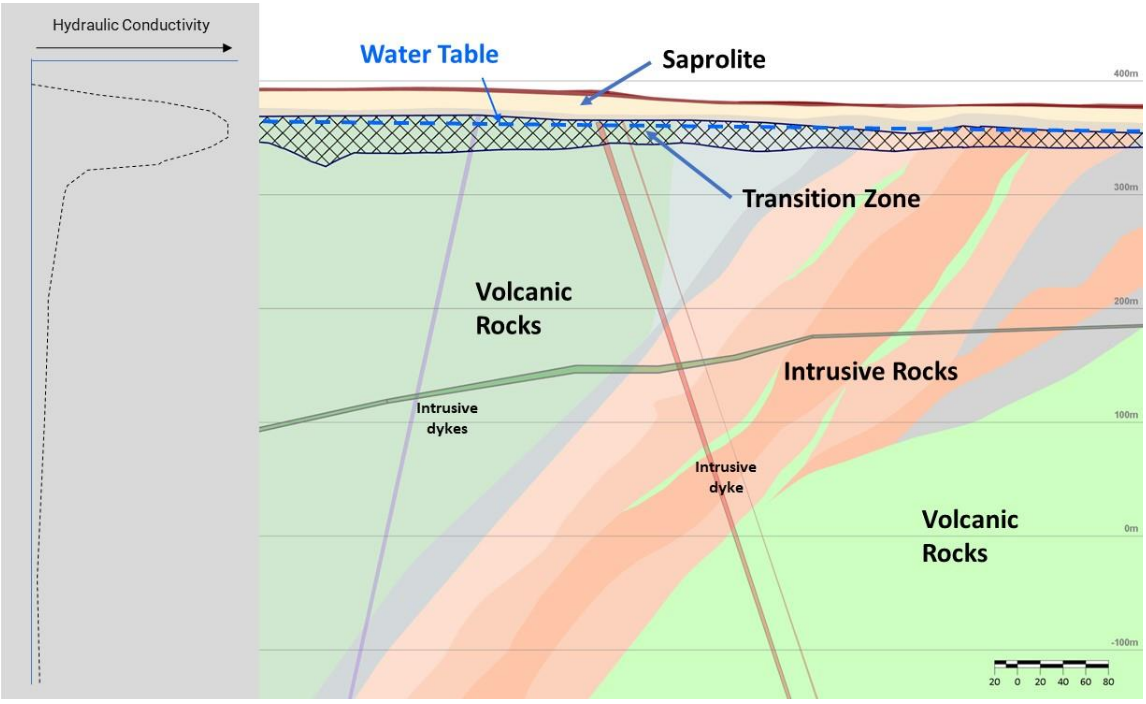
The most widespread aquifer type in Côte d'Ivoire is weathered and fractured basement rocks, which underlie most of the country. Gold projects in the region similar to the KGP reported on predicted groundwater inflows into pits ranging between 25 to 60 L/s.

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 (transition) and deep fractured rock hydrostratigraphic units generally have higher groundwater exploitable potential when comparing to the shallow regolith / saprolite 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.

The aquifer tests, from 15 boreholes at Koné show that hydraulic conductivity peaks at the transition zone and reduces with depth (Figure 18.1.6 and Figure 18.1.7). Generally, bore yields are below 1.5 L/s with only four of the 15 above. The majority of the hydraulic conductivity estimates are below 0.1 m/day (between 0.02 and 0.134) representing the various degrees of fractured diorite. Higher hydraulic conductivity estimates above 0.2 m/day (between 2.08 and 1.34) are evident for four boreholes representing multiple fracture zones associated with dyke contact zones, fractured footwall diorite, and the footwall shear zone and all more or less at the transition zone.

Four hydrogeological test bores were drilled at the Gbongogo Pit area in early 2023. The main water bearing zones were associated with the transition zone (30 to 45 m) and structural geology. Blow yields ranges between 0.3 to 4.1 L/s and hydraulic conductivity estimates between 0.005 and 0.8 m/day.

Figure 18.1.6 Example of Koné Saprolite Thickness and Geozones



Source: Montage 2023.

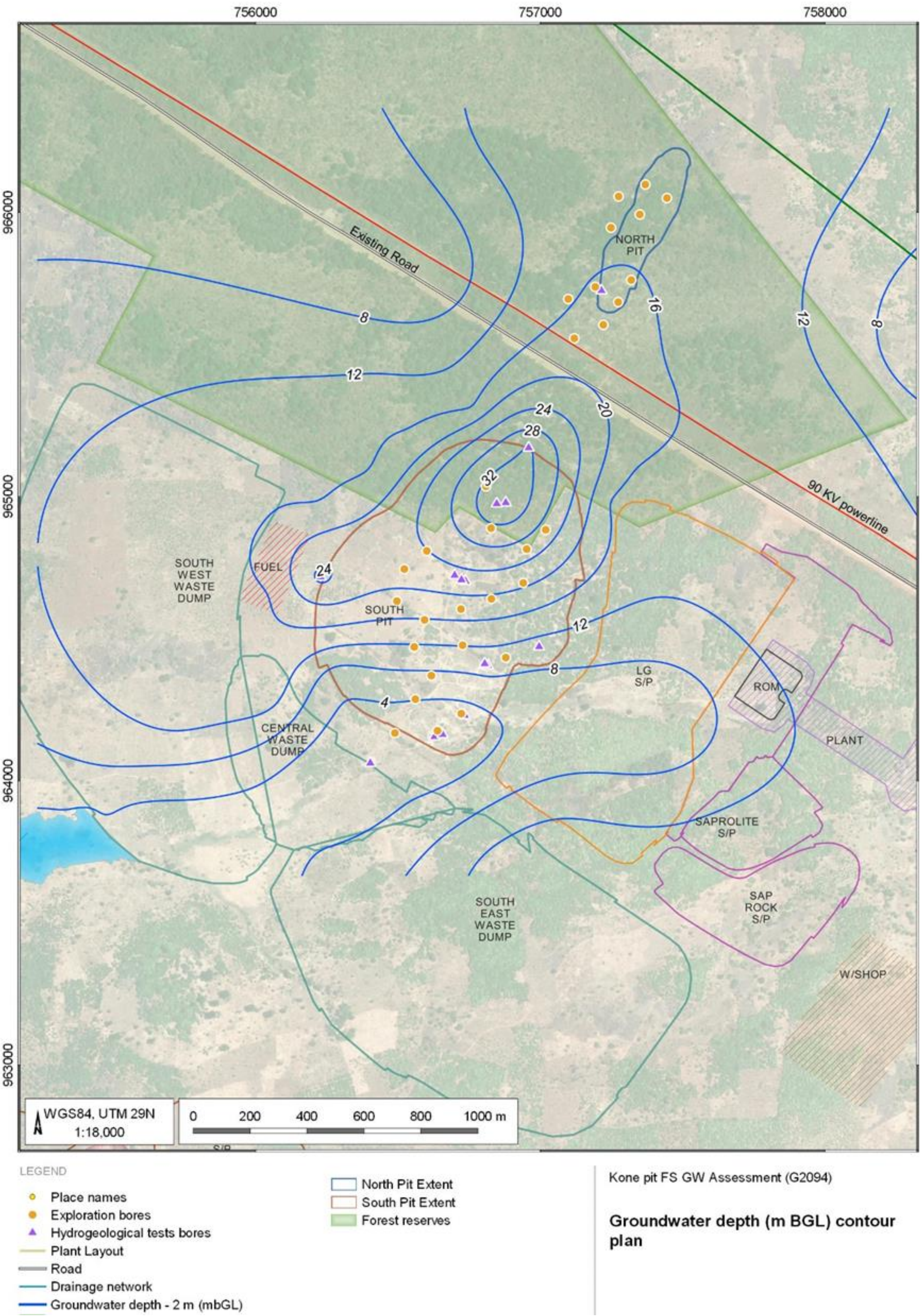
Groundwater Levels

The available data show the transition and deep aquifer zones are confined by the saprolite aquitard. The static groundwater levels at the Koné exploration site 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 18.1.7. 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 18.1.7 Groundwater Depth



Source: AGE, November 2021.

The static groundwater levels for the Gbongogo mine pit indicate that:

- The minimum groundwater level is: 2.1 mbgl.
- The maximum groundwater level is 24.6 mbgl.
- The average groundwater level is 16.75 mbgl or ~338 mAMSL.

18.1.3 Groundwater Quality

The groundwater has been sampled in the period 2020 to 2023 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 18.1.1.

Table 18.1.1 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
Fluroide	mg/L	0.3	0.1	0.8	1.50
Total Disolved Solids	mg/L	250.4	135	429.2	n/s
Aluminium	mg/L	0.3	<0.05	3.41	0.2

18.1.4 Hydrogeological Conceptual Model

The key conceptual information of the study area groundwater system used to construct the numerical groundwater model is summarised 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 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 Koné exploration site is 360 mAMSL 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 groundwater elevation at the Gbongogo area is 338 mAMSL.
- The average depth of weathering is ~30 m, approximately 10 m below the groundwater table. Saturation of the upper saprolite or weathering zone varies between 40 m to 5 m.
- Groundwater was consistently intersected in most of the boreholes at the transition aquifer zone and is on average 35 m to 45 m deep.

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 18.1.8).

18.1.5 Numerical Groundwater Model

Two separate numerical groundwater models were developed out of a conceptual model supported by the field observations, one for the Koné exploration site and one for the Gbongogo site. The main objectives for the Koné exploration site were to simulate regional dewatering impacts, assist with mine dewatering planning and backfilling of mine tailings back into the pit from Year 9. The Gbongogo model predicts regional dewatering impacts and assist with mine dewatering planning.

Numerical modelling has been undertaken using the MODFLOW 6 and MODFLOW USG codes. 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.

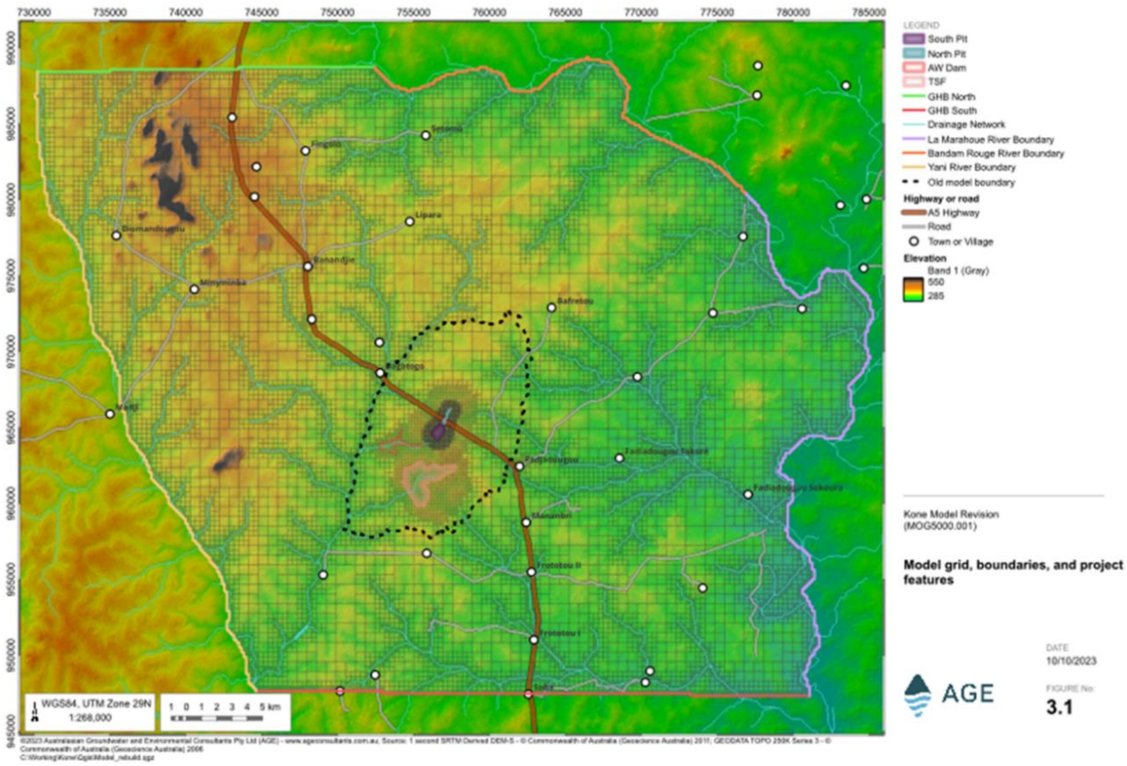
The Koné model was updated in 2023 and depicted in Figure 18.1.8 and Figure 18.1.9 shows the Gbongogo model grid.

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.

Mining will be four years at Gbongogo satellite pit. Following the completion of the mining at Koné in Year 9, tailings will be deposited into the pit. 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.

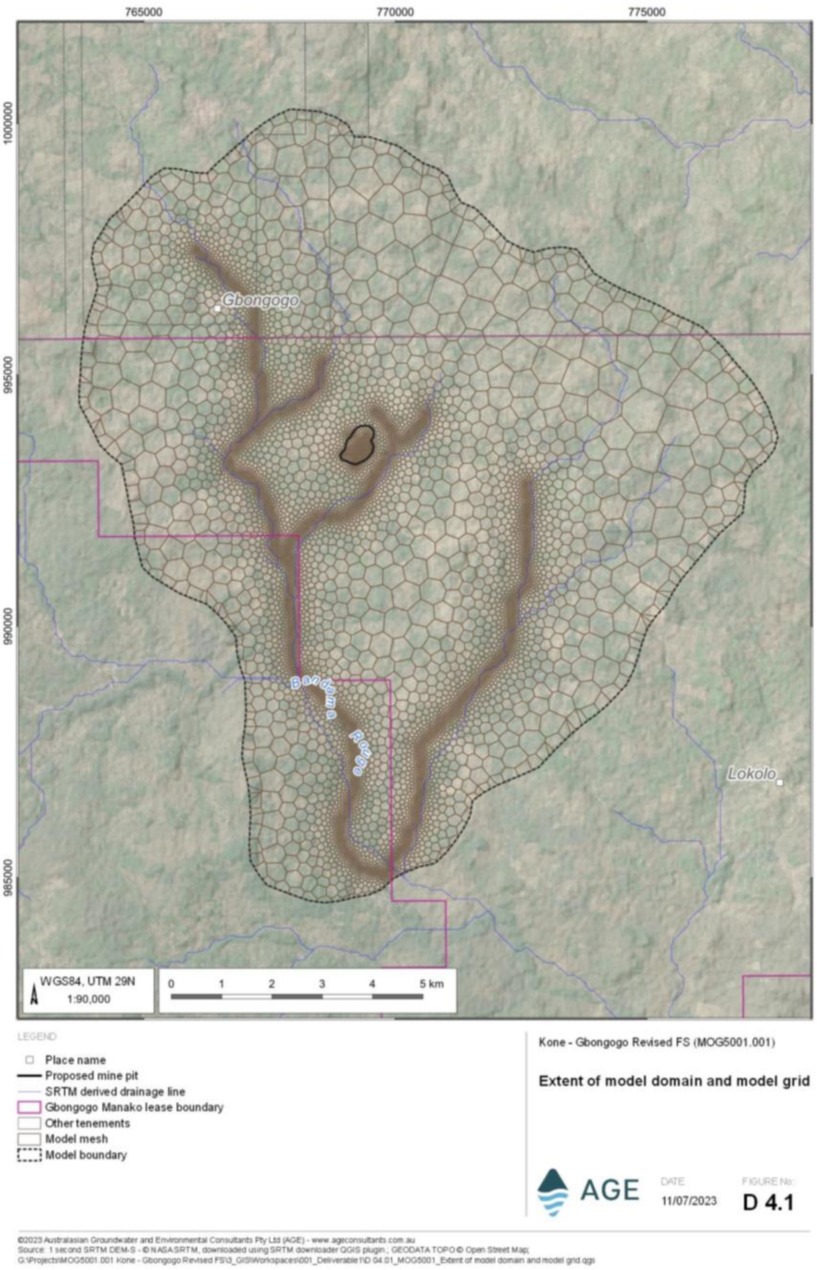
The dominant mechanism for recharge to the groundwater system is diffuse rainfall infiltration through the soil profile and subsequent drainage to underlying groundwater systems.

Figure 18.1.8 Koné Model Grid Structure



Source: AGE, November 2023.

Figure 18.1.9 Gbongogogo Model Grid Structure



Source: AGE, November 2023.

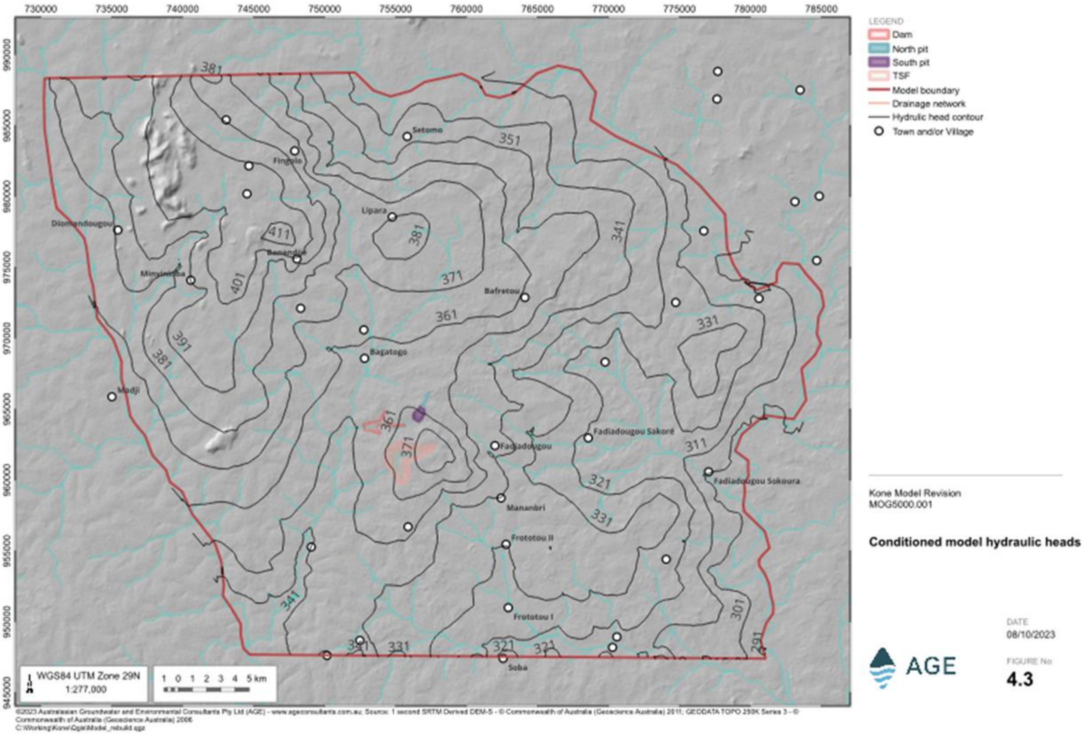
The following groundwater level maps present the different stages of pre-mining and operational groundwater levels for the Koné numerical model:

- Figure 18.1.10 shows the steady-state groundwater level for the model domain under the natural state.

The predicted drawdown by the end of mining is shown in

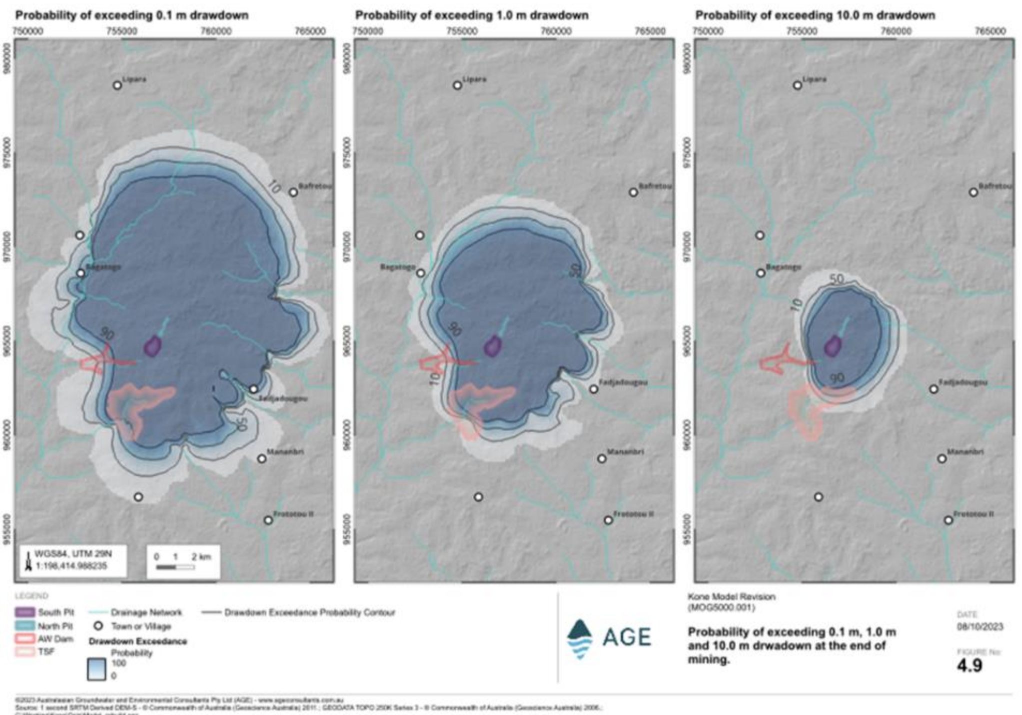
- Figure 18.1.11 shows the predicted 1 m drawdown zone of influence is approximately 4.5 km from the mine, and does not include any known community groundwater supply assets.

Figure 18.1.10 Pre-Mining Water Table



Source: AGE, November 2023.

Figure 18.1.11 Koné Predicted Drawdown Extent at the End of Mining in Year 9

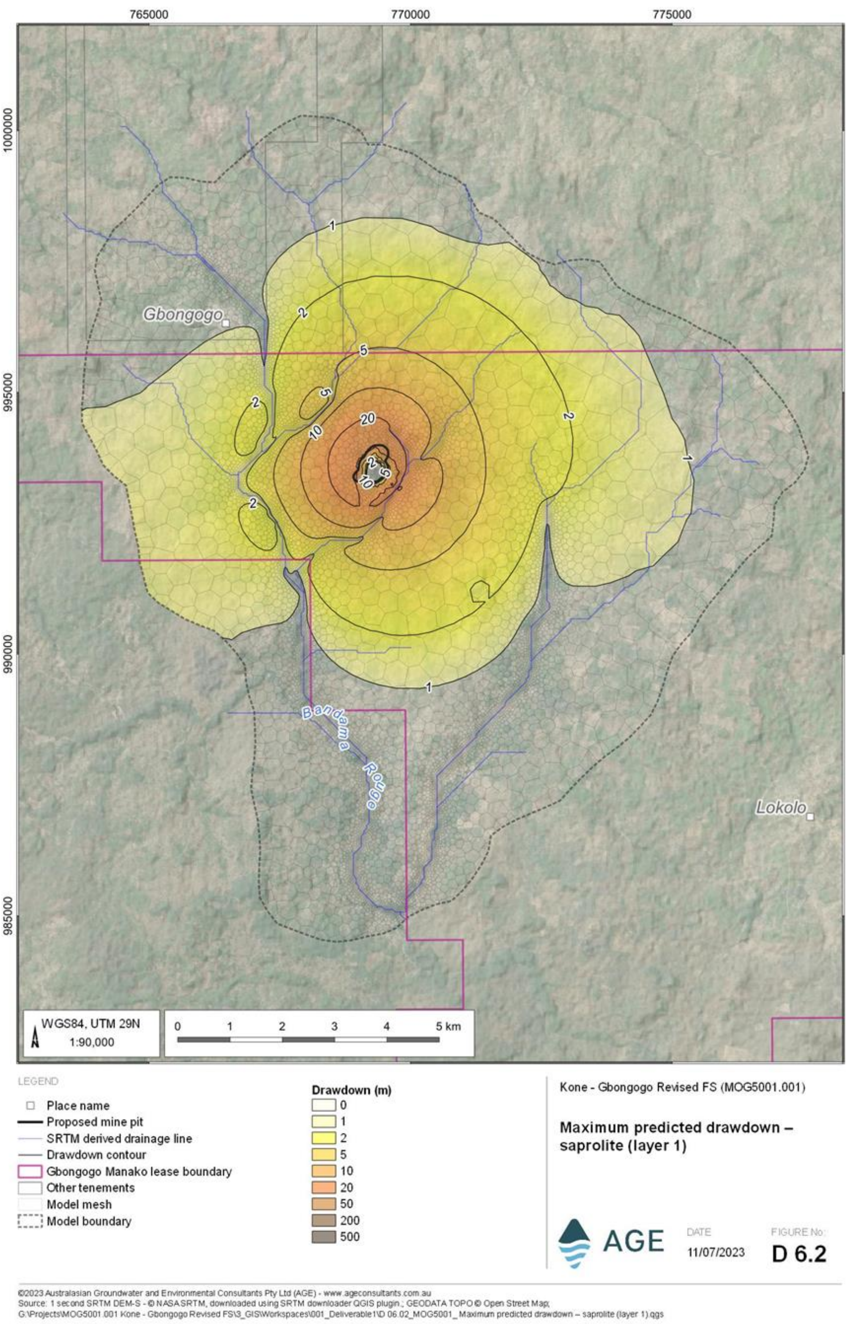


Source: AGE, November 2021.

The simulated range of the predicted Koné mine pit inflows range between 4 to 6 ML/day.

Predictions of maximum groundwater drawdown at the proposed Gbongogo mine pit area, during mining have been completed using the numerical groundwater model described above. As shown in Figure 18.1.12, only one community bore is predicted to have a drawdown of >1 m, Lokolo and Ouyiré will not be affected. The monitoring bores are included in the table to provide context.

Figure 18.1.12 Gbongogo Predicted Drawdown Extent at the End of Mining in Year 4

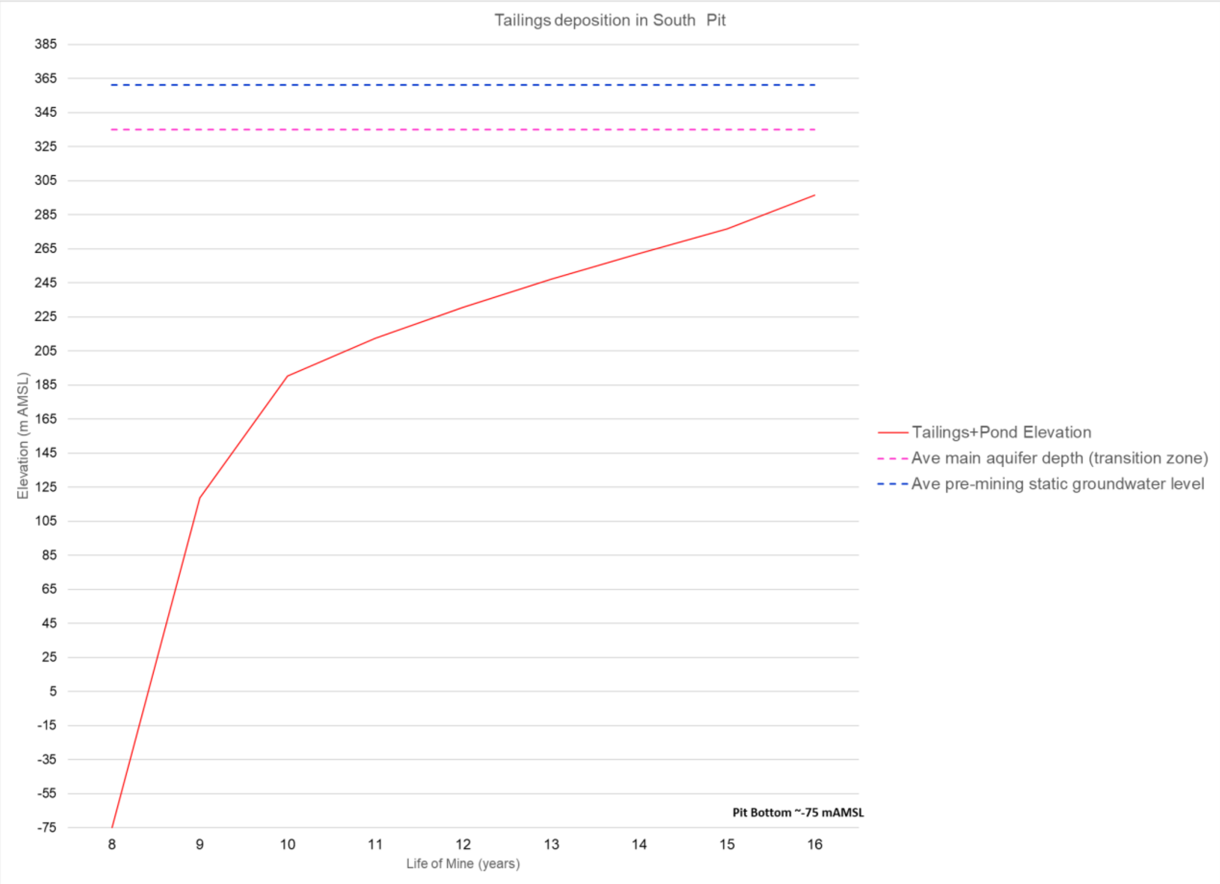


Source: AGE, 2023

18.1.6 Tailings Disposal in Koné South Pit

At the conclusion of mining at the Koné pit the maximum depth will be -75 mAMSL. Tailings disposal will start from Year 9, and the elevation for the rate of rise is shown in Figure 18.1.13. At the time of the mine closure, tailings will have reached 279 mAMSL, which is ~55 m below the transition aquifer and ~75 m below the average static groundwater level. The pre-mining groundwater level of ~361 mAMSL is reached between 17 to 25 years after dewatering is stopped. Due to the positive water balance, the groundwater level may rise above the original level as the mine pit becomes a water source during the rainy season or during above average rainfall years. This may lead to the decanting of the open pit after closure, from which the decant water will be captured in the WSF.

Figure 18.1.13 Tailings Deposition Elevation in South Pit



Source: AGE, 2022.

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.

The geochemical and mixing model data indicate that the operational pit lake nitrate concentration will be 0 mg/L when deposited in the pit and, following CN degradation, will peak at 100 mg/L and then reduce to <50 mg/L after about 35 years and <20 mg/L after about 100 years. The following provides an overview of the likely nitrate plume propagation over time.

35 Years Post Mining

Around this time the groundwater levels return to their original levels, and the pit environment will form an equilibrium state with the regional hydrogeological environment and climate. The primary mechanism for plume development before heads recover would be diffusion, which is demonstrated in Figure 18.1.14.

- As shown in Figure 18.1.14, the 10.0% threshold plot (left most plume map) in the 35 years post-mining plot reads as "…*the percentage chance that a concentration exceeding 10 mg/L*…" will be detected after 35 years – the extent of the 10 mg/L plume spreads out to the northeast. Although this is higher than the background natural groundwater nitrate concentrations (~0.4 mg/L), it is below the drinking water guideline concentration of 50 to 100 mg/L.

- The 25% (or 25 mg/L) plume will be limited to a small area northeast of the pit (the middle plume map in Figure 18.1.14).

- A 50% (or 50 mg/L) plume is restricted around the pit.

50 Years Post Mining

- As shown in Figure 18.1.15, the extent of the 10 mg/L plume spreads out to the northeast. Although this is higher than the background natural groundwater nitrate concentrations (~0.4 mg/L), it is below the drinking water guideline concentration of 50 to 100 mg/L.

- The 25% (or 25 mg/L) plume will be limited to the northeast (the middle plume map on Figure 18.1.15).

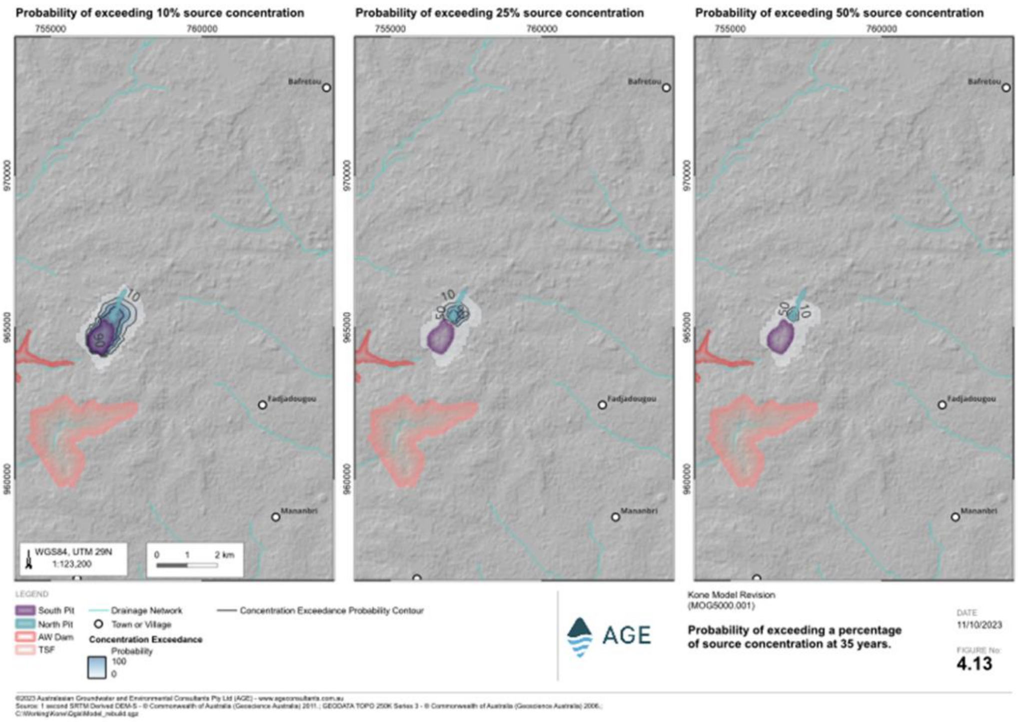
- A 50% (or 50 mg/L) plume is unlikely to develop to the northeast; only a small plume developed directly south of the pit.

100 Years Post Mining

- As shown in Figure 18.1.16, the 10.0% threshold plot (leftmost plume map) in the 100-year post-mining plot spreads out further to the northeast.

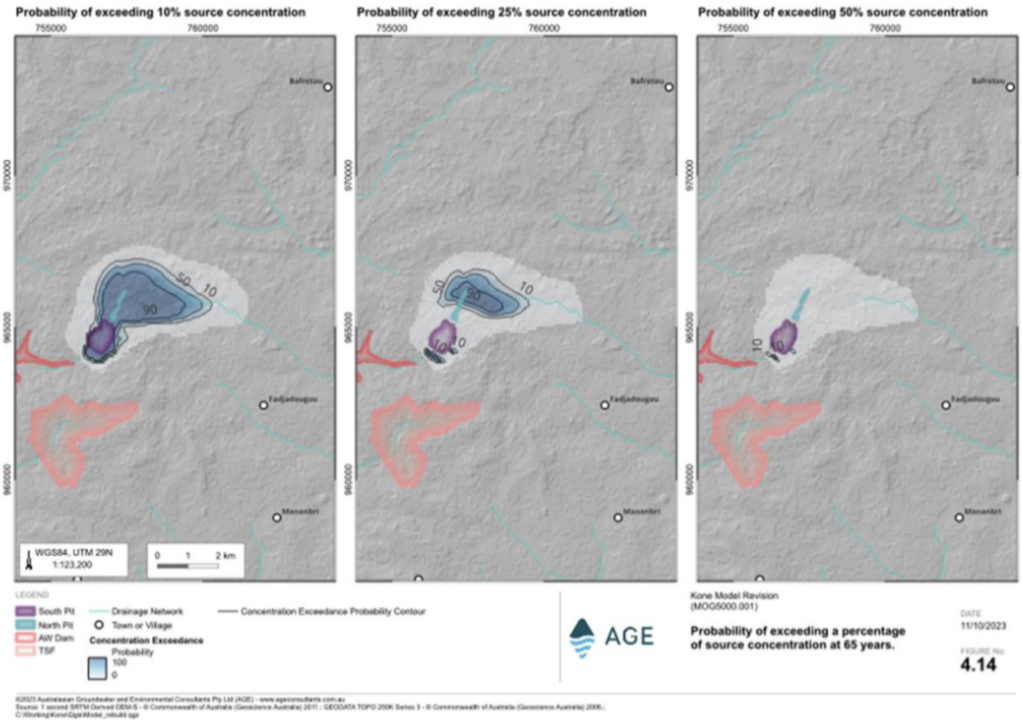
- The 25% (or 25 mg/L) plume will be limited to the south and southeast and in close proximity of the pit (the middle plume map in Figure 18.1.16); a 50% (or 50 mg/L) plume is unlikely to develop to the northeast; only a small plume developed directly south and southeast of the pit.

Figure 18.1.14 Probability of Exceeding 10%, 25% and 50% of Source Concentration at 35 Years



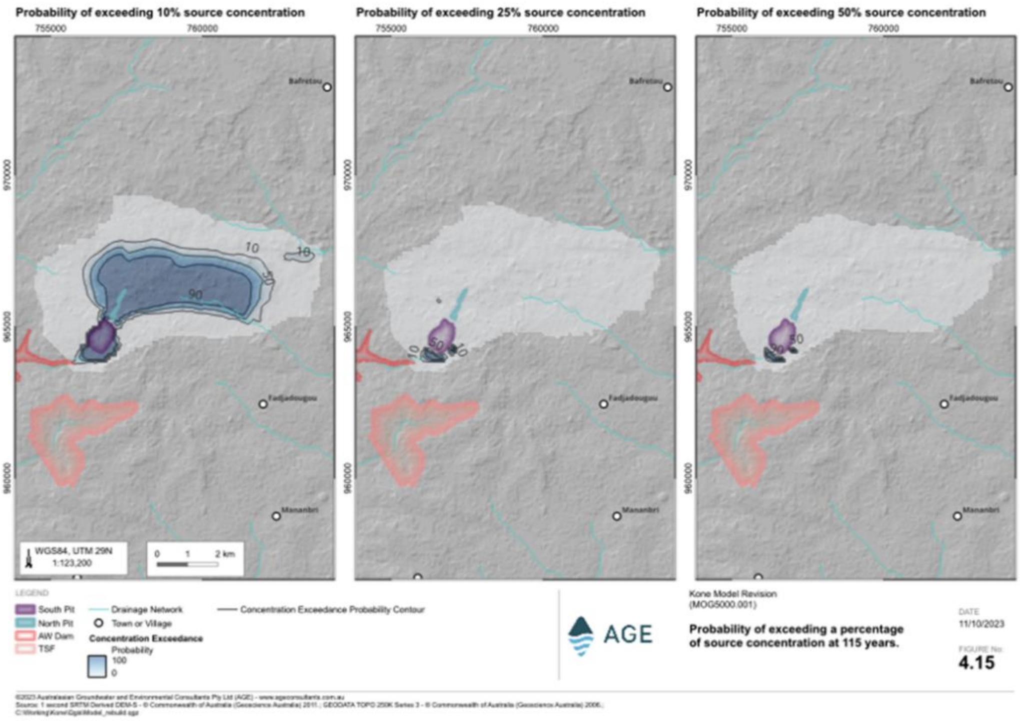
Source: AGE, 2023.

Figure 18.1.15 Probability of Exceeding 10%, 25% and 50% of Source Concentration at 65 Years



Source: AGE, 2023.

Figure 18.1.16 Probability of Exceeding 10%, 25% and 50% of Source Concentration at 115 Years



Source: AGE, 2023.

18.1.7 Environmental Assessment

Koné

The numerical groundwater model and work completed has not identified 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.

Gbongogo

Mine dewatering will have a limited to insignificant impact on regional communities and groundwater supply assets.

18.1.8 Mine Pit De-Watering

Pit dewatering for the Koné FS Project is discussed in Section 16.3.

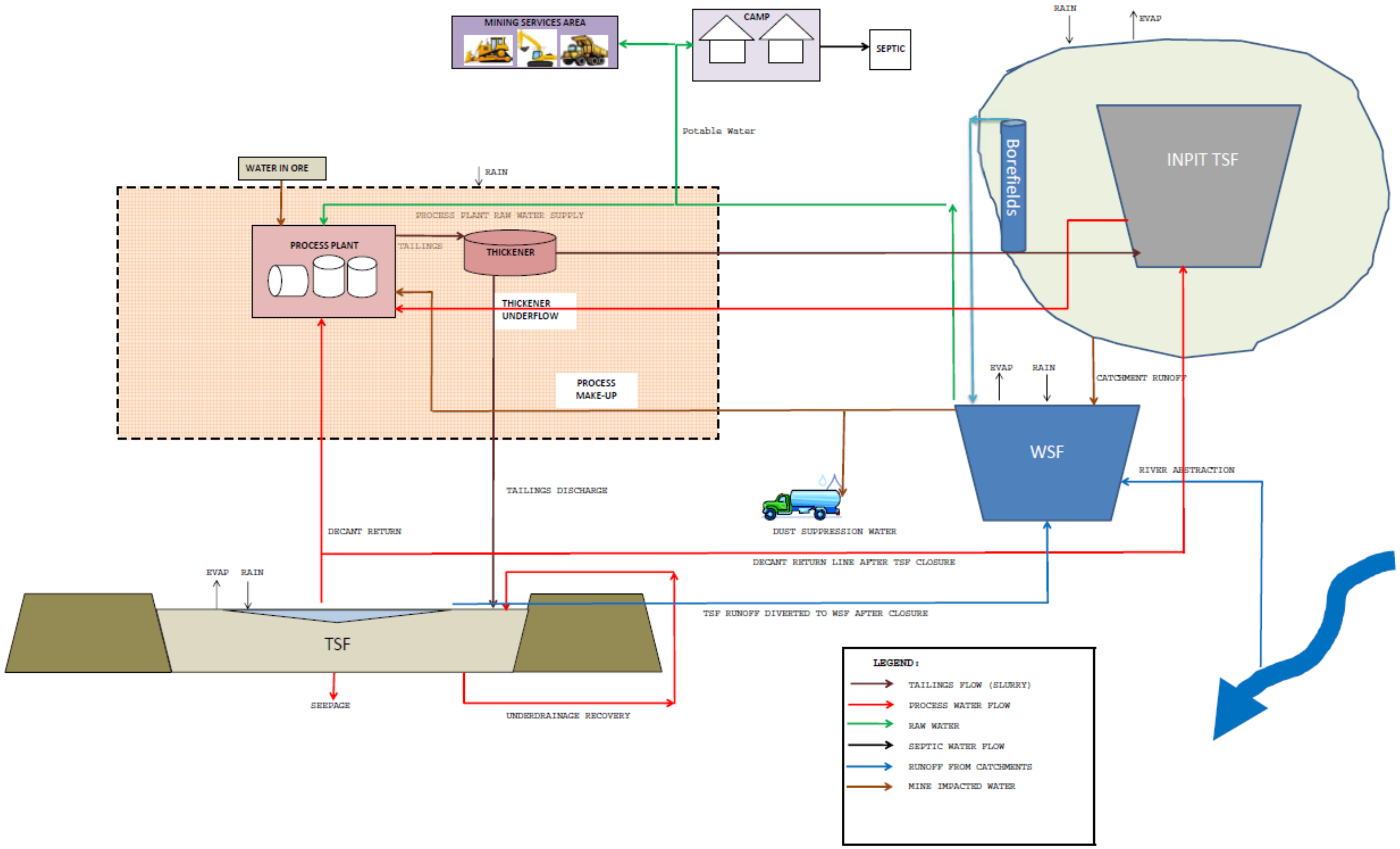
18.1.9 Water Balance Modelling

Water demand for the mineral processing, potable water and dust suppression demands will be supplied to the Project from TSF recycle, river harvesting and harvesting of run-off from the mining disturbance areas and the pit dewatering borefield for potable water supply.

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 Project's demand. The schematic flow diagram for the model in shown in Figure 18.1.17.

Figure 18.1.17 Water Balance Schematic

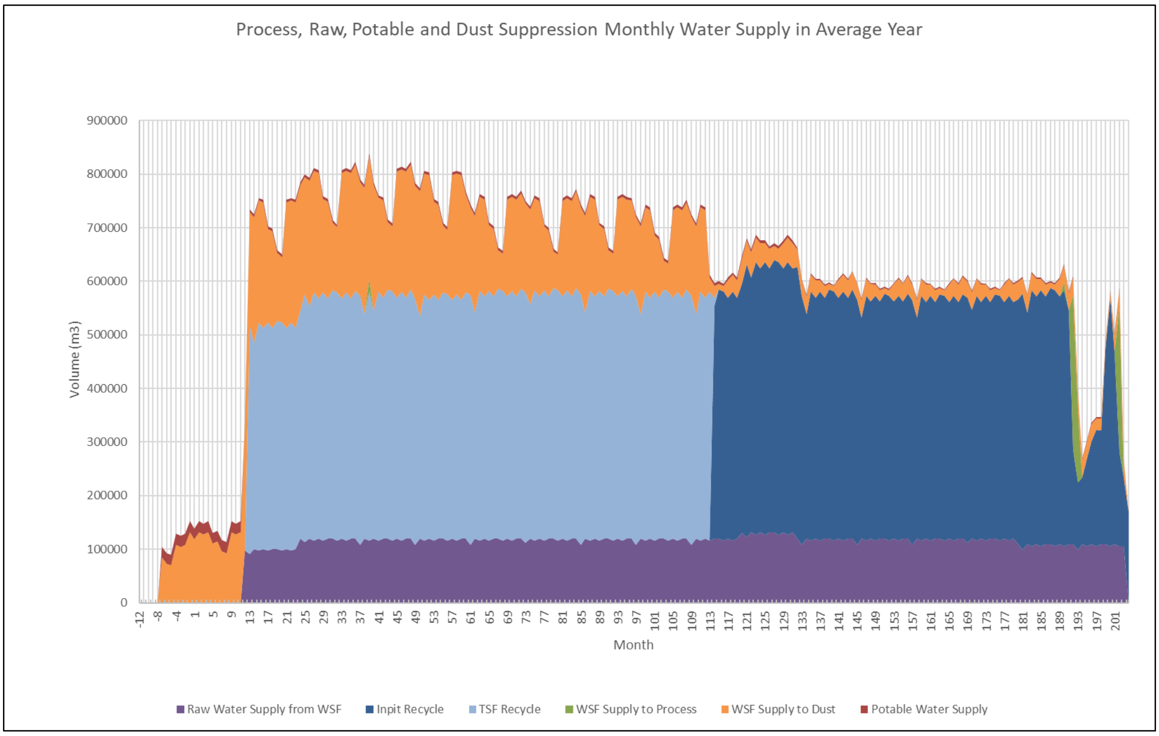


Source: KP, November 2021.

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 summarised in Figure 18.1.18 and

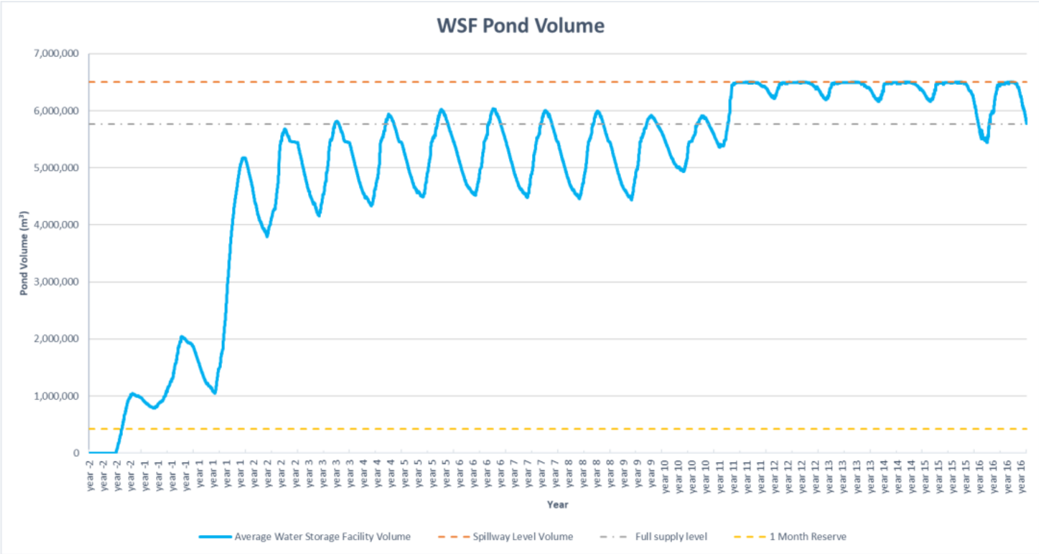
Figure 18.1.19 with statistical water abstraction rates provided in Figure 18.1.20.

Figure 18.1.18 Water Balance Results (Average Conditions)



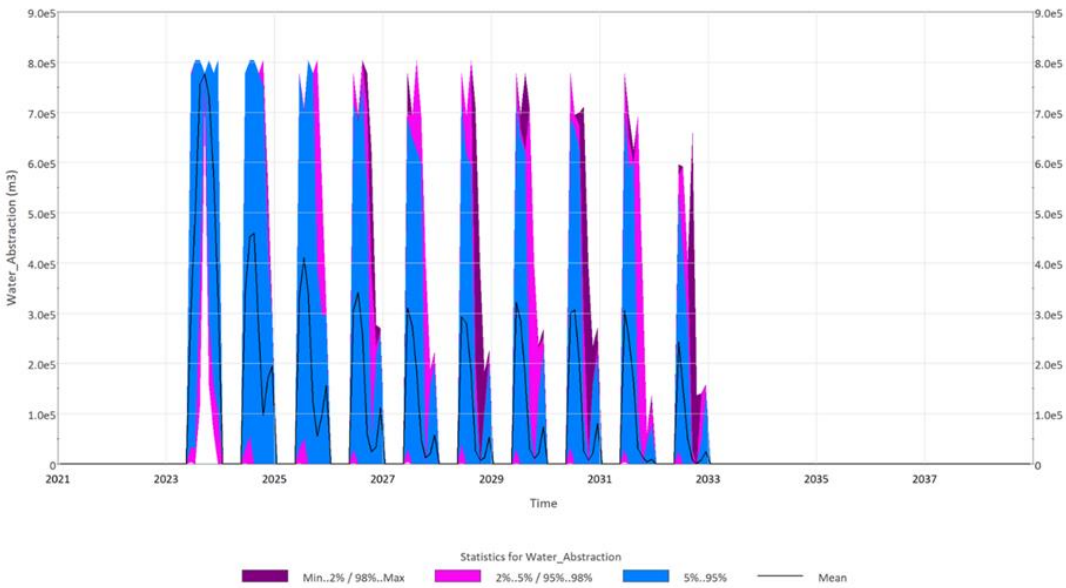
Source: KP 2024.

Figure 18.1.19 WSF Storage Facility (Average Conditions)



Source: KP 2024.

Figure 18.1.20 Water Abstraction Volume (Probabilistic)



Source: KP, 2024.

Table 18.1.2 shows the projected water supply and demand under average rainfall conditions.

Table 18.1.2 Koné Water Supply and Demand in Average Conditions

	Water Demand and Losses from WSF						Water Supply to WSF									TSF Water Supply and Demand			
Result:	Evaporation	Seepage	Potable Water Demand	Dust & Construction Demand	Process Raw Water Demand	Total	Run-Off and Rainfall	Marahoue River Abstraction	TSF Closure Pumped to WSF	Pit Run-Off	Borefields	Total	Shortfall	Spillway	WSF Pond Volume	Process Water Demand	Process Water Supply from TSF	Process Water Supply from Inpit	Process Shortfall from WSF
Unit:	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3	Mm^3
Year -2	0.18	0.06	0.12	0.55	0.00	0.91	1.21	0.00	0.00	0.20	0.73	2.13	0.74	0.00	1.23	0.00	0.00	0.00	0.00
Year -1	0.58	0.17	0.24	1.45	0.00	2.44	1.51	0.00	0.00	0.35	1.46	3.32	0.00	0.00	2.10	0.00	0.00	0.00	0.00
Year 1	0.91	0.27	0.06	2.44	1.18	4.86	1.51	44.17	0.00	0.35	2.19	8.22	0.00	0.00	5.46	5.01	5.01	0.00	0.00
Year 2	1.44	0.40	0.06	2.45	1.42	5.77	1.51	1.35	0.00	0.35	2.56	5.77	0.00	0.00	5.46	5.46	5.46	0.00	0.00
Year 3	1.44	0.40	0.06	2.45	1.41	5.77	1.44	1.24	0.00	0.71	2.48	5.88	0.00	0.00	5.54	5.47	5.44	0.00	0.03
Year 4	1.46	0.41	0.06	2.42	1.41	5.77	1.44	1.06	0.00	0.71	2.70	5.91	0.00	0.00	5.68	5.41	5.41	0.00	0.00
Year 5	1.49	0.42	0.06	1.81	1.41	5.20	1.44	0.88	0.00	0.71	2.37	5.40	0.00	0.05	5.84	5.49	5.49	0.00	0.00
Year 6	1.52	0.42	0.06	1.77	1.42	5.19	1.45	0.80	0.00	0.71	2.31	5.26	0.00	0.08	5.83	5.51	5.51	0.00	0.00
Year 7	1.51	0.42	0.06	1.81	1.41	5.20	1.44	0.86	0.00	0.71	2.19	5.20	0.00	0.03	5.79	5.49	5.49	0.00	0.00
Year 8	1.50	0.42	0.06	1.60	1.41	4.99	1.44	0.88	0.00	0.71	1.977	5.00	0.00	0.01	5.79	5.47	5.47	0.00	0.00
Year 9	1.50	0.42	0.06	0.96	1.41	4.35	1.44	0.83	0.00	0.31	1.97	4.55	0.00	0.00	5.99	5.47	2.28	3.199	0.00
Year 10	1.54	0.43	0.06	0.50	1.55	4.08	1.45	0.34	0.00	0.16	1.89	3.92	0.00	0.00	5.83	6.00	0.00	6.00	0.00
Year 11	1.62	0.45	0.03	0.32	1.41	3.83	1.44	0.00	6.70	0.16	1.97	10.28	0.00	5.79	6.50	5.47	0.00	5.47	0.00
Year 12	1.67	0.46	0.03	0.32	1.41	3.89	1.44	0.00	6.75	0.16	1.86	10.22	0.00	6.33	6.50	5.38	0.00	5.38	0.00
Year 13	1.67	0.46	0.03	0.32	1.41	3.89	1.44	0.00	6.74	0.16	1.75	10.10	0.00	6.21	6.50	5.37	0.00	5.37	0.00
Year 14	1.67	0.46	0.03	0.32	1.42	3.90	1.45	0.00	6.74	0.16	1.72	10.06	0.000\	6.16	6.50	5.39	0.00	5.39	0.00
Year 15	1.67	0.46	0.03	0.32	1.29	3.77	1.44	0.00	6.48	0.16	1.68	9.76	0.00	6.19	6.30	5.63	0.00	5.30	0.01
Year 16	1.57	0.44	0.03	0.32	1.28	3.63	1.44	0.00	4.88	0.16	1.60	8.08	0.00	4.16	5.89	5.60	0.00	2.70	0.70

18.1.10 Water Storage Facility

A WSF will be constructed downstream of the mining and processing area to act as the main WSF and sediment control dam. The facility will have a capacity of 6.4 Mm^3 (up to the spillway invert level) with a pond area of 158 ha. The WSF embankment will be a maximum of 16 m high and have a length of 615 m. A spillway will be provided to safely release excess water from the facility for events up to probable maximum precipitation. 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.

18.1.11 Water Harvesting / River Abstraction Facility

The river abstraction facility will be constructed at the bridge crossing along the Gbongogo Haul Road approximately 26 km north of the WSF. The abstraction system will comprise a system of pump(s) either suspended from or tethered to the bridge abutment which will allow abstraction of water from the river during the months of higher flow (nominally June through to December). A pipeline alignment has been nominated between the river abstraction location and WSF which will located within the Gbongogo haul road corridor to allow for inspection and maintenance.

18.2 Power Supply

Electric power consumption for the Koné Project is estimated to be:

- Connected load 60 MW.
- Maximum Demand 45 MW.
- Average annual demand 38 MW (at a load power factor of 0.95 lagging).
- Energy consumption 305 GWh/y.
- Largest motor 10.2 MW ball mill (dual 2 x 5.1 MW).
- Average grid power price US$0.115 /kWh.

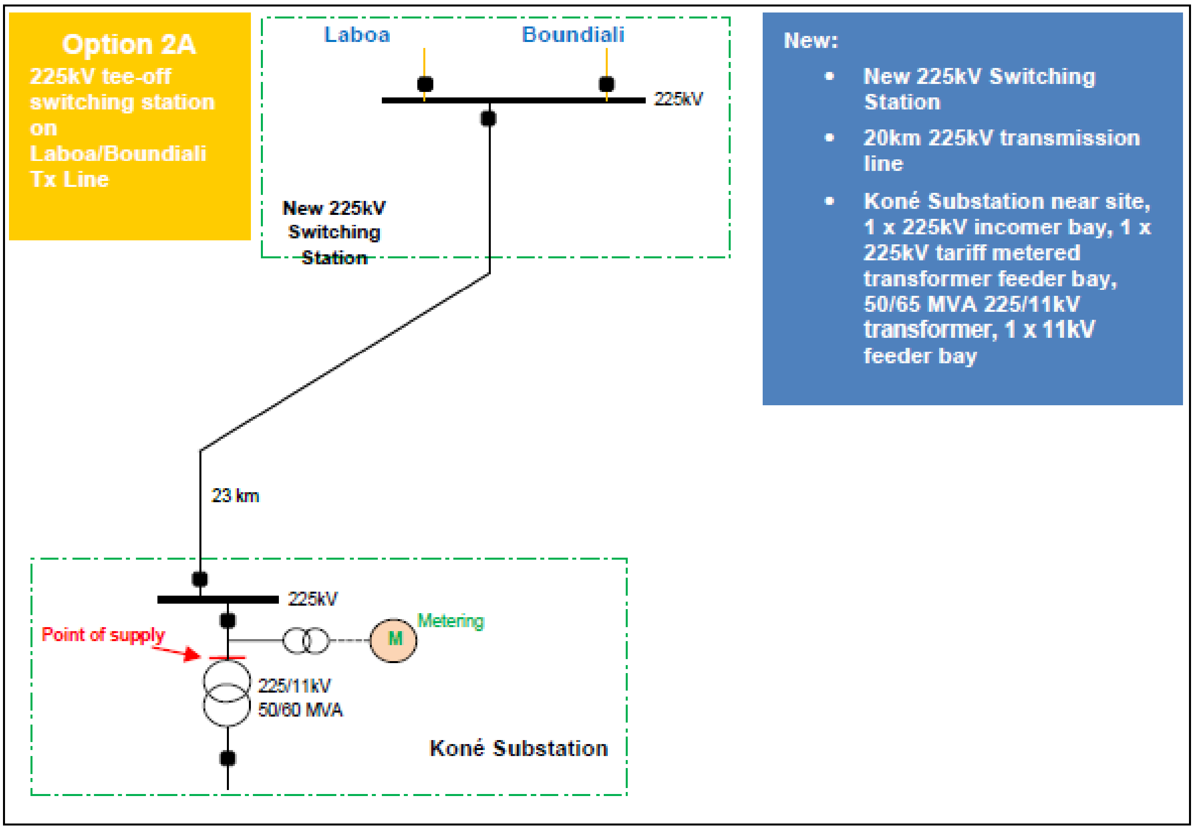
It is proposed that the power supply will be supplied by a dedicated 23 km 225 kV transmission line connecting to a switching station between Laboa and Boundiali (Figure 18.2.1 and Figure 18.2.2). The switching station would be located approximately 5 km from the Morondo town. The Koné plant substation would be owned and operated by CIE, and Koné mine would be supplied by a 225 kV tariff metered feeder, installing a 225 / 11 kV transformer in site substation with an 11 kV feeder to the plant main 11 kV switchboard.

The 225 kV supply at Koné should be of good quality and reliable. The 225 kV supply is from the 225 kV network and is part of a ring-main system which is reliable and based on a risk assessment, does not require a full back-up power station.

Figure 18.2.1 Cote d'Ivoire Electricity Network

Source: CIE, 2023.

Figure 18.2.2 225 kV Connection from Laboa Substation 225 kV Bus



18.3 Tailings Storage Facility Design

The design of the 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 analyses and stability analyses 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 the design report, 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 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 8.4 years). Tailings will be deposited in South Pit for the final 7.6 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 90.8 Mt which will be generated by the process plant over a period of 8.4 years at a rate of approximately 11 Mtpa after the initial ramp-up period.

The South Pit will store approximately 83.6 Mt and will be utilised from year 8.4 up to end of Year 16.

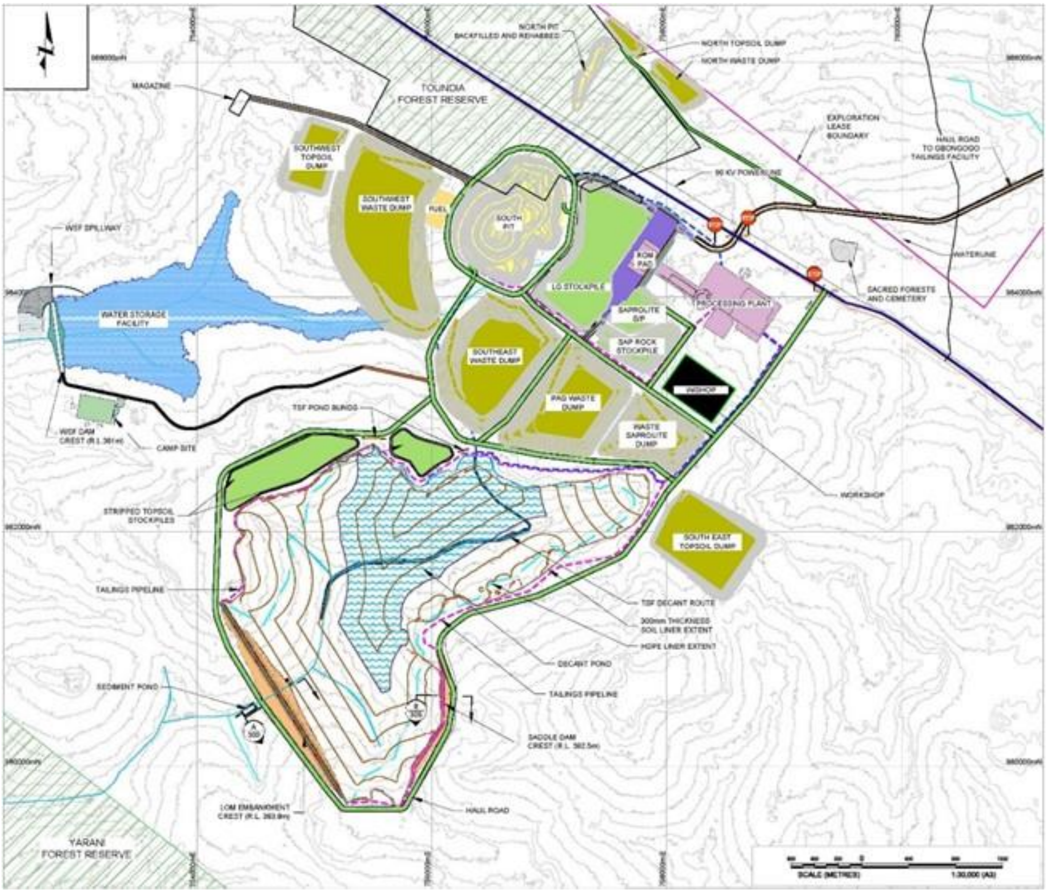
The TSF comprises one cell confined by a cross valley embankment and a saddle dam. The main embankment will initially be constructed, with the saddle dam constructed later in the mine life to provide enough capacity for the first stage of tailings management.

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 in Figure 18.3.1.

Figure 18.3.1 Tailings Storage Facility General Arrangement



Source: KP 2024.

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 two 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 two 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.

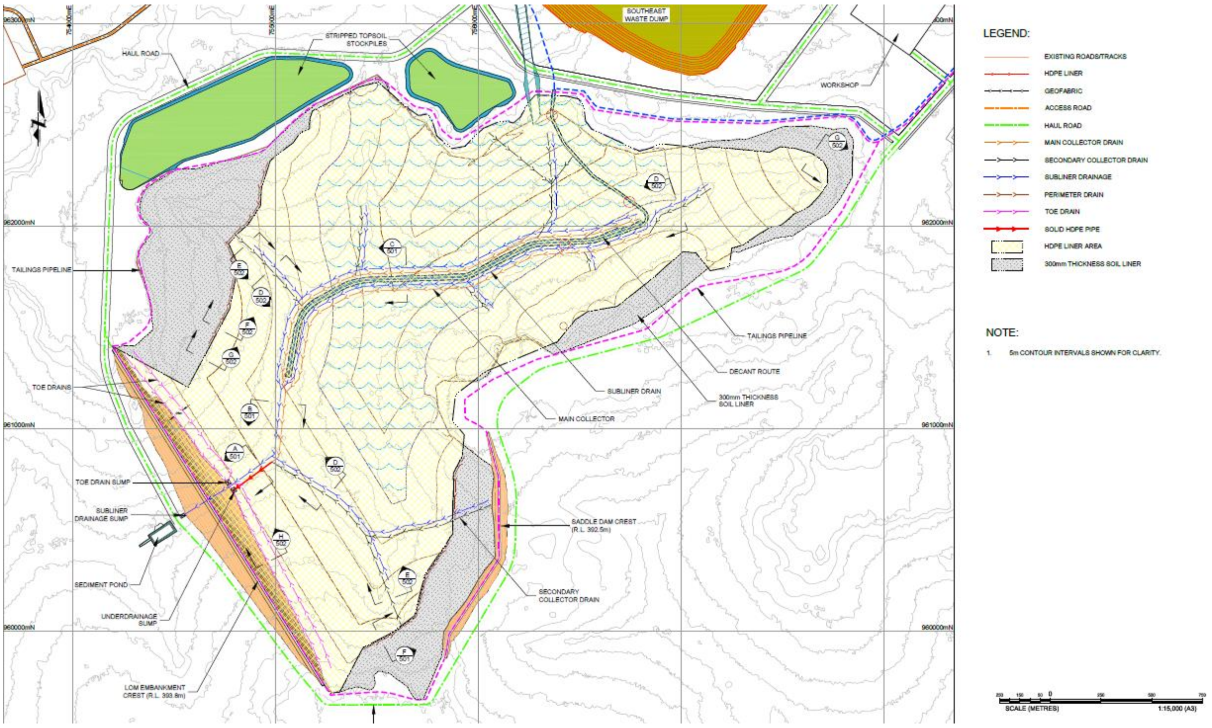
18.3.3 Tailings Geochemical Characteristics

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 130 mg/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 6 m 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 in Figure 18.3.2.

Figure 18.3.2 Tailings Storage Facility Liner and Drainage System



Source: KP, 2024.

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. As the extent of the high-resolution topography available downstream of the dams was limited at the time of assessment, very conservative estimates of the PAR have been employed in this study and it is conceivable that the consequence category may be reduced (in consultation with the TSF Independent Technical Review board) when additional topography can be acquired and the modelling has been re-run. The consequence category results are summarised in Table 18.3.1.

Table 18.3.1 Summary of Consequence Category Assessment

Description	ANCOLD Consequence	GISTM Consequence Category
Main Embankment Dam Failure Consequence Category	Extreme	Extreme
Saddle Dam – Dam Failure Consequence Category	Extreme	Extreme
TSF – Environmental Spill Consequence Category	Extreme	N/A
WSF – Dam Failure Consequence Category	High B	N/A

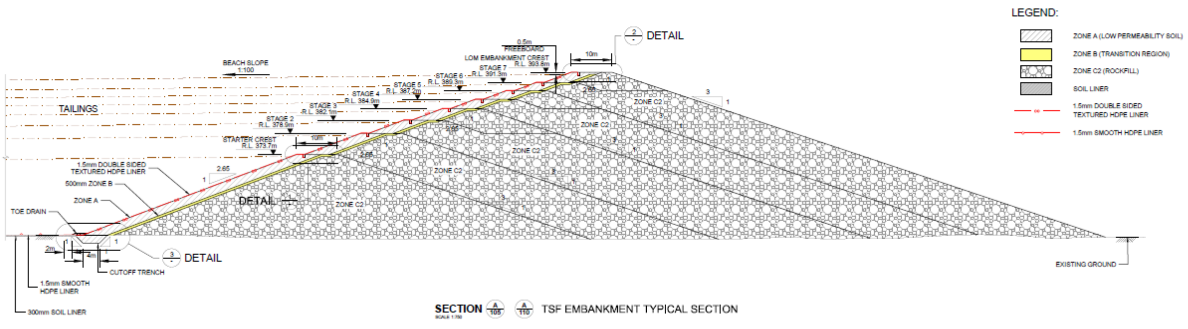
18.3.5 Embankment Configurations

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).

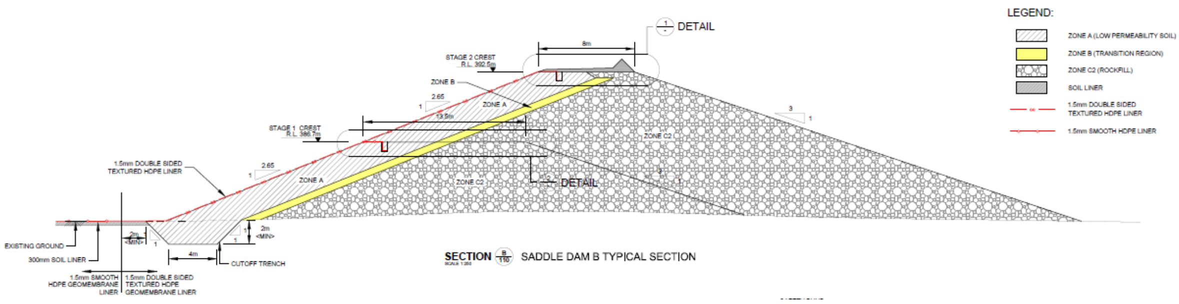
Figure 18.3.3 and Figure 18.3.4 show the typical details of the TSF embankment and saddle dam.

Figure 18.3.3 Main Embankment – Typical Section



Source: KP 2024.

Figure 18.3.4 Saddle Dam – Typical Section



Source: KP 2024.

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, even under extreme rainfall conditions. As such, only 0.5 m operational freeboard from the maximum tailings level on the embankment to the embankment crest was considered when defining the embankment crest RLs.

Embankment will be constructed with a 21 m high starter embankment and be raised annually in seven 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

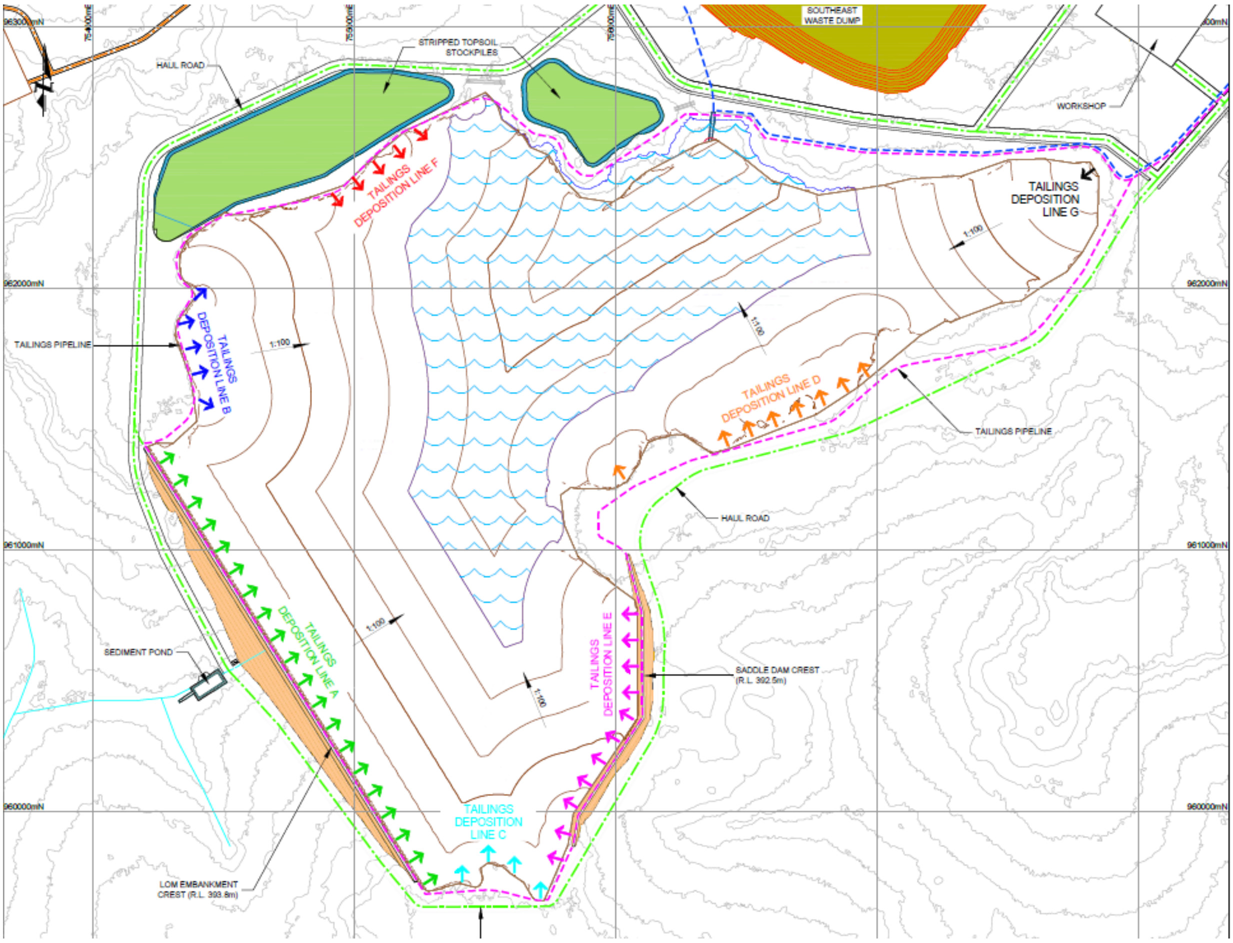
Tailings will be deposited subaerially with the supernatant pond located away from the embankment. Water will be recovered from the supernatant pond by a suction pump with floating intake located in a channel excavated adjacent to an access causeway.

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 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 the south ridgeline will start to guide the pond to the north, close to its final location. In Stage 4 and after construction of saddle dam, a deposition line from the saddle dam will be added to the deposition plan. The north line will be depositing in Stage 5 and an eastern line will be added in Stage 8. The deposition arrangement is presented in Figure 18.3.5.

Figure 18.3.5 Tailings Deposition Arrangement



Source: KP 2024.

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.

Table 18.1.2 shows the projected water supply and demand under average rainfall 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 34 boreholes to depths of between 18.9 m and 52.7 m.
- Excavation or drilling of 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 clayey residual soil (1 to 5 m), underlain by extremely weathered (XW) granite / basalt (5 to 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 clay rich residual soil (1 to 8 m), underlain by extremely weathered (XW) granite / basalt (8 to 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 to 6 m), underlain by extremely weathered (XW) granite / basalt / schist (6 to 12 m) and finally distinctly to slightly weathered granite / basalt / schist. Feasibility level foundation design has been completed which indicates that for the majority of the plant site structures, acceptable safe allowable bearing capacity values can be achieved through minor excavation and replacement of the residual soil with deeper foundations only required for the high load structures at the mills, HPGR and crushers.

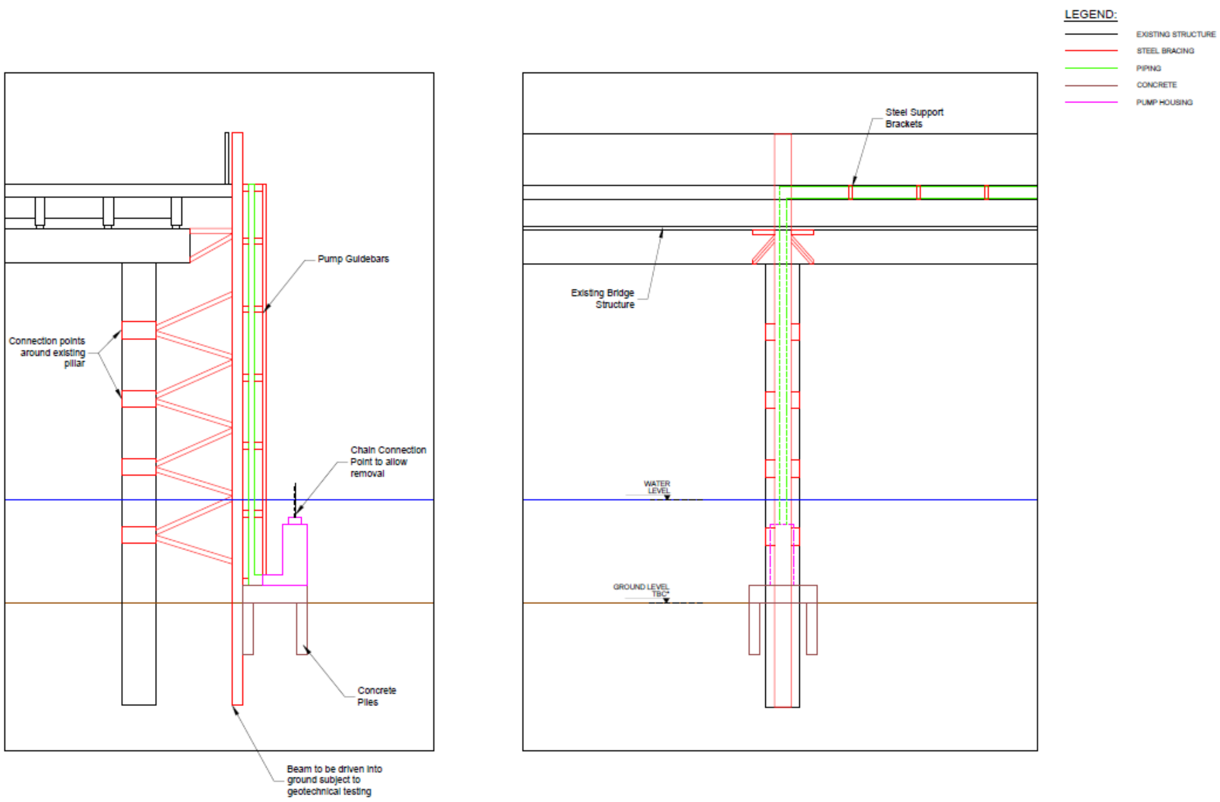
18.4.4 Water Storage Facility

A water dam will be constructed downstream of the mining and processing area to act as the main WSF and sediment control dam. The facility will have a capacity of 6.4 Mm^3 (up to the spillway invert level) with a pond area of 158 ha. The WSF 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.

Figure 18.4.1 Tailings Deposition Arrangement



Source: KP, 2024.

18.5 Gbongogo Haul Road

The Gbongogo haul road is 38.1 km in length and transverses a sparsely populated area between the two sites and has been designed to avoid villages, defined forest areas and minimise interactions with existing public roads. The road incorporates a pedestrian corridor leading to underpasses along the alignment. Access to the road will be restricted by construction of safety berms along the entire length of the road. Traffic control will be provided at all intersections with the public roads.

The haul road alignment has been designed to limit the number of water courses impacted by the road with culverts provided at all main water intersections and a bridge to be constructed at the crossing of the Marahoué River.

18.5.1 Haul Road Design

The design has been conducted in accordance with the Queensland Government's Recognised Standard 19 – Design and Construction of Mine Road, which, although not a Cote d'Ivoire standard, is one of the few comprehensive design standards specifically relating to mining haul roads.

No geotechnical investigation has been undertaken along the haul road alignment to date, therefore KP have applied the data from the geotechnical investigations at the Project and experience in similar west African tropical soils to estimate the geotechnical parameters for design of the road pavement system.

The pavement layer thickness defined was based on the heaviest equipment on the road (Cat 777), with the following thicknesses recommended:

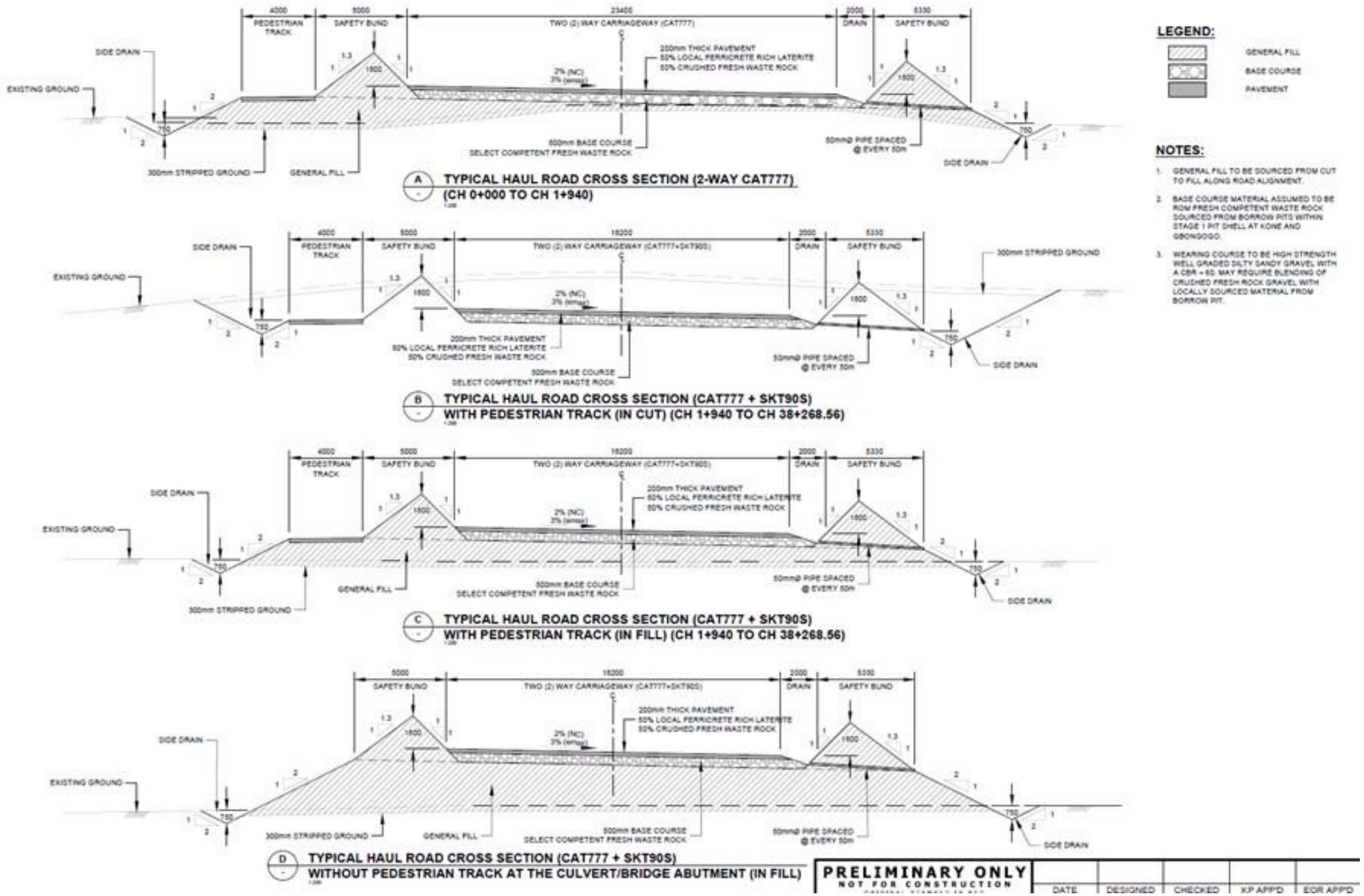
- Base coarse 500 mm.
- Wearing coarse 200 mm.

Geotechnical investigation is scheduled to be conducted along the haul road alignment during the dry season of early 2024 to confirm the geotechnical conditions and identify potential borrow material sources along the route.

Although the haulage will be carried out using Sany SKT90Sdump trucks with a 55 t payload, the haul road has been designed for the largest unit operating on the haulage road, a CAT 777 Water Truck. Provision has been made to incorporate safety berms with pedestrian access alongside the haulage road. The Marahoué pipeline will be incorporated in the eastern berm. Seventeen culverts will provide drainage and be sized to enable pedestrians to cross under the haul road.

Typical haul road sections are shown in Figure 18.5.1.

Figure 18.5.1 Typical Gbongogo Haul Road Sections



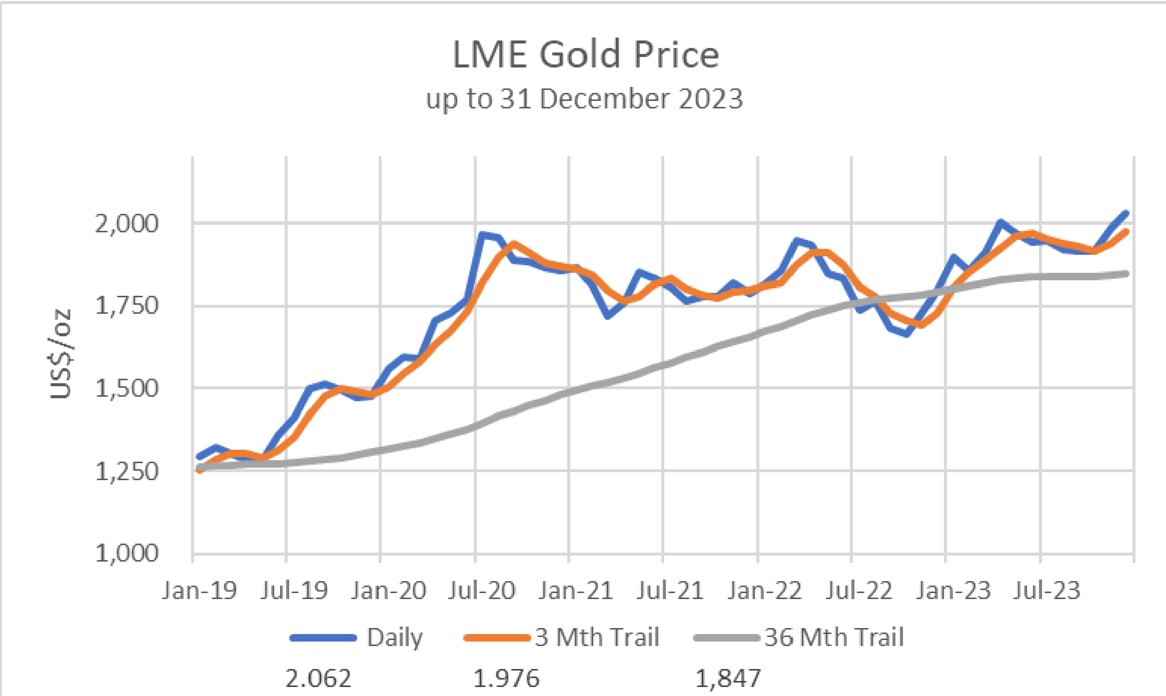
Source: KP, 2024.

19.0 MARKET STUDIES AND CONTRACTS

No market studies were carried out for this study. The final product of the 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 (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.

20.0 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 KGP, 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.
- Identify and review all requisite environmental data for the Project.
- Continue site sampling and analyses.
- Conduct detailed review of the type, scope and schedule for producing environmental and social government reports, including regulatory inspections, waste handling practices; management plans.
- Complete gathering and evaluation of baseline environmental and social conditions.
- Draft social, training, and health / safety programs.
- Submit the environmental and social impact assessment (ESIA) to regulatory authorities initiated.
- Define environmental characteristics used in the 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

  - closure plan
  - spill and emergency response plan.
- Complete environmental monitoring plan.
- Detailed evaluation of all pertinent authorisations 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 (EIA) prior to granting of any authorisation. 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 an EIA. The Mining Code requires that all mining title applicants (excluding artisanal) submit an ESIA to the DGMG and ANDE and all other institutions as required by the Mining Decree. This ESIA was submitted in December 2023. 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 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, provides 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 mitigation measures recommended where appropriate.

Under the Mining Code, all applicants for an exploitation licence must submit an ESIA to ANDE, which is the environmental authority in charge of supervising, validating and controlling environmental impact studies. The ESIA 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 ESIA 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 Administration, 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 20.5.1 and Figure 20.5.2. In summary, the main components of the operation are anticipated to comprise the following:

- Open pit mines – two open pits at Koné and a satellite pit at Gbongogo are to be developed using standard open pit mining techniques, treating 11 Mt/y.

- Processing plant – ore crushed prior to adsorption of gold onto activated carbon through CIP extraction methods. The process plant will be located near the Koné deposit.

- TSF – with capacity of 75 Mt and including tailings water drainage system, a recovery water basin and pipeline connecting the TSF to the plant. From Year 8, tailings are to be stored in the exhausted South Pit, storing 85 Mt. Waste rock dumps – for disposal of overburden and waste material from the open pits.

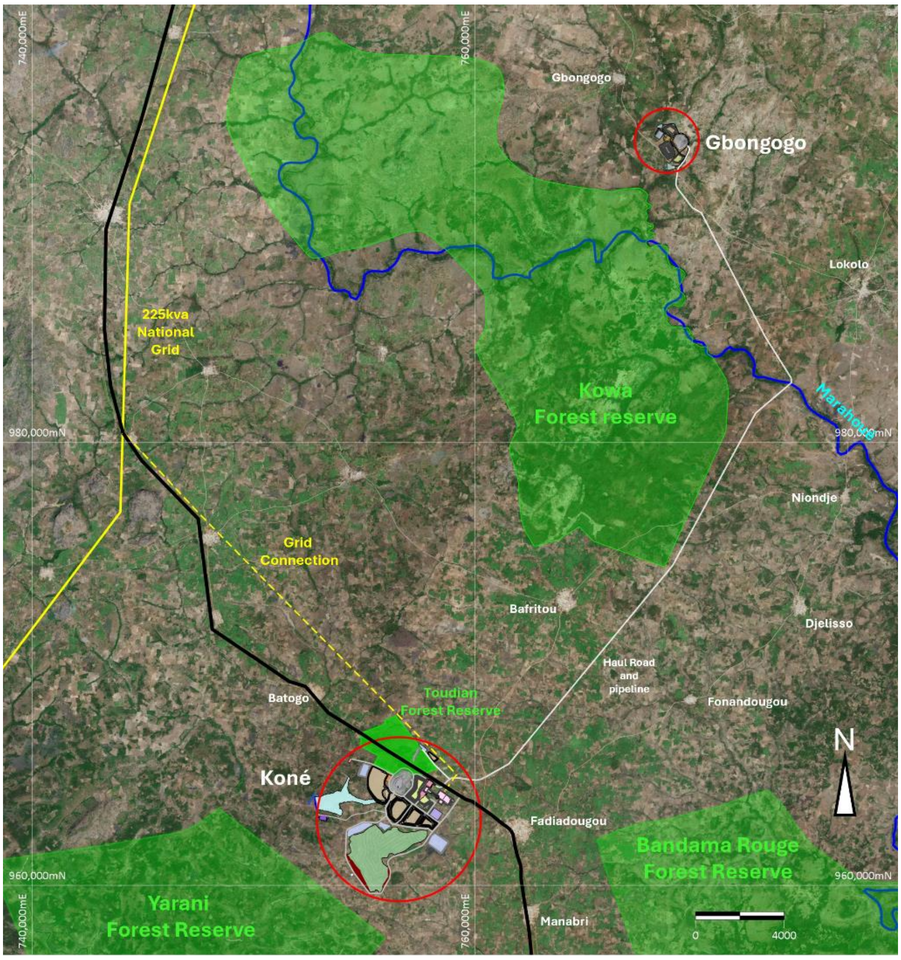
- Water supply and treatment – pipe work will be required to supply water from the Marahoué River to a raw WSF on the mine site, supplementing pit dewatering sources and rainfall harvesting.

- Power will be sourced from the grid via a 225 kV line.

- 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.

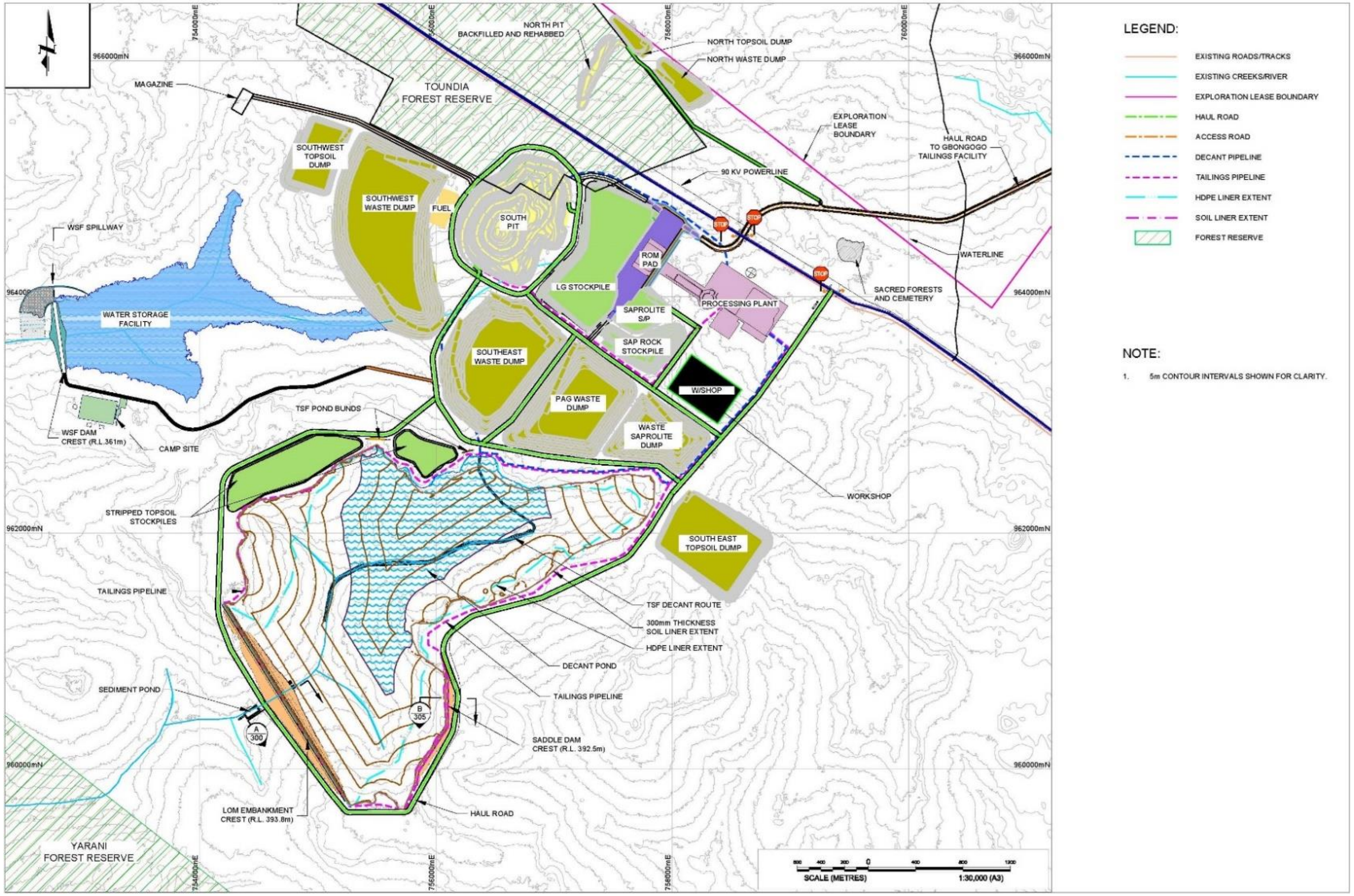
- Accommodation camps for construction workers and mine employees.

Figure 20.5.1 Site Layout



Source: Montage, 2024.

Figure 20.5.2 Central Site Layout



Source: KP, 2024.

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 in the Worodougou and Béré regions, covering the departments of Kani and Dianra. The capital of the Worodougou region is Séguéla and that of the Béré region is Mankono. The Project area covers the sub-prefectures of Fadiadougou, Kani, and Dianra. The ESIA study area covers an area of around 4,300 ha around Koné and 650 ha around Gbongogo. The two areas will be linked by a 38 km long haul road in a 50 m-wide corridor.

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 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. The smaller Koné northern pit and the northern edge of the larger pit are located within the Toudian classified forest. Permission is being sought to enable mining activities in this forest.

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 the Project footprint is the Kani-Bandama-Rouge or Bandama Rouge (UNEP-WCMC, 2019) Classified Forest Reserve, with an area of 1,055 km^2.

20.6.2 Baseline Environmental Setting

The climate of the 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°C and 32°C (77.0 and 89.6°F), and range from 10°C 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 centre 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 to Koné are Fadiadougou, Batogo and Manabri, while Gbongogo is the closest town to the satellite pit. These towns have light industry supporting the communities, but no heavy industry. Road traffic generates dust, which is captured by roadside vegetation, particularly along the haul road. 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 will be connected to the National Grid via a 225 kV line.

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 and stored in water tanks (Figure 20.6.1) indicated that the water was of good quality.

Figure 20.6.1 Water Tanks at Fadiadougou

Source: Montage, 2022.

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 20.6.2 to Figure 20.6.4). Monkeys and cattle have been observed in the early images captured by the passive cameras.

Figure 20.6.2 Panoramic of Pit Area, Showing Level Terrain

Source: Montage, 2022.

Figure 20.6.3 Panoramic of Area Between TSF and WRD, in South of Project

Source: Montage, 2022.

Figure 20.6.4 View from Camera Trap #2

Source: Montage, 2022.

20.6.3 Baseline Social Setting

The Project area is located relatively close to the communities of Batogo, Fadiadougou, Manabri and Gbongogo, 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 these communities regularly and the nearest communities have received specific support, such as beds for the clinic and construction materials for the school. The local communities provide 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 a 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, utilised for cotton and cashew. No sites of archaeological interest have been identified on the site and the Project has a 'Chance Finds Procedure' to mitigate against any impacts from their exploration activities. This procedure defines a series of steps to minimise 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 have gathered and evaluated data and produced a potential environmental and social impacts assessment, with support from international consultants as required. The consultant commissioned and completed studies and additional work necessary to produce the ESIA in accordance with national regulatory provisions and IFC Performance Standards. They have engaged with ANDE and other stakeholders relevant to the Project.

Terms of reference for the ESIA have been approved by ANDE, and the ESIA conformed to this terms of reference. Key components of the ESIA 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 hydrogeology, 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 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) was 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.
- Minimise emissions generated by exploration, particularly dust.

The Exploration Management Plan being implemented onsite will continue to be developed to inform the 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 and project development 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 and development 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 finalised 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 20.9.1 below.

A large proportion of the closure activities can be completed during mine operations. Progressive reclamation will be carried out during normal mine operations where circumstances allow. Many closure activities can be completed during mine operations. The Gbongogo Pit and associated infrastructure, North Pit with associated infrastructure, and the TSF will all be closed prior to the end of processing (Year 16). These costs are incorporated as sustaining capital costs, with residual closure costs of approximately US$4.7M incurred post-processing (Years 17 to 18).

Table 20.9.1 Schedule of Closure Costs

Description	US$M									
	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 handover 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.

- Minimise residual impacts requiring ongoing 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 onsite 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 TSF seven 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 minimise the infiltration of precipitation into the WRD and maximise water runoff and evapotranspiration.

Open pits will be made safe for closure during operations, with slopes being made stable and access points controlled. The Gbongogo Pit will cease operations in Year 3, while the Koné northern pit will be backfilled with waste rock, commencing in Year 7. The southern pit will cease operations in Year 8 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 0.5 m 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 run-off and minimise ponding.

Buildings and structures other than the haul road, water supply pipeline, and camp 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 offsite 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 offsite. 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. Data collection included 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 have been 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 ESIA is a multi-disciplinary and iterative process, and these baseline studies provided 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 was conducted in accordance with national requirements and best practice. In order to ensure that the Project is developed and operated in an appropriate manner, Montage 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, Manabri, and Gbongogo. Meetings have been held with the leaders of these communities already (Figure 20.12.1 and Figure 20.12.2) and ongoing meetings are planned throughout the Project life. A record of all meetings is being maintained, summarising the numbers of people engaged with, their activities, and any issues or concerns they may have with the Project.

Figure 20.12.1 Meeting with Fadiadougou Chief and Elders

Source: Montage, 2018.

Figure 20.12.2 Meeting with Batogo Chief and Elders

Source: Montage, 2018.

21.0 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. 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 21.2.1. All costs are estimated in US$ as at 4Q23.

21.2 Capital Cost Summary

The capital estimate is summarised in Table 21.2.1. The initial project capital cost is estimated at US$712.1M, including a contingency allowance of US$65.3M.

Table 21.2.1 Capital Estimate Summary (4Q23, ±15%)

Main Area	Koné	Gbongogo	US$M
Resettlement	7.4	2.0	9.4
Camp	6.4		6.4
Mine	45.2	11.9	57.1
GB Haul Road, Bridge		27.4	27.4
GB Surface Water		3.3	3.3
Grid	26.1		26.1
Process Plant	338.4		338.4
Infrastructure	26.5		26.5
TSF	41.4		41.4
WSF	13.6		13.6
EPCM	46.4		46.4
Owner	49.3	1.4	50.7
Pre-Production Sub Total	**600.8**	**46.0**	**646.8**
Contingency	61.3	4.0	65.3
Pre-Production CapEx	**662.1**	**50.0**	**712.1**

The total LOM cost is estimated at US$835.6M including sustaining capital costs of US$165.3M, as shown in Table 21.2.2.

Table 21.2.2 Sustaining Capital Estimate Summary (4Q23, ±15%)

Main Area	US$M
Camp	4.4
TSF	65.0
Process Plant	34.4
Closure	61.6
Grand Total	**165.3**

21.2.1 Capital Costs – Mining

With the use of a mining contractor, who will provide the mining fleet, the only capital costs included are for 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. Pre-strip costs of $56.2M 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.

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 21.2.3.

Table 21.2.3 TSF Capital Estimate Summary (4Q23)

Description	Pre-Production	Sustaining	Closure	LOM
Engineering and Design	1.7	5.5	1.4	10.8
Preliminary and General	4.7	5.6	0.8	15.1
TSF	31.7	63.9	13.0	102.4
WSF and River Abstraction	11.2	1.0	0.0	12.3
Fuel	5.7	9.5	8.2	23.4
Total	**55.0**	**85.6**	**23.3**	**163.9**
Total + Contingency	**62.5**	**97.8**	**26.7**	**187.0**

21.3 Operating Cost Summary

21.3.1 Operating Costs – Mining

A tender process was conducted to ascertain market rates for contract mining operations. Six contractors provided a tender response.

Fixed mining costs were calculated for the prescribed fleet dependant on material type (crusher feed / waste and oxide + transitional / fresh) and destination. These included components for:

- Loading costs.

- Fixed hauling costs.
- Drill and 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 21.3.1 shows the fixed mining costs for the Project based on the contractor bids. Table 21.3.2 shows the fixed costs for Gbongogo based upon the contractor bids.

Table 21.3.1 Kone Fixed Mining Costs

Ore / Waste	Ore	Ore	Ore	Waste	Waste	Waste
Material	Oxide	Transitional	Fresh	Oxide	Transitional	Fresh
Fixed Hauling	2.16	1.90	1.57	1.77	1.73	1.54
Drill and Blast	0.84	0.75	0.75	0.49	0.59	0.58
Ancillary	0.04	0.04	0.04	0.04	0.04	0.04
Mine Admin	0.34	0.35	0.36	0.34	0.35	0.36
Grade Control	0.28	0.24	0.24	0.00	0.00	0.00
Total Fixed Cost	**3.65**	**3.28**	**2.95**	**2.64**	**2.71**	**2.52**

Table 21.3.2 Gbongogo Fixed Mining Costs

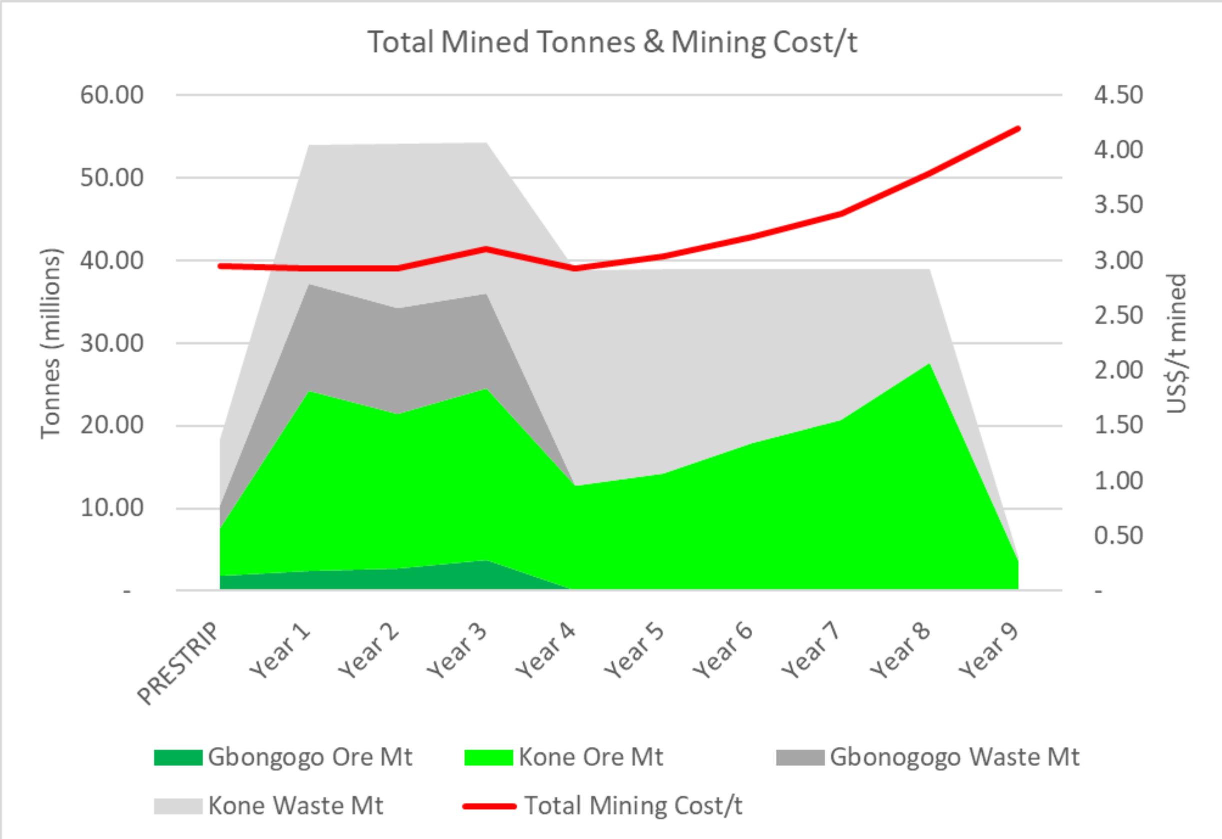
Ore / Waste	Feed	Feed	Feed	Waste	Waste	Waste
Material	Oxide	Transitional	Fresh	Oxide	Transitional	Fresh
Fixed Hauling	0.80	1.30	1.23	0.91	1.42	1.43
Drill and Blast	0.84	1.36	1.36	0.81	0.96	0.96
Ancillary	0.04	0.04	0.04	0.04	0.04	0.04
Mine Admin	0.31	0.31	0.31	0.31	0.31	0.31
Grade Control	0.28	0.24	0.24	0.00	0.00	0.00
Total Fixed Cost	**2.27**	**3.25**	**3.18**	**2.07**	**2.73**	**2.74**

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 calculated to be an incremental hauling cost of US$0.030 /t per 10 m of vertical haul and applied to all blocks within both deposits. At Koné, this was applied from a reference RL of 375m UTM and Gbongogo from a reference RL of 350 m UTM.

The fuel price used for mining optimization calculations was US$1.00 /L. Material rehandled from the stockpiles to the ROM pad was costed at US$0.93 /t. The haulage cost from Gbongogo to the Project plant is US$6.56 /t hauled.

Figure 21.3.1 shows the unit mining cost for the life of the operation compared to the total Koné and Gbongogo tonnes mined in each period. The unit costs increase gradually as the depth of the pit increases.

Figure 21.3.1 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 Opex is presented in Table 21.4.1, Table 21.4.2, and with an accuracy of ±15% based on pricing as at 4Q23. The process operating cost includes all direct costs to produce gold doré for the Project.

In general, costs have been built up from first principal estimates, with quotations obtained for major reagents and consumables and consumption rates based on metallurgical testwork, calculations or modelling. A quotation has been obtained for the laboratory costs based on an estimate of the sample analysis requirements for the project. Minor reagent, expatriate labour rates and a number of general and administration (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 utilisation applied and factored for the design throughput. The total power draw was used to calculate power costs based on the Government promulgated power price of US$0.1145 /kWh.

Total fixed G&A costs are estimated at US$12.3M annually.

Table 21.4.1 Koné South Pit Operating Cost per Material Type (4Q23 ±15%)

Cost Centre	Total				LOM	
	Oxide	Transition	Fresh	FW Fresh	Fix	Variable
	US$ /t	US$ /t	US$ /t	US$ /t	US$'000 /y	US$ /t
Power (exc. Crush and Grind)	0.61	0.61	0.61	0.61	8,397	0.61
Power (Crushing)	0.02	0.04	0.06	0.06		0.06
Power (Grinding)	0.56	1.23	1.74	1.74		1.64
Operating Consumables	4.01	3.59	4.05	4.05		4.03
Maintenance	0.15	0.15	0.15	0.15	6,378	0.15
Total Processing (Variable)	5.35	5.62	6.61	6.61	14,775	6.49
Laboratory	0.04	0.04	0.04	0.04	1,077	0.04
Process and Maintenance Labour					3,476	
Total Labour, Lab (Fix)	0.04	0.04	0.04	0.04	4,554	0.04
Total	**5.38**	**5.66**	**6.65**	**6.65**	**19,329**	**6.53**

Table 21.4.2 Koné North Pit Operating Cost per Material Type (4Q23, ±15%)

Cost Centre	Total				LOM	
	Oxide	Transition	Fresh	FW Fresh	Fix	Variable
	US$ /t	US$ /t	US$ /t	US$ /t	US$'000 /y	US$ /t
Power (exc. Crush and Grind)	0.61	0.61	0.61		8,397	0.61
Power (Crushing)	0.02	0.04	0.06			0.04
Power (Grinding)	0.56	1.23	1.74			0.99
Operating Consumables	4.01	3.59	4.05			3.92
Maintenance	0.15	0.15	0.15		6,362	0.15
Total Processing (Variable)	5.34	5.62	6.61		14,759	5.70
Laboratory	0.04	0.04	0.04		1,078	0.04
Process and Maintenance Labour					3,476	
Total G&A, Labour, Lab (Fix)	0.04	0.04	0.04		4,554	0.04
Total	**5.38**	**5.66**	**6.65**		**19.314**	**5.74**

Table 21.4.3 Gbongogo Pit Operating Cost per Material Type (4Q23, ±15%)

Cost Centre	Total				LOM	
	Oxide	Transition	Fresh	FW Fresh	Fix	Variable
	US$ /t	US$ /t	US$ /t	US$ /t	US$'000 /y	US$ /t
Power (exc. Crush and Grind)	0.61	0.61	0.61		8,397	0.61
Power (Crushing)	0.02	0.04	0.06			0.06
Power (Grinding)	0.56	1.23	1.74			1.63
Operating Consumables	4.44	3.45	5.10			4.97
Maintenance	0.15	0.15	0.15		6,377	0.15
Total Processing (Variable)	5.78	5.47	7.66		14,774	7.42
Laboratory	0.04	0.04	0.04		1,077	0.04
Process and Maintenance Labour					3,476	
Total G&A, Labour, Lab (Fix)	0.04	0.04	0.04		4,554	0.04
Total	**5.82**	**5.51**	**7.70**		**19,328**	**7.46**

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 at 0.5%.

- Licence fees.
- Royalties which are included in the financial model; Government at 4.0% and Triple Flag at 2.0% on the Koné pits, and Barrick / Endeavour at 2% on Gbongogo.
- 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 at US$4.71 /oz.
- Tailings storage costs, including future lifts and rehabilitation, which are included in sustaining capital.
- External government required tailings monitoring and compliance costs
- Tailings dust suppression costs.
- Any rehabilitation or closure costs which are included in sustaining capital.

22.0 ECONOMIC ANALYSIS

22.1 Introduction

The economic analysis is based on Inferred Resources and mine schedule as per Table 22.2.4.

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 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 summarised in Table 22.2.1 and, unless otherwise stated, is used in the model.

Table 22.2.1 Model Inputs and Assumptions

Model Inputs	**Unit / Value**
Base Currency	US$
Base Date	4th Quarter 2023
Côte d'Ivoire Royalty at US$1,600 /oz (charged against Revenue)	4.0%
Triple Flag Royalty on Koné pits (charged against Revenue)	2.0%
Barrick / Endeavour Royalty on GB pit (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$1,850 /oz
Refining Payability	99.9%
Refinery Charges and Shipping	US$4.71 /oz
Assumptions	
Capex excludes Finance Charges and Fees.	
Capex excludes Pre-production Investigations and Early Works.	
Capex Amortisation / Depreciation makes no allowance for potential salvage value.	
Capex excludes escalation.	
Tax paid on an Annual basis in the following year.	

22.2.1 Capital Costs

Pre-production capital expenditures are defined in Table 22.2.2. Sustaining capital for the plant, mining and TSF expansion costs have been phased over the life of the project and detailed in Table 22.2.3.

Table 22.2.2 Pre-Production Capital Expenditure

Item	Unit	Total	Year		
			-3	-2	-1
Resettlement	US$M	9.4			
Mine	US$M	57.1			
GB Haul Road and Bridge	US$M	27.4			
GB Water Ponds	US$M	3.3			
Process Plant	US$M	364.9			
Power	US$M	26.1			
TSF / WSF	US$M	55.0			
Camp	US$M	6.4			
EPCM	US$M	46.4			
Owner	US$M	50.7			
Construction Sub Total	**US$M**	**646.8**			
Contingency	US$M	65.3			
Construction Total	**US$M**	**712.1**	**43.8**	**246.2**	**422.1**

Table 22.2.3 Sustaining Capital Expenditure

Item	Unit	LOM
Camp	US$M	4.4
TSF	US$M	65.0
Plant	US$M	34.4
Mine Closure	US$M	61.6
Sustaining Total	US$M	**165.3**

22.2.2 Revenue

Revenue has been calculated allowing for 0.1% refinery loss.

22.2.3 Royalties

Royalties at 7.5% 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 five year starting with first year of production and can be summarised 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$58.3M to 31 October 2024.
- 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 five 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 21.3 and 21.4, are included in the cash flow.

22.2.8 Financial Model

Table 22.2.5 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 22.2.1 shows the annual and cumulative post-tax cash flow.

The pre-tax and post-tax financial results of the Project are summaries in Table 22.2.5. On a pre-tax basis, the Project has a Net Present Value (NPV) of US$1,437M at a discount rate of 5%, an Internal Rate of Return (IRR) of 34.6%; on a post-tax basis the NPV is US$71,089M at a discount rate of 5%, the IRR is 31.0% and the payback period is 2.6 years following commencement of production.

Figure 22.2.1 Cumulative Cash Flow

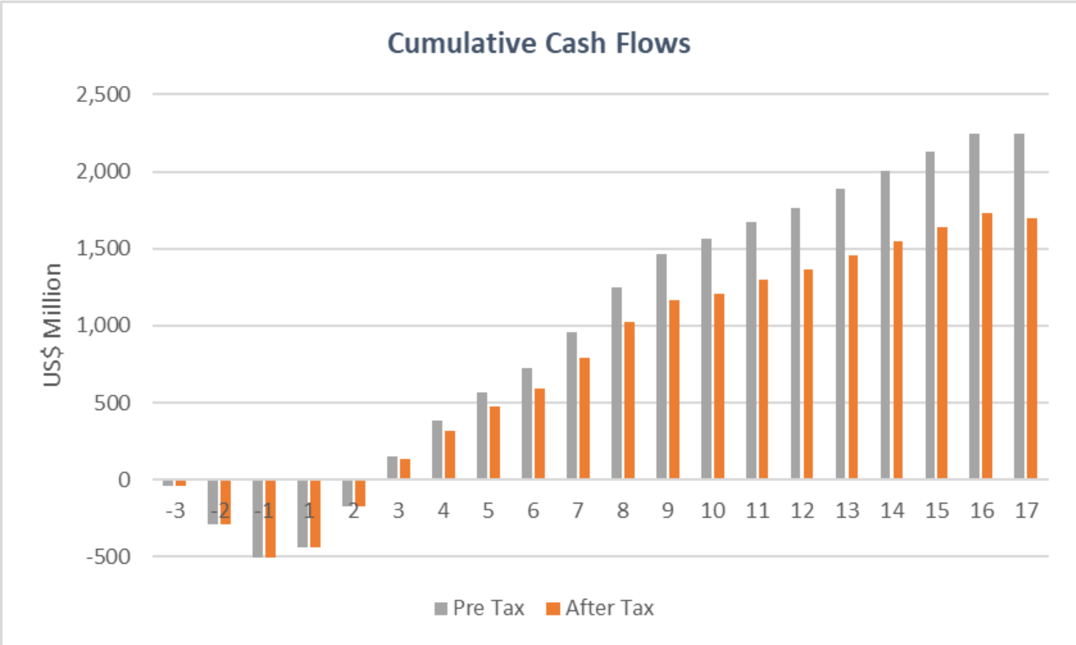


Table 22.2.4 Mine and Process Schedule

Description	Units	Total / Avg	Pre-Production	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Ore Mining																			
South Pit	Mt	161.9	5.7	21.7	18.7	20.7	12.5	13.7	17.4	20.1	27.7	3.6	-	-	-	-	-	-	-
South Pit Grade	g/t Au	0.67	0.69	0.67	0.68	0.71	0.58	0.61	0.62	0.69	0.71	0.82	-	-	-	-	-	-	-
North Pit	Mt	1.8	-	-	-	-	0.2	0.4	0.6	0.6	-	-	-	-	-	-	-	-	-
North Pit Grade	g/t Au	0.47	-	-	-	-	0.52	0.47	0.44	0.48	-	-	-	-	-	-	-	-	-
GB Pit	Mt	10.7	1.80	2.43	2.72	3.74	-	-	-	-	-	-	-	-	-	-	-	-	-
GB Pit Grade	g/t Au	1.43	1.47	1.46	1.28	1.52	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Mineralised Material	Mt	174.3	7.5	24.2	21.4	24.5	12.7	14.1	17.9	20.7	27.7	3.6	-	-	-	-	-	-	-
Total Grade	g/t Au	0.72	0.88	0.75	0.75	0.83	0.58	0.61	0.61	0.68	0.71	0.82	-	-	-	-	-	-	-
Waste Mining																			
South Pit Waste	Mt	162.8	7.9	16.8	20.0	18.3	25.5	24.3	20.6	17.7	11.3	0.5	-	-	-	-	-	-	-
North Pit Waste	Mt	2.1	-	-	-	-	0.5	0.6	0.4	0.6	-	-	-	-	-	-	-	-	-
GB Pit Waste	Mt	40.3	2.9	13.1	12.8	11.6	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Waste	Mt	205.3	10.8	29.8	32.7	29.8	26.1	24.8	21.1	18.3	11.3	0.5	-	-	-	-	-	-	-
Strip Ratio	W:O	1.18	1.43	1.23	1.53	1.22	2.05	1.76	1.18	0.88	0.41	0.13	-	-	-	-	-	-	-
Stockpile Rehandle	Mt	108.9	-	3.1	2.7	1.1	8.5	3.5	0.7	0.7	2.1	9.5	11.0	11.0	11.0	11.0	11.0	11.0	10.9
Processing																			
Oxide	Mt	11.3	-	0.6	0.7	0.8	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
Oxide Grade	g/t Au	0.63	-	1.35	1.18	0.88	0.96	0.96	0.77	0.57	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39
Transition	Mt	7.9	-	2.5	0.5	-	0.0	0.0	0.1	0.0	-	-	4.4	-	-	-	-	-	0.3
Transition Grade	g/t Au	0.63	-	1.03	0.71	-	0.93	0.70	0.50	0.61	-	-	0.41	-	-	-	-	-	0.38
Fresh	Mt	155.1	-	6.4	9.8	10.2	10.3	10.3	10.1	10.3	10.3	10.3	5.9	10.3	10.3	10.3	10.3	10.3	9.9
Fresh Grade	g/t Au	0.73	-	1.28	1.07	1.22	0.85	0.76	0.75	0.91	1.04	0.67	0.46	0.46	0.46	0.46	0.46	0.46	0.46
Total Mineralised Material	Mt	174.3	-	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	11.0	11.0
Total Grade	g/t Au	0.72	-	1.22	1.19	0.86	0.86	0.77	0.74	0.89	1.00	0.65	0.43	0.45	0.45	0.45	0.45	0.45	0.45
Production and Recoveries																			
Gold Production	koz	3,570	-	333.2	334.7	378.3	273.2	244.3	236.3	284.4	322.8	205.1	135.1	137.5	137.5	137.5	137.5	137.5	135.4
Processing Recoveries	%	89.0%	-	89.9%	89.3%	89.5%	90.2%	89.8%	89.8%	90.7%	91.3%	89.5%	88.6%	86.1%	86.1%	86.1%	86.1%	86.1%	85.9%

Table 22.2.5 Financial Model

Description	Units	Total / Avg	Production	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Gold Production and Sales																				
Gold Production	koz	3,570		333	335	378	273	244	236	284	323	205	135	137	137	137	137	137	135	-
Payable Gold Production	koz	3,567		333	334	378	273	244	236	284	322	205	135	137	137	137	137	137	135	-
Gold Price	US$ /oz	-		$1,850	$1,850	$1,850	$1,850	$1,850	$1,850	$1,850	$1,850	$1,850	$1,850	$1,850	$1,850	$1,850	$1,850	$1,850	$1,850	-
Net Revenue	US$M	$6,582		$614	$617	$697	$504	$450	$436	$524	$595	$378	$249	$253	$253	$253	$253	$253	$250	-
Direct Operating Costs																				
Mining	US$M	$1,164		$161	$161	$171	$116	$121	$128	$136	$150	$19	$1	$1	$1	$1	$1	$1	-	-
Road Haulage	US$M	$68		$21	$23	$24	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing	US$M	$1,456		$82	$94	$95	$92	$92	$91	$92	$92	$92	$87	$92	$92	$92	$92	$92	$90	-
Rehandle	US$M	$103		$5	$3	$1	$8	$3	$1	$1	$2	$9	$10	$10	$10	$10	$10	$10	$10	-
G&A	US$M	$171		$12	$12	$12	$11	$11	$11	$11	$11	$11	$10	$10	$10	$10	$10	$10	$9	-
Subtotal	US$M	***$2,962***		***$292***	***$292***	***$303***	***$227***	***$227***	***$231***	***$239***	***$255***	***$131***	***$108***	***$112***	***$112***	***$112***	***$112***	***$112***	***$110***	**-**
Royalties																				
Government Royalty	US$M	$330		$31	$31	$35	$25	$23	$22	$26	$30	$19	$12	$13	$13	$13	$13	$13	$13	-
Royalty on Koné Deposit	US$M	$116		$7	$8	$8	$10	$9	$9	$11	$12	$8	$5	$5	$5	$5	$5	$5	$5	-
Royalty on Gbongogo Main	US$M	$16		$5	$4	$6	$0	-	-	-	-	-	-	-	-	-	-	-	-	-
Community Development Fund	US$M	$33		$3	$3	$3	$3	$2	$2	$3	$3	$2	$1	$1	$1	$1	$1	$1	$1	-
Subtotal	US$M	***$495***		***$46***	***$46***	***$52***	***$38***	***$34***	***$33***	***$39***	***$45***	***$28***	***$19***	***$19***	***$19***	***$19***	***$19***	***$19***	***$19***	**-**
Capital and Closure Costs																				
Pre-Production Capital	US$M	$712	$712																	
Sustaining Capital	US$M	$104		$13	$15	$13	$10	$9	$11	$9	$6	$3	$4	$3	$3	$3	$3			-
Closure Costs	US$M	$62		-	-	-	-	$1	-	$0	-	-	$24	$9	$26	-	-	-	-	$1
Subtotal	US$M	***$877***	***$712***	***$13***	***$15***	***$13***	***$10***	***$11***	***$11***	***$9***	***$6***	***$3***	***$28***	***$11***	***$29***	***$29***	***$3***	***$0***	**-**	***$1***
Project Valuation																				
Net Cash Flow, Pre-Tax	US$M	$2,247	($712)	$274	$264	$329	$229	$179	$160	$237	$289	$216	$95	$111	$94	$120	$120	$122	$121	($1)
$NPV_{5\%}$	US$M	**$1,437**																		
IRR	%	**34.6%**																		
Payback Period	years	**2.5**																		
Net Cash Flow, After-Tax	US$M	$1,700	($712)	$274	$264	$305	$184	$161	$116	$197	$231	$145	$42	$88	$66	$97	$90	$93	$90	($31)
$NPV_{5\%}$	US$M	**$1,089**																		
IRR	%	**31.0%**																		
Payback Period	years	**2.6**																		
Cash Flows																				
EBITDA	US$M	$3,124.4		$287.1	$278.4	$341.9	$238.9	$189.9	$171.6	$245.9	$295.6	$219.0	$122.8	$122.5	$122.5	$122.5	$122.5	$122.5	$121.0	-
Cash Taxes	US$M	$547.5		-	-	$23.7	$44.5	$18.2	$44.0	$39.8	$58.8	$71.2	$53.0	$23.2	$27.4	$23.3	$29.9	$29.9	$30.6	$30.2
Sustaining Capital	US$M	$103.8		$12.8	$14.6	$13.3	$10.2	$9.3	$11.5	$8.6	$6.2	$3.2	$4.0	$2.5	$2.5	$2.5	$2.5	$0.0	-	-
Closure Cost	US$M	$61.6		-	-	-	-	$1.3	-	$0.4	-	-	$23.8	$8.6	$26.3	-	-	-	-	$1.2
Pre-Production Capital	US$M	$712.1	$712.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
After-Tax Net Cash Flow	US$M	$1,699.5	($712.1)	$274.4	$263.9	$304.9	$184.2	$161.1	$116.2	$197.2	$230..6	$144.6	$42.1	$88.1	$66.3	$96.7	$90.0	$92.5	$90.4	($31.5)

22.2.9 Financial Summary

The results of the financial model are summarised in Table 22.2.6.

Revenue generated per domain is shown in Figure 22.2.2. A breakdown of the total cash costs is shown in Figure 22.2.3.

Table 22.2.7 shows the breakdown of the LOM cash costs and unit costs per tonne processed.

Table 22.2.6 Financial Model Summary at US$1,850

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 22.2.2 Revenue Generated per Material Type

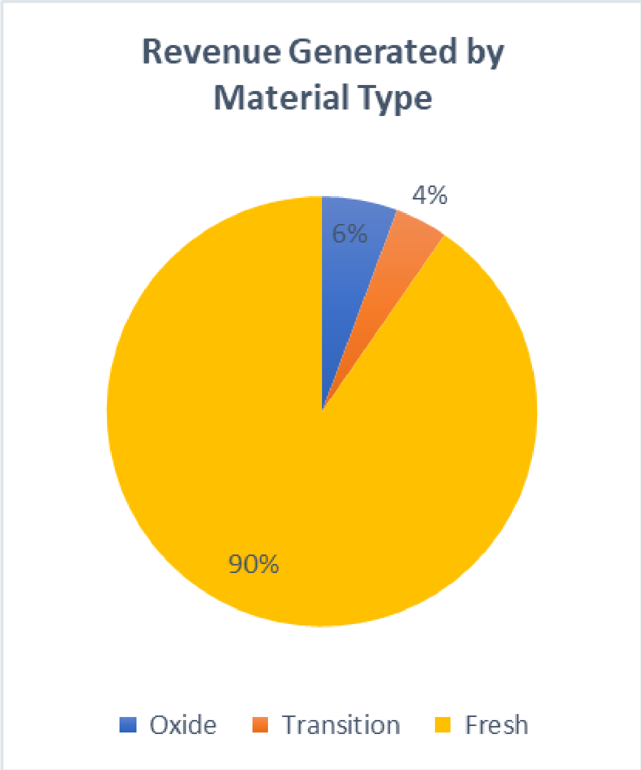


Figure 22.2.3 Operating Expense Split

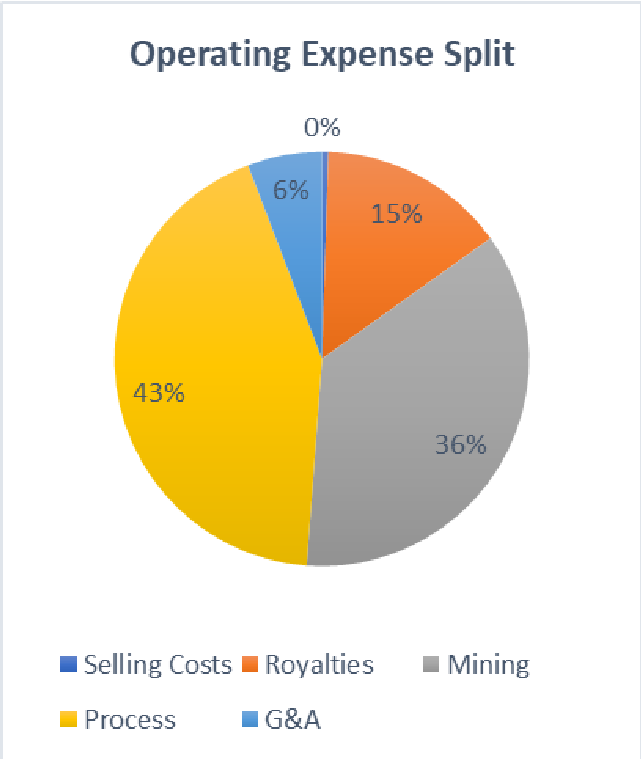


Table 22.2.7 Cash Cost and Unit Cost Summary

Description	LOM US$ /pay oz	LOM US$ /t processed
Mining	326	6.68
Road Haulage	19	0.39
Processing	408	8.35
Rehandle	29	0.59
G&A	48	0.98
Royalties	139	2.84
Total Cash Costs	969	19.83
Sustaining Capital	29	0.60
All-in Sustaining Costs	998	20.42

22.2.10 Single Parameter Sensitivities

Table 22.2.8 shows the changing post-tax $NPV_{5\%}$ 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. The post-tax IRR sensitivity to parameters shows that the NPV is most sensitive revenue / recovery.

Figure 22.2.4 NPV and IRR Sensitivity

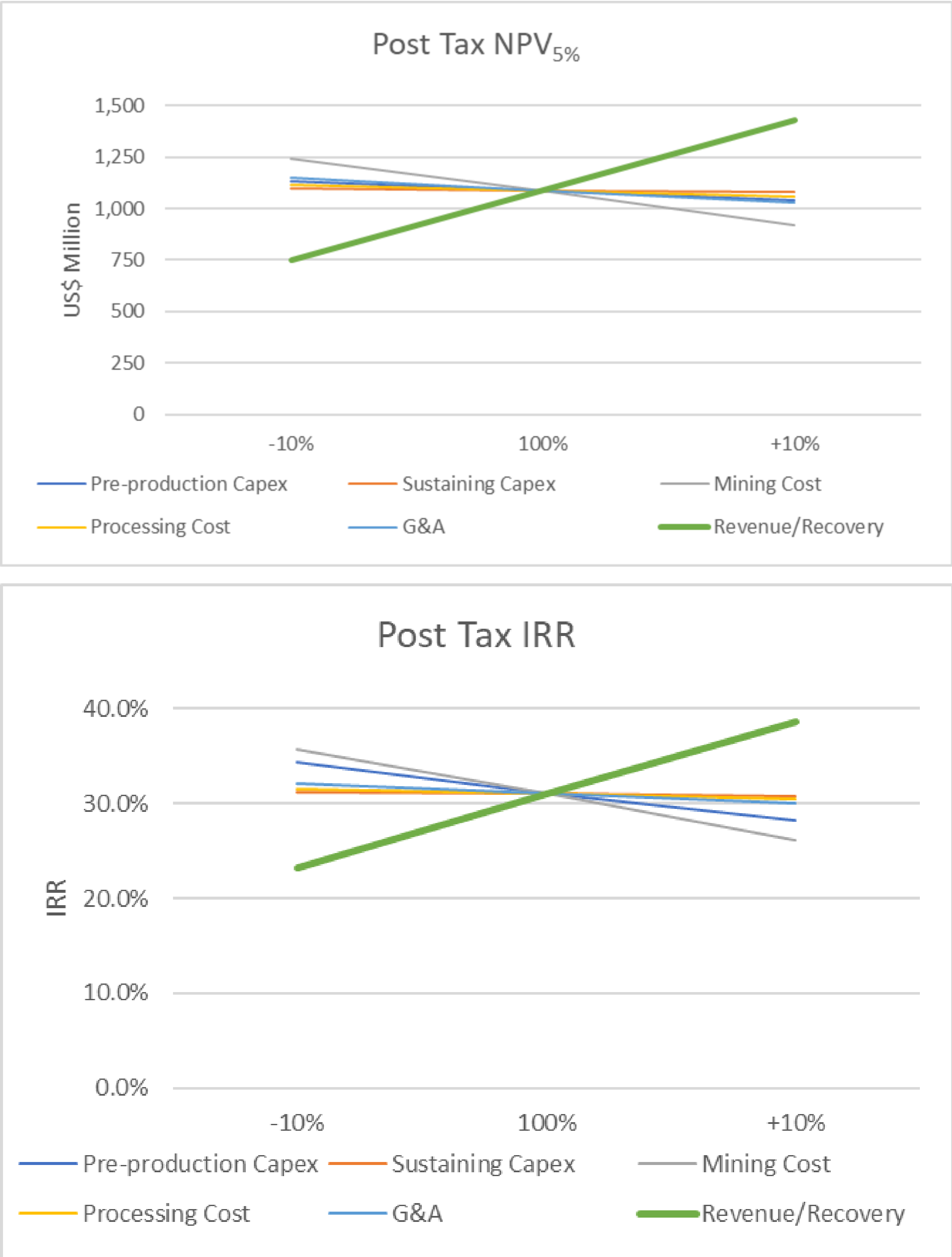


Table 22.2.8 shows the sensitivity of the NPV and IRR with gold price and discount rate.

Table 22.2.8 NPV and IRR Sensitivity

Gold Price	**1,650**	**1,750**	**1,850[1]**	**1,950**	**2,050[2]**
NPV $_{5\%}$	721	906	**1,089**	1,273	1,456
IRR	22.6%	26.9%	**31.0%**	35.2%	39.3%
Cash Cost	954	962	**969**	977	984
AISC	983	991	**998**	1,006	1,013
Payback	3.2	2.8	**2.6**	2.3	2.2

[1]**Three-year trailing average (31 December, 2023).**

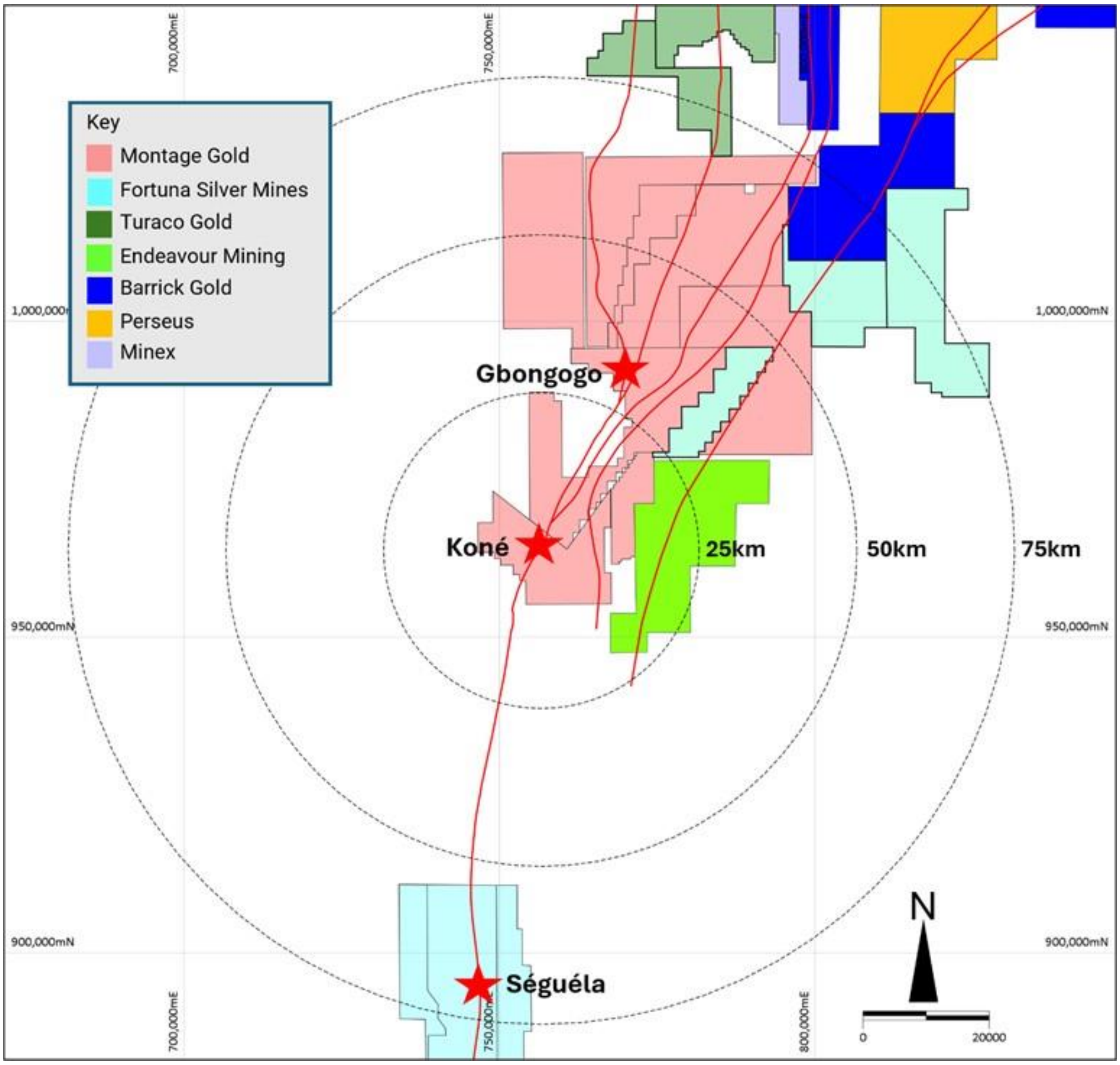
[2]**Spot (31 December, 2023).**

23.0 ADJACENT PROPERTIES

shows tenements held by other owners in the region of the KGP. This figure is derived from the Côte d'Ivoire Ministry of Mines and Geology's mining cadastral (Côte d'Ivoire Ministry of Mines, 2024). Immediately to the north and east of the KGP there are held by Fortuna Silver Mines (Fortuna), Barrick and Endeavour.

Figure 23.1 shows tenements held by other owners in the region of the KGP. This figure is derived from the Côte d'Ivoire Ministry of Mines and Geology's mining cadastral (Côte d'Ivoire Ministry of Mines, 2024). Immediately to the north and east of the KGP there are held by Fortuna Silver Mines (Fortuna), Barrick and Endeavour.

Figure 23.1 Adjacent Properties



Source: Côte d'Ivoire Ministry of Mines, 2024.

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no additional information or explanation required in order to make this report understandable and not misleading.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Assessment Status

25.1.1 Exploration and Drilling

The Indicated MRE for the Koné deposit are based on 102,249 m drilling (54,703.4 m of core and 45,545.3 m of RC) and the Gbongogo deposit was tested by 18,276.3 m drilling (6,119 m of RC and 12,157.3 m of core). The deposits have 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 QPs experience of good, industry standard practise.

The 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 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.

25.1.2 Mineral Resource Estimates

The QP considers that the sample preparation, security and analytical procedures adopted for the Koné drilling provide an adequate basis for the MRE.

The Koné MRE includes an Indicated Mineral Resource of 229 Mt grading 0.59 g/t for 4.34 Moz, and an Inferred Mineral Resource of 25 Mt grading 0.5 g/t for 0.4 Moz, both at 0.20 g/t cut-off. The Gbongogo MRE comprises an Indicated Mineral Resource of 11 Mt grading 1.48 g/t for 0.52 Moz at a 0.50 g/t cut-off. The combined Indicated MRE is 240 Mt, grading 0.63 g/t for 4.87 Moz.

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

25.2 Metallurgical Testing

A comprehensive comminution testwork programme has been carried out over all studies consisting of 68 JK Tech SMC, 68 Bond BWi, 17 Abrasion Work Index and 14 Bond Low Energy Impact tests and 146 leach variability samples using the optimum conditions from the PEA study.

Table 25.2.1 summarises the comminution testwork results. The predominant fresh mineralisation 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 25.2.1 Comminution Testwork

Ore Type	Deposit %	JK Tech SMC A x b			Ball Mill Work Index		Abrasion Index		Crusher Work Index	
		No. of Samples	Relative Density	JK SMC Axb	No. of Samples	Bond BWi kWh/t	No. of Samples	Bond Ai g	No. of Samples	Bond CWi kWh/t
Fresh	83%	52	2.74	31.9	52	11.8	11	0.5	13	16.4
FW Fresh	7%	3	2.77	31.1	3	9.7				
Trans	4%	9	2.69	76.5	9	7.8	4	0.2	1	8.5
Oxide	6%	4	2.54	488.9	4	3.9	3	0.1		
Total	**100%**	**68**	**2.68**	**59.6**	**68**	**33.6**	**18**	**0.4**	**14**	**11.9**

The metallurgical tests included oxide, transition and fresh mineralisation with results indicating that all material types are amenable to direct tank 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 25.2.2 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 (89%), but higher for the less dominant transition (5%) and oxide (6%) zones.

Table 25.2.2 Metallurgical Testwork Summary

# Samples	Deposit	Domain	Processed '000 t	Process g/t Au	Au Recovery %	kg/t NaCN	kg/t CaO
53	South	Fresh	129,510	0.69	89.1	0.22	0.47
13	South	FW Fresh	15,776	0.58	87.7	0.37	0.43
12	North	Fresh	416	0.51	77.1	0.23	0.45
8	GB	Fresh	9,427	1.46	86.1	0.42	0.55
17	South	Transition	6,957	0.60	91.3	0.18	0.99
5	North	Transition	425	0.44	88.0	0.35	0.75
4	GB	Transition	523	1.09	91.2	0.21	1.06
21	South	Oxide	9,628	0.59	93.9	0.13	2.79
9	North	Oxide	943	0.47	93.2	0.13	2.79
4	GB	Oxide	742	1.36	92.8	0.29	2.60
146	**Kone**	**LOM**	**174,345**	**0.72**	**89.0**	**0.24**	**0.62**

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

25.3 Mining

The mining of the Koné and Gbongogo deposits have been shown to be technically feasible through conventional open pit mining methods and are economically viable under the assumed economic and physical parameters. Using available geotechnical information and a series of pit optimisations and mining schedules, the study has shown that the Project can support an 11 Mtpa processing plant for 16 years. The crusher feed under this scenario is 174.3 Mt at a gold grade of 0.72 g/t. This is comprised entirely of Indicated material.

By implementing an elevated cut-off grade strategy, processing material above 0.65 g/t and stockpiling lower grade material, mining is completed in 8.4 years, during which the high grade material is processed. The stockpiled low grade material is processed after the completion of the open pit mining, which enhances the NPV of the Project considerably.

25.4 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 mineralisation 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 P_{80} grind size 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 the national grid.
- Sufficient automated plant control to minimise the need for continuous operator interface but allow manual override and control if and when required.

25.5 Tailings Storage

The design of the 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 analyses and stability analyses 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 the design report, 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 TSF comprises one cell confined by a cross valley embankment. The main embankment will initially be constructed, with the saddle dam constructed later in the mine life to provide sufficient capacity for the first stage of tailings management.

The TSF has been designed to store tailings capacity of approximately 90.8 Mt, which will be generated by the process plant over a period of 8.4 years at a rate of approximately 11 Mtpa after the initial ramp up period. The balance of the tailings, approximately 83.6 Mt, will be stored in the South Pit and will be utilised from Year 8.4 up to end of Year 16. The TSF will be closed and rehabilitated after tailings deposition is transferred to the pit.

Three scenarios were considered in dam break assessment to model failure of the main embankment, saddle dam and water dam. Breach assessment for 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 PAR in the event of failure of the facilities which will define the dam failure consequence categories. As the extent of the high-resolution topography available downstream of the dams was limited at the time of assessment, very conservative estimates of the PAR have been employed in this study and it is conceivable that the consequence category may be reduced (in consultation with the TSF Independent Technical Review Board) when additional topography is acquired and the modelling rerun. The consequence category results are summarised in Table 25.5.1.

Table 25.5.1 Summary of Consequence Category Assessment

Description	ANCOLD Consequence	GISTM Consequence Category
Main Embankment Dam Failure Consequence Category	Extreme	Extreme
Saddle Dam – Dam Failure Consequence Category	Extreme	Extreme
TSF – Environmental Spill Consequence Category	Extreme	N/A
WSF – Dam Failure Consequence Category	High B	N/A

25.6 Hydrogeology

Water will be sourced from the nearby Marahoué River rainfall harvesting and from pit dewatering. Hydrogeological assessment of the river catchment indicates that the river will have flow in excess of total water demand for seven months of the year.

The numerical groundwater model simulations concluded:

- For Koné concluded that pit de-watering will require abstraction in the order of 3,000 to 6,000 m^3/day (35 to 70 L/s).
- For Gbongogo that pit de-watering will require abstraction in the order of 3,000 m^3/day (35 L/s).

Harvested river water and Koné Pit de-watering will be pumped to an off-stream WSF, adjacent to the process plant. Surface run-off from the Koné mining area, ROM pad and stockpiles will gravity flow to this WSF. The WSF will have a capacity of approximately 6.4 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 TSF to the process water pond.

The Gbongogo Pit de-watering will be pumped to the two sediment settlement ponds and then overflow to the Marahoue River.

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 LOM with the proposed WSF, river harvesting, and pit de-watering.

25.7 Power

The Koné plant is estimated to have a maximum demand of 45 MW, an average annual demand of 38 MW with an expected energy consumption of 305 GWh/y.

Power will be supplied by a 23 km 225 kV connection to the National Grid.

25.8 Environment and Permitting

There are currently no objections to the development of the Project. The Project has completed the ESIA to inform designs and environment management plans and submitted the study in December 2023. 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, Manabri and Gbongogo, but is sufficiently remote that environmental impacts on these communities are likely to be minor. Montage provides support to local communities and the Company engages frequently with the local people. Engagement to date indicates that the local community is positive towards the Company. The Company records all contact with local communities through monthly records, including support provided. An EMP 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 ESMP. 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. Records shall be accurately maintained during exploration and development to monitor all activities and engagement. All development programs are under the control and responsibility of a designated qualified representative of the Company and audited to ensure that requirements are met.

Montage is committed to managing the impacts of its operations, in conformance with recognised international best practice. The Project aims to conform to the environmental and social requirements of the IFC Performance Standards, its associated Environmental Health and Safety guidelines, International Council of Metals and Mining, and Equator Principles where they are relevant.

26.0 RECOMMENDATIONS

26.1 Resource Definition

Future resource and definition drilling programs at the Project, consistent with Montage's planned work programme should reflect the following:

- 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 the potential for satellite deposits that could be trucked to Koné with the aim of enhancing the economics of the Project.

26.2 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.

The selection of the preferred mining contractor will be completed in 1Q24 to allow the contractor to meet the mining fleet delivery schedule to enable pre-stripping to commence as scheduled.

26.3 Metallurgical Testwork

The operating parameters and performance guarantees associated with the installation of HPGRs will be confirmed by pilot scale vendor testing in 1Q24.

Materials handling testwork will be carried out in 1Q24 to provide information about the ore handling characteristics for detailed design.

26.4 Hydrogeology

Investigations will be carried out to establish the availability of groundwater supplies in the Project area to reduce the volume of water pumped from the Marahoue River in the initial years. A geophysical survey will be completed to identify potential aquifers that would be investigated by pump tests from boreholes.

A comprehensive water monitoring program will be continued to monitor surface and groundwaters, to add reference data and improve the quantification of the impact of the mine on local water resources.

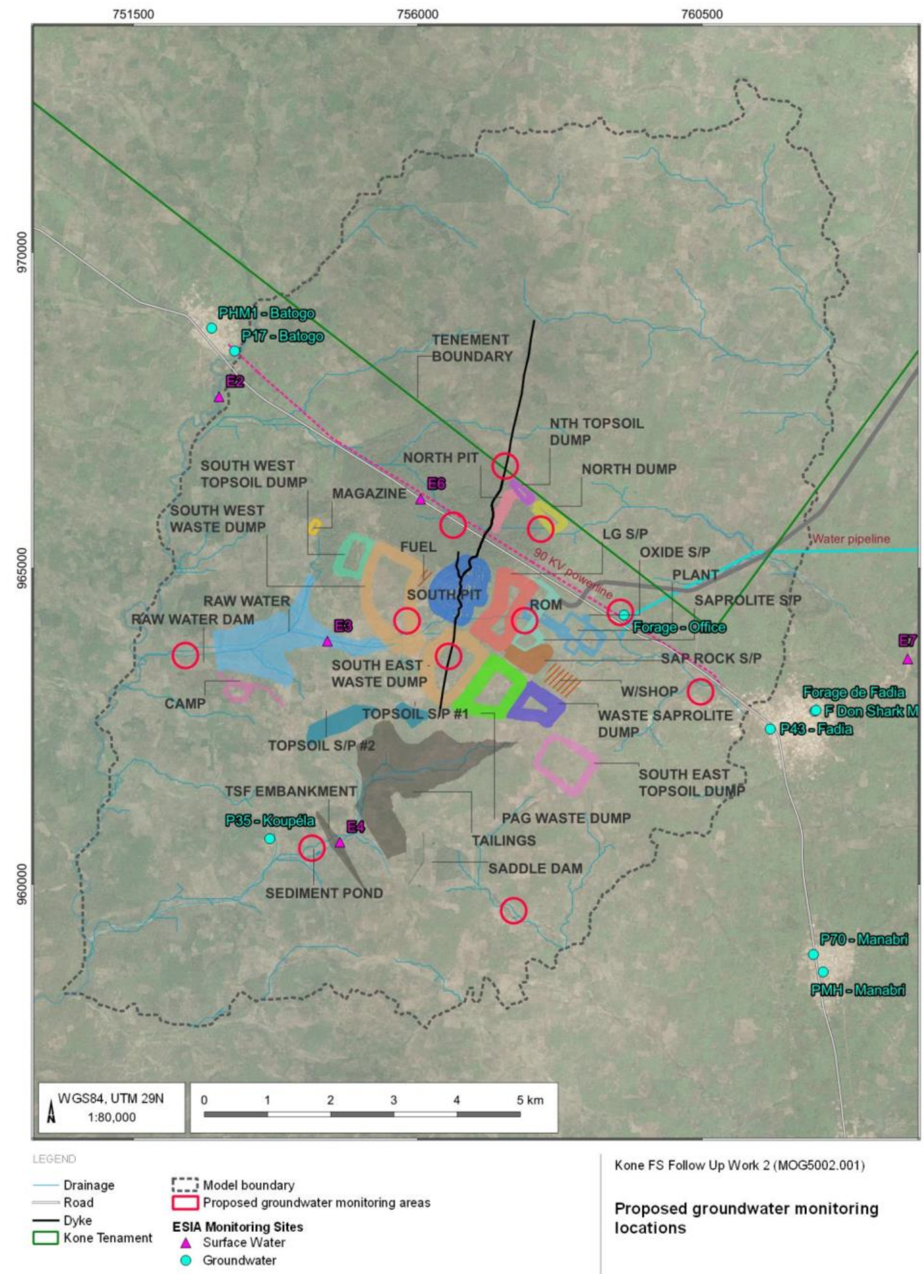
To further develop the understanding of the mobility of cyanide and other elements following tailings deposition in the pit:

- 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.

During operations, a series of additional groundwater monitoring locations are proposed (Figure 26.4.1).

Figure 26.4.1 Koné Proposed Groundwater Monitoring Locations



Source: AGE, 2024.

26.5 Tailings Storage Facilities and Water Management

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

- Expanding topography to include areas potentially impacted by a dam break. Re-run the dam break models as both Newtonian and non-Newtonian flow models.
- Re-assess the consequence category of the TSF based on the outcomes of the dam break models.
- Groundwater investigation to identify additional borefields to reduce the water demand from Marahoue River.
- Update the water balance model based on the finding of the proposed hydrogeological investigations and, if possible, reduce the size of the WSF and/or the instantaneous supply rate required from the Marahoué River
- Detailed design of WSF and Stage 1 of TSF.
- Revise site surface and sediment management designs based on any changes to the site layout which occur as part of the FEED study.
- Update of the design based on the findings of the above investigations.

26.6 Infrastructure

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

- Gbongogo haul road geotechnical investigation.
- Gbongogo haul road bridge geotechnical investigation.
- Gbongogo haul road detailed design.

26.7 Electric Power Supply

Complete design of 225 kV line from National Grid.

26.8 Environmental

The ESIA process has been used to improve the design of the Project, increasing the benefits of the study without incurring excessive costs. To support the implementation of the ESMP, 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.
- Monitoring of impacts to calibrate models, particularly for noise.
- 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.
- Monitoring of effectiveness of resettlement program.

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27.0 REFERENCES

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28.0 QP CERTIFICATES

I **Jonathon Robert Abbott** hereby state:

1. I am a Consulting Geologist, with the firm of Matrix Resource Consultants Pty Ltd, 6/32 Hulme Court, Myaree, WA 6154, Australia.
2. This certificate applies to the technical report with an effective date of 16th January, 2024, titled "Koné Gold Project Côte d'Ivoire Updated Feasibility Study National Instrument 43-101 Technical Report".
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 33 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 28 years, and NI43 101 guidelines for approximately 20 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 Koné Gold Project since July 2018 and visited the project site on the 23rd and 24th August 2018 and 25th to 28th September 2023.
7. I am responsible for Sections 1.7, 12.1, 14, and 25.1.2 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 Koné 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 and "Definitive Feasibility Study, Koné Gold Project, Côte d'Ivoire" with an effective date of 14th February, 2022.
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 15th day of February 2024

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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 16th January, 2024, titled "Updated 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 and a Registered Professional Engineer of Queensland.
4. I graduated in 2001 from the Murdoch University with a Bachelor of Science (Hons) in Mineral Science (Extractive Metallurgy). I have practiced as a metallurgist, metallurgy manager and process engineer continuously since 1996.
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, 1.11.3, 1.14 (except 1.14.2), 1.15, 17, 18.2, 21 (except 21.2.1, 21.2.3 21.3.1), 22 (overview), 25.4, 25.7 and 26.7.
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 "Definitive Feasibility Study, Koné Gold Project, Côte d'Ivoire" with an effective date of 14th March, 2022.
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 15th day of February 2024

Sandra Hunter

AusIMM
THE MINERALS INSTITUTE
CHARTERED PROFESSIONAL
METALLURGY
Sandy Hunter

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Sandra (Sandy) Hunter, FAusIMM(CP), RPEQ

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 16th January, 2024, and titled “Updated 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 43 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, 12.2, 13, 25.3 and 26.3 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 and “Definitive Feasibility Study, Koné Gold Project, Côte d’Ivoire” with an effective date of 14th March, 2022.
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 15th day of February 2024

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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 16th January, 2024 and titled "Updated 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 visited the Project site in January 2023.
7. I am responsible for sections 1.11.2, 1.16.5 (part), 12.4, 16.3, 16.5 (part), 18.1 (except 18.1.7, 18.1.9, 18.1.10, 18.1.11), 25.5 and 26.4
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 and "Definitive Feasibility Study, Koné Gold Project, Côte d'Ivoire" with an effective date of 14th February, 2022.
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 15th day of February 2024

____

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 16th January, 2024, titled “Updated 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 19 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, 25.8 and 26.8.
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 and “Definitive Feasibility Study, Koné Gold Project, Côte d’Ivoire” with an effective date of 14th March, 2022.
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 15th day of February 2024

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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 Croydon Road, Caterham, Surrey CR3 6PA, England.
2. This certificate applies to the technical report with an effective date of 16th January, 2024, and titled “Updated Definitive Feasibility Study, Koné Gold Project, Côte d’Ivoire”.
3. I am a practising Mining Engineer with over 17 years of relevant experience in open pit mining operations, 75 of which have been in open pit gold mines. I have over 23 years mining engineering experience spanning gold mines both underground and open pit.
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 and 10th to 12th July 2023.
7. I am responsible for sections 1.8, 1.9, 1.14.2, 1.16.3, 12.3, 15, 16, 21.2.1, 21.3.1, 25.3 and 26.2. I am independent of the issuer pursuant to Section 1.5 of NI 43-101.
8. I have had prior involvement with the Kone Gold Project as QP for the Technical Report titled “Updated Feasibility Study, Koné Gold Project, Côte d’Ivoire” with an effective date of 14th March, 2022.
9. 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.
10. 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 15th day of February 2024

J. McGrath

Joeline McGrath

I **Timothy Rowles** hereby state:-

1. I am employed as Regional Manager (Queensland), with the firm Knight Piésold Pty Limited, 36 Cordelia Street, South Brisbane, QLD 4101, AUSTRALIA.
2. This certificate applies to the technical report with an effective date of 16th January, 2024, titled “Updated Feasibility Study, Koné Gold Project, Côte d’Ivoire”.
3. I am a current Fellow and Chartered Professional of the Australian Institute of Mining and Metallurgy (No 227249), a current member of the Australian Institute of Geoscientists (No. 8161) and a Registered Professional Engineer of Queensland (No. 10166).
4. 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.
5. I have practised my profession continuously 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 on projects in Côte d’Ivoire, Ghana, Senegal, Republic of Guinea, Democratic Republic of Congo, Mali, Burkina Faso and Cameroon.
6. 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.
7. I personally inspected the project site from the 13th to 16th January 2023.
8. I am responsible for sections 1.11.1, 1.11.4, 1.16.5 (part), 18.1.7, 18.1.9, 18.1.10, 18.1.11, 18.3, 18.4,18.5, 21.2.3, 25.5, 26.5 and 26.6.
9. I am independent of the Issuer pursuant to 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 and “Definitive Feasibility Study, Koné Gold Project, Côte d’Ivoire” with an effective date of 14th February, 2022.
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 15th day of February 2024

Timothy Rowles, B.Sc., M.Sc., FAusIMM (CP), MAIG, RPEQ