PRELIMINARY ECONOMIC ASSESSMENT FOR THE KONÉ GOLD PROJECT

CÔTE D'IVOIRE

NI 43-101 TECHNICAL REPORT

Report Date: 7 June, 2021
Effective Date: 25 May, 2021

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Revision History

Rev	Date	Description
0	June 7, 2021	Issued for Use

Date and Signature Page

The independent Qualified Person (within the meaning of NI 43-101) for the purposes of this report is Dr Geoff Duckworth, representing Lycopodium. Further review as detailed below has been undertaken by Jon Abbott, Mike Hallewell, Pieter Labuschagne, Carl Nicholas, Chris Reardon and Ed Tuplin.

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

The overall report entitled Preliminary Economic Assessment Technical Report, Koné Gold Project, Côte d'Ivoire was collated and signed by the following author:

Signed, this 7th June, 2021.

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1. SUMMARY

1.1 Introduction

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

1.2 Property Description and Ownership

1.2.1 Property Description

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

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

1.2.2 Ownership

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

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

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

1.3 Accessibility, Climate, Local Resources, infrastructure and Physiography

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

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

Three seasons can be distinguished, namely: warm and dry (November to 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.

Power is supplied to the main communities by the national power grid.

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

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

The Project lies within the Guinean forest-savanna ecoregion of West Africa, a band of interlaced forest, savanna and grassland running from western Senegal to eastern Nigeria and dividing the tropical moist forests near the coast from the West Côte d'Ivoireian 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.

1.4 Geology and Mineralization

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

Much of the Project area is covered by duricrust with only very rare outcrop and deep weathering and local geology of the Koné deposit is not yet well understood. Local stratigraphy comprises a moderately westerly dipping sequence of mafic volcanics, which are intruded by an approximately 250m thick package of quartz diorites.

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

1.5 Exploration and Resource Definition

During 2009, 800m by 50m spaced soil sampling and subsequent local infill to 400m by 50m and 200m by 50m 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 December 2020 the Koné mineralization has been tested by 40,700m of drilling (25,545m of RC and 15,155m of core) on which the January 2020 Mineral Resource estimate has been based.

The interpreted mineralization had been tested by generally 100m spaced traverses of generally 50m and rarely 25m spaced holes extending to vertical depths of between 100m and 475m.

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 “inhouse” were limited to immersion density measurements by Company personnel.

Information available to demonstrate the reliability of sample preparation and assaying includes results for coarse blanks and reference standards along with interlaboratory repeat and duplicate assaying.

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

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

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

The author considers that the sample preparation, security and analytical procedures adopted for the 2010 to 2018 Koné drilling provide an adequate basis for the current Mineral Resource estimates and exploration activities.

1.6 Metallurgical Testing

Metallurgical test-work completed on samples of Koné mineralization includes scoping level bottle roll analyses undertaken on three samples of RC chips in 2014. A testwork programme on four composite diamond core samples was completed in 2018. In 2020 a comprehensive testwork programme was carried out on 43 comminution and 39 leach optimisation and variability samples

Table 1-1 shows that 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 1-1 Comminution Testwork

		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

The metallurgical tests included oxide, transition and fresh mineralization with results indicating that all material types are amenable to direct tank (CIP) cyanide leaching. Forecast gold recoveries were estimated based on predicted residue grades for average feed grades, solution loss of 0.01mg/l Au and carbon fines loss of 0.15%. Table 1-2 estimates the gold recoveries based on the average deposit grades, which are good due to consistently low tailings residues grades being observed. Cyanide consumptions are all low to very low and lime consumptions are low for the predominant fresh zone (88%), but higher for the less dominant transition (5%) and oxide (7%) zones.

Table 1-2 Metallurgical Testwork Summary

Oxidation Zone	**LOM % Plant Feed**	**Average LOM Grade Au g/t**	**Forecast Recovery % Au**	**NaCN Consumption kg/t**	**Lime Consumption kg/t**
Fresh	87%	0.67	89.1	0.18	0.22
Transition	5%	0.57	91.1	0.07	1.45
Oxide	8%	0.63	94.8	0.15	1.99

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

1.7 Mineral Resource Estimate

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

The Mineral Resource estimates have been classified and reported in accordance with NI 43-101 and classifications adopted by CIM Council in May 2014. They have an effective date of the 27th of January 2021.

Table 1-3 shows the Mineral Resource estimates for a range of cut off grades. The estimates are classified as Inferred, primarily reflecting the drill hole spacing. The figures in this tables are rounded to reflect the precision of the estimates and include rounding errors.

Estimated Mineral Resources include mineralization tested by generally 100 m spaced drilling traverses. More broadly sampled mineralization is too poorly defined for estimation of Mineral Resources.

Table 1-3 Inferred Resources (Jan 2021)

Cut Off Grade	Mt	Au g/t	Au Moz
0.1	255	0.51	4.18
0.2	**211**	**0.59**	**4.00**
0.3	161	0.69	3.57
0.4	123	0.80	3.16
0.5	95.6	0.90	2.77
0.6	74.1	1.0	2.38
0.7	57.5	1.1	2.03
0.8	44.7	1.2	1.72

Notes

1. The figures in this tables are rounded to reflect the precision of the estimates and include rounding errors.
2. Inferred Mineral Resources are reported in accordance with NI 43-101 with an effective date of January 27, 2021.
3. The Inferred Mineral Resources are reported on a 100% basis and are constrained within an optimal pit shell generated at a gold price of US$1,500/ounce.
4. The identified Mineral Resources are classified according to the "CIM" definitions of Inferred Mineral Resources.
5. The Inferred Mineral Resource statement was prepared by Mr. Jonathon Abbott of MPR Geological Consultants who is a Qualified Person as defined by NI 43-101.
6. Mineral Resources that are not Mineral Reserves do not necessarily demonstrate economic viability.
7. The estimates at 0.2g/t cut-off grade represent the base case or preferred scenario.

1.8 Mining

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

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

Pit optimisations were run using processing cost and recovery data. Mining costs were broken into base and incremental mining costs. Costs were built from first principles using knowledge of recent mining contracts operating under similar conditions in West Africa. 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 90t rigid body haul trucks with suitably sized loading unit.

The Koné deposit will be exploited through a two pits, a smaller northern pit which reaches a depth of 130m and a larger southern pit which extends to a depth of 470 metres deep. The overall strip ratio for the pits is 0.93:1. Based on the assumed mining equipment, a bench height of 5 meters in the oxide, 10 metres in the transition and 15m in the fresh rock was designed, although geotechnical conditions allowed for up to two benches to be excavated between safety berms, within the fresh rock. There may be some opportunity to mine to higher bench heights in areas of bulk waste.

A ramp up period of 18 months was assumed at the start of the schedule the total optimised production tonnage is 35 million tonnes per year. The target for Year 1 was 9.9mt and 11.0 mt of high grade crusher feed respectively, with all subsequent years targeting 11.0 mtpa inclusive of the lower grade stockpile material after year 9. Mining dilution and recovery were not included in the schedule, as these were included in the Resource model.

Table 1-4 shows the annualised mine production schedule with the extraction over nine year period, with an partial prestrip year at the start. Table 1-5 is the processing schedule which indicates the highest grade feed ore is processed first with the remaining stockpiled lower grade material processed after the mining operation has finished.

Table 1-4 Mine Production Schedule

Description	Units	Total/Avg	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Ore Mining												
South Pit Tonnes	Mt	153.3	6.1	15.2	21.4	20.1	14.5	9.8	15.5	16.2	15.1	19.4
South Pit Grade	Au g/t	0.66	0.70	0.70	0.70	0.73	0.58	0.56	0.59	0.62	0.61	0.72
North Pit Tonnes	Mt	7.7	-	-	-	-	1.0	1.4	1.5	1.3	2.5	0.1
North Pit Grade	Au g/t	0.50	-	-	-	-	0.40	0.44	0.47	0.49	0.59	0.75
Total Tonnes Mined	Mt	161.1	6.1	15.2	21.4	20.1	15.5	11.2	17.0	17.4	17.6	19.4
Total Grade	Au g/t	0.65	0.70	0.70	0.70	0.73	0.57	0.55	0.58	0.61	0.60	0.72
Waste Mining												
South Pit	Mt	138.5	7.2	15.3	13.5	14.9	16.3	21.6	15.9	15.2	14.9	3.6
North Pit	Mt	11.1	-	-	-	-	2.0	2.2	2.1	2.3	2.4	0.0
Total Waste	Mt	149.6	7.2	15.3	13.5	14.9	18.3	23.7	18.0	17.6	17.4	3.6
Strip Ratio	w:o	0.93	1.18	1.01	0.63	0.74	1.18	2.11	1.06	1.01	0.99	0.19

1.9 Recovery Method

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, (Primary and Secondary crushing, SABC, CIP). The metallurgical testwork conducted to date, has confirmed that the Koné gold 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 µm.
- Process plant availability of 91.3% supported by the selection of standby equipment in critical areas, reputable western vendor supplied equipment and connection to an onsite LNG fired power station.
- Sufficient automated plant control to minimize the need for continuous operator interface but allow manual override and control if and when required.
- The treatment plant design incorporates the following unit process operations:

- Primary and secondary crushing using a gyratory crusher and two cone crushers to produce a crushed product size P_{80} of approximately 64mm. Due to the high competency of this rock, SAG milling is inherently energy inefficient and secondary crushing has been included in the flowsheet to mitigate this competancy.
- A crushed ore stockpile with a nominal live capacity of nominally 21,000 wet tonnes. Three reclaim apron feeders will deliver ore to the milling circuit via conveyor.
- A grinding circuit configured as a two stage circuit with a SAG mill, two closed circuit ball mills and two recycle pebble crushers (SABC). The circuit will produce a P_{80} grind 75 µm. A single ball mill is not possible as this would exceed the maximum power that is currently achievable using a twin-pinion ball mill.
- Pre-leach thickening to increase the slurry density feeding the leach and carbon in pulp (CIP) circuit to minimise tankage and reduce overall reagent consumption.
- Leach circuit incorporating 14 leach tanks, arranged in two parallel trains of 7 each in series, to provide 36 hours leach residence time, and equipped with external oxygen contacting.
- A Kemix Pumpcell CIP circuit 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 tonne split AARL elution circuit, electrowinning and gold smelting to recover gold from the loaded carbon to produce doré.
- Tailings thickening to recover and recycle process water from the CIP tailings.
- Tailings pumping to the tailings storage facility (TSF).

Table 1-5 shows the annualised processing schedule.

Table 1-5 Mine Processing Schedule

Description	Units	Total/Avg	Year -1	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
Processing																		
Stockpile Rehandle	Mt	73.9	-	1.7	0.8	0.5	0.8	5.4	0.3	0.2	0.4	0.4	11.0	11.0	11.0	11.0	11.0	8.2
Oxide Tonnes	Mt	12.4	-	1.5	1.1	0.5	1.0	0.1	0.3	0.1	0.4	0.4	0.4	0.4	1.6	1.7	1.7	1.2
Oxide Grade	Au g/t	0.56	-	0.98	0.97	0.97	0.61	0.73	0.46	0.43	0.40	0.40	0.40	0.40	0.39	0.39	0.39	0.39
Transition Tonnes	Mt	8.6	-	2.6	0.1	-	0.6	0.2	-	0.1	-	-	1.0	-	-	-	-	4.1
Transition Grade	Au g/t	0.57	-	0.87	1.00	-	0.70	0.60	-	0.72	-	-	0.41	-	-	-	-	0.37
Fresh Tonnes	Mt	140.0	-	5.8	9.8	10.5	9.5	10.7	10.7	10.8	10.5	10.5	9.6	10.6	9.4	9.4	9.3	2.9
Fresh Grade	Au g/t	0.66	-	0.91	0.95	0.96	0.69	0.59	0.67	0.73	0.71	0.95	0.45	0.45	0.45	0.45	0.44	0.42
Total Tonnes	Mt	161.1	-	9.9	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	8.2
Total Grade	Au g/t	0.65	-	0.91	0.96	0.96	0.68	0.59	0.66	0.73	0.70	0.92	0.45	0.45	0.44	0.44	0.43	0.39
Production and Recoveries																		
Gold Production	koz	3,012	-	265.4	307.8	308.3	216.9	183.7	209.3	231.3	219.9	295.7	137.9	138.5	136.7	136.1	132.5	91.4
Processing Recoveries	%	89.4%		91.6%	91.1%	90.8%	89.7%	88.3%	89.1%	89.5%	89.4%	90.6%	86.9%	86.6%	87.1%	87.1%	87.0%	88.8%

1.10 Project Infrastructure

1.10.1 Water Supply

Subject to final approval by government authorities, water will be sourced from the nearby Marahoué river, from pit dewatering and a supplementary borefield. Hydrological assessment of the river catchment indicates that the river will have flow in excess of total water demand for 8 months of the year.

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

Eight hydrogeological exploration boreholes were drilled to determine the preliminary aquifer characteristics at the proposed Koné Gold Project and was focused on the main South pit. Four aquifer tests (pump tests) were conducted and interpreted to derive aquifer parameters for three auifer systems. The aquifer parameters obtained suggest overall low aquifer transmissivity with higher transmissivity associated with fracturing along geological structures.

- The upper saturated weathered and laterite/saprolite zone (between 5 and 40m below surface) - groundwater flow potential is low, and boreholes may have probable range between 0.1 and 0.5 l/sec.
- The transition zone aquifer (Between 25 to 60m) - borehole yields are likely to be between 0.1 and 2.5 l/sec.
- The deeper fractured rock aquifer - groundwater borehole yields are likely to be between 0.1 and >5 l/sec.

The preliminary numerical model simulations concluded that pit de-watering will require abstraction in the order of 2,000 to 6,000 m3/day (23 l/sec to 70 l/sec). The overall mine pit de-watering will be supplemented by perimeter de-watering boreholes that will increase the overall water-make from the mining activities slightly. It is not expected that mining will supply more than 15 to 25% of the total water balance.

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

Harvested river water, pit de-watering and supplementary borefield water will be pumped to an off-stream water storage facility (WSF), adjacent to the process plant. Surface runoff from the mining area, ROM pad and stockpiles will gravity flow to this WSF. The WSF will have a capacity of approximately 3.0Mm^3 and will enable accumulation of water during the wet season and a gradual drawdown in the dry season. In addition, water will be recycled from the tailings storage facility to the process water pond.

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

1.10.2 Power Supply

An evaluation of power supply options for the development of the Koné Gold plant in Côte d'Ivoire has been undertaken. The options considered Diesel, HFO and LNG with varying levels of solar PV and battery energy storage integration. A Build Own Operate Transfer (BOOT) is the preferred commercial arrangement for the power station supply. A LNG/Solar Hybrid power plant has been assessed as the preferred power supply combination.

The Koné Plant is estimated to have a Maximum Demand of 47.4 MW, an average annual demand of 39 MW and an expected energy consumption of 342GWhr/yr.

The up-front capital cost estimate for this LNG/Solar hybrid power station estimated at US$2.5M with annual repayments of $14.04M over 10 years. The annual operating cost is estimated at $0.076/kWhr. The solar PV and Battery Energy Storage Systems integration is expected to save in the order of 50,000 tonnes/year of CO2 emissions compared to the stand-alone LNG power plant. Dedicated hybrid power station control systems will be utilised to optimise the renewable energy yield whilst ensuring the security and reliability of the power supply is maintained at a very high level.

1.10.3 Tailings Storage Facility

The tailings management arrangement comprises two cells separated by a natural ridgeline, with each cell confined by a cross valley embankment. Initially the northern cell (TSF A) will be constructed, with the southern cell (TSF B) constructed later in the mine life to reduce the capital expenditure early in the mine life. Towards the end of the mine life, the two cells will merge into a single facility (Combined Facility), with the confining embankments raised concurrently.

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

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.

1.11 Market Studies and Contracts

No formal market studies have been undertaken. Koné will produce gold doré which is readily marketable on an ‘ex-works’ or ‘delivered’ basis to a number of refineries in Europe and Africa. There are no indications of the presence of penalty elements that may impact the price or render the product unsaleable.

1.12 Environmental

Environmental issues are administered by the Ministry of Environment, Urban Sanitation and Sustainable Development and by the National Environmental Agency (Agence Nationale de L’Environnement (ANDE)).

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

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

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

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

- Ongoing monitoring of wildlife presence in the Project area, such that management measures can be adapted to reflect changing conditions;
- Assessing requirements of each of the classified forest reserves;
- Ongoing community engagement, including information sharing as well as support initiatives and infrastructure development;

- Optimising the energy mix between LNG and solar; and
- Maintaining a grievance procedure to identify and pre-empt potential issues.

1.13 Capital and Operating Costs

1.13.1 Capital Cost

The capital estimate is summarized in Table 1-6 and Table 1-7. The initial project capital cost is estimated at US$489.9 M, including a contingency allowance of US$65.1 M.

Table 1-6 Capital Estimate Summary (1Q21, ±20/-10%)

Main Area	US$'M
Mine	32.0
Process Plant	263.3
Power	2.5
TSF	53.7
Camp	1.5
EPCM	40.2
Owners Costs	31.6
Subtotal	424.8
Contingency	65.1
Grand Total	489.9

The duration of the detailed design and construction phase of the project has been estimated to be 27 months. The site accommodation camp size has been selected, by taking the manning demands for the various overlapping activities into account and the use of local villages.

The total LOM capital cost is estimated at US$934.8M, including sustaining capital costs of US$444.9M, as shown in Table 1-5Table 1-7. The LNG power plant and camp will be financed under a Build Own Operate Transfer (BOOT) contract. The duration of the contracts will be 10 and 5 years respectively.

Table 1-7 Sustaining Capital Estimate Summary (1Q21, ±20/-10%)

Main Area	US$M
Camp	4.5
TSF	205.8
Power	140.4
Process Plant	27.5
Closure	66.6
Grand Total	**444.9**

1.13.2 Operating Cost Mining

Contract open pit mining costs were derived from first principles based on equipment required and include pit and dump operations, road maintenance, mine supervision and technical services cost. The average open pit operating cost (US$ /t mined) is shown Table 1-8.

Table 1-8 Mining Costs

	Mineralized Rock (US$/t)	Waste Rock (US$/t)	Total Rock (US$/t))
Total	**3.19**	**2.58**	**2.90**

A diesel price of $0.75/l was used.

1.13.3 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 bullion for the Project. The battery limits are the ROM feed into the primary crusher (ROM loader by Mining), production of dore in the gold room and discharge of tailings at the TSF.

Operating costs are presented in United States Dollars (US$), to an accuracy of ±35% and are based on pricing obtained during the second quarter of 2021. Process operating costs have been developed for each major domain. Operating costs were developed using the plant parameters specified in the process design criteria. Table 1-9 presents the operating cost summary by material type. In addition to the processing costs LOM rehandle costs equate to $0.40/t processed.

Table 1-9 Operating Cost per Material Type, Main Zone (2Q21, ±20/-10%)

Cost Centre	Fixed US$'000/y	Oxide	Transition	Fresh	LOM
		Variable US$/t	Variable US$/t	Variable US$/t	Fix & Var US$/t
TOTAL	**11,707**	**4.71**	**4.29**	**5.87**	**6.80**

G&A costs have been estimated at $9.4M/yr.

Table 1-10 shows the LOM cash cost and unit cost.

Table 1-10 Cash Cost and Unit Cost Summary (@$1,600/oz)

Description	LOM (AISC $/oz)	LOM ($/t processed)
Mining	289	5.39
Processing	385	7.20
G&A	46	0,86
Royalties	103	1.93
Total Cash Cost	823	15.39
Sustaining Capital	126	2.35
Closure	22	0.41
All-in Sustaining Costs	971	18.15

1.14 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 and refining charges, royalties and capital expenditures (both initial and sustaining). Unless otherwise stated all currencies refer to US$.

The financial analysis used a base price of US$1,600 /oz. The financial assessment of the project is carried out on a "100% equity" basis and the debt and equity sources of capital funds are ignored. No provision has been made for the effects of inflation. Current Côte d'Ivoire tax regulations are applied to assess the tax liabilities. Discounting has been applied mid year from the first year of operation.

The results of the financial model are summarized in Table 1-11. A breakdown of the annualised operating and economic details can be found in Tables 22.4 and 22.5.

Table 1-11 Financial Model Summary @ $1,600/oz

Description	Units	LOM
Feed Tonnage	Mt	161.1
Waste Rock	Mt	149.6
Total Mined	Mt	310.6
Strip Ratio	W:O	0.93:1
Feed Grade Processed (average)	g/t	0.66
Gold Recovery (average)	%	89.4
Gold Production	'000 oz	3,012
Annual Gold Production (average)	'000 oz/y	205
Pre-production Capital Cost	US$M	(490)
Sustaining Capital Cost	US$M	(445)
Total Capital Cost	US$M	(935)
Net Revenue	US$M	4,782
Selling Costs	US$M	(12)
Royalties	US$M	(312)
Total Operating Costs	US$M	(2.167)
Tax	US$M	(352)
EBITDA*	US$M	2,304
$NPV_{5\%}$ After Tax	US$M	652
Cash Cost	US$/oz	827
AISC	US$/oz	975

* EBITDA is a non GAAP financial measure

Table 12 shows the project sensitivity of the NPV, IRR, Cash Cost and AISC with gold price.

Table 12 Project Sensitivity

Gold Price	1,400	1,500	1,540*	**1,600**	1,700	1,800	2,000
$NPV_{5\%}$	359	495	585	**682**	781	1,054	1,294
IRR	16.2%	24.7%	23.1%	**25.9%**	36.1%	36.6%	43.2%
Cash Cost	810	821	819	**823**	851	857	869
AISC	953	969	963	**966**	999	1,001	1,102
Payback	4.9	3.2	3.0	**2.8**	2.5	2.2	2.0

*** Three year trailing average (May 18, 2021)**

1.15 Recommendations

1.15.1 Environmental

Montage has developed and implemented an environmental and social monitoring plan, including appropriate sampling procedures. The objective of monitoring is to characterise environmental conditions, including surface and groundwater, air quality (specifically airborne dust) and ecology. Monitoring should continue through the life of the Project to observe any changes in the environment. This information will be used to inform the EIA and the environmental management of the Project. Data will support action levels and response plans for future monitoring of construction, operation and closure phase impacts.

1.15.2 Geology

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

The proposed work program would include 12,500 of RC and 37,990m of core drilling with an estimated cost of US$10.6M.

1.15.3 Mining

The next phase of work will cover:

- Proceed with a budget tender exercise for the mining contract to confirm the assumptions in mining costs.
- Review geotechnical information and complete additional geotechnical investigations (including drilling).
- Complete hydrogeological study in conjunction with geotechnical work to confirm the effects of groundwater on both wall angles and operating costs due to dewatering.
- Complete waste dump and haul road design to allow for more accurate estimate of haulage requirements.

1.15.4 Metallurgical Testwork

The next phase of testwork will require the following flowsheet development activities:

- Additional comminution testing to increase the variability sampling particularly on oxide mineralisation.
- Further variability testing, specifically on low grade samples to verify metallurgical response at the lower grades at site ambient temperature and design DO conditions.
- Further metallurgical testing to evaluate the potential for the addition of a gravity stage.

1.15.5 Process

The flowsheet incorporates primary crushing followed by open circuit full secondary crushing prior to a SAG, Ball Mill, pebble crushing circuit. It is recommended that, in the next study phase, sensitivity analysis be conducted on SAG mill pebble extraction rates to determine the impact on the pebble crushing circuit.

The next study phase should consider alternative options for feeding oxide ore when a better definition of the split between saprolite, saprock and transition like oxide material is available to ensure the best flowsheet option is incorporated.

1.15.6 Infrastructure

Water

The hydrogeological and environmental geochemistry components to undertake PFS level hydrogeological characterization for both water supply and pit de-watering purposes will be in the order of US$250k.

The following work requirements will be required for the FS:

- Test boreholes should continue to be monitored for groundwater level monthly and groundwater quality on a quarterly basis
- Three additional deep test boreholes should be drilled, two at the north pit and one at the south pit to a depth of at least 180m to 250m. A 4th intermediate depth borehole should be drilled east of GT005 to a depth of 100m. The locations are selected based on the identification of areas where groundwater was intersected by exploration RC drilling and diamond core geotechnical drilling. The boreholes will be pump tested to derive aquifer parameters and yield potential.

- The numerical groundwater model will be updated, and the de-watering system revised for project infrastructure and outlay.

Tailings Storage Facilities and Water Management

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

- Expanding topography to include areas potentially impacted by a dam break.
- Sterilisation of infrastructure footprints.
- Site inspection visit by KP project manager, COVID-19 permitting.
- Expansion of geotechnical investigation to include TSF areas, WSF and river abstraction location.
- River flow monitoring at proposed abstraction point.
- Stream gauging in WSF and TSF drainage lines to verify runoff co-efficients.
- Further groundwater assessment and verification of pit dewatering volumes.
- Probabilistic water balance model.
- Update of the design based on the findings of the above investigations.

Electric Power Supply

Further optimisation of the hybrid LNG/Solar/BESS power station is recommended during the next phase to minimise the overall cost of energy over the life of mine. Further options for the LNG supply chain are also to be explored in the next phase, including gas storage and back-up power options.

2. INTRODUCTION

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

The Project is envisaged to comprise open pit mining operations with the process plant, water storage dam and tailings storage facility located near the pit.

2.1 Basis of Technical Report

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

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

- Jonathon Abbott (MPR Geological Consultants Pty Ltd), responsible for report Sections: 1.7, 12 and 14.
- Geoff Duckworth (Lycopodium Minerals Pty Ltd), responsible for report Sections: 1.9, 1.10.2, 1.13, 17, 18.2, 21, 22 (overview), 25.6 and 26.5.3.
- Michael Hallewell (MPH Minerals Consultancy Ltd), responsible for report Sections: 1.6, 1.15.4, 13, 25.3 and 26.4.
- Pieter Labuschagne (GCS), responsible for report Sections: 1.10.1, 1.15.5, 18.1, 25.5 and 26.5.1.
- Carl Nicholas (Mineesia Ltd), responsible for report Sections: 1.12, 1.15.1, 20, 25.7 and 26.6.
- Chris Reardon (Orca), responsible for report Sections: 1.8, 1.15.3, 16, 21.3, 21.5, 25.4 and 26.3.
- Ed Tuplin (Knight Piésold Pty. Ltd.) responsible for report Sections: 1.10.3, 1.15.5, 18.1.9, 18.1.10, 18.1.11, 18.3 and 25.5.2.
- Other sections have been provided by the Company.

2.2 Property Inspections by QPs

A summary of the QP site visits is detailed in Table 2.1.

Table 2-1 Summary of QP Site Visits

Qualified Person	Site Visit
Jonathon Abbott	23/08/18 – 24/08/18
Carl Nicholas	13/03/21 – 18/03/21

2.3 Effective Dates

The Effective Date of this report is 25 May, 2021. 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

a	annum
AAS	Atomic Absorption Spectrometry
ANDE	Agence Nationale de L'Environnement
As	Arsenic
Au	Gold
BOO	Build Own Operate
BOOT	Build Own Operate Transfer
CA	Concession Agreement
CAE Fusion	Geological Data Management System
CIL	Carbon-in-Leach
CIP	Carbon-in-Pulp
CRM	Certified Reference Material
˚C	Degree Celsius
EITI	Extractive Industries Transparency Initiative
EPL	Exclusive Prospecting Licence
EMP	Environmental Management Programme
ESIA	Environment and Social Impact assessment
F_{80}	80% of a unit process feed particle size is below a given size, based on particle size distribution (PSD)
g	grams
g/l	grams per litre
g/t	grams per tonne
HQ	Exploration drill size (96 mm OD / 63.5 mm ID)
GAT	Gravity Amenability Test
GDMS	Geographical Data Management System
GED	General Exploration Drilling
GPS	Global Positioning System
ha	Hectare
HARD	Half Normal Distribution
HLS	Heavy Liquid Separation
hr	Hour
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
IRR	Internal Rate of Return
IRS	Intact Rock Strengths
JC	Joint Conditions (JC)
JS	Joint Spacing
Km	kilometer
km^2	square kilometres
kV	kilovolt
kWh	kilowatt hour
l	litre
l/s	litre per second
M	million
masl	metres above sea level
MIK	Multiple Indicator Kriging
Min	minutes
Mm^3	million cubic metres
MoM	Ministry of Minerals
mS/m	milli Siemens per metre
Mtpa	million tonne per annum

Mt	million tonne
NPV	Net Present Value
NQ	Exploration drill size (75. 5mm OD / 47.6 mm ID)
oz	31.10348 grams
PSD	Particle Size Distribution
PFS	Pre-Feasibility Study
PIR	Passive Infra-red
ppm	parts per million
PQ	Exploration drill core size (122.6 mm OD / 85 mm ID)
P_{80}	80% of a unit process product particle size is below a given size, based on particle size distribution (PSD)
QAQC	Quality Assurance Quality Control
RC	Reverse Circulation
RF	Revenue Factor
RMR	Rock Mass Rating
ROM	Run-of-Mine
RQD	Rock Quality Designation
SABC	Semi Autogenous Ball mill Crushing
SAG mill	Semi Autogenous Grinding mill
SD	Standard Deviation
SG	Specific Gravity
t	metric tonne (1,000 kg)
TDS	Total Dissolved Solids
TOR	Terms of Reference
TSF	Tailings Storage Facility
TTG	Tonalite-Trondhjemite-Granodiorit
VMS	Volcanogenic Massive Sulphide
WHF	Water Harvest Facility
WSF	Water Storage Facility
µS/cm	micro Siemens per centimetre
µm	micron

3. 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.

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

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

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

Metallurgical Testwork: the Author has relied on information provided by MPH Minerals Consultancy Ltd for Sections 1.6, 1.15.4, 13, 25.3 and 26.4. Lycopodium has reviewed the metallurgical testwork results and concurs with their interpretation.

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

Mining: the Author has relied on information provided by Orca Gold for Sections 1.8, 1.15.3, 16, 21.3, 21.5, 25.4, and 26.3.

Hydrogeology: the Author has relied on information provided by GCS Water and Environmental Consultants for Sections 1.10.1, 1.15.3, 18.1, 25.5 and 26.5.1.

Tailings Storage: the Author has relied on information provided by Knight Piésold Pty. Ltd. for Sections: 1.10.3, 1.15.5, 18.1.9, 18.1.10, 18.1.11, 18.3 and 25.5.2.

Environment and Social: the Author has relied on information provided by Mineesia Limited for Sections 1.12, 1.15.1, 20, 25.7 and 26.3.

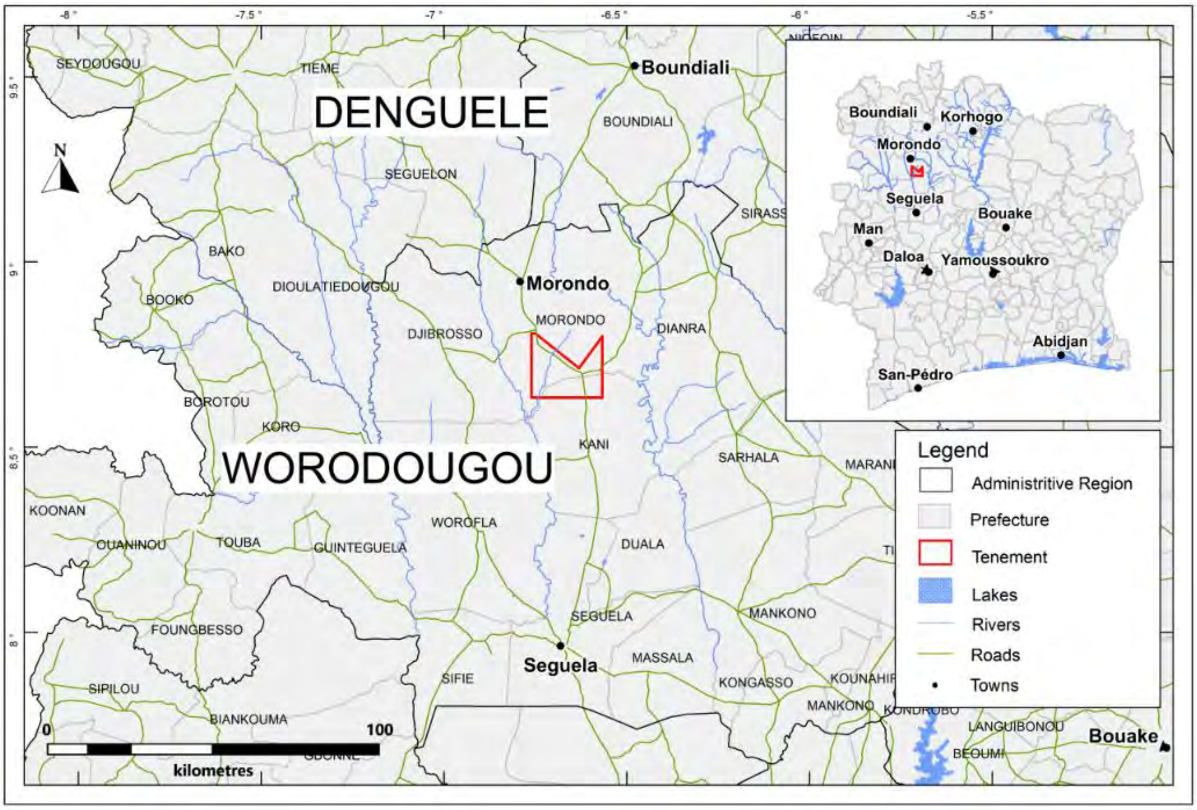
Financial: the Author has relied upon the financial analysis by Montage in Sections 1.13 and 22 of this report. Lycopodium has reviewed the inputs and basis for the financial analysis.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

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

Figure 4-1 Project Location Map



Source Montage

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

4.2 Mineral Tenure

4.2.1 Mineral Tenure Framework

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

- Prospecting Permit - Up to 2,000 km^2, non-exclusive and granted for one year.
- Exploration Permit - Up to 400 km^2, exclusive and granted for 4 years, plus 2 renewals of 3 years with the possibility of a third renewal for 2 years under extraordinary circumstances.
- Mining Licence - Granted for up to 20 years with option of 10-year renewals.

- Semi Industrial Mining Licence - Ivorian nationals or Ivorian majority cooperatives of companies only, up to 1 km, 4-year period, renewable.
- Artisanal Mining Licence - Ivorian Nationals or Ivorian majority co-operatives only, maximum of 25 Ha. 2-year period, renewable.

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

For a company to take a mining licence, the company must form a local entity and the state can take up to 10% free carry in any mining operation and up to 15% with further financial contribution. Mining royalties for gold extraction vary with gold price (Table 4.1).

Table 4-1 Summary of Royalties

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

4.2.2 Project Mineral Tenure and Ownership

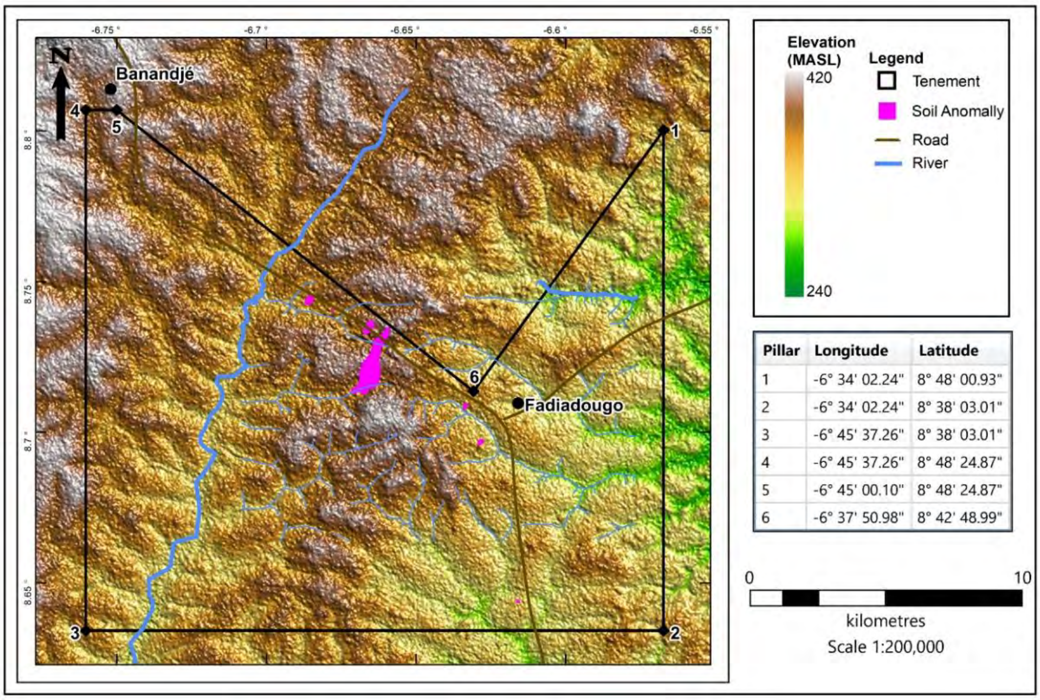
The Koné Exploration Permit number 262 (PR 262) was granted to Red Back on 22nd March 2013 under the 1995 Mining Code. It was renewed in March 2016 and March 2019 under the 2014 Mining Code for three years committing Montage to the expenditure requirements in Table 4.2.

Table 4-2 Exploration Permit Expenditure Commitments

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

The Koné Exploration Permit will expire in March 2022 but can be renewed for a further two years if Feasibility Studies are in progress. Figure 4.2 shows the lease boundary relative to the SRTM elevation along with latitude and longitude of the lease corners.

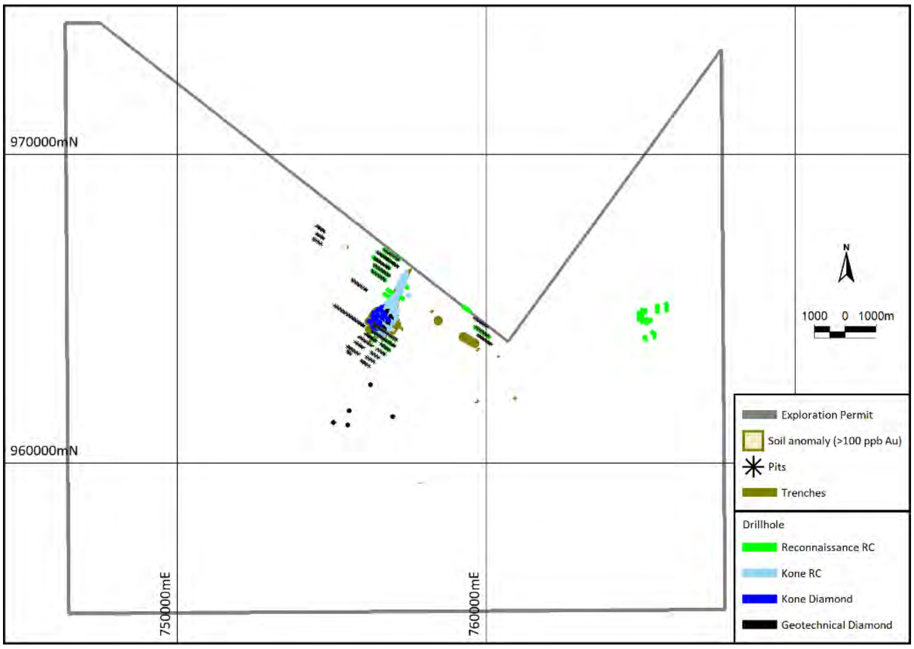
Figure 4-2 Exploration Permit Boundaries and SRTM Elevation

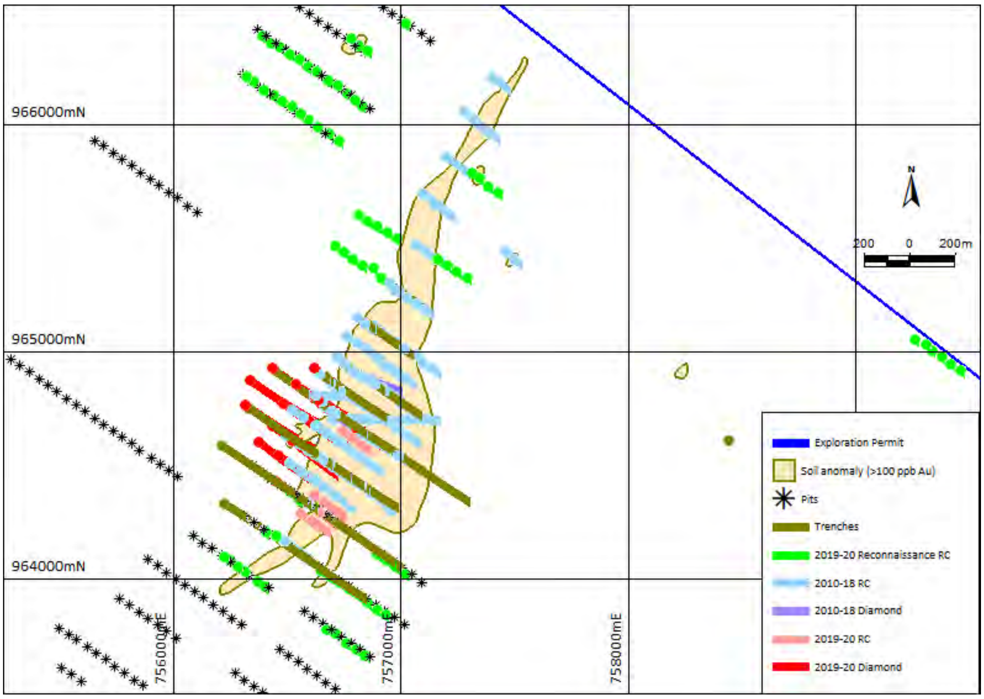


Source: Montage

Figure 4.4 presents the locations of trench and drill hole sampling relative to the soil anomaly and Exploration Permit which is shown as a thick black line. The coordinate system used in this figure and throughout this report is World Geodetic System (WGS84) Zone 29 N coordinates. The Project is centred at around 757,000 mE, 963,300 mN.

Figure 4-3 Soil Anomaly, Trenching and Drilling Locations





Produced by MPR from information supplied by Montage

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

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

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

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

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

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

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

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

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

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

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

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

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

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

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

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, PHYSIOGRAPHY

5.1 Accessibility

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

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

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

5.2 Climate

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

5.3 Local Resources, Infrastructure

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

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

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

5.4 Physiography

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

The Project area is characterized by moderate relief between 200m and 420m above sea level (Figure 4.2, Figure 5.1). The Marahoué and Yarani rivers are the main drainages in the area but the bulk of the project is cut by shallow seasonal drainages that only show significant flow in the wet season.

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

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

Source: Montage

6. HISTORY

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

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

Table 6-1. Field exploration undertaken by previous owners

Activity	Red Back 2009-10	Sirocco 2013-14	Orca 2017-2019
Worldview imagery (km²)	230	-	-
Ground magnetics (km²)	4.68	-	-
Soil samples	4,877	-	-
Rock chip samples	61	2	6
Trenching (number/metres)	9/4,155 m	3/610 m	-
RC drilling (holes/metres)	8/943	43/3,431	64/13,360
Diamond drilling (holes/metres)	-	-	2/527.8

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

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

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

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

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

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

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

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

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

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

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geological Setting

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

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

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

Three main intrusive episodes have been identified:

- Calc-alkaline biotite and amphibole bearing TTG suites, forming large generally elongate and irregularly shaped regions of granitic gneiss that were syn/post tectonically emplaced into the greenstone belts. 2,250 to 2,120 Ma.
- Calc-alkaline potassic granodiorite-granite suites, biotite and K-feldspar bearing with rare amphibole and muscovite, undeformed and sub-circular or elliptical which cross cut older units, but are locally affected by shear zones. 2,120 to 2,090 Ma.
- Undeformed potassic granites, occasionally metaluminous or syenitic with abundant K-feldspar often with a biotite association, amphibole is usually absent. 2,110 to 2,070 Ma.

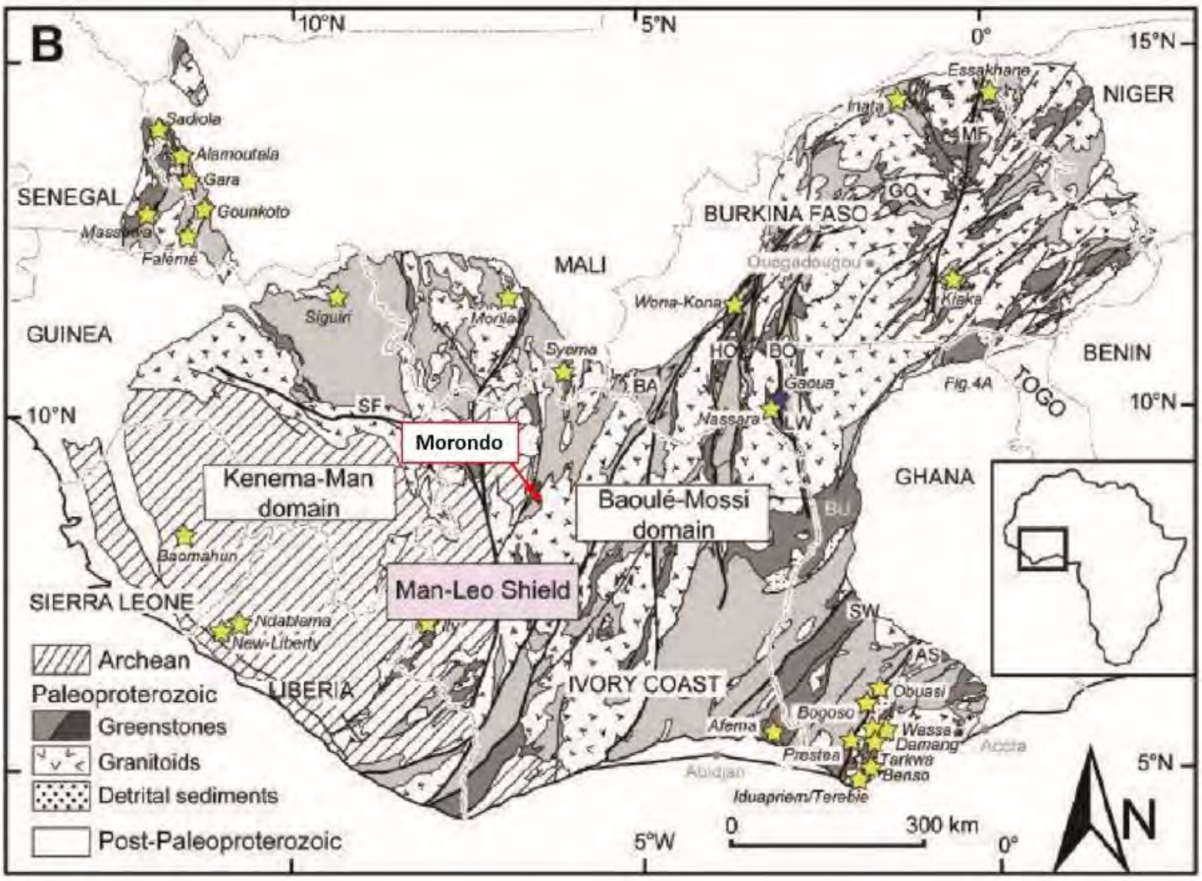
The TTG suites are commonly referred to as 'Belt Type' granites and the potassic suites are referred to as 'Basin-Type' granites reflecting the source and age of the intrusive suites. The TTG suites are derived from melting during subduction and form elongate domes or antiforms between and around the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

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

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

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

Figure 7-1 Geology of the Man leo Shield



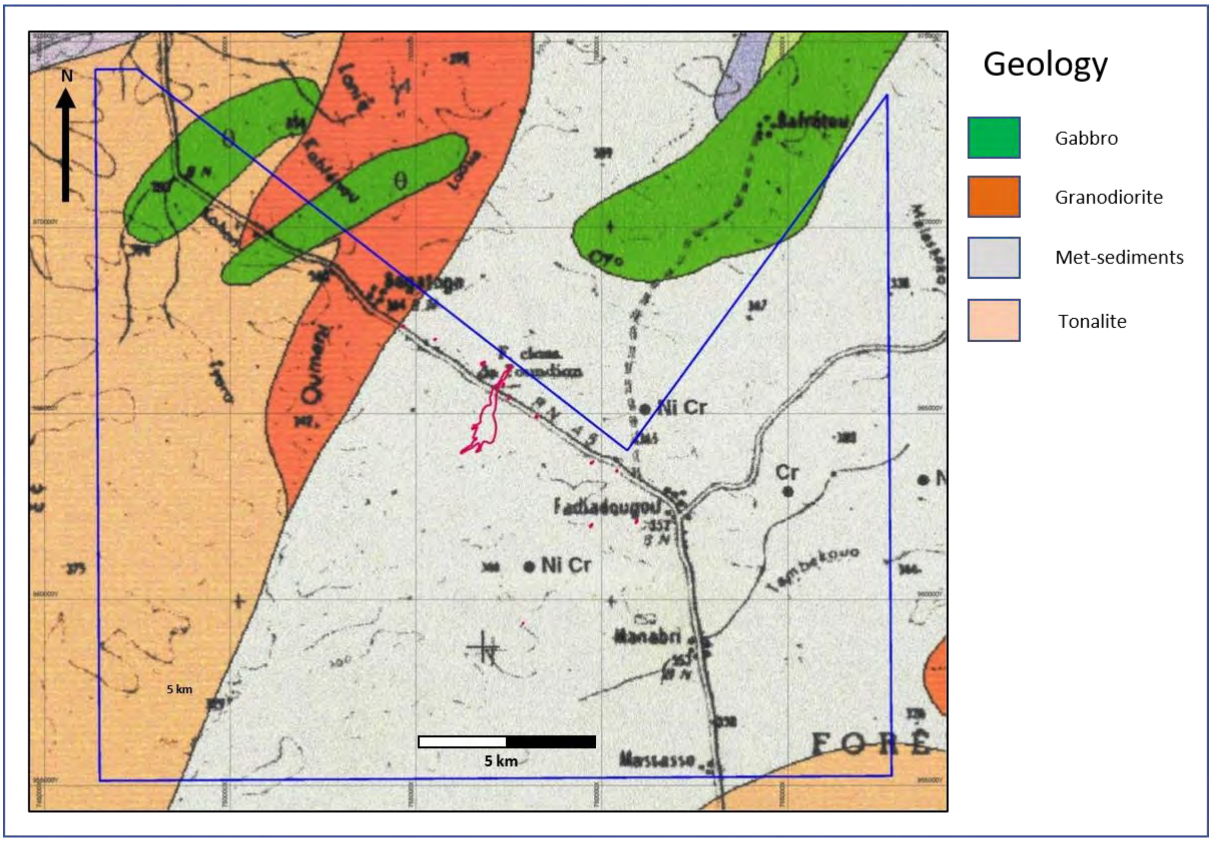
Base Map modified form Goldfarb et al, 2017 Source: Montage.

7.2 Koné Exploration Permit Geological Setting

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

Regional mapping indicates the project overlies Birimian sediments, volcaniclastics and volcanics flanked to the west by basement tonalite and diorites (Figure 7.2). The rocks have been metamorphosed to greenschist facies. Regional aeromagnetic data shows strong north east – south west trends interpreted to reflect the distribution of underlying rock units.

Figure 7-2 Geological Map of the Koné Exploration Permit



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

7.3 Koné Deposit Geological Setting and Mineralization

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

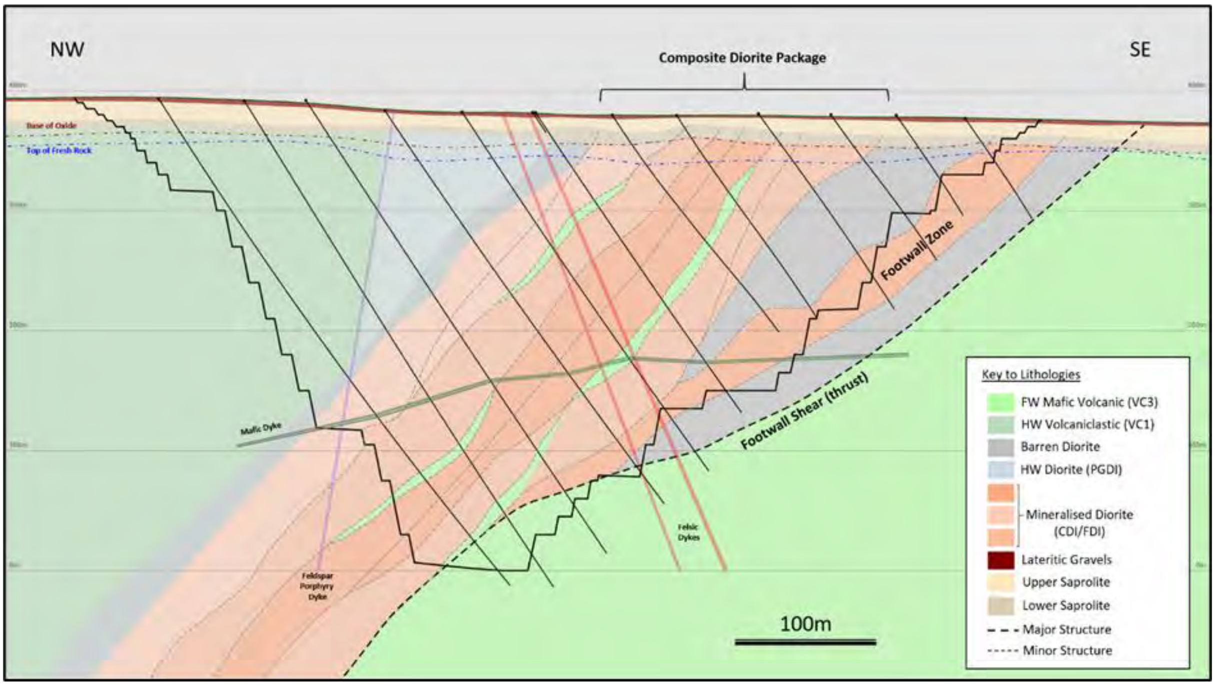
The diorite bodies at Koné have intruded into the contact zone between two different sequences of mafic volcaniclastic rocks which form the hangingwall and footwall of the deposit. The diorite intrusions have been age dated to 2168 ± 5 Ma indicating they were emplaced during the later stages of the Eburnian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

The hangingwall volcaniclastics are 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 5-20m wide zone of shallowly plunging/horizontal tight folds can be observed within volcanic rocks immediately below the footwall contact of the diorite. This folding progressively decreases as you move outward from the contact into the footwall volcanic sequence. The upper contact of the diorite displays little to no deformation and has no mineralisation related to it.

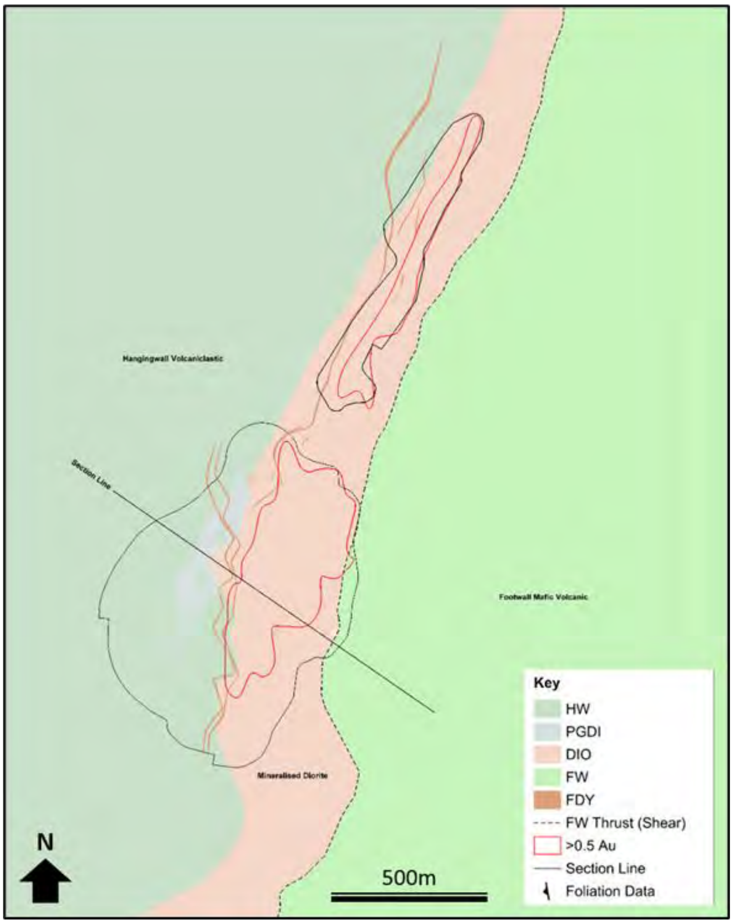
Within the diorite, higher gold grades (>1g/t) are associated with swarms of foliation parallel, 1-2mm quartz + pyrite ± chalcopyrite veinlets which form distinct corridors of mineralisation. Recent observations from drill core indicate these higher grades are related to discrete zones of more intense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

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



Source: Montage

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



Source: Montage

Regional metamorphism in the Boundiali belt is greenshist 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.

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 7-5).

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

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

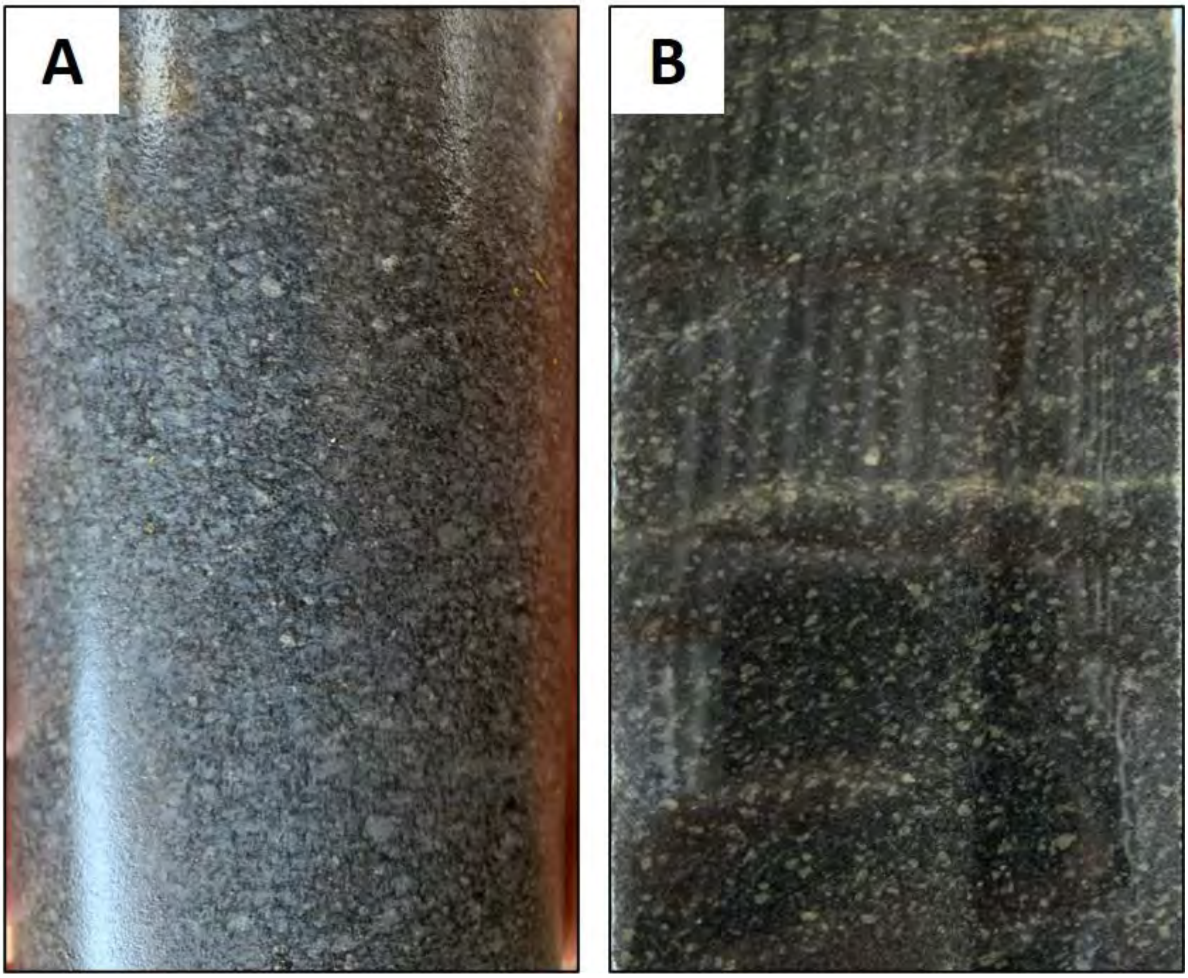
Sharp contact between two diorite bodies highlighted by appearance of porphyritic texture in MDD017 at 214.15m depth. The contact is dipping 53° towards 279° (striking 009°). Numerous observations of this nature indicate a complex diorite package with stacked diorite bodies Source: Montage

7.3.1.1 Coarse Grained Diorite (CDI)

Diorite with up to 2mm grain size composed of domains of fine plagioclase ± quartz and domains of mafic minerals – predominantly biotite. This lithology is moderate/strongly magnetic but in localised patches/zones in the core. The main textures observed under this code are porphyritic, caused by albitised plagioclase phenocrysts and equigranular texture highlighted by plagioclase crystals (Figure

7-6). This unit hosts gold mineralisation, associated with 1-2mm, foliation parallel sulphide bearing quartz veinlets and disseminated pyrite due to the brittle nature of deformation compared to more ductile deformation seen in other units.

Figure 7-6 Coarse Grained Diorite



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

7.3.1.2 Fine Grained Diorite (FDI)

The fine diorite unit is very closely related to the coarse diorite (Figure 7-7). Its composition is the same as the coarse diorite and locally it is simply a finer grained, recrystalised 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 often 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 1-2mm sulphide bearing veinlets and disseminated pyrite.

Figure 7-7 Fine Grained Diorite

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

7.3.1.3 Black Siliceous Diorite (BSD)

One distinctive fine grained diorite is observed. Characterised by its' very fine crystal size, dark grey/black colour and siliceous nature (Figure 7-8). This unit contains abundant magnetite and often hosts quartz-sulphide veinlets containing 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 7-8 Black Siliceous Diorite

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

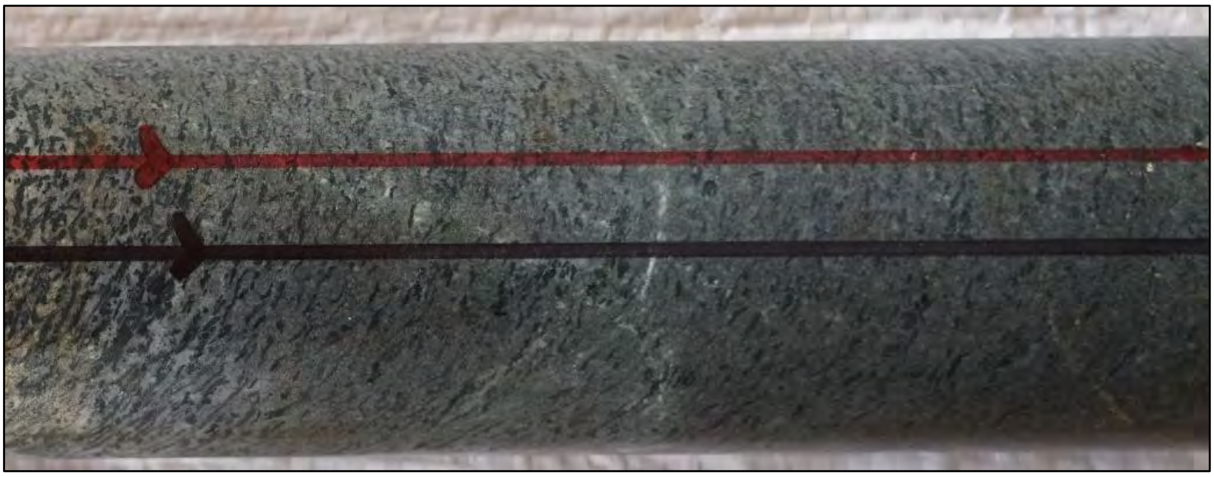
7.3.2 Hanging Wall Geology

7.3.2.1 PGDI – Pale green Diorite

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

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

Figure 7-9 Pale green Diorite



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

7.3.2.2 Mafic Volcaniclastic type 1 (VC1)

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

Figure 7-10 Mafic Volcaniclastic type 1

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

Figure 7-11 Mafic Volcaniclastic type 1

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

7.3.3 Footwall Geology

7.3.3.1 Mafic Volcaniclastic type 3 (VC3)

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

Figure 7-12 Mafic Volcaniclastic type 3

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

7.3.4 Mineralisation

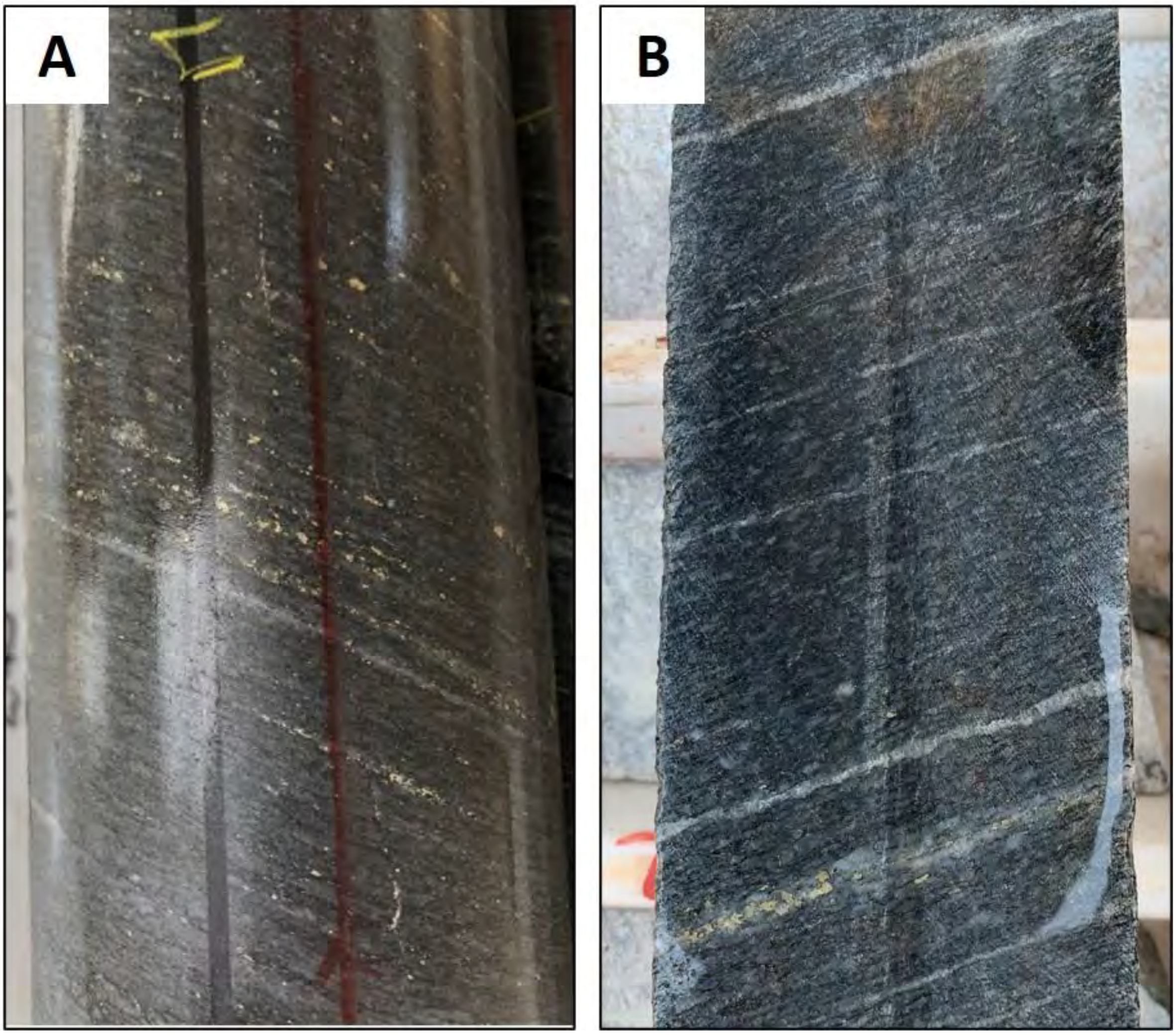
The Koné Gold deposit is hosted by the composite diorite body and is characterised by very large mineralised true widths, averaging >200m over the southern half of the deposit, with a maximum of up to 330m (MDD015B, 330.7m grading 0.58g/t) i.e. almost the entire width of the composite diorite body.

Higher grades (1-1.5 g/t) are associated with high density "swarms" of 2-5mm thick, foliation parallel (figure 7.14), translucent white to smoky quartz veinlets containing fine grained sulphide (Figure 7-13). Lower grades are related to dissminated 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 7-15). The volcaniclastic rocks in the footwall of the thrust have deformed plastically whereas the more rheolgically competent diorite has developed brittle/ductile shears. This is expressed in drill core as brecciated zones of diorite, associated slivers of internal volcanic rocks, and localised shears with very localised folds of foliation within the diorite and footwall volcanic (Figure 7-16 and Figure 7-17).

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

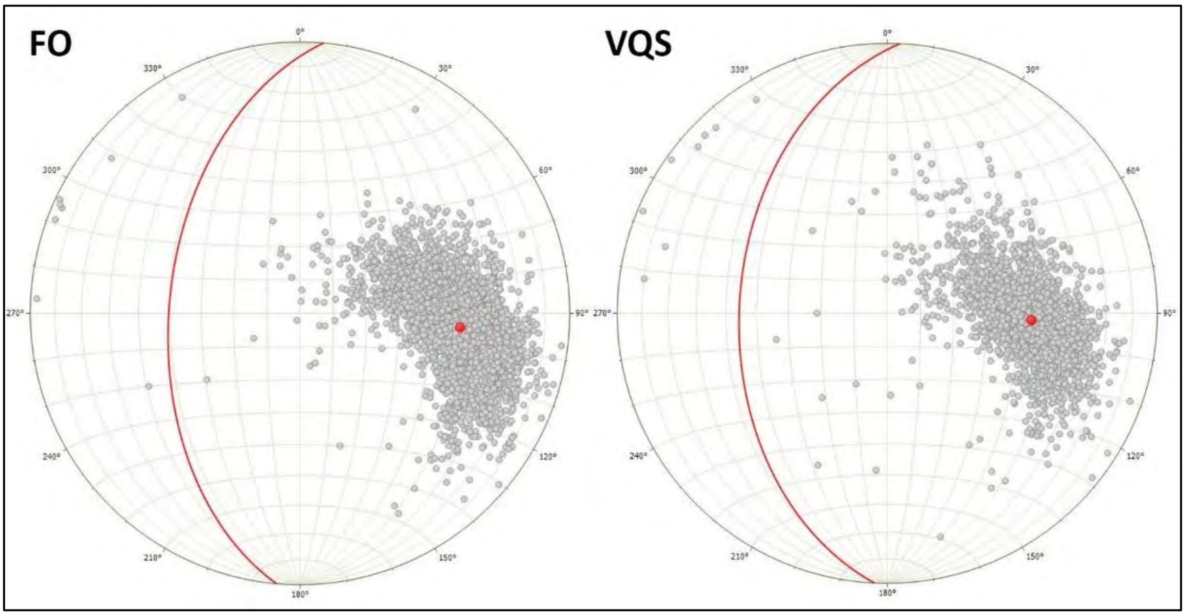
Figure 7-13 Mineralised Foliation & VQS vein swarm



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

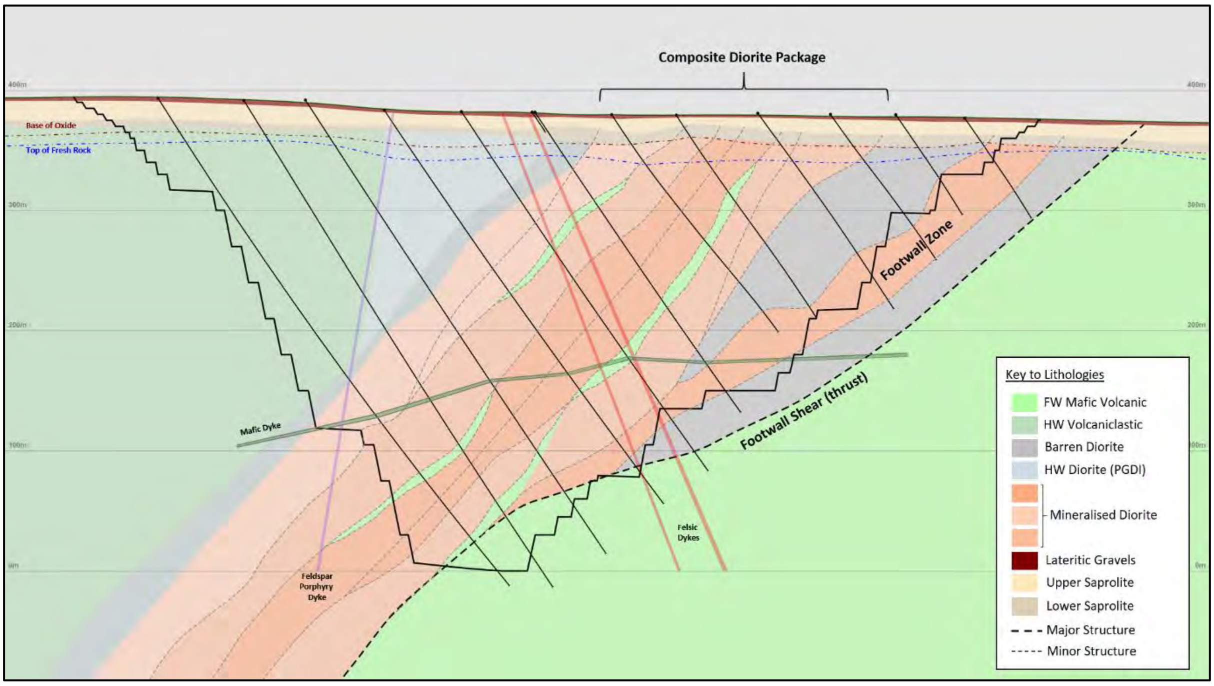
A phase of barren pyrite is present in the hanging wall that has a 1-2mm grainsize and euhedral cubic in form. Pyrite related to the mineralisation is either <0.5mm disseminated globules of ultrafine pyrite that appear to be replacing magnetite locally, or internally with the VQS veins.

Figure 7-14   Foliation orientations



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

Figure 7-15   Section through the centre of the deposit



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

Figure 7-16 Ductile shear within diorite

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

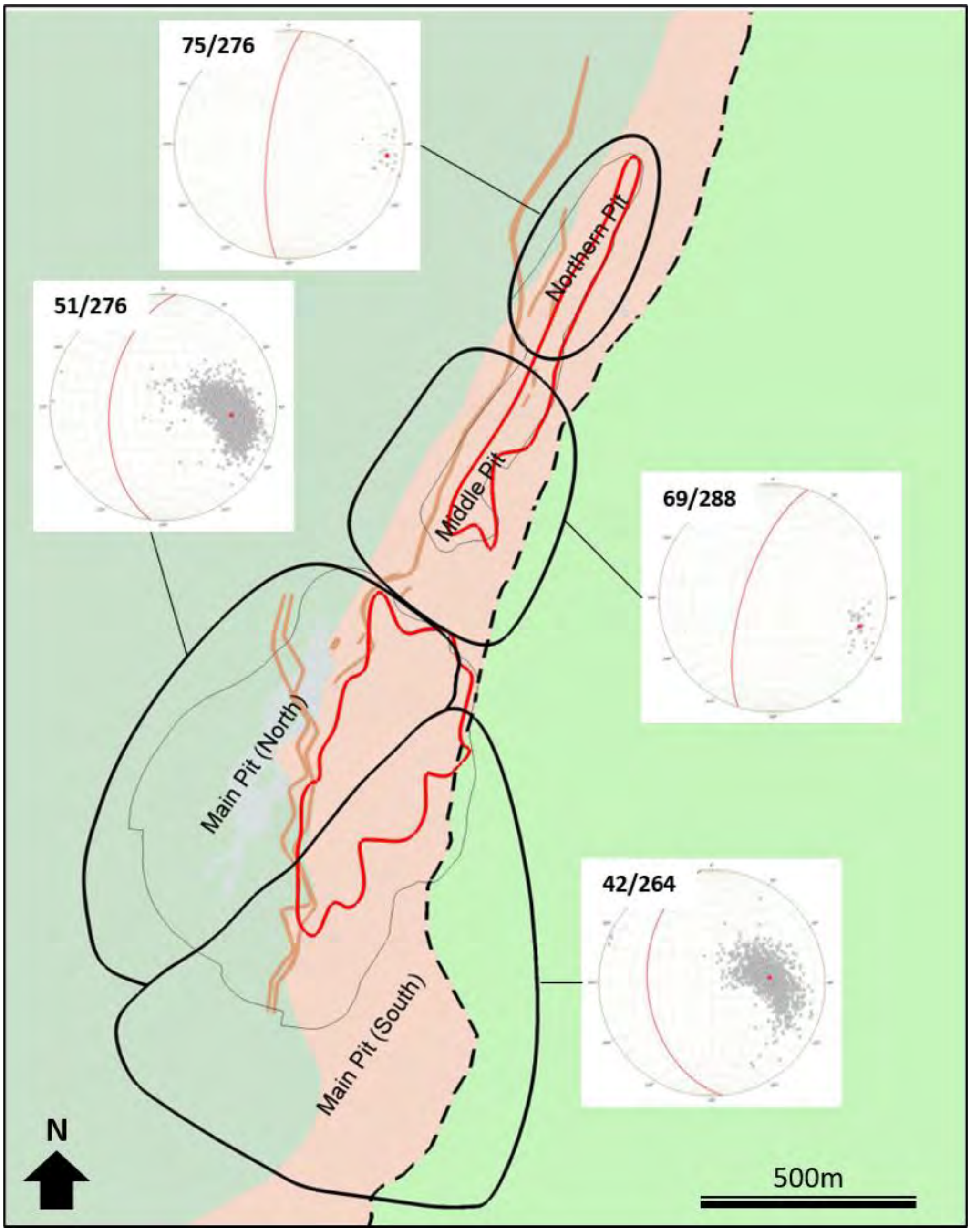
Figure 7-17 Buckle folds within the diorite

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

7.3.5 Structure and Deformation

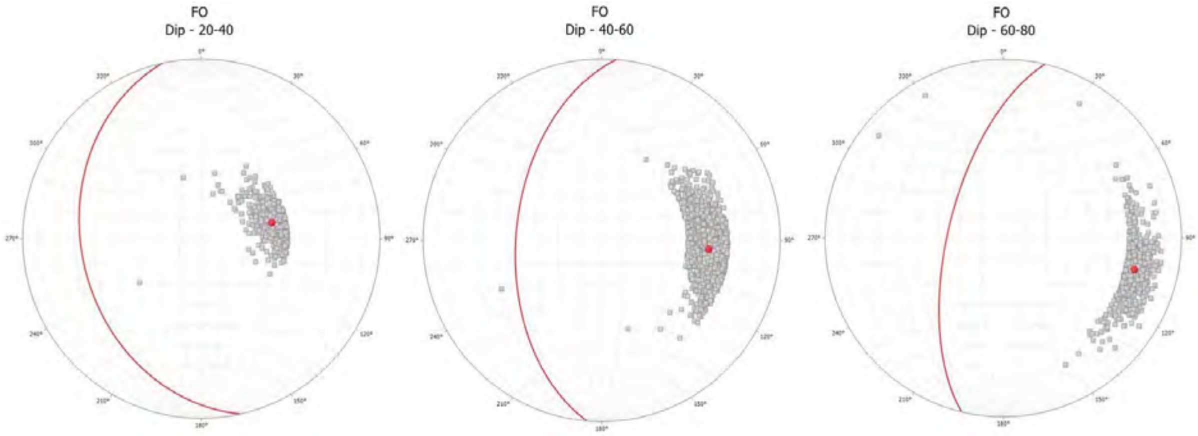
A correlation between the dip of the foliation and the strike of the host rocks can also be observed (Figure 7-19), with foliations dipping 20-50° averaging a strike of ~348° in the southern part of the deposit and and dipping 60-80° striking ~014° in the northern sector (Figure 7-16). This change in 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-south-east (Figure 7-19).

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



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

Figure 7-19 Stereo-nets of foliations



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

7.3.6 Dykes

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

7.3.6.1 Early Green Dykes (EGD)

Intermediate to mafic foliated dykes displaying strong chlorite alteration with local biotite alteration/metamorphism related to regional greenschist/amphibolite metamorphism (Figure 7-20). The foliation is often deformed and the dykes regularly return above detection limit for gold, meaning these dykes are interpreted to be pre-/syn- 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.

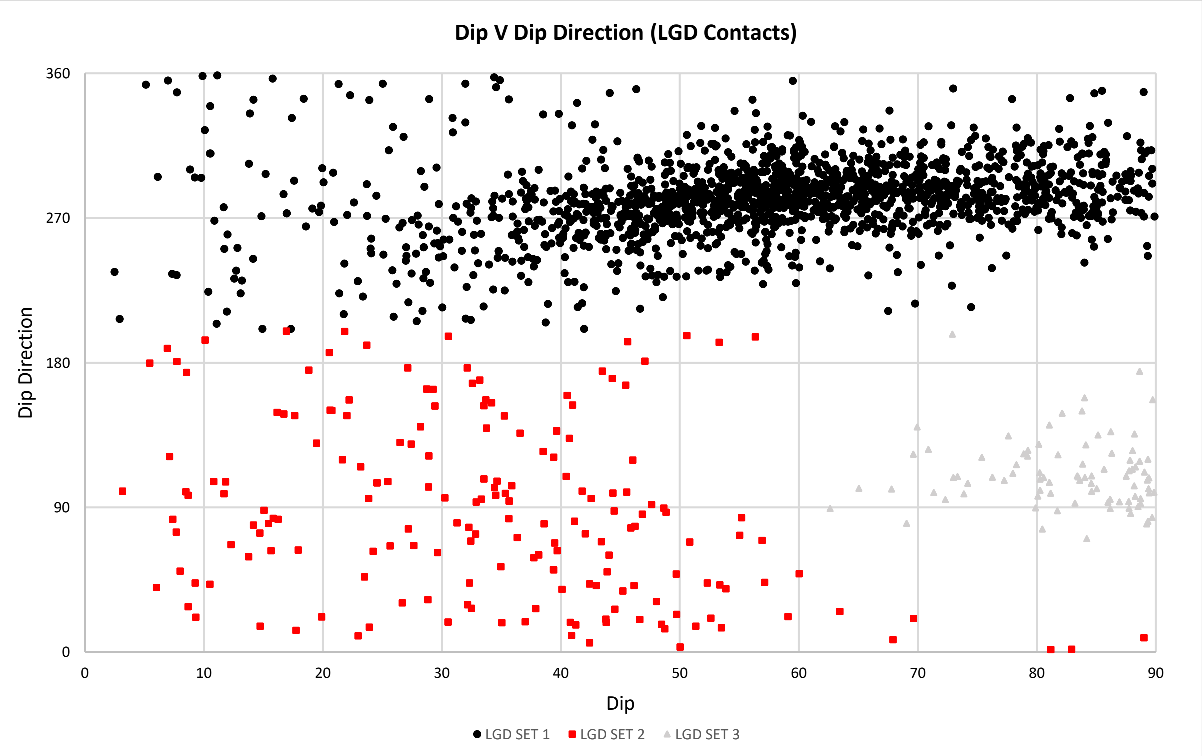
Figure 7-20 Chlorite/biotite altered foliations

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

7.3.6.2 Late Green Dykes (LGD)

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

Figure 7-21 Late Green dykes dip and dipderction



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

Figure 7-22   Late Green dykes

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

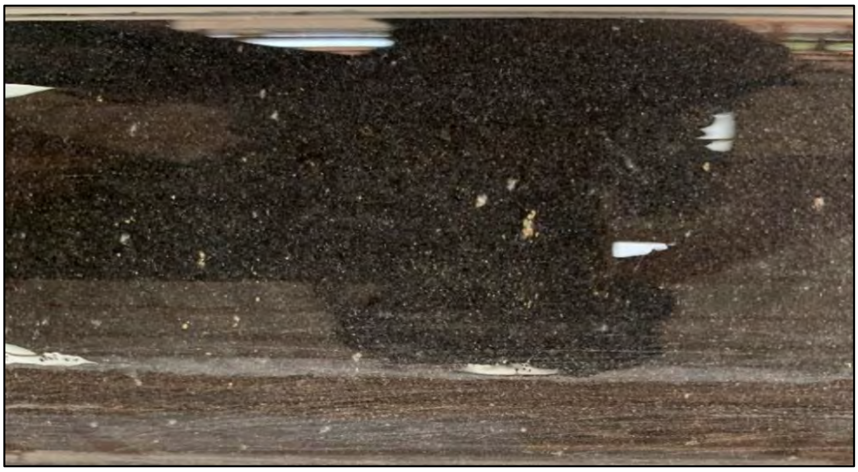
7.3.6.3 Mafic Dykes (MDY)

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

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

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

Figure 7-23   Massive Mafic dyke

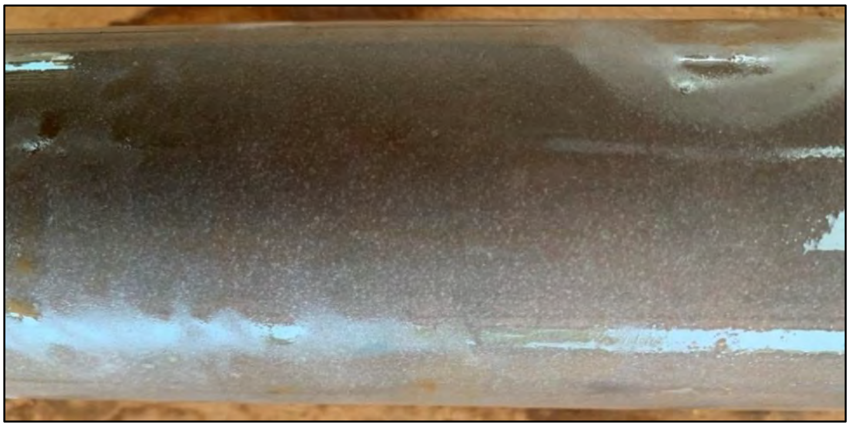


Example of the sub-horizontal massive mafic dyke, MDD069 at 312.6m depth. Source: Montage

7.3.6.4 Felsic Dykes (FDY)

Felsic dykes are light grey in colour, aphanitic, massive and cross-cut the foliation at a high angle. 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 ~75-80°. Two main felsic dykes have been logged in the main pit, with increased frequency seen to the north .

Figure 7-24 Felsic dyke



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

7.3.6.5 Intermediate Dyke (IDY)

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

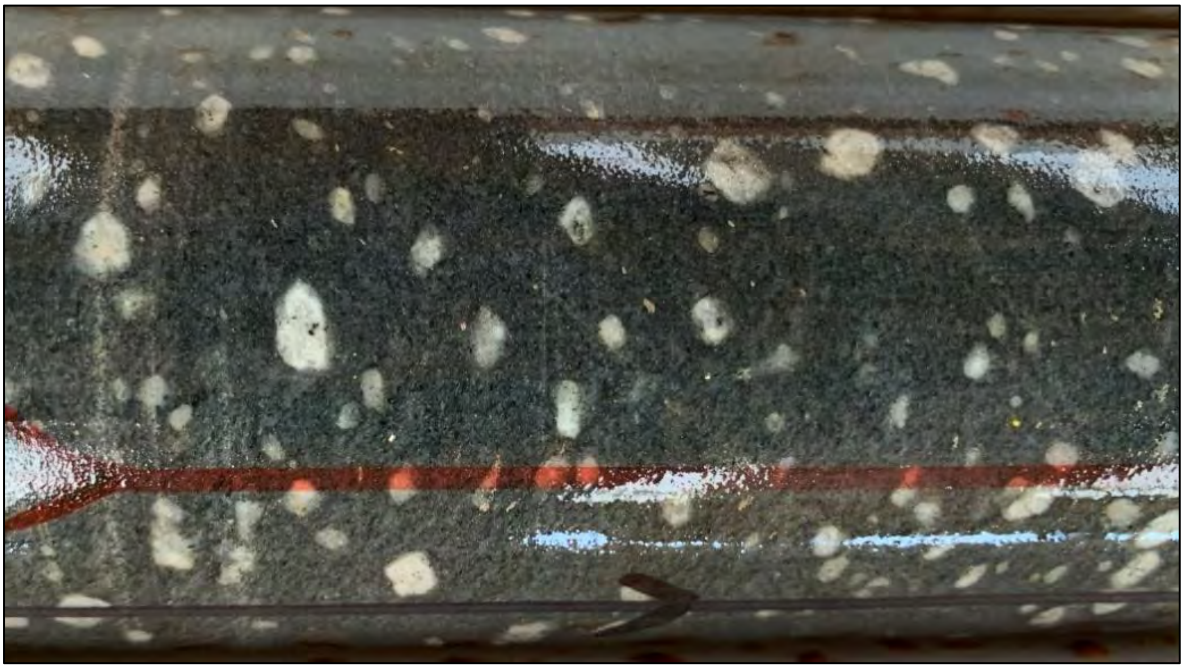
Figure 7-25 Felsic dyke

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

7.3.6.6 Feldspar Porphyry Dyke (FPR)

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

Figure 7-26 Felsic Porphyry dyke



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

7.3.7 Post Mineral deformation

7.3.7.1 Faults

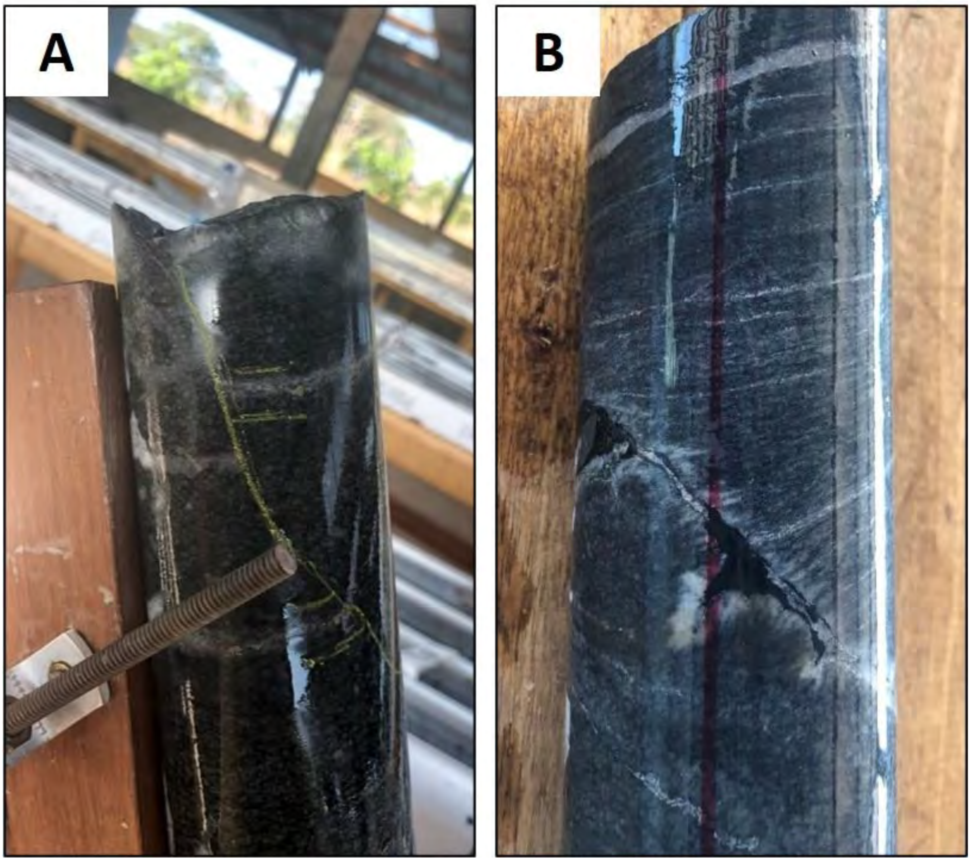
Very few larger fault, clays/breccias are observed within the deposit with the majority of faulting observed being minor faults displaying both normal and reverse movement (Figure 7-27), leading to high core recoveries (average 98% across the entire deposit). Therefore, any displacement and offset of the 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 7-28):

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

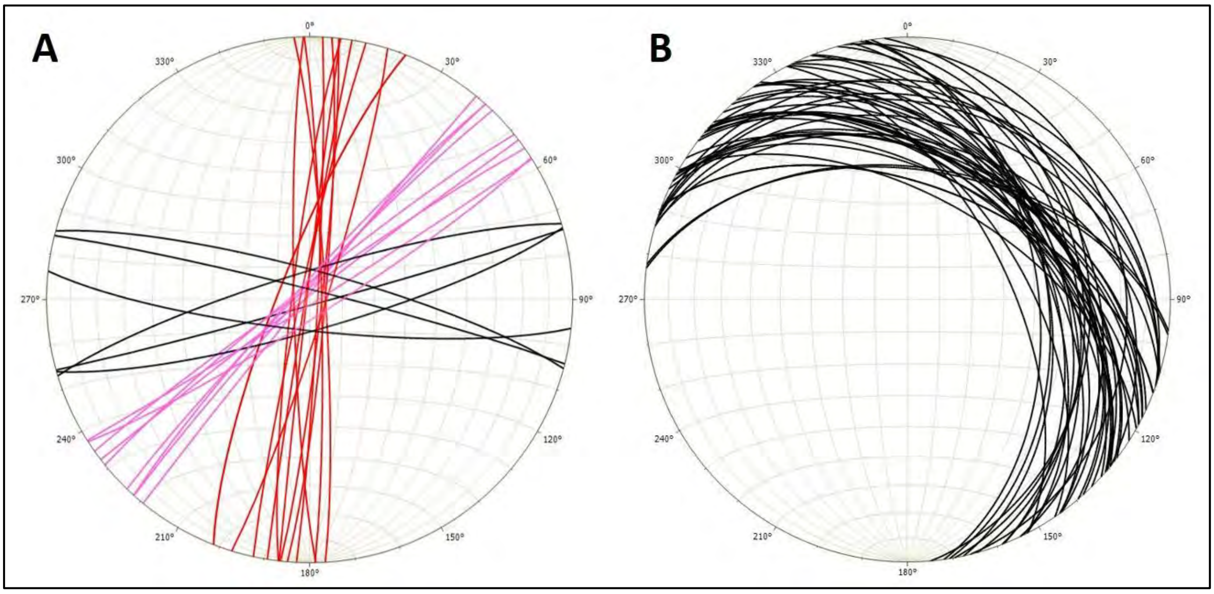
One further set of NW-SE striking faults averaging ~40° dip, sub-parallel to drill fences is present. Further study of the fault sets needs to be conducted in order to define the movement on these faults. Minor faults can be seen offsetting mineralised veins, therefore showing that faulting occurred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

Figure 7-27 Minonr healed fault



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

Figure 7-28 Minor faults stereo net

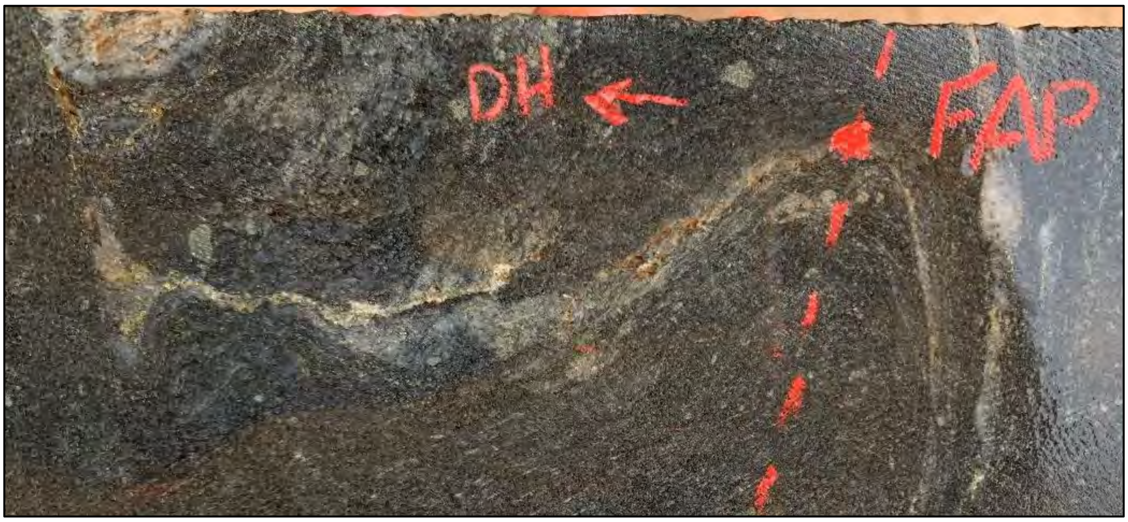


A – Sub-vertical faulting showing two clear orientations and a conjugate set of faults.
B – Shallow dipping fault set striking sub-parallel to drill fences.
Source: Montage

7.3.7.2 Folding

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

Figure 7-29 Folded VQS



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

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

Figure 7-30 Ductile Folding

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

8. DEPOSIT TYPES

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

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

Table 8-1 Ground Selection Criteria

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

9. EXPLORATION

9.1 Introduction

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

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

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

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

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

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

Table 9-1 Exploration Activities to Date

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

9.2 Soil Sampling

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

During 2019 and 2020 a further 3,137 soil samples were collected on the Koné Exploration Permit both infilling and extending previous grids. This sampling led to the delineation of the Petit Yao anomaly

8km east of the Koné deposit. Figure 9-1 shows the locations of soil samples relative to the Koné Exploration Permit, with sample locations coloured by assayed gold grade.

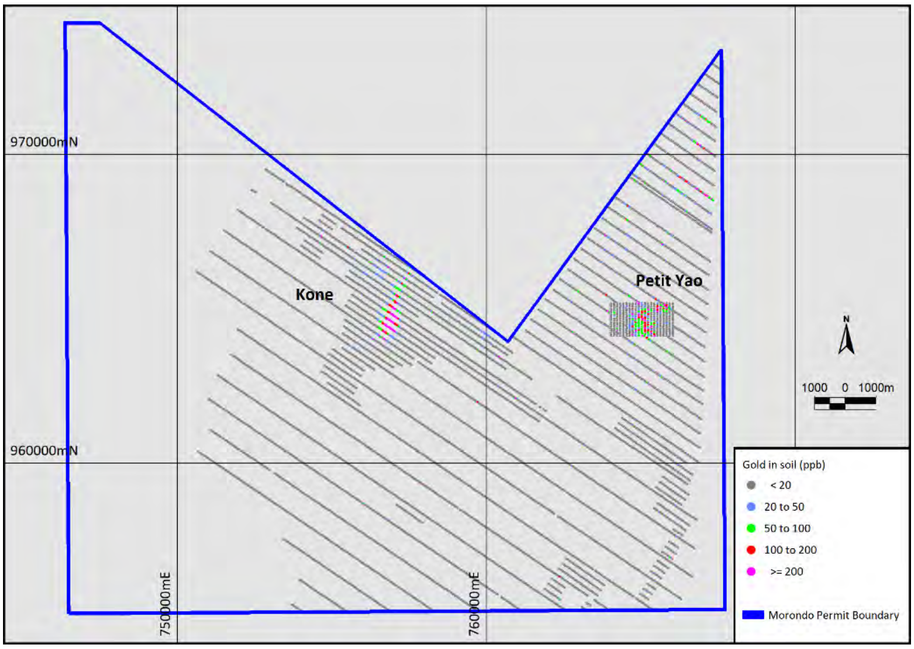
The 2009 and 2010 soil sampling phases utilized 20 to 30 cm diameter hand held augers to collect generally 2.5 to 3.0 Kg samples from depths of 50 to 60 cm, below the organic layer. Soil sampling in 2019 and 2020 was based on approximately 50 cm deep pits from which a 1 Kg sample was collected in the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

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

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

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

Figure 9-1 Soil Sampling Distribution



Source: Montage

9.3 Trenching

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

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

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

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

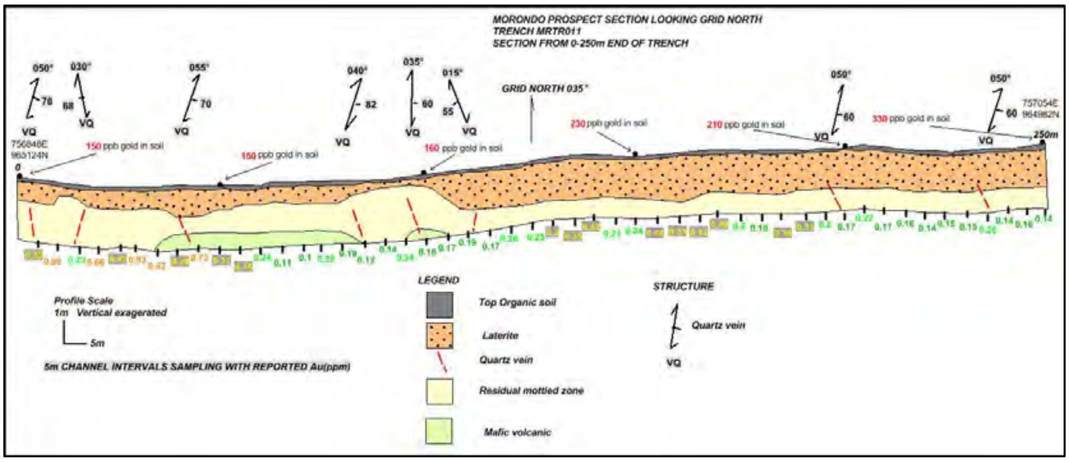
Significant intercepts from the trenching at Koné are shown in Table 9-2. True intercept thicknesses are interpreted to approximate 75% of interval lengths.

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

Table 9-2. Significant intercepts for 2009 and 2010 trenching

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

Figure 9-2. Example annotated trench section



Trench MRTR010. Source: Montage

9.4 Pit Sampling

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

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

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

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

9.5 Magnetic Survey

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

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

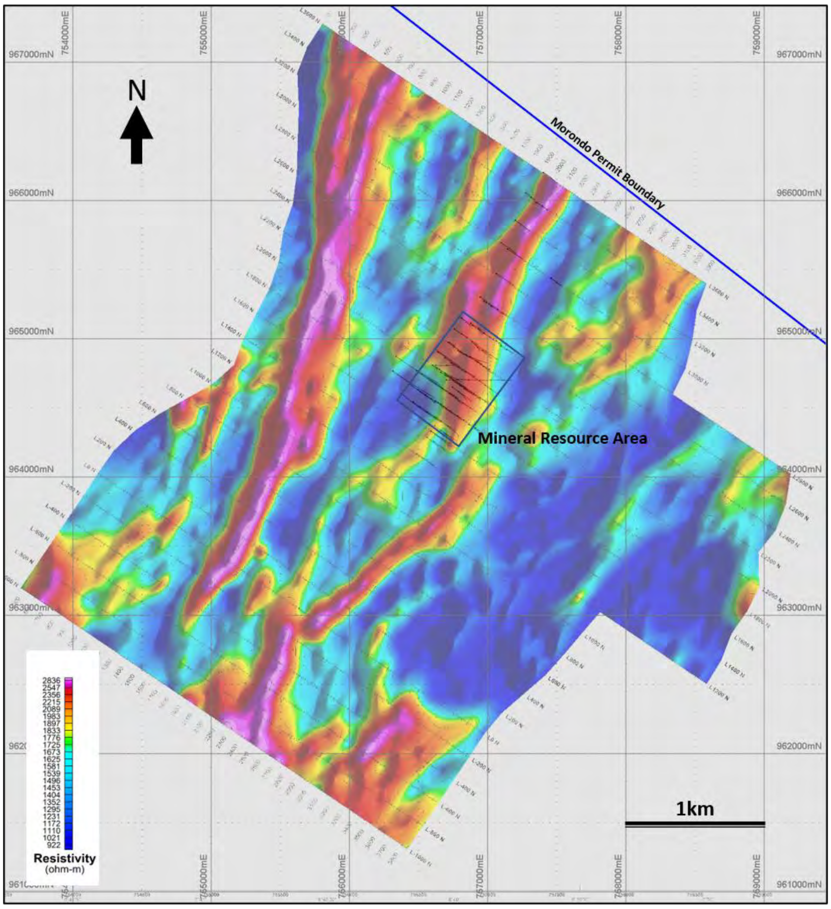
The ground magnetics are dominated by three, east-west trending magnetic highs that are considered to be mapping the extent of surficial duricrust and as a result the survey has been of limited use.

9.6 Gradient Array Induced Polarisation Survey

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

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

Figure 9-3. Induced Polarisation survey



Apparent Resistivity. Source: Montage

10. DRILLING

10.1 Introduction and Summary

As summarized in Table 10-1, drilling to date at Koné totals 353 RC and 50 diamond holes for 50,016.8 m. The RC drill metres shown in Table 10-1 for 2019 to 2020 Koné area drilling include 493.3 m of pre-collared portions of seven diamond holes .

In addition to RC and diamond drilling in the Koné area, which informs Mineral Resource estimates, Montage's drilling at the Koné includes shallow reconnaissance RC drilling testing exploration targets identified by soil and rock chip sampling, and 11 diamond holes drilled for geotechnical investigations, for which no samples have been submitted for gold analysis. Information from these drill holes does not inform resource modelling.

Central portions of the currently interpreted Koné mineralization have been tested by generally 100 m spaced northwest southeast traverses (125° UTM) of RC and rare diamond holes generally inclined to the southeast at around 55 degrees. These holes are generally spaced at around 50 and rarely 25 m along the traverses with each traverse extending to vertical depths of around 60 to 490 m.

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

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

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

The 2019 and 2020 reconnaissance RC holes, which were drilled to average depths of 39m are not intended for use in resource estimation and these programs do not include such rigorous surveying, or sampling and assaying procedures as adopted for resource drilling. Drilling completed in 2019 focussed on the general area surrounding the Koné mineralization and returned several low tenor anomalies (<0.2 g/t Au). The 2020 reconnaissance drilling targeted the Petit Yao prospect and intersected narrow mineralized zones.

Table 10-1. Koné drilling campaigns

Company	Phase	Holes			Metres		
		RC	Diamond	Total	RC	Diamond	Total
Red Back	2010 Koné area	8	-	8	943	-	943
Sirocco	2013 Koné area	43	-	43	3,341	-	3,341
Orca	2017-2018 Koné area	64	2	66	13,360	528	13,888
Montage	2019 – 2020 Reconnaissance	187	-	187	7,339	-	7,339
	2019 – 2020 Koné area	51	37	88	7,901	14,627	22,528
	2019 – 2020 Geotechnical	-	11	11	-	1,978	1,978
Subtotal resource drilling		**166**	**39**	**205**	**25,545**	**15,155**	**40,700**
Total		**353**	**50**	**403**	**32,884**	**17,133**	**50,017**

10.2 Koné RC Drilling

10.2.1 Drilling and sampling procedures

The RC drill rigs (Figure 10-1) generally utilized 140mm (5.5 inch) face sampling bits.

Figure 10-1 Drilling at Koné in 2013

Source: Montage

Samples were collected over 1m down-hole intervals from the base of the cyclone with a systematic procedure adopted for sample handling from collection at the cyclone to the laboratory dispatch stage as follows:

- Each metre sample was collected from the cyclone in a new 55 by 100 cm plastic sample bag labelled with the hole number and interval and weighed at the rig with the weight recorded on the drill log sheet.
- The bulk sample was then passed through a three-tier riffle splitter with an approximately 3kg primary "original" sub-sample collected in a plastic bag which was then sealed.
- The bulk sample was passed through riffle splitter a second time to produce an approximately 3kg archive sample with the remaining bulk sample stored in the original bag.
- Duplicates were collected by passing the bulk sample through the riffle splitter a third time producing another approximately 3 kg sub-sample.
- Samples tags were added to each sub-sample from numbered ticket books, with the hole number and interval clearly written on the ticket stub for reference.
- The 100 cm x 55 cm plastic bags containing the bulk reject sample were left at the drill site in ordered lines.

- The riffle splitter was cleaned thoroughly with compressed air between samples.
- All sub-samples (original, archive and duplicate) were transported to the field office at the end of the shift, where the archive sample is stored and original and duplicates prepared for despatch to the analytical laboratory.
- All assay pulps were returned to the field office from the laboratory and stored for future reference

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

All RC holes were geologically logged over 1m intervals with logging information recorded on paper drill log sheets by the field geologists including recording 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.

10.2.2 Collar and down-hole surveying

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

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

- 2010 RC holes were generally surveyed with a single shot Camteq Pro shot instrument at intervals of around 30m.
- 2013 RC holes were generally surveyed at intervals of around 80m with a Reflex Ez-Trac single-shot survey tool (Reflex).
- 2017 RC holes were generally surveyed at intervals of around 40m with a Reflex tool.
- 2018 RC and diamond holes were generally surveyed at intervals of around 30m with a Reflex tool.
- 2019 and 2020 Koné RC holes were generally surveyed with a Reflex Gyro tool at 5m intervals

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

10.2.3 Sample representivity

RC sample condition

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

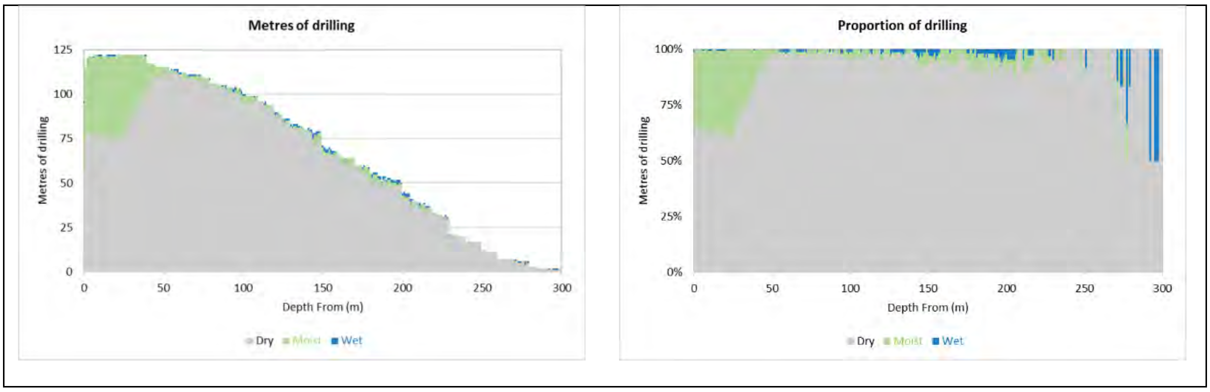
For all RC drilling field geologists recorded sample condition with samples assigned to dry, moist, or wet categories. Site visit observations suggest that samples logged as moist have little apparent moisture and, in terms of sample quality can be considered as effectively dry.

Table 10-2 summarizes sample condition logging for the 2018 drilling and Figure 10-2 shows the number and proportion of samples by sample condition category versus drilling. This table and figure demonstrate that wet samples provide only a small proportion of the 2018 RC drilling and any uncertainty over the reliability of these samples does not significantly affect confidence in resource estimates.

Table 10-2. Sample condition logging for 2018 RC drilling

Sample Condition	Number of Samples	Proportion of samples
Dry	18,393	89.1%
Moist	2,112	10.2%
Wet	146	0.7%
Subtotal	**20651**	**100%**

Figure 10-2. Sample condition logging for Koné RC drilling



RC Sample recovery

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

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

Field procedures for the 2017 and 2018 RC drilling programs included weighing recovered sample material, with weights available for around 99.7% of this drilling. No sample weights are available for the 2010 and 2013 RC campaigns which represent around one quarter of the Koné RC drilling available for resource estimation.

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

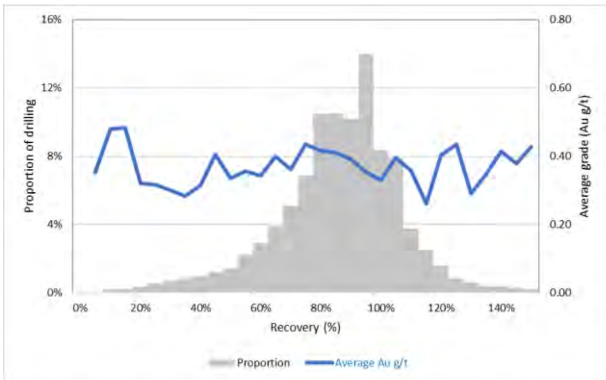
Table 10-3 summarizes RC sample recovery estimates by logged sample condition and Figure 10-3 shows average gold grade for increments of sample recovery. Notable features of this table and figure include the following:

- At 88%, average estimated RC sample recovery is consistent with the good quality RC drilling.
- Samples logged as moist or wet show proportionally lower average recoveries than dry samples.
- There is no notable association between estimated recovery and average gold grade.

Table 10-3. RC sample recovery estimates for 2017 and 2018 drilling

Sample Condition	Number Samples	Average Recovery
Dry	18,393	88%
Moist	2,112	79%
Wet	146	51%
Total	**20,651**	**88%**

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



10.3 Diamond Drilling

10.3.1 Drilling and sampling procedures

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

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

Core handling and sampling procedures included the following:

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

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

10.3.2 Collar and down-hole surveying

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

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

- 2018 holes were generally surveyed at intervals of around 30m with a Reflex tool.
- 2019 and 2020 holes were generally surveyed with a Reflex Gyro tool at 5m intervals

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

10.3.3 Sample representivity

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

Core recoveries for these intervals average 99.1% (Table 10-4 with only approximately 4% of composites showing recoveries of less than 90%. These recoveries are consistent with the author's experience of high-quality diamond drilling. Although lower than for fresh rock, average core recoveries for oxidized and transitional intervals are within the range shown by the author's experience of good quality diamond drilling.

Table 10-4. Core recovery for 3m composites from 2018 diamond drilling

Oxidation Zone	Number	Minimum	Average	Maximum
Oxide	330	17%	87%	140%
Transitional	183	46%	92%	109%
Fresh	4,547	50%	100%	141%
Total	**5,060**	**17%**	**99%**	**141%**

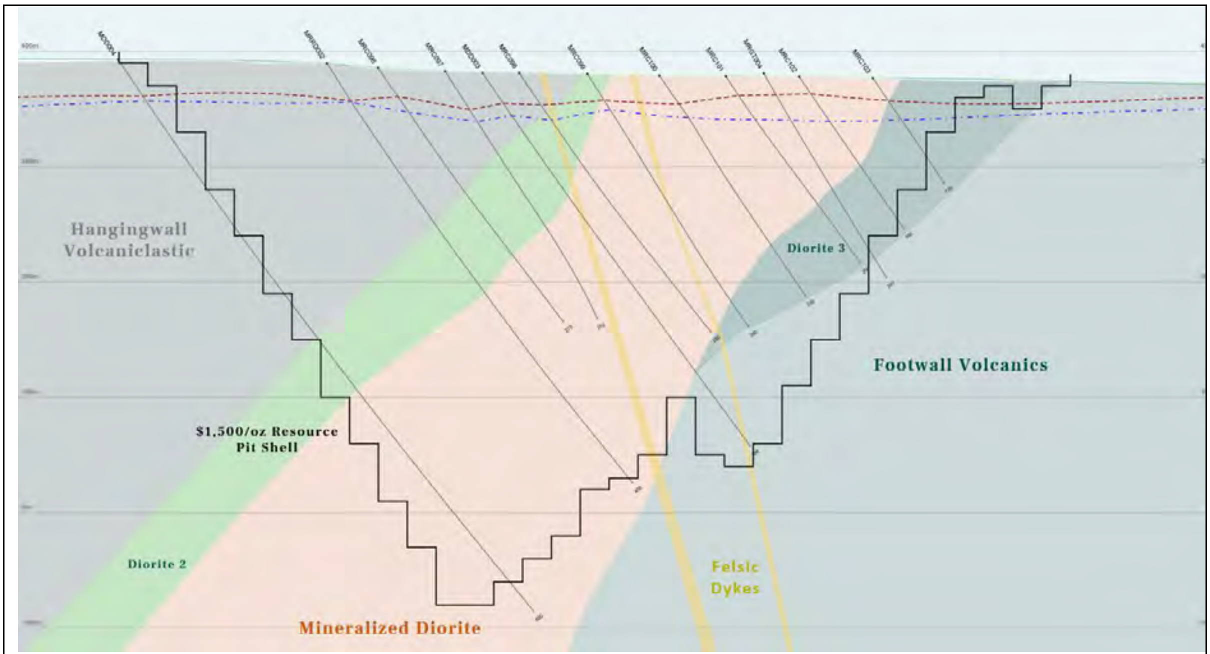
10.4 2019 and 2020 Reconnaissance RC Drilling

Shallow reconnaissance RC drilling was completed during 2019 and 2020 with average hole depths of 41m. This exploratory drilling tested several exploration targets identified by soil and rock chip sampling. Drilling completed in 2019 focussed on the general area surrounding the Koné mineralization and returned several low tenor anomalies (<0.20g/t Au). The 2020 reconnaissance drilling targeted the Petit Yao prospect and intersected narrow mineralized zones.

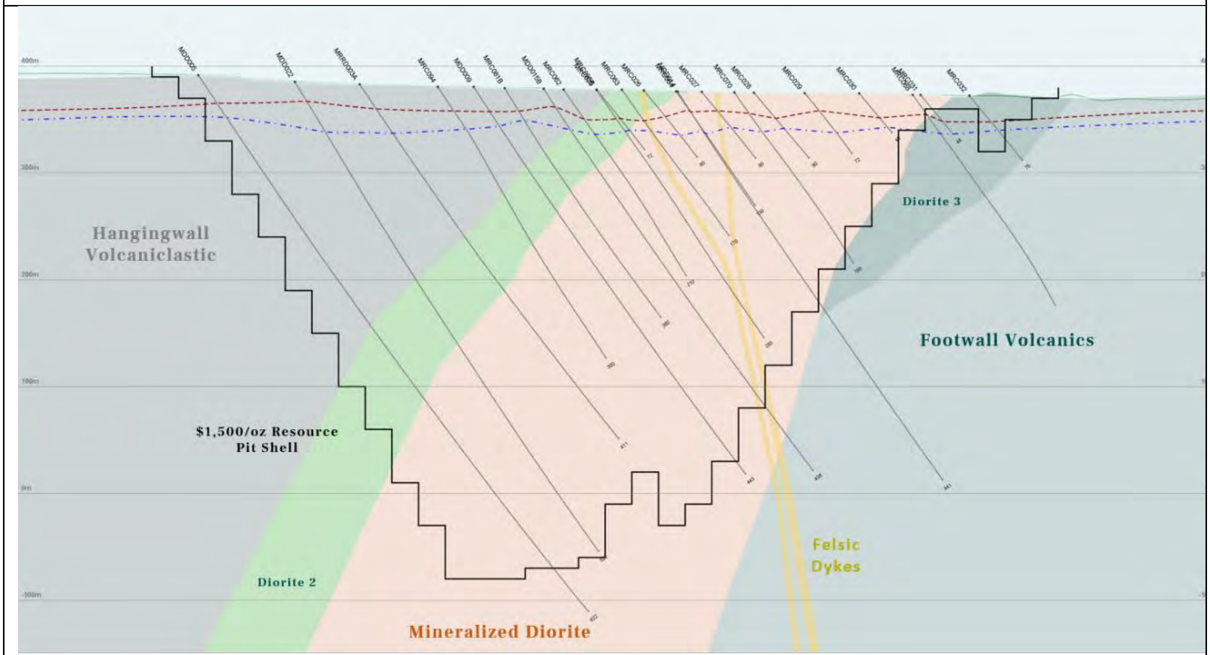
The reconnaissance RC holes were inclined at 50 or 55° at orientations and hole spacings reflecting interpreted local mineralization trends and previous exploration sampling. Hole spacings vary from rarely around 20m to around 180m spaced traverses.

These exploration holes are not intended for use in resource estimation and drilling and sampling did not include such rigorous surveying, or sampling and assaying procedures as adopted for resource drilling. The report author concurs with this approach, and considers it appropriate for the such drilling.

Figure 10-4. Koné representative cross sections



Section A



Section B

Section lines shown in Figure 7-2**. Looking northeast: Source: Montage, 2020**

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Introduction and Summary

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

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

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

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

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

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

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

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

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

11.2 Sample Submission Procedures and Security

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

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

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

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

Figure 11-1. Fadiadougou sample organisation and storage facility

Source: Montage

11.3 Primary Assay Laboratories and Accreditation

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

SGS

Samples from the 2009 to 2010 soil sampling, 2009 trenching and 2013 RC drilling were analysed by SGS with sample preparation performed by SGS in Yamoussoukro Côte d'Ivoire and 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.

Bureau Veritas

Samples from the 2010 RC drilling, 2013 and 2019 trenching, 2017 RC drilling, 2018 diamond drilling, 2019 reconnaissance RC drilling, 2019 to 2020 soil and pit sampling and primary RC and Diamond drilling in 2019 and 2020 were submitted to Bureau Veritas in Abidjan, Côte d'Ivoire for analysis.

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.

Intertek

Primary samples collected from 2018 Koné RC drilling and 2020 reconnaissance RC drilling were prepared and analysed 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.

11.4 Exploration Sampling

11.4.1 Soil Sampling

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

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

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

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

Assay results for coarse blanks and Rocklabs (2009-10) and Geostats (2019-20) standards included in batches of soil samples provide adequate confirmation of the reliability of sample preparation and analysis (Table 11-1 and 11.2).

Table 11-1. Coarse blanks and reference standards included soil samples

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

Reference Standards				
Reference Standard	Number Samples	Gold grade (ppb) Expected	Avg. Assay	Avg. vs. Expected
GLG302-3	4	30.8	28.3	-8%
GLG305-1	5	101.6	99.8	-2%
GLG305-3	5	55.5	52.4	-6%
GLG310-3	10	119.3	113.5	-5%
GLG313-5	10	83.4	66.7	-20%
GLG908-4	13	65.9	64.0	-3%
GLG910-2	13	24.7	21.6	-13%
GLG914-3	5	205.8	205.2	0%
GLG916-1	12	5.1	8.6	70%
OXA26	38	79.8	82.4	3%
OXA45	27	81.1	99.3	22%
OXA71	2	84.9	86.5	2%
OXD43	5	401	462	15%
OXD57	33	413	407	-1%
OXE42	8	610	605	-1%
OXE56	24	611	592	-3%

11.4.2 Trenching

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

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

Assay results for coarse blanks included in batches of trench samples at an average frequency of around one blank per 18 primary samples are summarized in **Table 11-2** with samples assaying at below the detection limit of 0.01 g/t assigned gold grades of half the detection limit. This table demonstrates that coarse blank assays show very low gold grades relative to typical Koné mineralization with no indication of significant contamination or sample misallocation.

Samples of Rocklabs certified reference standards were routinely included in batches of trench samples at an average frequency of around 1 standard per 45 primary sample. As shown in Table 11-2, although, as expected there is some variability for individual samples, average assay results closely match expected values.

Table 11-2. Coarse blanks and reference standards included with 2009-10 trench samples

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

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

11.4.3 Pit Sampling

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

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

Coarse blanks and OREAS certified reference standards were submitted in batches of pit samples at an average frequency of around 1 per 26 and 57 primary samples respectively. Gold assay grades reported for these samples are summarized in Table 11-3 with assays reported as below the detection limit of 0.01 g/t assigned values of half the detection limit.

Table 11-3 demonstrates that coarse blank assays show very low gold grades, and average assay results for standards closely match expected values, supporting the reliability of sample preparation and assaying for the pit samples.

Table 11-3. Coarse blanks and reference standards included with 2019 pit samples

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

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

11.5 Koné RC and Diamond Drilling

11.5.1 Sample Preparation and Analysis

Primary analyses of samples from the RC and diamond drilling in the Koné area, which provide the basis for the current Mineral Resource estimate was undertaken by several laboratories as follows:

- Samples from the 2010 RC drilling were submitted to Bureau Veritas in Abidjan, Côte d'Ivoire for analysis.
- Primary samples from the 2013 RC drilling were analysed by SGS with sample preparation in Yamoussoukro, Côte d'Ivoire and analysis by fire assay at SGS Tarkwa, Ghana for most samples, with proportionally few samples from four holes analysed at SGS Ouagadougou, Burkina Faso.
- Samples from September 2017 RC drilling, 2018 Diamond drilling and RC and diamond drilling in 2019 and 2020 were submitted to Bureau Veritas in Abidjan, Côte d'Ivoire for preparation and analysis.
- Primary RC samples collected in 2018 were prepared and analysed by Intertek in Tarkwa, Ghana.

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

- Each batch received was laid out in sequence, weighed and checked in to the Bureau Veritas system. Inhouse geologists responsible for sample submission to the laboratory were informed of any missing samples or extra samples not listed on the submission form, and a replacement or corrected submission form prepared by inhouse personnel.
- Each, nominally 3 Kg sample was jaw crushed to >80% passing 2 mm and riffle split to produce two 1.5 kg sub-samples. After every twentieth sample and at the end of each assay batch a crusher flushing sample of barren vein quartz was used to clean the crusher plates.

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

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

11.5.2 Monitoring of Sampling and Assay Reliability

Field Duplicates

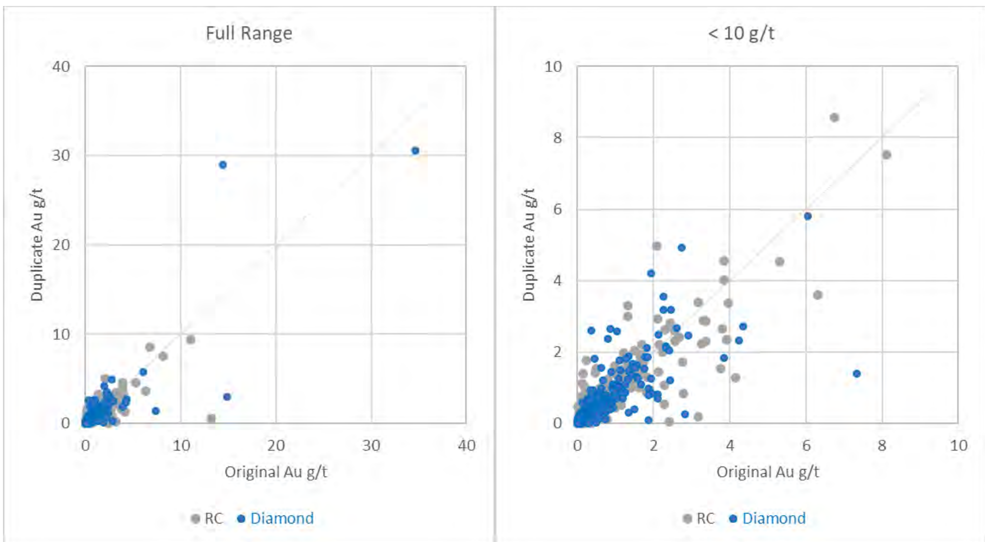
Field duplicates were collected for Koné RC and diamond drilling at average frequencies of around one duplicate per 20 primary samples for both drill types. Field duplicates were collected consistently with and assayed in the same batch as original samples.

The summary statistics in Table 11-4 and scatter plots in Figure 11-2 demonstrate that although there is some scatter for individual pairs duplicate assay results generally correlate reasonably well with original results demonstrating the adequacy of field sub-sampling procedures.

Table 11-4. Field duplicates for Koné RC and diamond drilling

Au g/t	RC				Diamond			
	Full Set		0.1 to 10 g/t		Full Set		>0.1 g/t	
	Orig.	Dup.	Orig.	Dup.	Orig.	Dup	Orig.	Dup
Number	1,116		605		530		293	
Average	0.38	0.35	0.62	0.60	0.52	0.50	0.90	0.86
Difference		-6%		-2%		-5%		-4%
Variance	0.70	0.48	0.67	0.60	3.47	3.68	5.95	6.37
Coef. Variation.	2.23	1.97	1.33	1.29	3.56	3.86	2.71	2.93
Minimum	0.01	0.01	0.10	0.10	0.01	0.01	0.10	0.10
1st Quartile	0.03	0.03	0.19	0.19	0.04	0.04	0.21	0.19
Median	0.13	0.13	0.35	0.35	0.14	0.14	0.38	0.37
3rd Quartile	0.40	0.39	0.69	0.72	0.45	0.43	0.87	0.78
Maximum	13.19	9.39	8.12	8.57	34.69	30.55	34.69	30.55
Correl. Coef.	0.82		0.89		0.87		0.87	

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



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 21 primary samples.

Table 11-5 summarizes gold assays for these blanks by assay laboratory with samples assaying at below the detection limit of 0.01 g/t assigned values of half the detection limit. This table excludes two anomalous samples from the 2018 drilling with gold grades of 0.56 and 1.10 g/t which appear to reflect misallocation.

Table 11-5 demonstrates that coarse blank assays show very low gold grades relative to typical Koné mineralization with no indication of significant contamination or sample misallocation.

Table 11-5. Coarse blanks included with Koné drill samples

Laboratory	Number Blanks	Gold assay (g/t)			Proportion > Detection
		Minimum	Average	Maximum	
Bureau Veritas	1,043	0.005	0.008	0.04	21%
Intertek	563	0.005	0.006	0.12	8%
SGS	48	0.005	0.010	0.05	21%
Combined	**1,654**	**0.005**	**0.007**	**0.12**	**16%**

Reference Standards

For all phases of Koné RC and diamond drilling samples of certified reference standards prepared by commercial standards suppliers were inserted in assay batches at an average rate of around 1 standard per 23 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 and 2020 drill programmes Geostats standards were used. Expected gold grades for the standards range from around 0.3 to 6.1 g/t covering the range of typical gold grades shown by Koné drill hole samples.

Table 11-6 summarizes assay results for standards included in batches of drill samples by assay laboratory. This table excluded a small number of standards for which fewer than five samples were analysed by each laboratory. Table 11-6 demonstrates that although, as expected there is some variability for individual samples, average assay results closely match expected values.

Table 11-6. 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	80	1.11	1.07	-4%
	G314-1	142	0.75	0.77	3%
	G315-4	85	0.32	0.32	1%
	G316-8	37	6.11	6.16	1%
	G319-2	41	3.96	4.01	1%
	G908-4	90	0.96	0.97	1%
	G910-10	94	0.97	0.98	1%
	G912-7	61	0.42	0.42	1%
	G913-2	36	2.40	2.44	2%
	G916-2	115	1.98	1.99	1%
	G916-4	36	0.51	0.51	0%
	OREAS-210	56	5.49	5.53	1%
	OREAS-214	52	3.03	3.06	1%
	OREAS-250	8	0.31	0.38	23%
	OREAS-251	42	0.50	0.51	2%
	OREAS-502b	6	0.50	0.51	2%
	OREAS-504b	8	1.61	1.63	1%
	Combined	**989**	**1.67**	**1.68**	**1%**
Intertek	OREAS-210	166	5.49	5.50	0%
	OREAS-214	150	3.03	3.08	2%

	OREAS-250	12	0.31	0.32	3%
	OREAS-251	51	0.50	0.51	2%
	OREAS-502b	19	0.50	0.48	-3%
	OREAS-504b	146	1.61	1.61	0%
	Combined	**544**	**3.01**	**3.03**	**0%**
Laboratory	**Reference Standard**	**Number Samples**	**Gold grade (g/t) Expected**	**Avg. Assay**	**Avg. vs. Expected**
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%**

Intertek Screen Fire and Cyanide Leach Duplicates

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

As summarized in Table 11-7, 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 11-7. Alternate method duplicate assays versus original assays for 2010 to 2018 drill samples

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

ALS Inter Laboratory Repeats

Information available to demonstrate the accuracy of primary gold assaying for Koné drill samples includes fire assays of pulp samples performed by ALS in Rosia Montana, Romania during August 2018 including:

- 239 samples originally assayed by Bureau Veritas in 2017 comprising 228 original, or field duplicate samples and 11 coarse blanks, and
- 649 samples originally assayed by Intertek in 2018 comprising 618 original, or field duplicate samples and 31 coarse blanks, and
- 38 samples of reference standards.

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).

Average assay results for reference standards closely match expected values supporting the general accuracy of ALS assaying (Table 11-8).

ALS reported very low gold grades for each of the coarse blanks, which provide little information about general accuracy of the original assaying and these results were excluded from the review dataset.

The summary statistics in Table 11-9 and scatter plots in Figure 11-3 demonstrate that although there is some scatter for individual pairs the ALS repeat assay results generally correlate reasonably well with original results providing additional confidence in the accuracy of the primary Bureau Veritas and Intertek assaying.

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

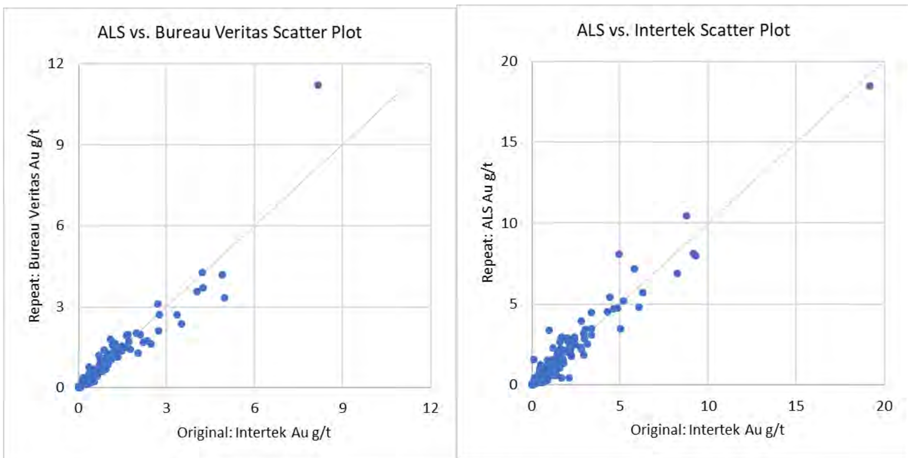
Table 11-8. Reference standards assays included with interlaboratory repeats

Reference Standard	Number Samples	Gold grade (g/t) Expected	Avg. Assay	Avg. vs. Expected
OREAS 210	14	5.490	5.505	0%
OREAS 214	10	3.030	3.077	2%
OREAS 502b	6	0.495	0.505	2%
OREAS 504b	8	1.610	1.626	1%
Combined	**38**	**3.237**	**3.260**	**1%**

Table 11-9. Interlaboratory repeat assays of 2010 to 2018 drill samples

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

Figure 11-3. Interlaboratory repeat assays of drill samples



11.6 Reconnaissance RC Drilling

11.6.1 Sample Preparation and Analyis

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

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

11.6.2 Monitoring of Sampling and Assay Reliability

Routine Field Duplicates

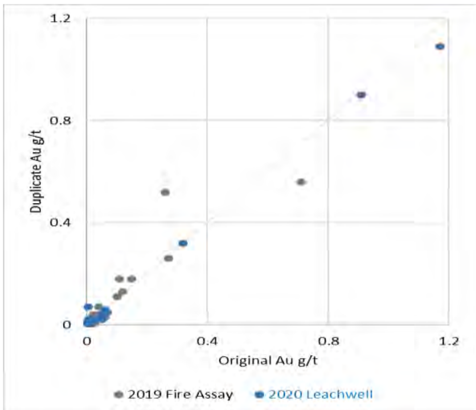
Routine field duplicates were collected for the 2019 and 2020 RC reconnaissance drilling at average frequencies of around one duplicate per 29 and 39 primary samples respectively. These samples were collected consistently with and assayed in the same batch as original samples providing an indication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

Table 11-10. Field duplicates for reconnaissance RC drilling

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

Figure 11-4. Field duplicates for reconnaissance RC drilling



Coarse Blanks and Reference Standards

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

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

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

Coarse Blanks					
Assay Group	**Number Samples**	**Gold assay (g/t) Minimum**	**Average**	**Maximum**	**Proportion > 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) Expected**	**Avg. Assay**		**Avg. vs. Expected**
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.

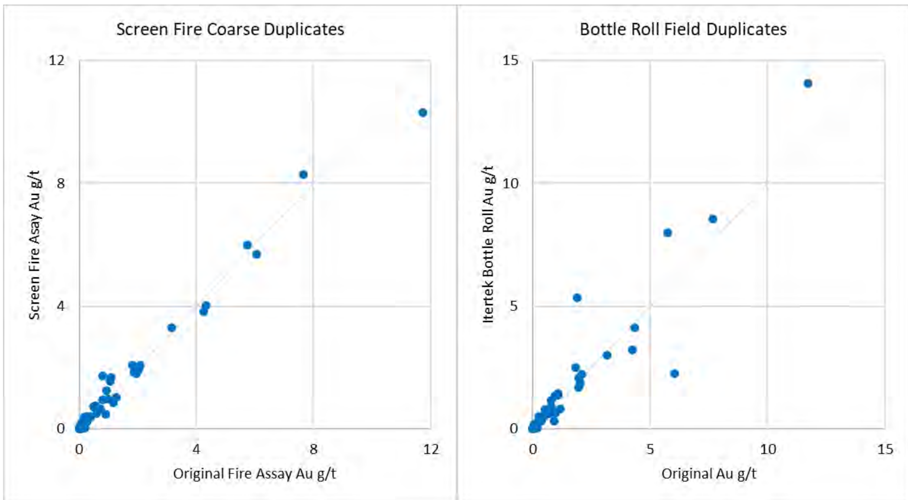
The summary statistics in Table 11-12 and scatter plot in Figure 11-5 demonstrate that although there is some scatter for individual pairs the screen fire and bottle roll duplicate assays correlate reasonably well with original results providing additional confidence in the accuracy of the primary Bureau Veritas fire assaying.

Table 11-12. Alternative method and interlaboratory duplicates for 2019-20 reconnaissance RC 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	

Figure 11-5. Alternative method and interlaboratory duplicates for reconnaissance RC drilling



11.7 Density Measurments

Bulk density measurements available for the Koné drilling comprise 1,867 immersion measurements performed by inhouse personnel.

The density measurements were carried out on 10 to 15 cm lengths of core which were oven dried for 24 hours at 100^{0}C and wax coated to prevent water absorption. Densities were measured by the Archimedes method with allowance for the wax coating.

Table 11-13 summarizes the density measurements by mineralization and oxidation domain excluding three anomalous measurements with supplied densities of less than zero.

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

Table 11-13. Density measurements

Mineralized Domain	Oxidation Zone	Number	Density (t/m^3) Minimum	Average	Maximum
Background	Completely Oxidized	93	1.23	1.67	2.79
	Transitional	44	1.62	2.48	2.90
	Fresh	429	2.33	2.84	3.39
Mineralized	Completely Oxidized	51	1.16	1.70	2.58
	Transitional	34	1.91	2.56	2.83
	Fresh	1,213	1.73	2.82	3.64
Combined	Completely Oxidized	144	1.16	1.68	2.79
	Transitional	78	1.62	2.52	2.90
	Fresh	1,642	1.73	2.83	3.64

12. DATA VERIFICATION

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

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

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

The consistency checks showed no significant inconsistencies.

While visiting Montage’s field office in Fadiadougou, the author compared original field records with database entries. These checks included 180 down hole survey table records and down hole depths and sample identifiers for 5,523 assay intervals representing approximately 25% and 33% of database entries respectively at that time. Relative to the drill holes informing the current estimates, these checks represent 4% and 15% of down hole survey and assay records respectively. These spot checks showed no significant inconsistencies.

For all routine assays from RC and diamond drilling, the author compared database assay entries with gold grades in laboratory source files supplied by Orca or Montage (Table 12-1). These checks showed no inconsistencies.

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

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

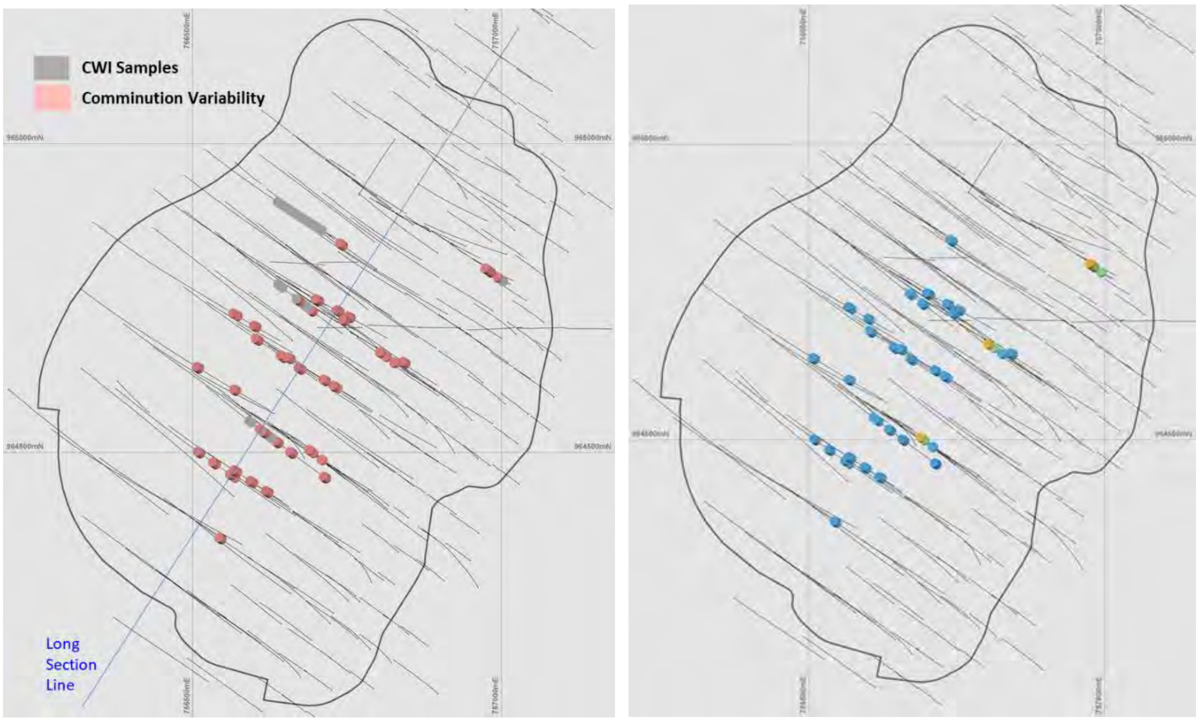
Group	Number of assays		Proportion
	In database	Checked	Checked
Reconnaissance RC holes	4,487	4,487	100.0%
RC holes included in estimation data set	23,586	23,586	100.0%
Diamond holes included in estimation data set	12,307	12,307	100.0%
Combined	**40,380**	**40,380**	**100.0%**

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Samples

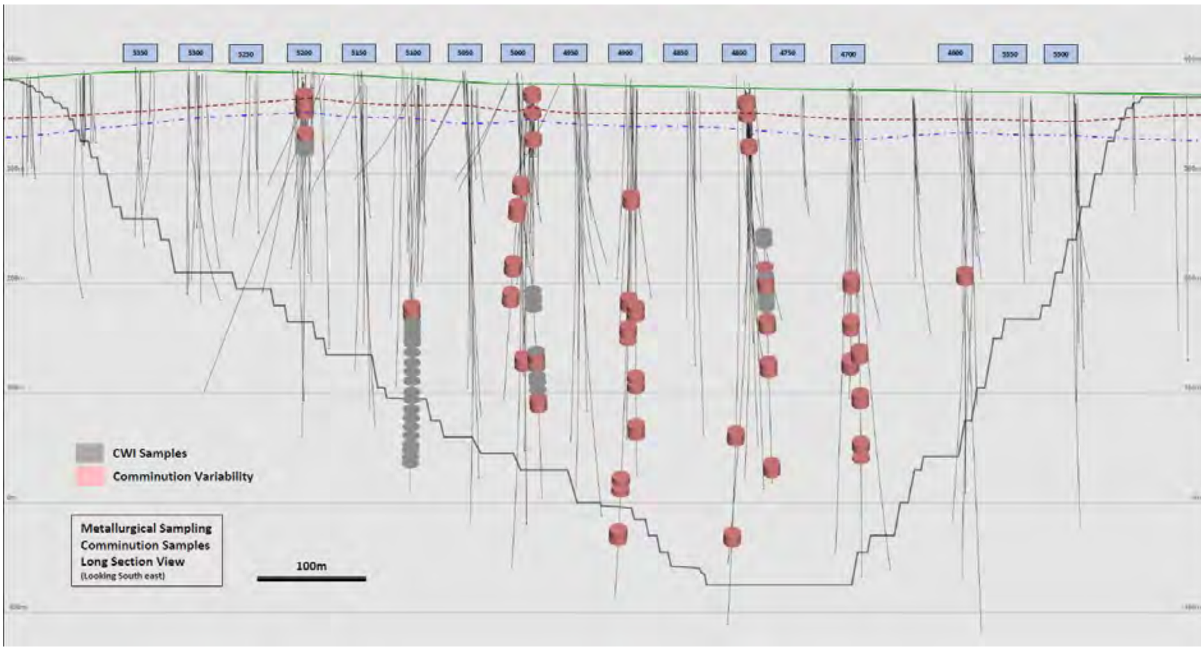
Figure 13-1,Figure 13-1 and Figure 13-3 show the locations of the metallurgical test samples relative to the extents of mineralization included in Mineral Resource estimates.

Figure 13-1. Metallurgical sample locations (Plan)



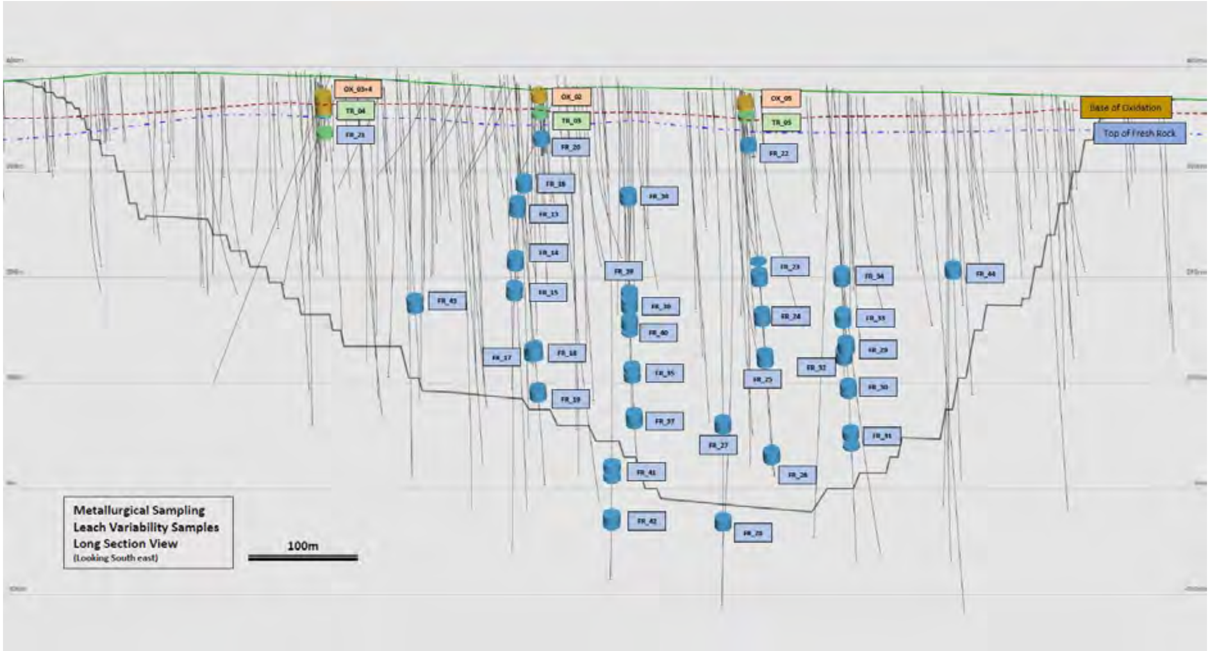
Source: Montage.

Figure 13-2. Comminution sample locations (Long Section)



Source: Montage.

Figure 13-3. Leach sample locations (Long Section)



Source: Montage.

13.2 Preliminary Bottle Rolls 2014

In 2014, SGS Minerals Services UK Ltd, Cornwall performed bottle roll tests on three composite RC samples of fresh mineralization. The samples were ground to 90-microns and leached for 48 hours at 40% solids, 0.5 g/l and an average pH of 10.7.

As summarized in Table 13-1 recoveries for the three samples ranged from 96.4% to 97.6% and averaged 96.9%. The information in this table was derived from a summary spreadsheet of test results produced by SGS.

Table 13-1 2014 Bottle Roll Results

Parameter	METSAMP_001	METSAMP_002	MATSAMP_003	Average
Head Assay g/t Au	1.11	0.82	2.71	1.55
Leach Extraction, %	96.4	97.6	96.7	96.9
CN Consumption kgs/t	0.12	0.06	0.15	0.11
CaO Consumption kgs/t	0.50	0.52	0.45	0.49

13.3 Diamond Drill Core Testing 2018

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

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.

Results of the head-assay and BWi determinations are summarized in Table 13-2, with notable features described by ALS 2018 including the following:

- Variability in the gold head assays and screened fire assay data indicate the composites are likely to contain coarse gold, particularly the oxide composite.

- Cyanide consumption is likely to be highest for the FW fresh composite due to the higher cyanide soluble copper and iron content but is overall relatively low.
- All composites contain some mercury, with the oxide and transition composites containing slightly elevated levels of greater than 1ppm.
- The BWi result for the oxide composite is likely to be significantly overstated, due to excessive fines in the feed material to this test. The BWi was determined at a closing screen size of 106µm.

Table 13-2 Diamond Core Matallurgical Sample Head Assay and BWi

Analyte	Method	Units	Oxide	Transition	Fresh	FW Fresh
Au	Fire assay	g/t	1.18	1.19	0.92	1.80
Au	Fire assay	g/t	1.56	1.21	1.32	1.82
Au	Screen Fire	g/t	1.30	0.98	0.82	n/a
CNsCu	D13	ppm	16	6	8	38
CNsFe	D13	ppm	42	115	140	170
Hg	D1/ICP	ppm	1.6	1.3	0.2	0.3
Bond BWi		kWh/t	10.9	5.2	9.8	10.7

Sub samples of each composite were submitted to coarse crush leach tests at various crush sizes to determine amenability to heap leaching. The samples were ground to 80% passing 75 microns and leached for 48 hours at 40% solids w/w, 0.5 g/CN and an average pH of 10.7.

Tests were also conducted to compare gold extraction via 'direct' cyanide leaching with gold extraction under CIL conditions. Additional tests were conducted to determine the impact of gravity gold recovery prior to cyanide leaching. Results are summarized in Table 13-3, with observations by ALS, 2018 including the following:

- For all composites, gold extraction under CIL conditions was very similar to that achieved via direct leaching at P80 75µm, indicating the samples are not preg robbing.
- For the oxide composite
  - Overall gold extraction was high for all tests, at 95% or higher.
  - Despite gravity gold recovery of around 39%, removal of gravity gold did not appear to improve leach kinetics.
- For the Transition Composite:
  - Approximately 30% of the gold was recovered by gravity at P_{80} 75µm. Removal of this gold improved leach kinetics.
- For the Fresh rock composite:
  - Approximately 23% gravity gold recovery was achieved at P_{80} 75µm. Removal of this gold did not improve leach kinetics.
  - Coarse crush leach results followed the expected trend, with average gold extraction highest for the finest crush size.

Table 13-3 Diamond Core Leaching Test Summary

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

13.4 Diamond Drill Core Metallurgical Testing 2020

The scope of work was designed to advance the flowsheet development and optimisation.

13.4.1 Variability Comminution & Physical Testing

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

This phase of testwork focussed on the Fresh domain which comprises 87% of total resource tonnage and 89% of the resource ounces. The results are summarised in Table 13-4.

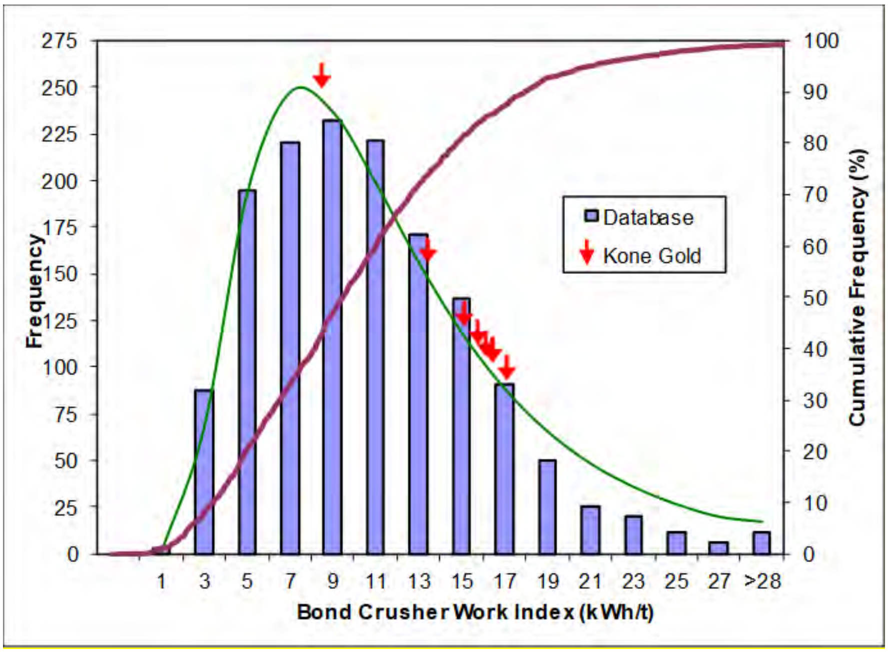
The variability within the Fresh samples was narrow, with A x b’s varying from 36.3 (moderately hard) to 26.0 (very hard), while the Transition samples were more widely spread, with A x b’s varying 176 (very soft) to 77.6 (soft).

Table 13-4 Comminution Testwork Summary

Sample Name	Sample Type	Rock Type	Hole ID	Interval (m) From	To	Relative Density	JK Parameters A x b	SCSE	Work Indices (kWh/t) CWI	BWI	AI (g)
MR_FR_007	Fresh	CDI/FDI	MDD008	239.75	349.67	2.76	32.1	11.1	16.5	10.5	0.368
MR_FR_008	Fresh	FDI/CDI	MDD009	161.90	264.50	2.75	30.9	11.3	16.2	12.1	0.500
MR_FR_009	Fresh	CDI/FDI	MDD011	270.80	426.22	2.72	32.0	11.0	15.7	10.8	0.458
MR_FR_010	Fresh	VC	MDD012	57.47	77.12	2.78	28.1	12.0	15.2	11.7	0.339
MR_FR_011	Fresh	CDI/FDI	MDD013	56.32	74.92	2.72	31.0	11.2	16.5	9.9	0.532
MR_FR_012	Fresh	CDI/FDI	MDD014	51.52	75.98	2.72	28.8	11.6	13.3	10.5	0.446
MR_FR_013	Fresh	CDI	MRRD001	134.00	150.00	2.80	29.4	11.7	-	10.1	-
MR_FR_014	Fresh	FDI	MRRD001	198.35	212.30	2.74	30.0	11.4	-	10.7	-
MR_FR_015	Fresh	FDI	MRRD001	233.70	247.30	2.73	31.9	11.1	-	10.6	-
MR_FR_016	Fresh	CDI	MRRD006	109.00	121.00	2.73	28.4	11.7	-	10.0	-
MR_FR_017	Fresh	VC	MRRD006	307.70	321.00	2.77	27.5	12.0	-	9.4	-
MR_FR_018	Fresh	CDI	MDD008	309.75	324.14	2.70	30.8	11.2	-	10.3	-
MR_FR_019	Fresh	CDI/VC	MDD008	358.10	370.05	2.74	33.4	10.8	-	9.9	-
MR_FR_020	Fresh	CDI	MDD013	52.45	60.71	2.72	26.0	12.2	-	11.2	-
MR_FR_021	Fresh	VC	MDD012	53.85	59.55	2.77	33.3	10.9	-	8.7	-
MR_FR_022	Fresh	FDI/VC	MDD014	56.90	65.16	2.76	28.4	11.8	-	10.7	-
MR_FR_023	Fresh	CDI	MDD009	199.50	224.42	2.74	28.6	11.7	-	12.9	-
MR_FR_024	Fresh	CDI	MDD009	256.90	270.50	2.72	30.3	11.3	-	11.2	-
MR_FR_025	Fresh	FDI	MDD009	304.70	319.05	2.75	26.3	12.3	-	14.3	-
MR_FR_026	Fresh	FDI/CDI +/- VC	MDD009	419.90	432.35	2.78	32.6	11.1	-	11.1	-
MR_FR_027	Fresh	CDI	MDD005	397.90	410.20	2.75	26.0	12.3	-	13.4	-
MR_FR_028	Fresh	FDI/CDI/VC	MDD005	514.20	525.65	2.76	32.9	11.0	-	10.9	-
MR_FR_029	Fresh	FDI/VC	MDD006	293.25	306.40	2.76	27.4	12.0	-	13.6	-
MR_FR_030	Fresh	CDI/FDI/VC	MDD006	343.87	357.25	2.77	28.0	11.9	-	12.9	-
MR_FR_031	Fresh	FDI/CDI	MDD006	399.20	420.05	2.73	27.2	12.0	-	15.4	-
MR_FR_032	Fresh	CDI/FDI	MRRD004	311.70	326.00	2.74	26.0	12.3	-	15.3	-
MR_FR_033	Fresh	CDI	MRRD004	265.25	280.00	2.75	31.0	11.3	-	10.9	-
MR_FR_034	Fresh	CDI/FDI	MRRD004	215.00	230.00	2.73	31.0	11.2	-	12.1	-
MR_FR_035	Fresh	CDI/FDI	MRRD002	333.80	350.00	2.74	34.1	10.7	-	9.3	-
MR_FR_036	Fresh	CDI	MRRD002	253.50	268.00	2.74	28.0	11.8	-	10.9	-
MR_FR_037	Fresh	FDI/VC/CDI	MRRD002	390.15	404.55	2.75	27.9	11.9	-	11.9	-
MR_FR_038	Fresh	CDI	MDD003	121.00	133.80	2.74	26.0	12.3	-	13.9	-
MR_FR_039	Fresh	CDI/FDI	MDD003	236.20	249.35	2.69	34.1	10.6	-	9.8	-
MR_FR_040	Fresh	FDI/CDI	MDD003	269.00	286.65	2.73	36.3	10.4	-	9.4	-
MR_FR_041	Fresh	CDI/FDI	MDD004	457.95	476.65	2.74	31.8	11.1	-	11.0	-
MR_FR_042	Fresh	VC/FDI	MDD004	519.60	533.40	2.84	31.8	11.4	-	10.5	-
MR_FR_043	Fresh	FDI	MRRD005	259.60	274.20	2.74	30.3	11.4	-	10.6	-
MR_FR_044	Fresh	FDI/CDI	MDD007	198.75	210.70	2.77	30.0	11.5	-	11.6	-
MR_FR_045	Fresh	FDI	MDD015B	174.11	218.86	2.73	29.0	11.6	17.1	11.1	0.515
MR_TR_002	Transition	CDI/FDI/VC	MDD013	20.69	42.00	2.64	77.6	7.5	8.5	7.4	0.123
MR_TR_003	Transition	CDI/VC	MDD013	23.93	30.00	2.65	91.5	7.0	-	6.7	-
MR_TR_004	Transition	VC	MDD012	30.00	35.85	2.65	84.0	7.3	-	7.4	-
MR_TR_005	Transition	CDI/VC	MDD015	24.52	30.53	2.62	176	5.8	-	6.7	-

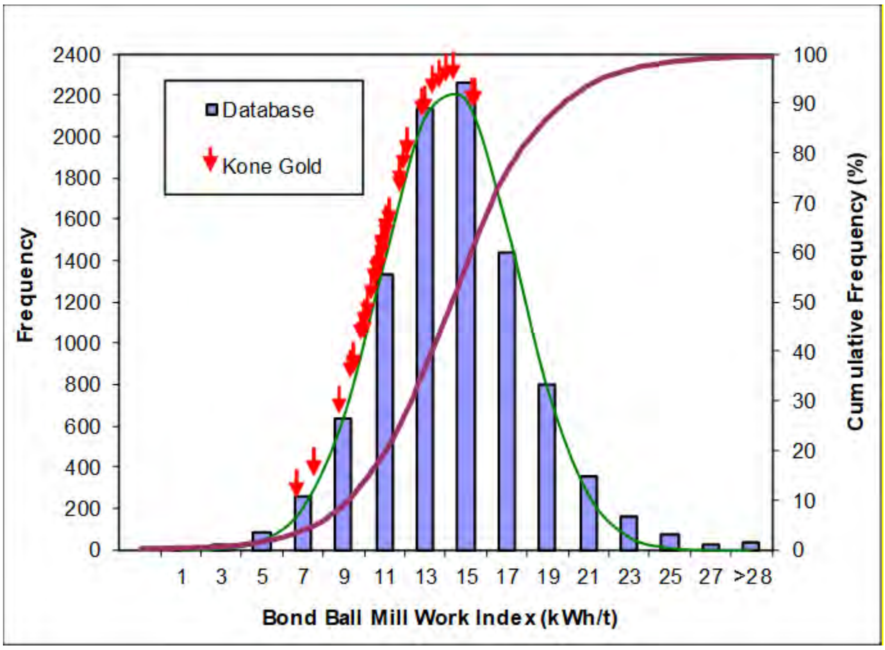
The Bond Crusher Work Indices, Figure 13-4, typically fell in the moderately hard to hard range of hardness of the SGS database, with CWI values ranging from 13.3 to 17.1 kWh/t. The Transition sample MR_TR_002 was found to be significantly softer, with a CWI of 8.5 kWh/t, placing the sample in the moderately soft range of hardness of the SGS database.

Figure 13-4. Bond Crusher Work Index Variability



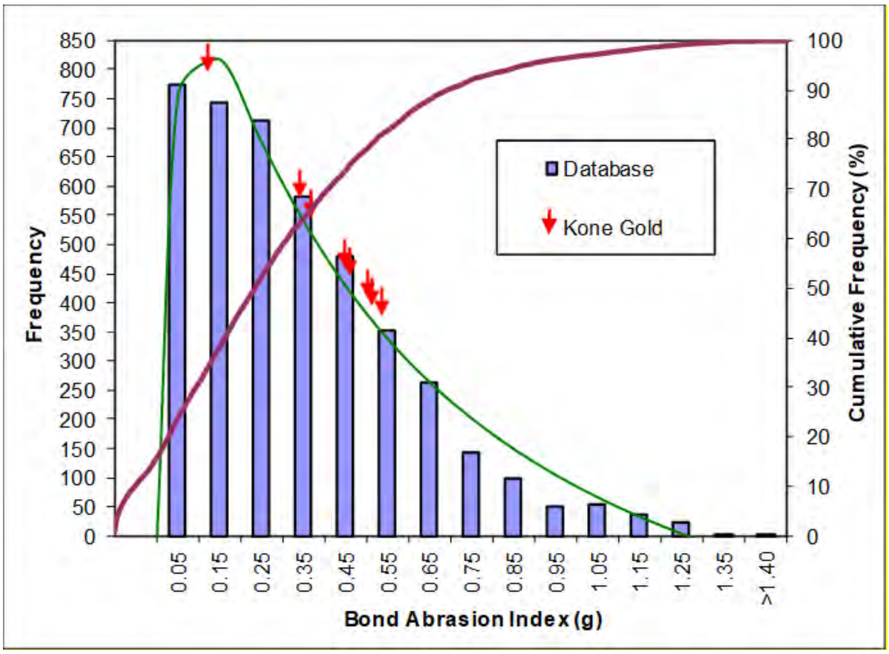
The Bond Ball Mill Work Index (BWI) values, Figure 13-5, ranging from 6.7 kWh/t to 15.4 kWh/t, the samples covered the very soft to moderately hard range of hardness of the SGS database. However, as seen with the SMC test results, the four Transition samples were significantly softer than the Fresh samples. The BWI from the Transition samples ranged from 6.7 to 7.4 kWh/t, which categorized the four samples as very soft. For the 39 Fresh samples, the BWI ranged from 8.7 kWh/t to 15.4 kWh/t and covered the very soft to moderately hard range of hardness. The attained P_{80} values varied from 83 to 89 microns.

Figure 13-5. Bond Ball Mill Work Index Variability



The Abrasivity (Ai) values, Figure 13-6, ranged from 0.34 g to 0.53 g, placing the samples from the medium to abrasive range of abrasivity of the SGS database. The Transition sample MR_TR_002 was found to be significantly less abrasive, with an AI of 0.123 g, placing the sample in the mild range of abrasivity of the SGS database.

Figure 13-6. Bond Ball Mill Work Index Variability



13.4.2 Leach Optimisation Testing

In total, 36 leach optimization tests, Table 13-5 Fresh Composite Optimisation Test Results, were completed using the Fresh Master Composite Sample. The objective of the tests was to determine the optimum:

- Grind size P_{80}
- Cyanide concentration
- Pulp density
- Aeration conditions (air vs. oxygen and impact of pre-aeration)
- Leach retention time

A single test was conducted to evaluate the effect of leaching at higher (40°C) pulp temperatures indicative of West African site conditions. A single test was conducted to evaluate the effect of operating at 15ppm DO as opposed to 25-30ppm DO.

The target pH for all tests during the program was ~10.5-10.7, which was maintained with lime.

The Fresh composite head assay was 0.88g/t Au, 0.10%S with below detection level quantity of Copper, Tellurium, Mercury and Organic Carbon.

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

- Grind Size P80 target of ~75 µm
- Pulp Density = 50% solids (w/w)
- Pulp pH = 10.5-10.7 (maintained with lime)
- 36-hour leach retention time
- Cyanide concentration of 0.5 g/L NaCN (only maintained for 8 hours)
- Dissolved oxygen concentration of ~20-30 mg/L (tests sparged with oxygen)
- No pre-aeration
- No lead nitrate addition

Table 13-5 Fresh Composite Optimisation Test Results

Test Description	CN Test No.	Feed Size P_{80}µm	NaCN g/L	% Solids	Aeration PA h	Aeration CN (type)	Pb $(NO^3)_2$ g/t	DO Target mg/L	CN Consumption kg/t	CaO Consumption kg/t	Au Extraction, % 2 h	4 h	8 h	16 h	24 h	32 h	40 h	**48 h**	Residue, Avg.	Au Head, g/t Calc.	Au Head, g/t Assay	Normalised Extraction **Au %**
SET #1 - Effect of Grind	CN-1	**241**	0.50	50		Air		5-8	0.45	0.40	17.9	30.6	46.0	60.6	66.3	69.7	74.0	**75.8**	**0.19**	0.78	0.88	**76.8**
SET #1 - Effect of Grind	CN-2	**140**	0.50	50		Air		5-8	0.61	0.43	20.6	35.4	51.5	69.5	75.2	75.6	77.8	**79.5**	**0.14**	0.68	0.88	**82.9**
SET #1 - Effect of Grind	CN-3	**122**	0.50	50		Air		5-8	0.62	0.38	18.5	35.7	54.0	68.9	76.6	76.8	77.6	**83.5**	**0.13**	0.76	0.88	**84.7**
SET #1 - Effect of Grind	CN-4	**93**	0.50	50		Air		5-8	0.67	0.38	18.6	32.0	51.0	70.7	80.6	82.0	85.2	**90.2**	**0.10**	0.97	0.88	**88.4**
SET #1 - Effect of Grind	CN-5	**72**	0.50	50		Air		5-8	0.83	0.31	19.2	38.5	61.8	77.5	84.9	86.2	86.1	**90.4**	**0.08**	0.78	0.88	**90.8**
SET #1 - Effect of Grind	CN-6	**57**	0.50	50		Air		5-8	0.74	0.40	23.0	44.6	66.2	80.1	85.2	87.2	90.7	**91.2**	**0.07**	0.74	0.88	**92.1**
SET #1 - Effect of Grind	CN-5R	**73**	0.50	50		Air		5-8	0.54	0.39	19.1	34.9	53.3	70.7	81.8	87.4	91.5	**93.0**	**0.06**	0.85	0.88	**92.7**
SET #2 - Effect of NaCN	CN-7	68	**0.75**	50		Air		5-8	0.96	0.35	14.0	34.8	55.2	69.1	79.3	84.6	85.5	**90.3**	**0.09**	0.88	0.88	**89.6**
SET #2 - Effect of NaCN	CN-8	69	**0.25**	50		Air		5-8	0.34	0.52	13.5	32.3	50.7	69.6	80.0	85.5	88.9	**89.9**	**0.08**	0.74	0.88	**90.8**
SET #3 - "Oxidant" tests	CN-9	72	0.50	50	**4**	Air		5-8	0.47	0.60	33.9	50.0	61.1	71.7	84.7	84.8	83.8	**89.0**	**0.08**	0.68	0.88	**90.8**
SET #3 - "Oxidant" tests	CN-10	72	0.50	50	**4**	Air		5-8	0.47	0.61	31.6	46.0	60.3	73.8	82.4	82.3	84.4	**89.3**	**0.09**	0.79	0.88	**89.6**
SET #3 - "Oxidant" tests	CN-11		0.50	50		**Oxygen**		20-30	0.36	0.21	46.4	70.1	79.8	89.1	90.5	91.6	93.1	**95.8**	**0.07**	1.54	0.88	**92.1**
SET #3 - "Oxidant" tests	CN-12	70	0.50	50		**Air**	**500**	5-8	0.57	0.59	67.7	79.4	82.8	88.4	87.7	86.5	87.7	**93.2**	**0.07**	0.96	0.88	**92.1**
SET #3 - "Oxidant" tests	CN-11R	72	0.50	50		**Oxygen**		20-30	0.14	0.24	37.5	48.8	61.2	74.1	78.2	84.0	86.7	**92.3**	**0.07**	0.85	0.88	**92.1**
SET #4 - Pulp Density	CN-13	72	0.50	**55**		Air		5-8	0.45	0.46	23.6	34.3	53.3	69.4	75.3	79.5	83.7	**88.1**	0.09	0.71	0.88	**89.6**
SET #4 - Pulp Density	CN-14	75	0.50	**55**		Oxygen		20-30	0.25	0.25	31.7	54.7	67.8	76.0	82.3	86.0	86.6	**91.2**	0.07	0.74	0.88	**92.1**
SET #3 - PD & Oxidant tests	CN-22	67	0.50	50		Oxygen	**150**	20-30	0.24	0.27	71.6	83.4	85.4	88.7	86.7	87.9	87.0	**92.0**	0.06	0.75	0.88	**92.7**
SET #3 - PD & Oxidant tests	CN-23	66	0.50	55		Oxygen	**150**	20-30	0.34	0.32	65.1	79.9	87.5	88.0	88.7	89.6	90.0	**92.3**	0.07	0.84	0.88	**92.1**
SET #5 - Hard Stops	CN-28	68	0.50	50		Oxygen		20-30	0.22	0.24					88.6				0.08	0.66	0.88	**90.8**
SET #5 - Hard Stops	CN-29	69	0.50	55		Oxygen		20-30	0.19	**0.18**					85.7				0.12	0.84	0.88	**85.3**
SET #5 - Hard Stops	CN-30	70	0.50	50		Oxygen		20-30	0.16	0.25						91.6			0.06	0.71	0.88	**92.7**
SET #5 - Hard Stops	CN-31	71	0.50	55		Oxygen		20-30	0.19	**0.23**						91.1			0.08	0.90	0.88	**90.2**
SET #5 - Hard Stops	CN-32	68	0.50	50		Oxygen		20-30	0.39	0.25							92.4		0.06	0.78	0.88	**92.7**
SET #5 - Hard Stops	CN-33	69	0.50	55		Oxygen		20-30	0.27	**0.22**							91.3		0.06	0.69	0.88	**92.7**
SET #5 - Hard Stops	CN-34	70	0.50	50		Oxygen		20-30	0.19	0.23								92.4	0.07	0.86	0.88	**92.1**
SET #5 - Hard Stops	CN-35	71	0.50	55		Oxygen		20-30	0.15	0.22								92.6	0.06	0.81	0.88	**92.7**
SET #5 - Hard Stops	CN-36	68	0.50	50		Oxygen		20-30	0.14	0.26						90.8			0.06	0.66	0.88	**92.7**
SET #5 - Hard Stops	CN-37	67	0.50	55		Oxygen		20-30	0.17	**0.26**						90.9			0.07	0.77	0.88	**91.4**
SET #5 - Hard Stops	CN-38	67	0.50	50		Oxygen		20-30	0.13	0.21									0.07	0.88	0.88	**92.1**
SET #5 - Hard Stops	CN-39	71	0.50	55		Oxygen		20-30	0.13	**0.23**									0.06	0.80	0.88	**92.7**
SET #5 - Hard Stops	CN-40	73	0.50	50		Oxygen		20-30	0.13	0.25							91.3		0.07	0.74	0.88	**92.1**
SET #5 - Hard Stops	CN-41	69	0.50	55		Oxygen		20-30	0.21	**0.24**							92.5		0.06	0.80	0.88	**92.7**
SET #5 - Hard Stops	CN-42	67	0.50	50		Oxygen		20-30	0.13	0.25								92.8	0.06	0.83	0.88	**92.7**
SET #5 - Hard Stops	CN-43	65	0.50	55		Oxygen		20-30	0.18	0.26								92.9	0.06	0.85	0.88	**92.7**

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

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

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

Test 84 performed at 40°C pulp temperature, which is the expected operating temperature in Côte D'Ivoire, showed that this has no effect on gold dissolution but cyanide consumption increased when compared with the standard laboratory pulp temperature of circa 20°C. Test 85 performed at 20°C with oxygen but at lower oxygen dosages (15-20ppm DO) also had no effect on gold dissolution but cyanide consumption increased. Further optimisation of ppm DO at elevated site pulp temperature is recommended.

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

13.4.3 Fresh Leach Variability Testing

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

Table 13-6 summarises all Fresh variability tests. The suffix "R" denotes repeat tests. Residue assays varied from 0.02g/t Au to 0.20g/t gold, with an average of 0.09g/t Au.

The average back calculated head assay was 1.10g/t Au which compares well with the measured head assay 1.10g/t Au. However there is clearly some sample splitting errors that are associated with free gold. Screened fire assay protocol is potentially a better technique for any future phases of work.

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

Table 13-6 Fresh Sample Variability Test Results

Sample	CN Test No.	Feed Size P_{80}, µm	NaCN g/L	% Solids	Aeration CN (type)	CN Consumption kg/t	CaO Consumption kg/t	Au Extraction, % 2 h	4 h	8 h	12 h	24 h	32 h	36 h	Average Residue g/t	Au Head, g/t Calc.	Direct	**Au Extraction %**
MR_FR_013	CN-44R2	66	0.50	50	Oxygen	0.12	0.21	29.6	45.9	70.6	89.0	96.3	93.7	95.2	0.05	0.93	1.20	96.2
MR_FR_014	CN-45R	67	0.50	50	Oxygen	0.16	0.12	41.5	62.9	81.7	89.8	91.0	91.6	92.6	0.05	0.61	0.56	92.0
MR_FR_015	CN-46R	70	0.50	50	Oxygen	0.33	0.11	43.4	61.6	78.0	84.7	87.2	86.2	91.6	0.15	1.72	1.17	87.6
MR_FR_016	CN-47	71	0.50	50	Oxygen	0.25	0.17	28.9	46.5	61.5	72.4	85.9	88.3	92.6	0.10	1.26	1.13	91.1
MR_FR_017	CN-48	74	0.50	50	Oxygen	0.20	0.20	50.0	61.3	74.0	80.1	87.9	89.1	93.4	0.10	1.44	1.03	90.8
MR_FR_018	CN-49	74	0.50	50	Oxygen	0.21	0.18	51.1	70.1	81.8	84.0	85.9	85.7	91.6	0.08	0.96	0.78	89.7
MR_FR_019	CN-50	70	0.50	50	Oxygen	0.25	0.21	47.9	61.9	76.6	83.1	94.0	93.6	94.2	0.08	1.39	0.94	91.4
MR_FR_020	CN-51	72	0.50	50	Oxygen	0.19	0.20	40.4	55.0	69.3	77.1	84.4	84.4	89.5	0.07	0.62	0.46	85.9
MR_FR_021	CN-52	72	0.50	50	Oxygen	0.08	0.23	38.0	51.6	69.0	77.3	90.5	93.8	94.6	0.07	1.29	4.33	98.4
MR_FR_022	CN-53R	69	0.50	50	Oxygen	0.10	0.17	28.5	46.2	64.7	74.9	88.9	90.7	93.2	0.15	2.21	2.42	93.8
MR_FR_023	CN-54	80	0.50	50	Oxygen	0.10	0.21	48.4	68.3	78.9	85.5	91.1	92.4	91.9	0.08	0.99	0.81	90.1
MR_FR_024	CN-55R	75	0.50	50	Oxygen	0.12	0.19	39.4	60.2	74.7	85.4	95.7	96.2	95.2	0.06	1.14	0.82	93.3
MR_FR_025	CN-56	81	0.50	50	Oxygen	0.17	0.29	55.5	67.7	81.0	83.6	85.6	84.6	87.9	0.09	0.70	0.57	85.0
MR_FR_026	CN-57	79	0.50	50	Oxygen	0.10	0.19	30.5	40.1	58.1	67.8	82.4	92.1	94.4	0.08	1.42	0.79	90.5
MR_FR_027	CN-58	66	0.50	50	Oxygen	0.23	0.28	40.9	55.2	69.6	79.5	88.3	91.5	92.3	0.05	0.65	0.45	88.8
MR_FR_028	CN-59R	74	0.50	50	Oxygen	0.25	0.19	54.7	65.0	74.8	78.9	86.4	84.7	87.2	0.13	1.02	0.97	86.6
MR_FR_029	CN-60	77	0.50	50	Oxygen	0.18	0.29	60.6	72.5	80.9	83.4	88.2	88.6	89.4	0.11	0.99	0.76	86.1
MR_FR_030	CN-61	71	0.50	50	Oxygen	0.14	0.26	40.8	50.7	69.4	74.8	87.3	91.3	93.0	0.09	1.28	0.76	88.2
MR_FR_031	CN-62	77	0.50	50	Oxygen	0.15	0.27	51.5	67.6	82.6	85.1	90.2	91.6	91.3	0.08	0.92	1.46	94.5
MR_FR_032	CN-63R2	69	0.50	50	Oxygen	0.28	0.17	36.1	54.8	71.5	82.2	88.0	89.8	91.6	0.17	1.97	1.35	87.8
MR_FR_033	CN-64R2	69	0.50	50	Oxygen	0.16	0.22	50.4	68.7	81.4	83.9	92.2	92.8	93.2	0.03	0.44	0.41	92.6
MR_FR_034	CN-65	71	0.50	50	Oxygen	0.09	0.23	58.9	67.1	79.1	83.1	89.7	91.0	88.6	0.06	0.53	0.46	86.8
MR_FR_035	CN-66	70	0.50	50	Oxygen	0.13	0.25	42.7	54.5	68.9	75.4	86.1	90.1	90.8	0.09	0.79	1.30	93.4
MR_FR_036	CN-67	72	0.50	50	Oxygen	0.13	0.23	65.3	76.2	84.2	86.5	91.8	93.1	92.7	0.08	1.09	1.19	93.3
MR_FR_037	CN-68R	73	0.50	50	Oxygen	0.20	0.23	35.2	54.0	69.2	77.4	91.8	89.3	93.5	0.09	1.39	1.96	95.4
MR_FR_038	CN-69	75	0.50	50	Oxygen	0.12	0.25	45.2	62.1	76.6	77.2	87.8	88.6	92.4	0.07	0.92	1.64	95.7
MR_FR_039	CN-70	70	0.50	50	Oxygen	0.13	0.16	40.2	54.7	68.0	75.9	84.8	87.8	90.3	0.09	0.93	0.67	86.5
MR_FR_040	CN-71	76	0.50	50	Oxygen	0.17	0.20	39.1	57.6	75.6	10.3	94.2	90.8	94.2	0.12	2.06	1.35	91.1
MR_FR_041	CN-72	76	0.50	50	Oxygen	0.18	0.19	41.7	60.8	72.8	78.9	84.8	91.1	94.3	0.04	0.71	0.71	94.3
MR_FR_042	CN-73	74	0.50	50	Oxygen	0.25	0.22	60.3	59.7	60.0	64.6	54.9	75.2	86.4	0.12	0.88	0.71	83.1
MR_FR_043	CN-74	68	0.50	50	Oxygen	0.23	0.25	47.2	65.6	75.6	81.3	83.0	84.6	91.6	0.08	0.95	1.17	93.2
MR_FR_044	CN-75	72	0.50	50	Oxygen	0.32	0.38	53.5	68.1	77.7	82.0	86.0	86.3	88.5	0.09	0.78	0.87	89.6
Minimum		66				0.08	0.11							86.4	0.03	0.44	0.41	83.1
Maximum		81				0.33	0.38							95.2	0.17	2.21	4.33	98.4
Average		73				0.18	0.22							91.9	0.09	1.09	1.10	90.7

Figure 13-7 indicates that there is a linear relationship between feed and residue grades for the Fresh variability samples. The data trend confirms consistency of this relationship. The LOM average Fresh feed grade is 0.67 g/t, will generate a residue of 0.063 g/t Au. This relationship between Head and Residue grade has been used for financial modelling.

Figure 13-7. Fresh Samples Head and Residue Gold Grades

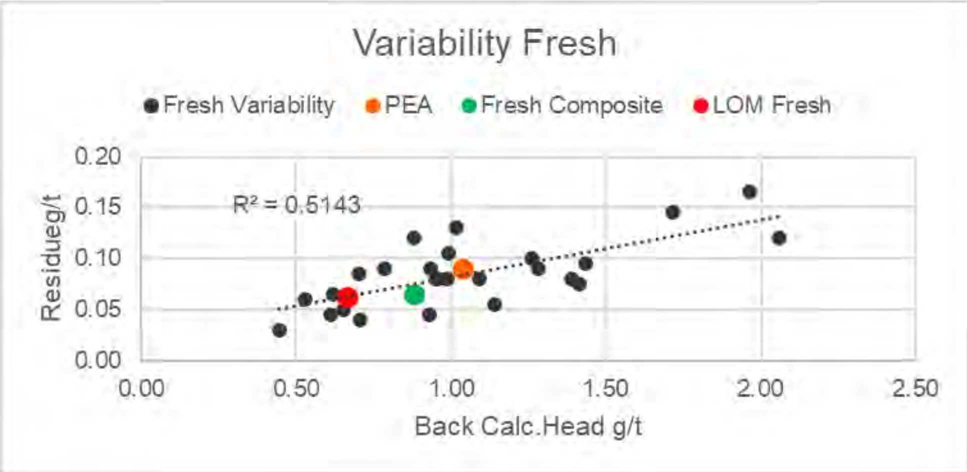


Figure 13-8 indicates that there is a linear relationship between feed sulphur and residue grade. This is consistent across all samples tested.

Figure 13-8. Fresh Samples Head Sulphur and Residue Gold Grade

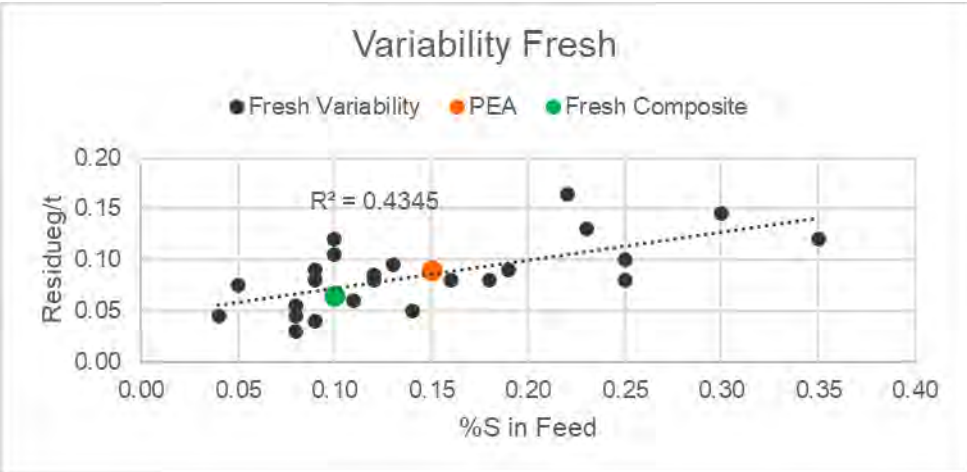


Figure 13-9 indicates that there is a linear relationship between feed sulphur and cyanide consumption. This is consistent across all samples tested.

Figure 13-9. Fresh Samples Head Sulphur and Cyanide Consumption

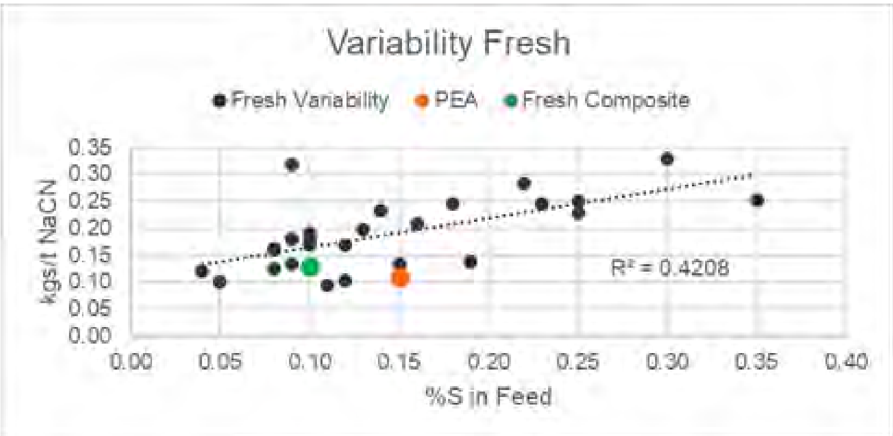
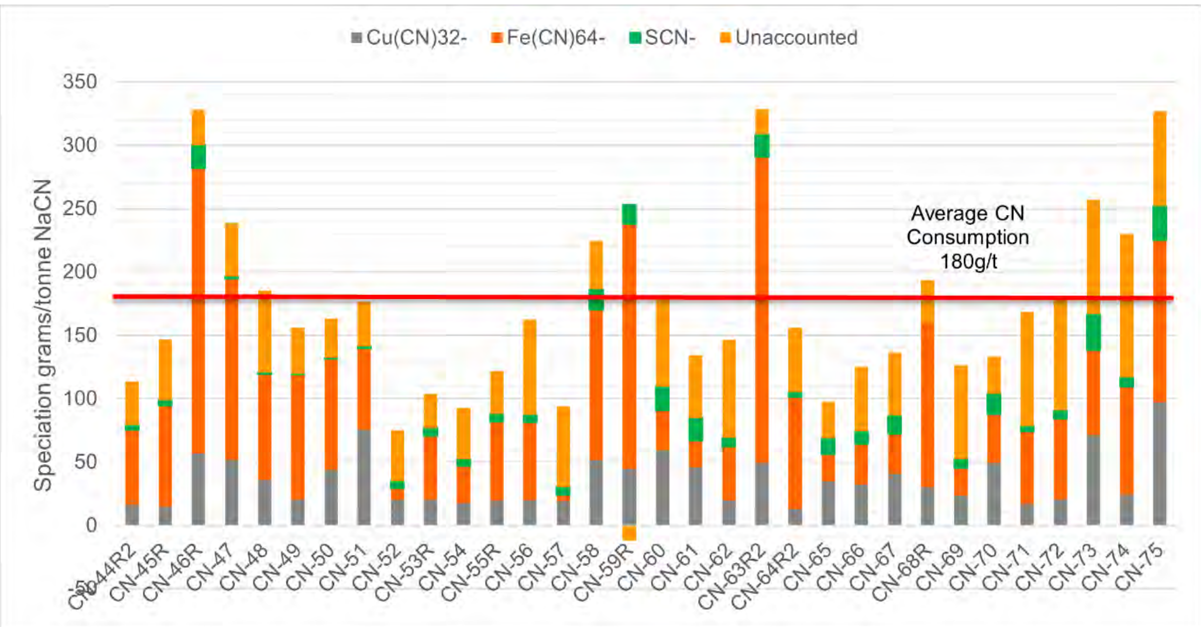


Figure 13-10 shows the cyanide consuming species for the fresh variability samples. The Unnacounted element is considered to be gaseous loss of cyanide as HCN and more specifically hydrolysis and oxidation of gaseous bi-products such as ammonia. Iron appears as the dominant consumer followed by copper. The higher cyanide consuming samples tend to correspond to higher sulphur in feed, which has been determined to be pyrite.

Figure 13-10. Fresh Sample Cyanide Speciation

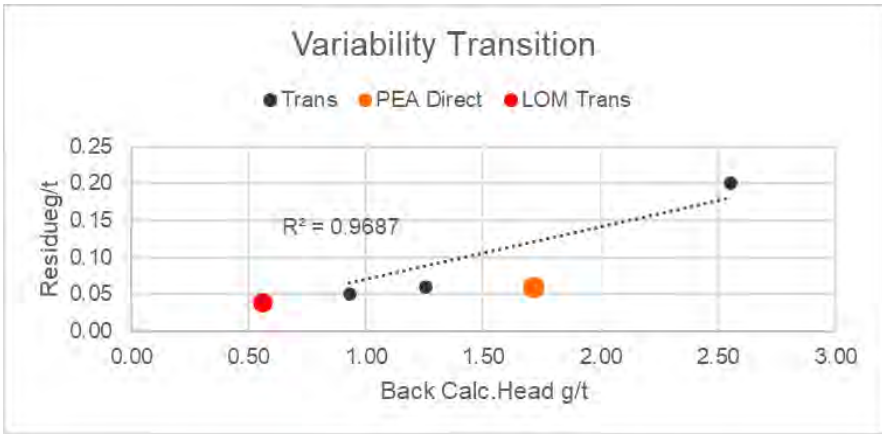


13.4.4 Transition Leach Variability Testing

Table 13-7 presents the Transition variability results, residue gold grades ranged from 0.05 g/t to 0.20 g/t and averaged 0.13 g/t. The average cyanide consumption was very low, 0.08 kg/t NaCN and the average lime consumption was 1.41 kg/t CaO, higher than the Fresh samples. The leach kinetic results indicated that the Transition samples do not require a 36-hour retention time.

Fugure 13-11 shows the linear relationship between Transition Feed grade and the Transition leach Residues. The LOM average Transition grade is 0.57g/t Au and based upon the relationship, the projected g/t Au in tailings is 0.040g/t Au. This relationship between Head and Residue grade has been used for financial modelling. More data at lower head grades is required to verify this relationship.

Figure 13-11. Transition Sample Head and Residue Grades



Variability Transition
Trans
PEA Direct
LOM Trans
0.25
0.20
0.15
0.10
0.05
0.00
Residueg/t
R² = 0.9687
0.00
0.50
1.00
1.50
2.00
2.50
3.00
Back Calc.Head g/t

Table 13-7 Transition Samples Variability Test Results

Sample	CN Test No.	Feed Size P_{80}, µm	NaCN g/L	% Solids	Aeration CN (type)	CN Consumption kg/t	CaO Consumption kg/t	Au Extraction, % 2 h	4 h	8 h	12 h	24 h	32 h	36 h	Average Residue g/t	Au Head, g/t Calc.	Au Head, g/t Direct	**Au Extraction %**
MR_TR_003	CN-76	78	0.50	50	Oxygen	0.01	1.31	37.1	65.2	88.1	94.8	95.6	96.7	95.2	0.06	1.25	1.14	94.7
MR_TR_004	CN-77R	74	0.50	50	Oxygen	0.09	1.26	63.9	78.8	90.4	96.5	100.8	102.3	92.2	0.20	2.55	2.21	91.0
MR_TR_005	CN-78	75	0.50	50	Oxygen	0.10	1.77	71.3	79.4	95.0	85.3	85.8	89.3	94.6	0.05	0.93	0.83	93.9
Minimum		74				0.01	1.26							92.2	0.05	0.93	0.83	91.0
Maximum		78				0.10	1.77							95.2	0.20	2.55	2.21	94.7
Average		76				0.07	1.45							94.0	0.10	1.58	1.39	93.2

Table 13-8 Oxide Samples Variability Test Results

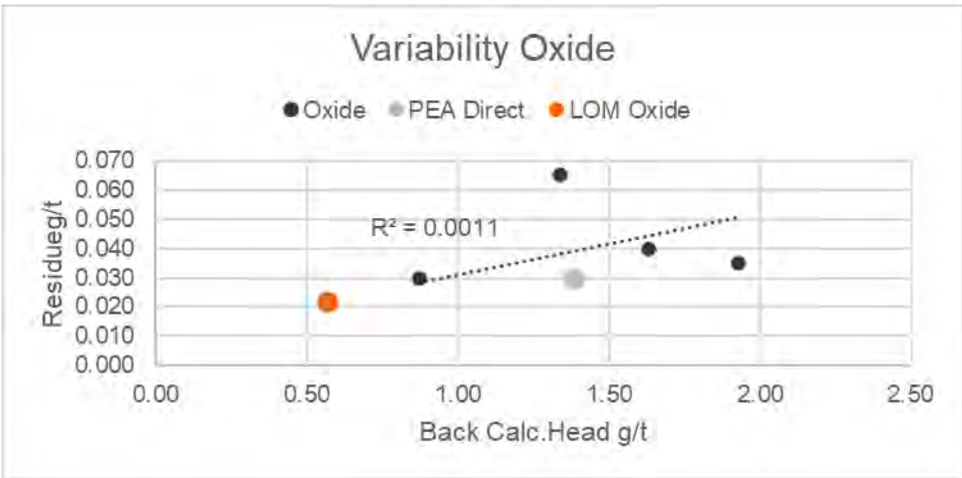
Sample	CN Test No.	Feed Size P_{80}, µm	NaCN g/L	% Solids	Aeration CN (type)	CN Consumption kg/t	CaO Consumption kg/t	Au Extraction, % 2 h	4 h	8 h	12 h	24 h	32 h	36 h	Average Residue g/t	Au Head, g/t Calc.	Au Head, g/t Direct	**Au Extraction %**
MR_OX_002	CN-79R	67	0.50	50	Oxygen	0.11	1.49	56.1	76.9	88.6	93.9	99.2	98.3	98.2	0.035	1.93	1.57	97.8
MR_OX_003	CN-80	68	0.50	50	Oxygen	0.12	1.35	54.7	84.4	91.1	95.1	97.8	94.0	96.6	0.030	0.87	0.77	96.1
MR_OX_004	CN-81	65	0.50	50	Oxygen	0.13	2.59	74.8	89.3	95.0	89.5	95.0	92.4	95.1	0.065	1.34	1.33	95.1
MR_OX_005	CN-82	66	0.50	50	Oxygen	0.22	2.53	68.9	82.4	92.7	97.0	102.0	101.8	97.5	0.040	1.63	0.88	95.5
Minimum		65				0.11	1.35							95.1	0.03	0.87	0.77	95.1
Maximum		68				0.22	2.59							98.2	0.07	1.93	1.57	97.8
Average		67				0.15	1.99							96.9	0.04	1.44	1.14	96.1

13.4.5 Oxide Leach Variability Testing

Table 13-8 presents the Oxide variability results, the residue gold grades were all very low and ranged from 0.03 g/t to 0.07 g/t, averaging 0.04 g/t.

Figure 13-2 shows the relationship between Oxide feed grade and leach residue grade. The LOM average grade is 0.63g/t Au and a residue assay of 0.023g/t. This relationship between Head and Residue grade has been used for financial modelling. More data at lower head grades is required to verify this.

Figure 13-12. Transition Sample Head and Residue Grades



13.4.6 Carbon Modelling

Full Carbon Modelling testwork was performed on the Fresh Composite sample and the 40% Oxide blend sample and leach & adsorption kinetics conducted only on the 20% Oxide Blend sample.

The Kk constants are summarised below in Table 13-9. Both mineralisation types show moderate adsorption kinetics.

Table 13-9 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 <0.10mgs/l are achievable and 0.10mgs/l Au has been assumed for financial modelling. Loss of gold in carbon fines is assumed to be 0.15% of gold in feed for financial modelling.

13.4.7 Tailings Sample Generation

Three samples were generated for Geotechnical and Envonmental testing by Knight Piesold. The samples were generated using the Fresh, Transition and Oxide Composites. The samples were generated using the following leach conditions:

- Grind Size P_{80} target of ~75 µm

- Pulp Density = 50% solids (w/w) for the Fresh and Transition Comps, 39% solids (w/w) for the Oxide Comp
- Pulp pH = 10.5-10.7
- 48-hour leach retention time
- Cyanide concentration of 0.5 g/L NaCN
- Dissolved oxygen concentration of ~5-8 mg/l
- SGS Lakefield tap water

13.4.8 Thickener Tests

Twenty kilograms of the Fresh, Transition, Oxide and 20% & 30% Oxide Blend Composite samples were also provided to Outotec for a full suite of 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 13-10.

Table 13-10 Dynamic Thickeming Testwork Results

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

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

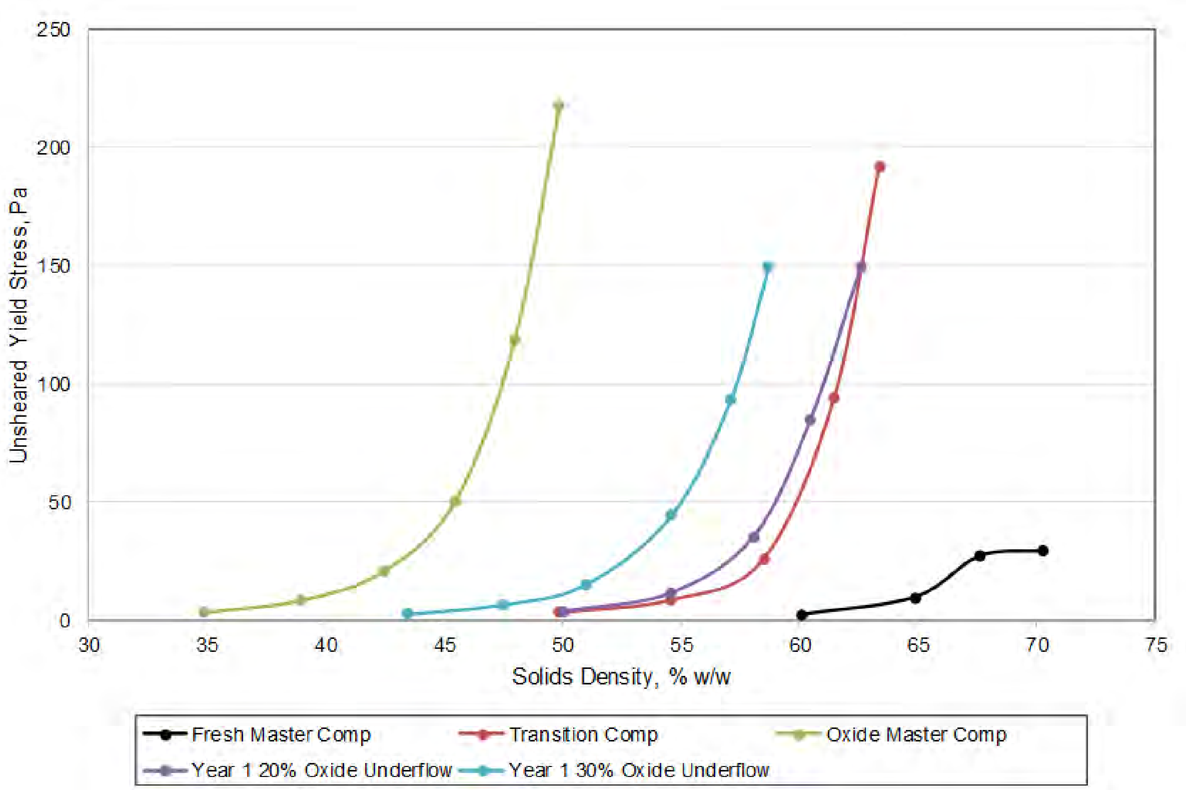
13.4.9 Rheological Tests

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 13-13. Thickener Test Yield Stress Curvesummarises the relationship between unsheared yiled stress and pulp denisty 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 13-13. Thickener Test Yield Stress Curve



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

13.4.10 Metallurgical Results Summary

A comprehensive testwork programme was carried out on 43 comminution and 39 leach optimisation and variability samples

Table 13-11 Comminution Testwork 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 13-11 Comminution Testwork

		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

Table 13-12 Comminution Testwork SGS Classification summarises the comminution SGS classifications.

Table 13-12 Comminution Testwork SGS Classification

Oxidation Zone	SCSE	CWi	BWi	Ai
Fresh	Moderate	Medium	Soft	Medium
Transition	Soft	Very Soft	Soft	Low

The metallurgical tests included oxide, transition and fresh mineralization with results indicating that all material types are amenable to direct tank (CIP) cyanide leaching. Table 13-13 sumarises the leach performance of the samples tested.

Table 13-13 Leach Testwork Summary

Oxidation Zone	Average Test Grade Au g/t	Leach Extraction Recovery % Au	NaCN Consumption kg/t	Lime Consumption kg/t
Fresh	1.09	90.8	0.18	0.22
Transition	1.58	93.2	0.07	1.45
Oxide	1.44	96.0	0.15	1.99

Forecast gold recoveries were estimated based on predicted residue grades for average feed grades, solution loss of 0.01g/t and carbon fines loss of 0.15%. Table 13-14 estimates the gold recoveries based on the average deposit grades, which are good considering the low head grades due to consistently low tailings residues grades being observed. Cyanide consumptions are all low to very low and lime consumptions are low for the predominant fresh zone (88%), but higher for the less dominant transition (5%) and oxide (7%) zones.

Table 13-14 Metallurgical Testwork Summary

Oxidation Zone	LOM Plant Feed	Average LOM Grade Au g/t	Recovery % Au	NaCN Consumption kg/t	Lime Consumption kg/t
Fresh	87%	0.67	89.1	0.18	0.22
Transition	5%	0.57	91.1	0.07	1.45
Oxide	8	0.63	94.8	0.15	1.99

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

Future testwork will be carried out using:

- anticipated site ambient temperature of 35°C - 40°C
- dissolved oxygen concentration of 20mg/L to reflect the use of dedicated oxygen contactor slurry pumps in the leach circuit

14. MINERAL RESOURCE ESTIMATES

14.1 Introduction

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

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

Micromine software was used for data compilation, domain wire framing and coding of composite values and GS3M was used for resource estimation. The resulting estimates were imported into Micromine for resource reporting.

The Mineral Resource estimates have been classified and reported in accordance with NI 43 101 and the classifications adopted by CIM Council in May 2014. The estimates are classified as Inferred, primarily reflecting the drill hole spacing.

The estimates are constrained within an optimal pit generated at a gold price of $US 1,500/oz below a topographic wire frame produced by Montage from DGPS surveys and include mineralization tested by generally 100 m spaced drilling traverses. More broadly sampled peripheral mineralization is too poorly defined for estimation of Mineral Resources.

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

Table 14-1. WGS84 to local grid transformation

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

14.2 Geological Interpretation and Domaining

Drilling to date at Koné has delineated a north easterly trending mineralized zone interpreted to dip to the northwest at around 50º. The transition from gold mineralization to barren host rock is generally characterized by diffuse grade boundaries.

The mineralized domain used for the current estimates was interpreted by MPR on the basis of composited gold grades and captures continuous intervals of greater than 0.1 g/t. Domain boundaries were digitized on cross sections, snapped to drill hole traces where appropriate, then wire framed into a three dimensional solid.

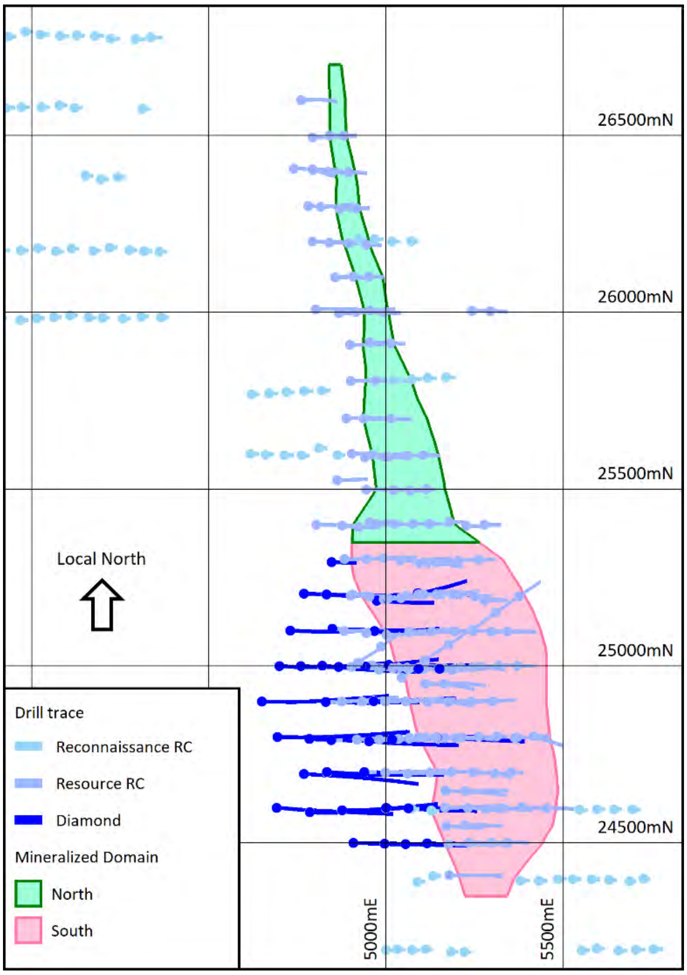
The mineralized envelope is interpreted over 2.4 km of strike with horizontal widths ranging from around 35 to 450 m and averaging around 215 m. It strikes north-north east (350) and dips to the west an average of around 50º. It extends to well below the base of drilling. In the southern portion of the deposit, where mineralization is notably broader than in the north, average drill hole composite gold grades higher in the western portion of domain than in the east.

For resource modelling the mineralized envelope was subdivided into two mineralized domains comprising a southern domain encompassing comparatively higher average drill hole composite gold grades, and a northern domain of lower average composite gold grades.

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

Figure 14.1 shows the surface expression of the mineralized domain relative to drill hole traces of RC and diamond drilling utilized for Resource estimation. Figure 14.3 shows example cross sections of the estimation domains relative to drill hole traces coloured by composited gold grades. and block model estimates. These plots demonstrate that in the southern portion of the deposit many resource drill holes do not penetrate the full width of the mineralized enevelope. In this area drilling prefferentially tests the western, generally higher average gold grade portions of the mineralized enevelope. This selective clustering of drill holes in higher grade areas impacts comparison of model estimates and composite gold grades

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



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

14.3 Estimation Dataset

The estimates are based on two metre down hole composited gold grades from RC and diamond drilling comprising 19,619 composites with gold grades ranging from 0.000 to 51.16 g/t and averaging

0.36 g/t. Samples from RC and diamond drilling provide 33% and 67% of the combined mineralized domain composites respectively.

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

- At 0.03 g/t, the mean gold grade for the background domain composites is notably lower than for the mineralized domain demonstrating that the domaining has been effective in assigning most mineralized composites into the mineralized domains.
- For each mineralized domain there is comparatively little variability in average gold grade between oxidation zones.
- Gold grades show strong positive skewness with a coefficient of variation of around two indicating that MIK is an appropriate estimation technique.

Table 14-2. Estimation dataset statistics

Au g/t	Background Domain				Combined Mineralized Envelope			
	Comp. Ox	Trans.	Fresh	Total	Comp. Ox.	Trans.	Fresh	Total
Number	1,310	446	3,520	5,276	1,790	997	11,556	14,343
Mean	0.04	0.02	0.03	0.03	0.53	0.45	0.48	0.48
Variance	0.01	0.00	0.01	0.01	0.68	0.42	1.01	0.93
Coef. Var.	2.85	2.15	2.56	2.70	1.57	1.45	2.10	1.99
Minimum	0.000	0.000	0.000	0.000	0.005	0.005	0.000	0.000
1st Quartile	0.01	0.00	0.00	0.00	0.12	0.10	0.10	0.10
Median	0.01	0.01	0.01	0.01	0.26	0.24	0.23	0.24
3rd Quartile	0.04	0.02	0.03	0.03	0.63	0.56	0.53	0.55
Maximum	1.87	0.41	1.86	1.87	17.86	11.16	51.16	51.16
Au g/t	**Southern Mineralized Domain**				**Northern Mineralized Domain**			
	Comp. Ox	**Trans.**	**Fresh**	**Total**	**Comp. Ox.**	**Trans.**	**Fresh**	**Total**
Number	523	207	1,418	2,148	1,267	790	10,138	12,195
Mean	0.35	0.23	0.30	0.30	0.60	0.51	0.50	0.51
Variance	0.43	0.07	0.45	0.41	0.77	0.50	1.08	1.01
Coef. Var.	1.88	1.16	2.24	2.10	1.46	1.40	2.06	1.96
Minimum	0.005	0.005	0.003	0.003	0.005	0.005	0.000	0.000
1st Quartile	0.09	0.06	0.05	0.06	0.14	0.11	0.11	0.11
Median	0.19	0.16	0.14	0.15	0.32	0.28	0.25	0.26
3rd Quartile	0.36	0.32	0.32	0.33	0.79	0.67	0.57	0.59
Maximum	8.96	2.17	12.17	12.17	17.86	11.16	51.16	51.16

14.4 Estimation Parameters

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

For each domain, composites from all three oxidation subdomains were combined for determination of indicator thresholds and class mean gold grades. This approach reflects the 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. In the author's experience this approach is appropriate for MIK modelling of highly variable mineralization such as Koné.

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

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

As an example of the variogram models,

Figure 14-2 presents a three dimensional variogram surface map of the median indicator variogram model at variogram value of 0.95.

The four progressively more relaxed search criteria used for MIK estimation are presented in Table 14-4. Search ellipsoids were aligned with dominant domain mineralization orientation and inclined towards the west at 50 °. Search pass 4 informs a small number of panels in broadly sampled areas not informed by Search passes 1 to 3. Panels informed by this search pass represent around 0.3% of estimated mineral resources and reliability of these estimates does not significantly impact confidence in estimated resources.

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

Bulk densities were assigned to the block model by oxidation zone with densities of 1.6, 2.4 and 2.8 t/bcm assigned to completely oxidized, transitional and fresh material respectively. These values reflect the average of the available measurements.

Table 14-3. Indicator thresholds and bin mean grades

Percentile	Background Domain		North Mineralized Domain		South Mineralized Domain	
	Threshold (Au g/t)	Mean (Au g/t)	Threshold (Au g/t)	Mean (Au g/t)	Threshold (Au g/t)	Mean (Au g/t)
10%	0.000	0.000	0.018	0.007	0.043	0.022
20%	0.000	0.000	0.043	0.030	0.085	0.064
30%	0.005	0.001	0.070	0.057	0.130	0.107
40%	0.005	0.005	0.110	0.090	0.185	0.157
50%	0.005	0.005	0.150	0.129	0.255	0.220
60%	0.013	0.009	0.205	0.177	0.355	0.304
70%	0.025	0.019	0.280	0.239	0.490	0.418
75%	0.030	0.029	0.330	0.302	0.590	0.537
80%	0.043	0.037	0.390	0.357	0.715	0.651
85%	0.060	0.051	0.500	0.441	0.892	0.801
90%	0.080	0.070	0.650	0.576	1.175	1.023
95%	0.140	0.104	0.970	0.787	1.770	1.420
97%	0.215	0.172	1.317	1.132	2.245	1.992
99%	0.395	0.281	2.610	1.721	3.635	2.784
100%	1.865	0.736	12.170	5.032	51.160	6.836
		0.395		4.185		5.030
		(Bin Threshold)		(Bin Median)		(Bin Median)

Figure 14-2. Three dimensional variogram plot

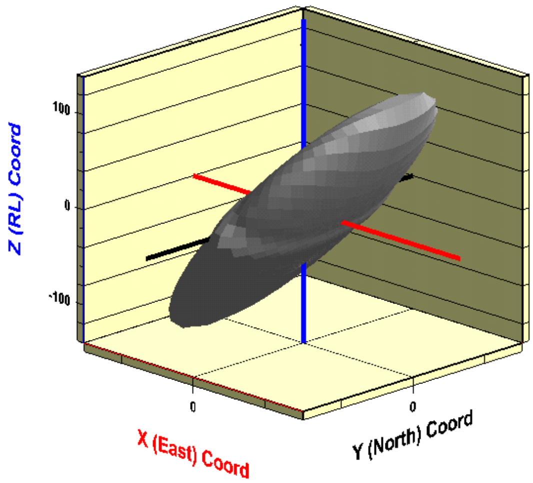


Table 14-4. Search criteria

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

Table 14-5. Variance adjustment factors

Domain	Block/ Panel	Information Effect	Total Adjustment
All domains	0.197	0.833	0.164

14.5 Resource Classification

The current Mineral Resource estimates are classified as Inferred, primarily reflecting the drill hole spacing. The estimates are restricted to model panels within the mineralized envelope tested by generally 50 by 100 m to 100 by 100 m and locally closer spaced drilling defined by polygons digitized for each block model row (Figure 14-3). More broadly sampled peripheral mineralization is too poorly defined for estimation of Mineral Resources and is not included in estimated resources.

14.6 Model Reviews

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

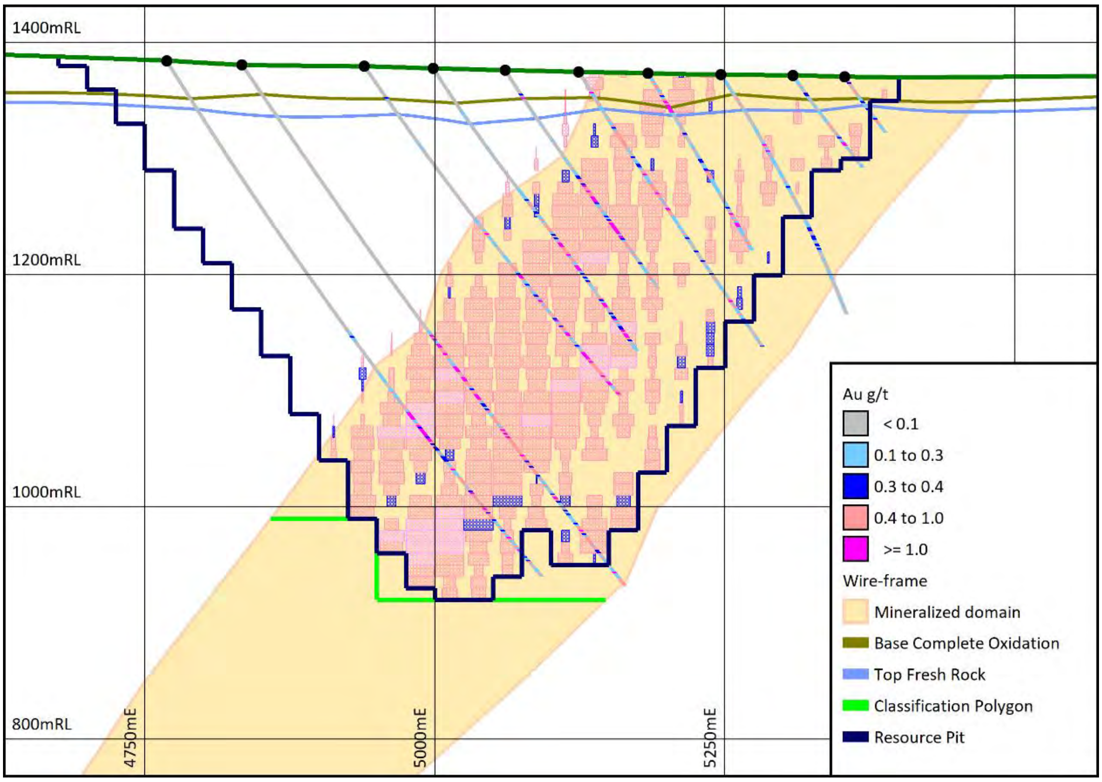
Figure 14.3 shows representative cross sections of the Koné block model. These plots show model panels scaled by the estimated proportion above 0.4 g/t cut off and coloured by the estimated gold grade above this cut off relative to the estimation domains and drill holes traces coloured by two metre composited gold grades. The model panels shown in this figure are restricted to those within the optimal pit used for constraining Mineral Resource estimates.

The plots in Fugure 14-3 include instances where model blocks appear to poorly correlated with mineralized intercepts in nearby drill holes. This reflects the way the resource models have been presented. Only model blocks that contain an estimated resource above 0.4 g/t gold cut off are plotted and the proportion above cut off has been used to scale the east dimension of the blocks for presentation purposes. The scaling occurs about the model block centroid coordinate and therefore introduces the apparent miss match between data and the resource model blocks.

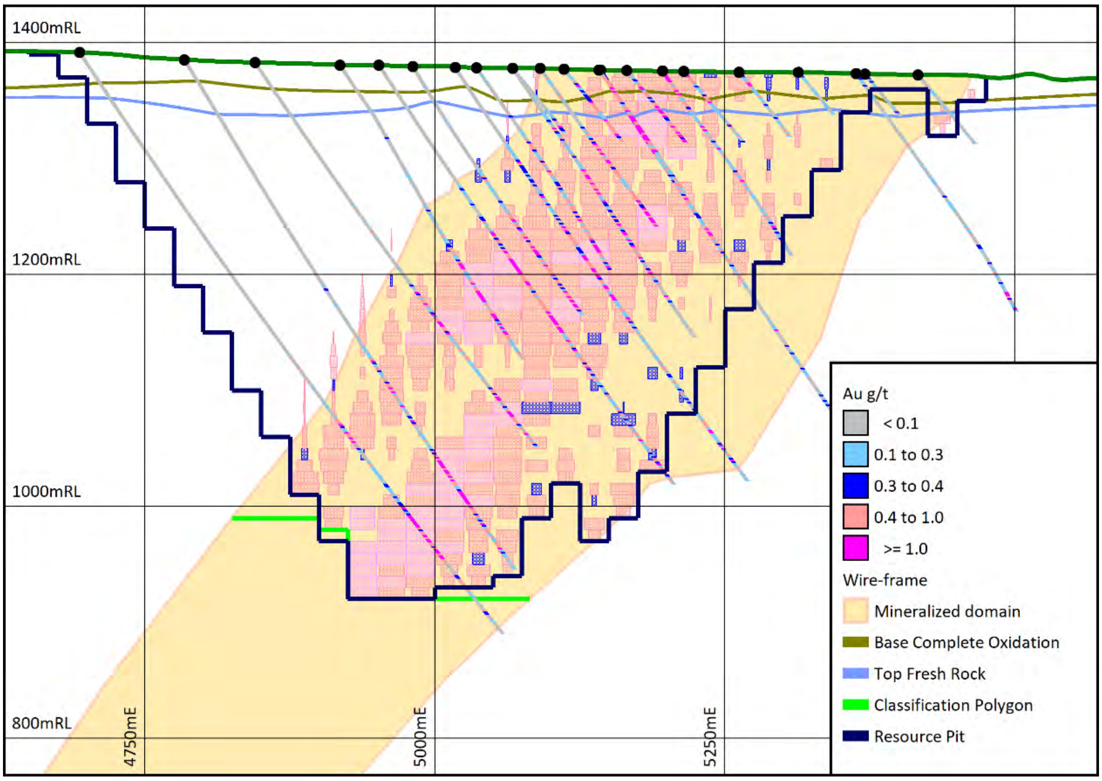
The swath plot in Figure 14.4 compares average mineralized domain estimated panel grades for Inferred Resources and average composite grades by local grid northing. For preparation of this plot average composite gold grades from the southern and northern mineralized domains include upper cuts of 7.3 g/t and 5.7 g/t respectively representing the 99.75th percentile of each dataset reducing the impact of a small number of outlier composite grades.

Figure 14.4 shows that although, as expected average MIK panel grades are smoothed relative to the average composite grades they generally closely follow the trends shown by the composite mean grades with the exception of peripheral areas of limited and variably spaced sampling at the extremities of the deposit. The figure shows local apparent deviations between model and composite trends which are influenced by the variability in drill hole spacing, such as clustering of drilling in areas of higher average grade mineralization around 24,900 mN. These features reflect the distribution of drilling and do not represent biases in the model estimates.

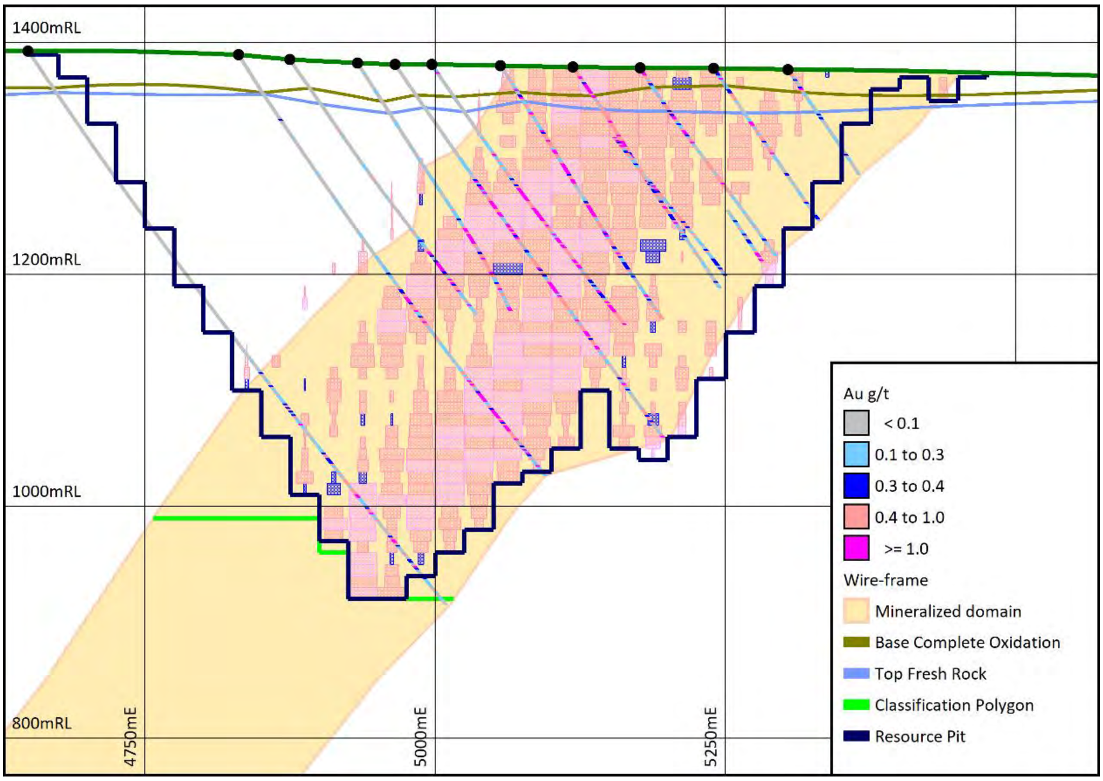
Figure 14-3 Model Blocks at 0.5 g/t cut off



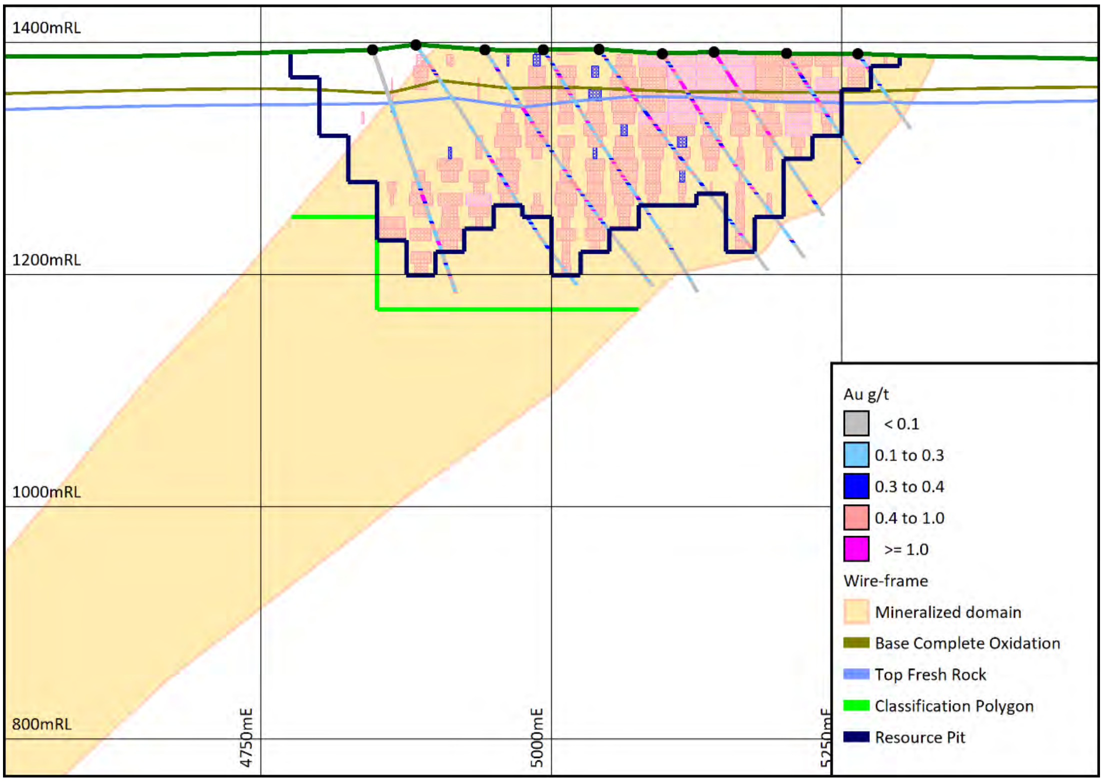
24,700m N



24,900m N



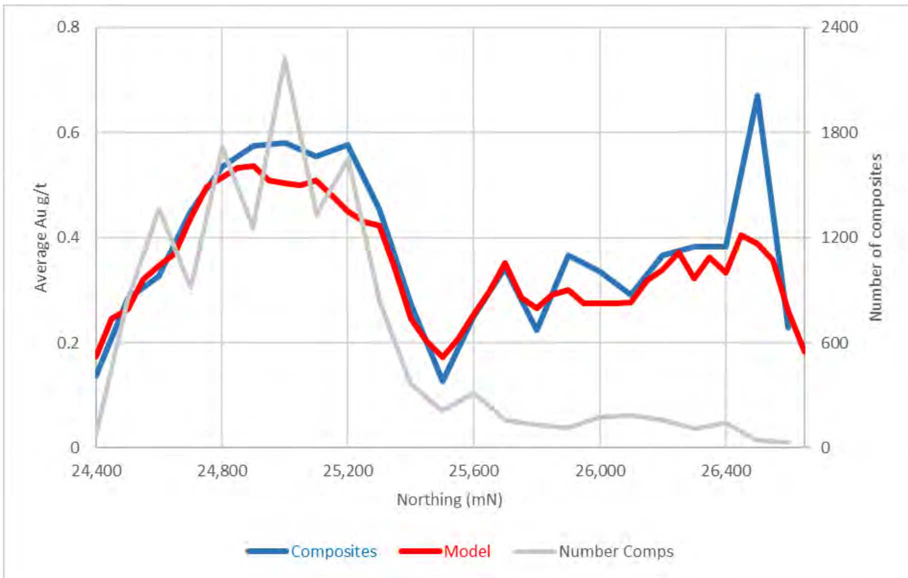
25,100m N



25,300m N

Source MPR, January 2021, Local Grid

Figure 14-4 Estimated Panel Grades Versus Composite Grades



14.7 Resource Estimates

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

The optimal pit shell generated for constraining the Inferred Mineral Resource has dimensions of approximately 800 metres by 2,350 metres, with a maximum depth of around 460 metres.

Table 14-6. Resource pit shell optimization parameters

Gold Price	US$ 1,500/oz			
	Oxide	**Transition**	**Fresh**	**Total**
Wall angle	30º	40º	60º	
Average mining cost	US$ 2.25/t	US$ 2.34/t	US$ 2.90/t	US$ 2.80/t
Mill processing cost	$US 8.86/t	$US 8.07/t	$US 8.93/t	$US 8.89/t
Mill recovery	97.8%	96.50%	91.40%	92.0%
Government royalty	4%	4%	4%	4%
Maverix royalty	2%	2%	2%	2%
Selling costs	US$ 95/oz	US$ 95/oz	US$ 95/oz	US$ 95/oz

Table 14-7 shows the Koné Inferred Mineral Resource Estimates for a range of cut off grades. The author considers the estimates at 0.2 g/t represent the base case or preferred scenario.

Cut off Au g/t	Mt	Au g/t	Au moz
0.1	255	0.51	4.18
0.2	**211**	**0.59**	**4.00**
0.3	161	0.69	3.57
0.4	123	0.80	3.16
0.5	95.6	0.90	2.77
0.6	74.1	1.0	2.38
0.7	57.5	1.1	2.03
0.8	44.7	1.2	1.72

Table 14-8 shows the estimates at 0.2 g/t cut off subdivided by oxidation type. The figures in these tables are rounded to reflect the precision of the estimates and include rounding errors.

The Mineral Resource estimates have an effective date of the 27th of January 2021.

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

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

Cut off Au g/t	Mt	Au g/t	Au moz
0.1	255	0.51	4.18
0.2	**211**	**0.59**	**4.00**
0.3	161	0.69	3.57
0.4	123	0.80	3.16
0.5	95.6	0.90	2.77
0.6	74.1	1.0	2.38
0.7	57.5	1.1	2.03
0.8	44.7	1.2	1.72

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

Oxidation Zone	Mt	Au g/t	Au moz
Oxidized	13.7	0.55	0.24
Transition	8.8	0.56	0.16
Fresh	189	0.59	3.59
Total	**211**	**0.59**	**4.00**

15. MINERAL RESERVE ESTIMATES

There are no Mineral Reserves.

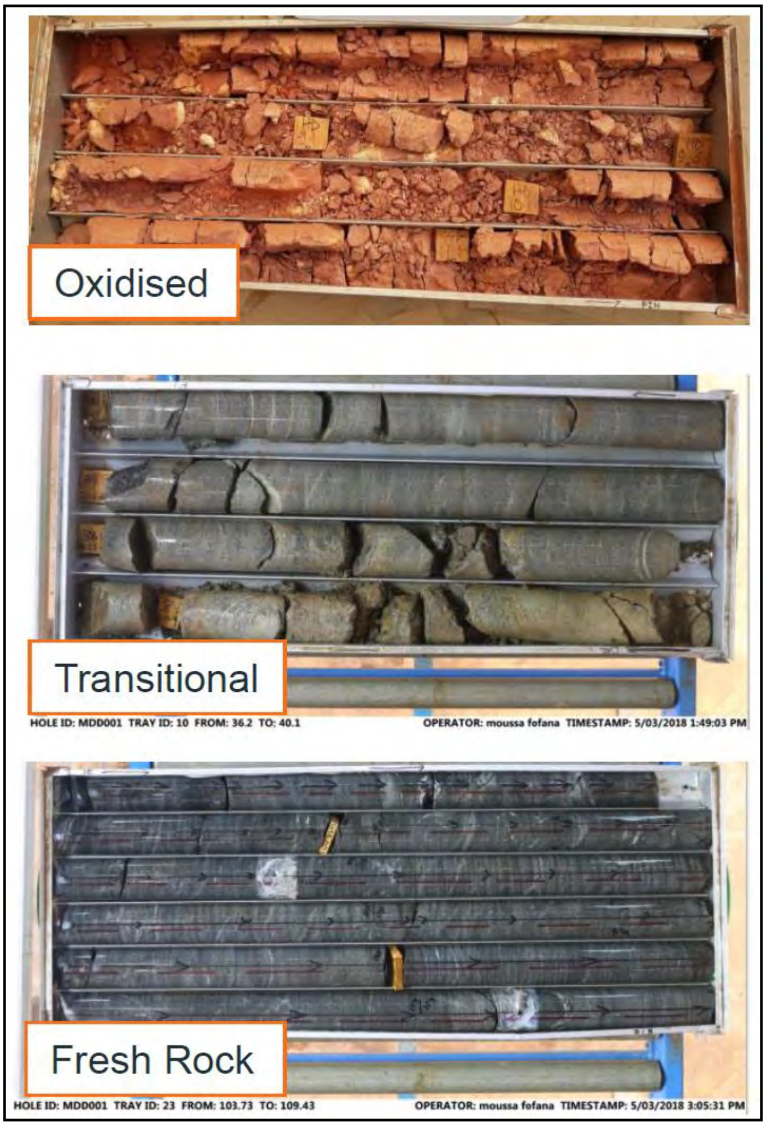
16. MINING METHODS

16.1 Geotechncial considerations

A site visit and review of existing geotechnical data was conducted by SRK Consulting (UK) Limited in January 2021. The report from this review has been used as a basis for subsequent optimisation and pit design work discussed in this study.

There are three clear rock mass domains: oxidised, transitional and fresh rock (Figure 16-1).

Figure 16-1 Rock Mass Domains present within the Koné deposit



Source: Montage

The rock mass within which the pits will be mined is strong, competent and homogeneous. It has therefore been possible to recommend relatively steep slope angles in the unweathered rock (

Table 16-1).

Table 16-1 Preliminary recommended slope angles for Koné deposit

Domain	Description	Range	Overall Slope Angle
Oxidised	Completely weathered/Highly weathered zone	Surface to +/- 35m depth	36°
Transitional	Moderately Weathered/Slightly Weathered	+/- 35m to +/- 60m depth	65°
Fresh Rock - Hangingwall	Unweathered Rock	+/- 60m depth and below	59°
Fresh Rock – Footwall &end walls	Unweathered Rock	+/- 60m depth and below	60°

16.2 Hydrogeology and Mine Dewatering

Hydrogeology is discussed in detail in Section 18.1.3. Groundwater is expected to be encountered at approximately 19 metres below the surface. Groundwater and mine dewatering is assumed to be manageable though the use of sub-horizontal gravity drains in areas where seepage from the pit walls is observed. Dewatering boreholes around the perimeter of the pit may also be required. Groundwater from the pit will be discharged into the WSF and will supplement water pumped from the river, reducing the overall extraction requirements from that water source.

16.3 Mining Method Selection

Given the gold grades contained within the orebody and its proximity to surface, the deposit will be mined via a conventional truck and excavator open pit mining method. The deposit will be mined via two open pits. The main pit to the south contains the bulk of the deposit and is some 475 metres deep, divided into several cutbacks. The smaller northern pit is composed primarily of oxide and transitional material and is approximately 110 metres deep at it's maximum point.

16.4 Mine Optimisation

16.4.1 Scenarios Examined

Previous work has identified that sourcing electric power from the national grid was the preferred option. This was used as a base assumption in this work and other power sources were not considered.

A range of processing throughput rates were examined. Varying the throughput rate had the effect of varying the processing costs which in turn influenced the final shells. Optimisations were conducted with processing costs reflecting a range of throughput rates from 8 mtpa to 11 mtpa. Total annual material movement rates were also considered as part of this study. Sensitivities on gold price were conducted with the gold price varied in $50/oz increments from $1,200/oz to $1,350/oz. For reporting and pit design purposes a gold price of $1,250/oz was used.

16.4.1 Processing Recovery

Metallurgical testwork showed that at very low grades, the recovery dropped due to a semi fixed tails grade. Recovery formulae were developed which considered these tails grades on lower grade portions of the deposit.

Table 16-2 Recovery Calculation shows the formulae used for calculating the tail grade and the recovery based on the head grade and assumed values for solution and carbon fines loss of 0.005 g/t and 0.08% respectively.

Table 16-2 Recovery Calculation

Material	Residue	Recovery
Oxide	0.054 * Head Grade + 0.0267	$\frac{Head\ Grade - Residue - Solution\ Loss}{Head\ Grade} - Carbon\ Fines$
Transitional	0.0706 * Head Grade	$\frac{Head\ Grade - Residue - Solution\ Loss}{Head\ Grade} - Carbon\ Fines$
Fresh	0.25 * Head Grade + 0.004	$\frac{Head\ Grade - Residue - Solution\ Loss}{Head\ Grade} - Carbon\ Fines$

16.4.2 Cut-off grade determination

The model supplied by the geological group was a partial percentage or proportional model. Grade bins were spaced at 0.05g/t increments from 0.1 g/t to 0.7 g/t and then in 0.1g/t increments from 0.7 g/t to 1.5 g/t. In order to process the model it was necessary to determine the cut-off grades prior to running the optimisations.

Cut-off grades were calculated for oxide, transitional and fresh material for the Koné deposit, with the fresh material being further divided into footwall and hanging wall material. Table 16-3 shows the revenue and selling parameters used in the optimisations while Table 16-4 shows the processing costs used for the 11mtpa optimisation run.

Table 16-3 Revenue and selling parameters

Item	Value
Gold Price	$1,250/oz
Refining and Selling Cost	$5/oz
Royalty	3.5%

Table 16-4 Processing costs ($/t processed)

Material	Fixed Costs (annual)	Variable Cost ($/t)	Total Unit Cost ($/t)
Oxide	$21,261k	$6.76	$8.69
Transitional	$21,261k	$5.92	$7.85
Fresh Hanging Wall	$21,261k	$7.05	$8.98
Fresh Footwall	$21,261k	$7.06	$8.99

Table 16-5 shows the calculated cut-off grades for the 11mtpa case at different gold prices. A price of $1,250/oz has been used for the optimisation. The cut-off grades for this price correspond to the grade bins shown in Table 16-6.

Table 16-5 Cut-off grade calculations

	$1,200	**$1,250**	$1,300	$1,400	$1,500	$1,700
Oxide	0.24	**0.23**	0.22	0.22	0.20	0.17
Transitional	0.22	**0.22**	0.21	0.20	0.18	0.16
Fresh Hanging Wall	0.27	**0.26**	0.25	0.24	0.22	0.19
Fresh Footwall	0.28	**0.27**	0.26	0.25	0.23	0.20

Table 16-6 Cut-off grade bins

	Cut-off Grade	Model Grade Bin
Oxide	0.23	0.20
Transitional	0.22	0.20

Fresh Hanging Wall	0.26	0.25
Fresh Footwall	0.27	0.25

16.4.3 Cost data for optimisation

It is assumed that mining is conducted by a mining contractor. Mining costs were broken into base and incremental mining costs. Costs were built from first principles using knowledge of several mining contracts in similar countries in West Africa and recent contractor quotes from Côte d'Ivoire.

For the purposes of the optimisatrion, the mining fleet was assumed to comprise of 90t rigid body haul trucks with suitably sized loading units.

Unit costs were determined for the following items:

- Loading
- Fixed hauling component
- Drill & Blast
- Ancillary
- Grade Control
- Mine Admin

Ancillary and Mine Admin costs were fixed for all material types while loading, hauling and drill & blast costs were varied to reflect oxide/fresh rock and surface haulage distances for crusher feed and waste. Grade control costs were applied to mineralisation only.

Table 16-7 Summary of Fixed Mining Costs

Ore/Waste	Feed	Feed	Feed	Waste	Waste	Waste
Material	Oxide	Transitional	Fresh	Oxide	Transitional	Fresh
Loading	0.32	0.34	0.36	0.32	0.34	0.36
Fixed Hauling	0.34	0.34	0.34	0.36	0.36	0.36
Drill & Blast	0.89	0.89	1.00	0.74	0.74	0.91
Ancillary	0.50	0.50	0.50	0.50	0.50	0.50
Mine Admin	0.14	0.14	0.14	0.14	0.14	0.14
Grade Control	0.26	0.26	0.26	0.00	0.00	0.00
Total Fixed Cost	2.45	2.47	2.60	2.06	2.08	2.28

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

16.4.4 Optimisation Scenarios

All optimisations were performed in the Deswik software using the Pseudoflow tool. Initial optimisations examined the effects of differing throughput rates on the economics.

Figure 16-2 and Figure 16-3 show a comparison of various optimisation results normalised to the 10mtpa case. Figure 16-4 shows the optimisation results by Revenue Factor for the 11 Mtpa base case.

The options analysis showed that the 11 Mtpa case provided the best value and a reasonable mine life of 14.3 years. A high level NPV analysis showed that the case also provided an NPV that was 9.5% greater than the second most valuable scenario.

Based on current market conditions and future projections, a gold price of $1,250/oz was chosen as

the optimisation price. The shells generated were examined to determine the appropriate shell to guide design. The optimisatio graph shows a fairly consistent increase in ore and waste tonnes, with no significant change in strip ratio or value (Figure 16-4). Based on this information, the shell representing revenue factor 1.00 was chosen for the basis of design.

Figure 16-2 Comparison of physical results based on throughput

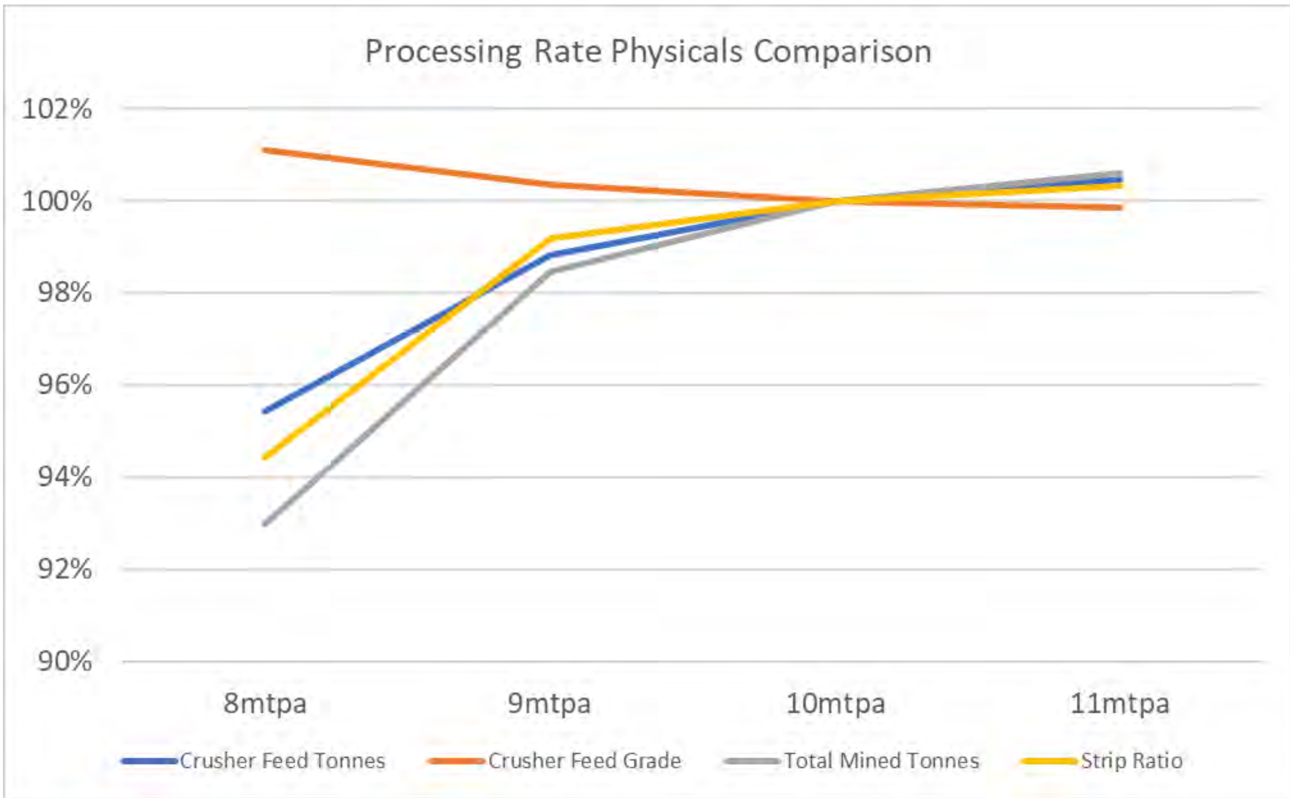
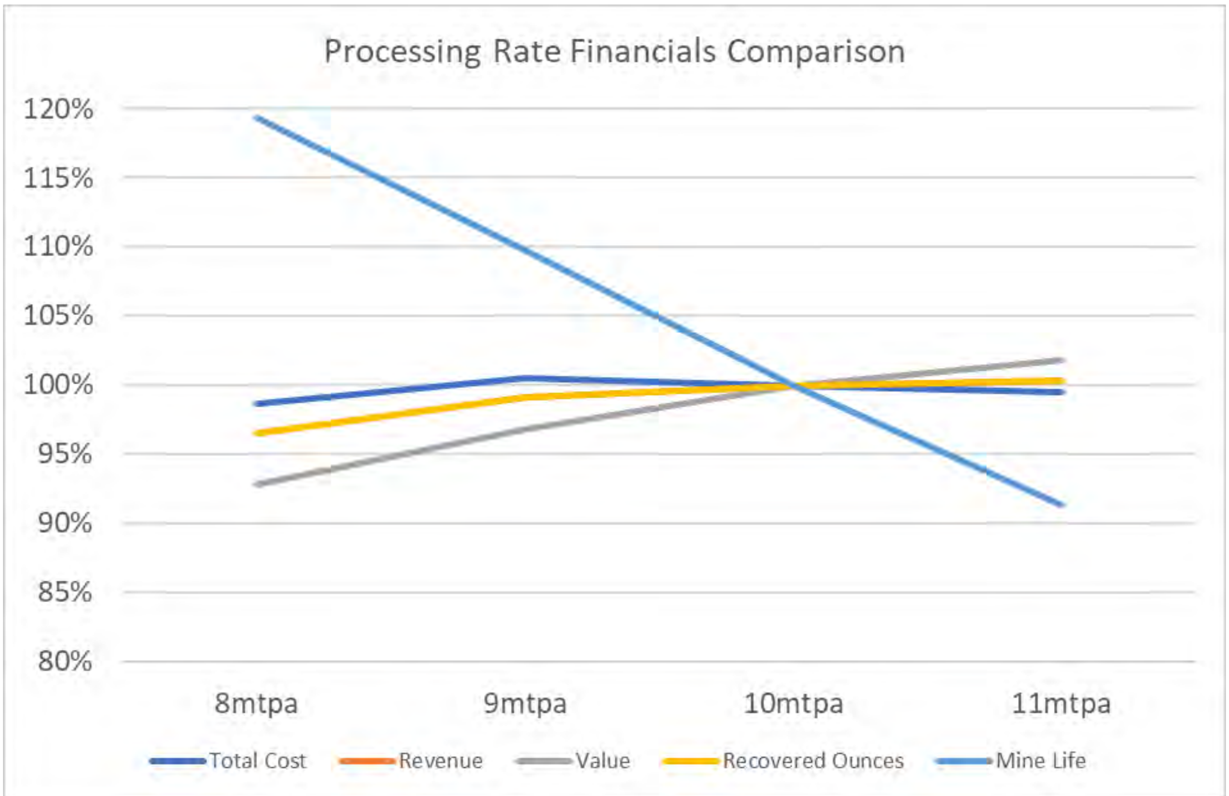


Figure 16-3 Comparison of production and financial results based on throughput



16.5 Mine Design and Sequencing

16.5.1 Mine Design

The Koné deposits will be mined by two open pits, with the bulk of the mill feed contained within the southern pit with the smaller northern pit contributing mainly oxide and transitional ore. The pit inventory contains a total of 161.1 million tonnes of millfeed at a grade of 0.65 g/t Au. This is associated with 149.6 million tonnes of waste rock, providing a strip ratio of 0.93:1 (tonnes of waste:tonne of millfeed) (Table 16-8).

Table 16-8 Koné Pit Inventory

		South Pit	North Pit	Total Pit Inventory
Oxide Millfeed Tonnes	*Mt*	10.08	2.34	12.42
Oxide Millfeed Grade	*g/t*	0.59	0.42	0.56
Transitional Millfeed Tonnes	*Mt*	7.47	1.12	8.59
Transitional Millfeed Grade	*g/t*	0.58	0.46	0.57
Fresh Footwall Millfeed Tonnes	*Mt*	4.95	0	4.95
Fresh Footwall Millfeeed Grade	*g/t*	0.61	0	0.61
Fresh Hanging Wall Millfeed Tonnes	*Mt*	130.80	4.29	135.09
Fresh Hanging Wall Millfeed Grade	*g/t*	0.67	0.55	0.67
Total Fresh Millfeed Tonnes	*Mt*	135.76	4.29	140.05
Total Fresh Millfeed Grade	*g/t*	0.67	0.55	0.66
Total Millfeed Tonnes	***Mt***	**153.31**	**7.75**	**161.06**
Total Millfeed Grade	***g/t***	**0.66**	**0.50**	**0.65**
Oxide Waste Tonnes	*Mt*	16.25	4.53	20.78
Transitional Waste Tonnes	*Mt*	9.90	1.31	11.21
Fresh Footwall Waste Tonnes	*Mt*	2.31	0	2.31
Fresh Hanging Wall Waste Tonnes	*Mt*	110.02	5.26	115.28
Total Fresh Waste Tonnes	*Mt*	112.33	5.26	117.59
Total Waste Tonnes	***Mt***	**138.48**	**11.10**	**149.58**
Total Tonnes	***Mt***	**293.74**	**19.39**	**310.63**
Strip Ratio	*t waste : t ore*	0.90:1	1.43:1	0.93:1

Figure 16-4 and Figure 16-5 show the engineered designs for the North and South pits respectively.

Figure 16-4   North Pit Engineered Design

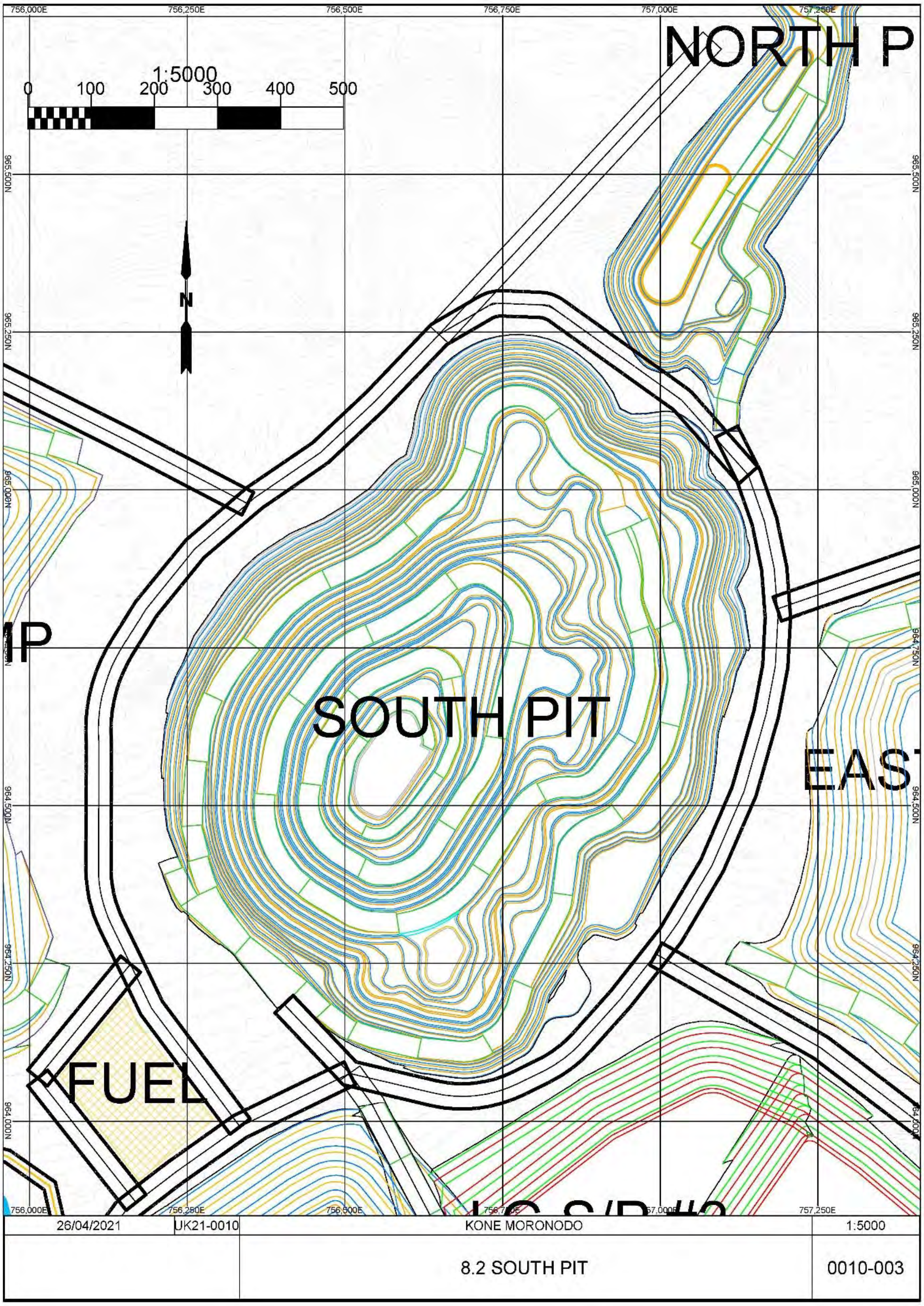
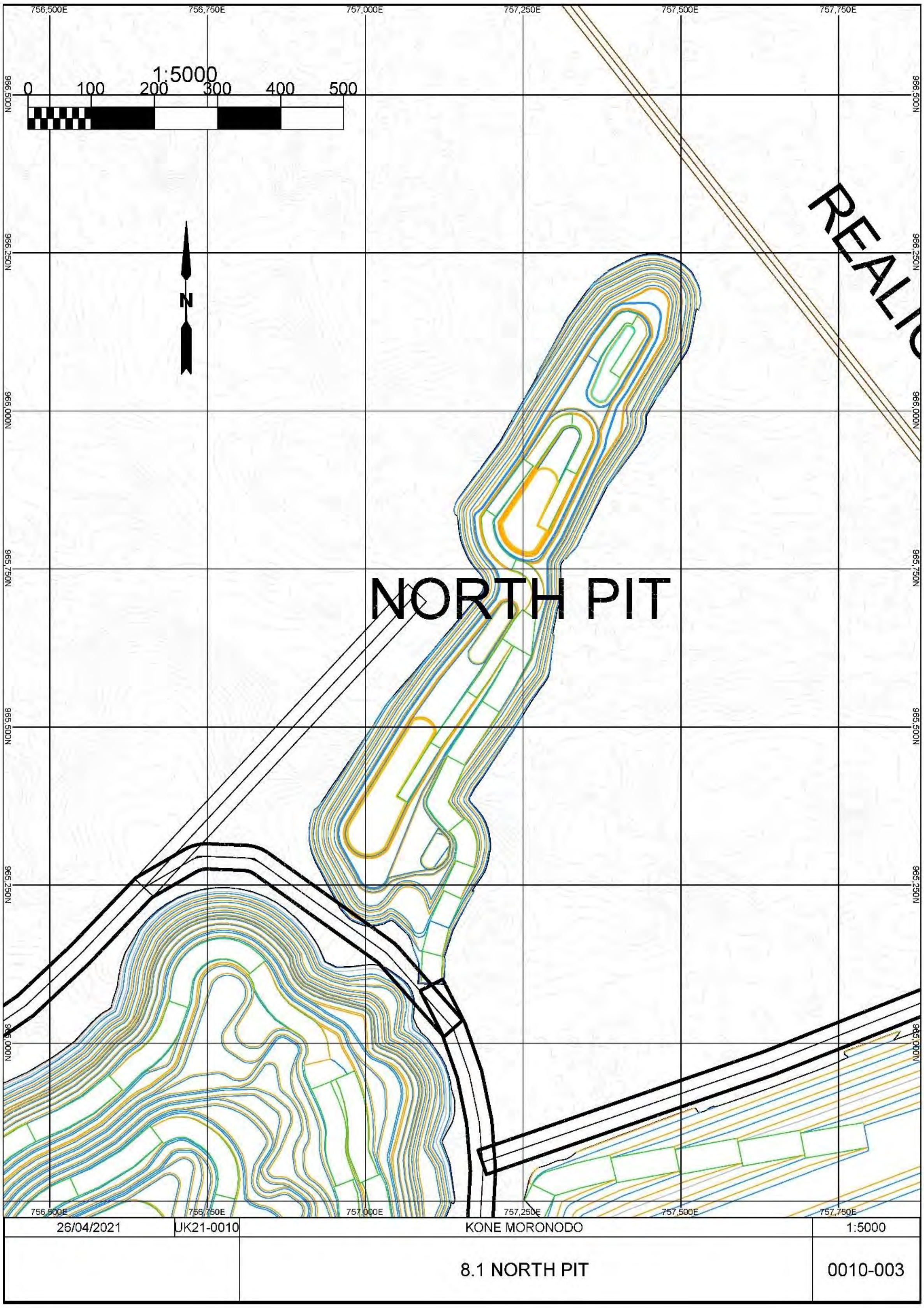
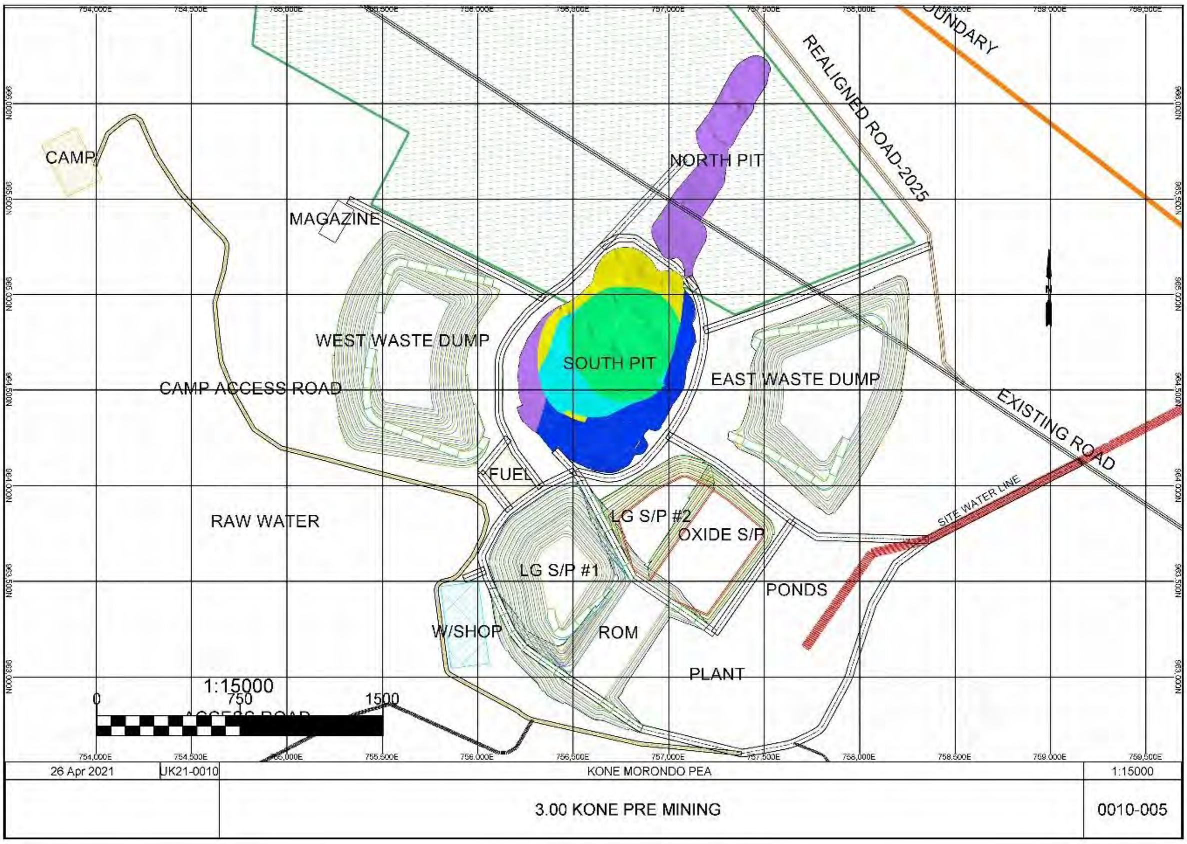


Figure 16-5 North Pit Engineered Design



Waste rock dumps, stockpiles and a Run of Mine (ROM) pad were also designed and used in subsequent scheduling and haulage activities. The designs can be seen in Figure 16-6

Figure 16-6 Starting Pit Surfaces Prior to Mining Pre Strip



16.5.2 Mine Schedule

Earlier studies had identified that an accelerated mining schedule with stockpiling and subsequent rehandling and processing of low grade material provided the best NPV for the project. The highest NPV came from a combination of a 35 mpta mining rate option with an 11 mpta mill throughput. This combination was used for this study.

Figure 16-6 shows the pit surfaces prior to the commencement of mining in year 0. The shaded areas indicate the materials to be mined whereas the outlines indicate the designs of dumps, roads, stockpiles and ROM to be built. Figure 16-7 shows the resulting surfaces after the initial prestrip year, as can be seen the ROM pad has been constructed from waste material and the oxide ore material is stockpiled on the Oxide S/P adjacent to the ROM. Roads and infrastuture pads for the workshop, plant and fuel farm are also constructed during this period.

Figure 16-8 shows the end of the mine life after the initial year of prestrip and 9 years of mining. This figure shows the mined pits, with the full lower grade stockpiles. The stockpiles contain the lower grade material mined but not processed, with the oxide stockpiled separately to the fresh and transitional materials.

Figure 16-7 End of Prestrip – Construction of ROM Pad

Figure 16-9 shows the end of project surface will all of the low grade stockpiles reclaimed and processed. In practice, progressive rehabilitation activites would be conducted so that most of the rehabilitation is completed before this point in time.

Figure 16-8 End of Mine Life Year 9– Pit & Dump Surfaces

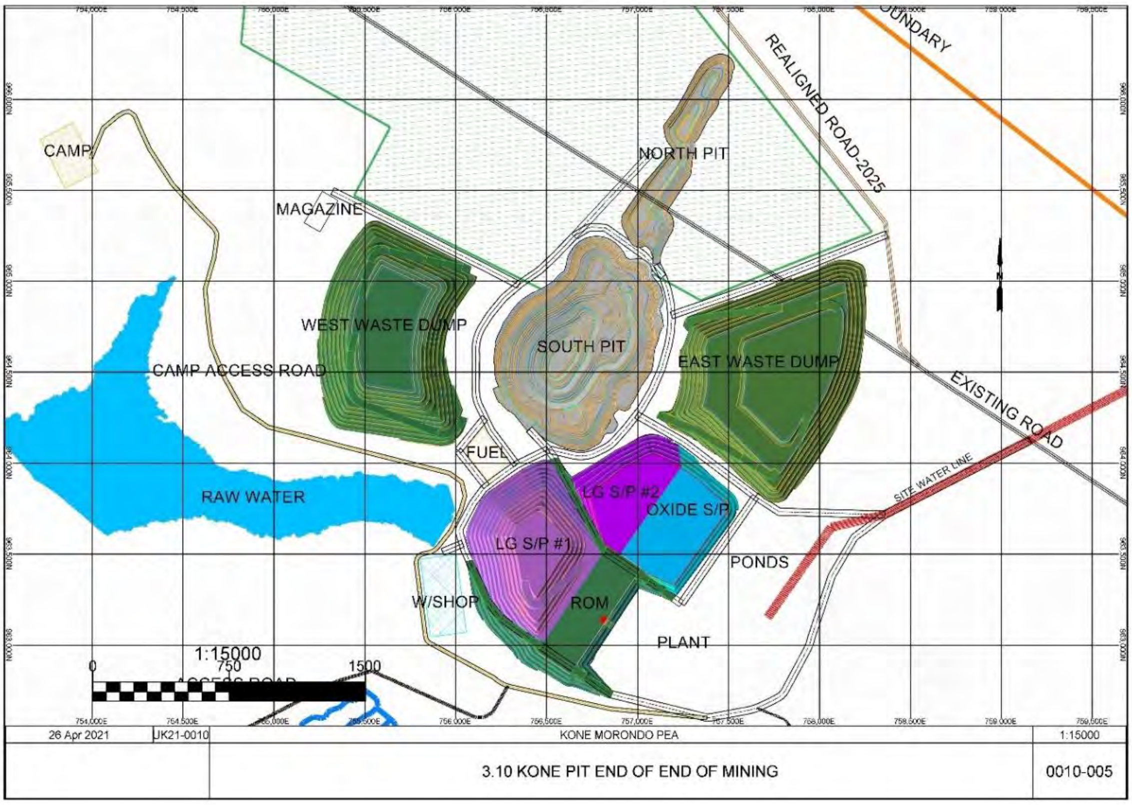


Figure 16-9 End of Processing Life – Pit & Dump Surfaces

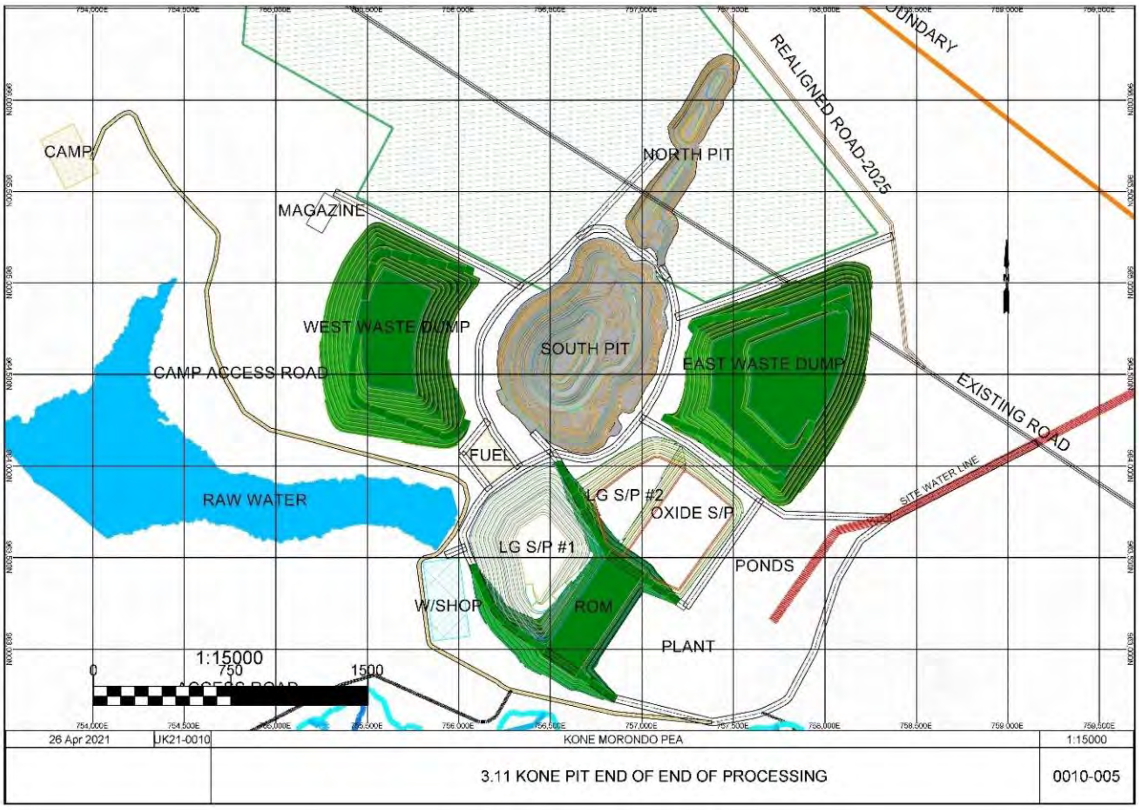


Table 16-9 and Figure 16-10 shows the mining schedule indicating a mine life of 9 years plus six months of pre-strip. Figure 16-11 show the processing schedule which highlights the high grade feed material for the first 9 years while the pit is in operation.

Table 16-9 11Mtpa Mining 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
Mineralized Material Mining																		
South Pit Mineralized Material	Mt	153.3	6.1	15.2	21.4	20.1	14.5	9.8	15.5	16.2	15.1	19.4	-	-	-	-	-	-
Grade	Au g/t	0.66	0.70	0.70	0.70	0.73	0.58	0.56	0.59	0.62	0.61	0.72	-	-	-	-	-	-
North Pit Mineralized Material	Mt	7.7	-	-	-	-	1.0	1.4	1.5	1.3	2.5	0.1	-	-	-	-	-	-
Grade	Au g/t	0.50	-	-	-	-	0.40	0.44	0.47	0.49	0.59	0.75	-	-	-	-	-	-
Total Mineralized Material	Mt	161.1	6.1	15.2	21.4	20.1	15.5	11.2	17.0	17.4	17.6	19.4	-	-	-	-	-	-
Total Grade	Au g/t	0.65	0.70	0.70	0.70	0.73	0.57	0.55	0.58	0.61	0.60	0.72	-	-	-	-	-	-
Waste Mining																		
South Pit Waste	Mt	138.5	7.2	15.3	13.5	14.9	16.3	21.6	15.9	15.2	14.9	3.6	-	-	-	-	-	-
North Pit Waste	Mt	11.1	-	-	-	-	2.0	2.2	2.1	2.3	2.4	0.0	-	-	-	-	-	-
Total Waste	Mt	149.6	7.2	15.3	13.5	14.9	18.3	23.7	18.0	17.6	17.4	3.6	-	-	-	-	-	-
Strip Ratio	w:o	0.93	1.18	1.01	0.63	0.74	1.18	2.11	1.06	1.01	0.99	0.19	-	-	-	-	-	-
Stockpile Rehandle	Mt	73.9	-	1.7	0.8	0.5	0.8	5.4	0.3	0.2	0.4	0.4	11.0	11.0	11.0	11.0	11.0	8.2
Processing																		
Oxide Mineralized Material	Mt	12.4	-	1.5	1.1	0.5	1.0	0.1	0.3	0.1	0.4	0.4	0.4	0.4	1.6	1.7	1.7	1.2
Grade	Au g/t	0.56	-	0.98	0.97	0.97	0.61	0.73	0.46	0.43	0.40	0.40	0.40	0.40	0.39	0.39	0.39	0.39
Transition Mineralized Material	Mt	8.6	-	2.6	0.1	-	0.6	0.2	-	0.1	-	-	1.0	-	-	-	-	4.1
Grade	Au g/t	0.57	-	0.87	1.00	-	0.70	0.60	-	0.72	-	-	0.41	-	-	-	-	0.37
Fresh Mineralized Material	Mt	140.0	-	5.8	9.8	10.5	9.5	10.7	10.7	10.8	10.5	10.5	9.6	10.6	9.4	9.4	9.3	2.9
Grade	Au g/t	0.66	-	0.91	0.95	0.96	0.69	0.59	0.67	0.73	0.71	0.95	0.45	0.45	0.45	0.45	0.44	0.42
Total Mineralized Material	Mt	161.1	-	9.9	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	8.2
Total Grade	Au g/t	0.65	-	0.91	0.96	0.96	0.68	0.59	0.66	0.73	0.70	0.92	0.45	0.45	0.44	0.44	0.43	0.39
Production and Recoveries																		
Gold Production	koz	3,012	-	265.4	307.8	308.3	216.9	183.7	209.3	231.3	219.9	295.7	137.9	138.5	136.7	136.1	132.5	91.4
Processing Recoveries	%	89.4%		91.6%	91.1%	90.8%	89.7%	88.3%	89.1%	89.5%	89.4%	90.6%	86.9%	86.6%	87.1%	87.1%	87.0%	88.8%

Figure 16-10 Scheduled material mined

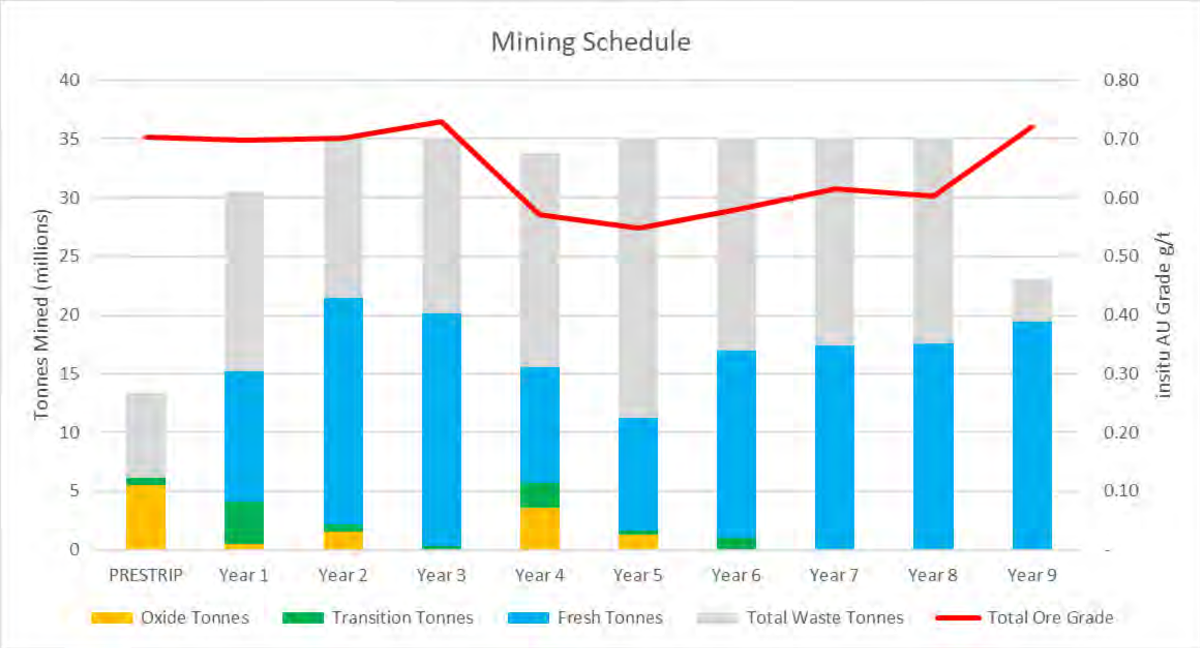
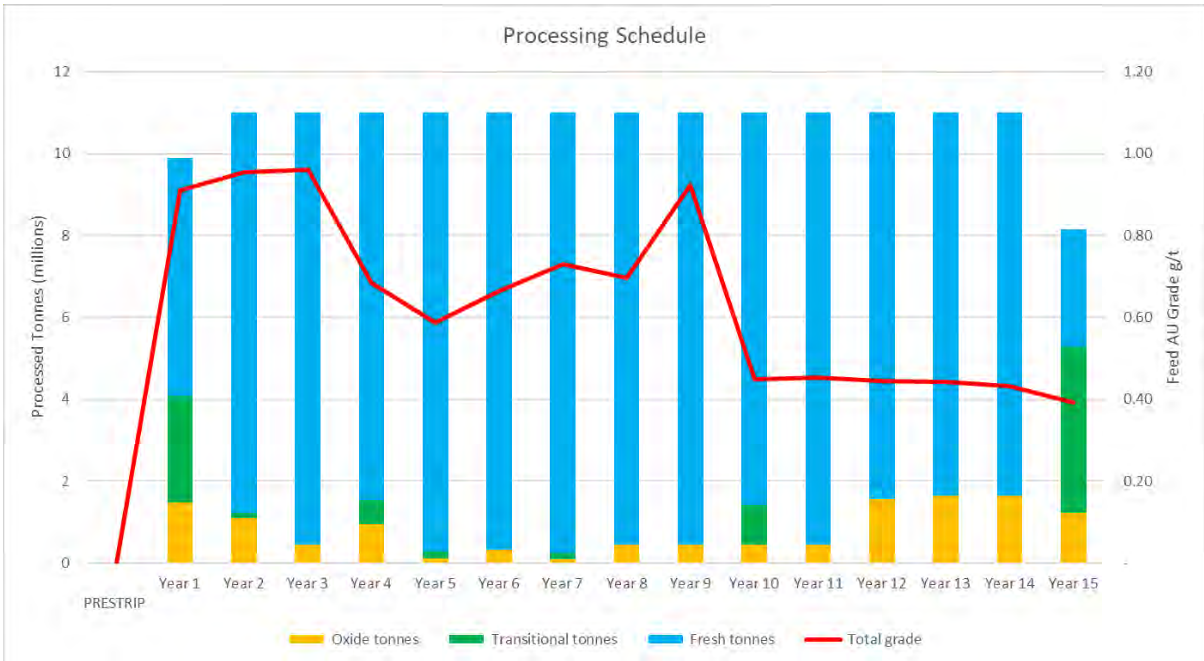


Figure 16-11 Scheduled material Processed



16.6 Operating Strategy

The project has been developed around the assumption that the mining operations will be carried out by a contractor, on a cost per tonne basis. Explosives and fuel are included in this rate, although supply of one or both of these may be through the Owner.

The Owner will be responsible for all geological, geotechnical and mine engineering activities. The Owner's team will include:

- A Mine Manager and Alternate Mine Manager
- Geologists (resource and grade control)
- Mining Engineers (scheduling will be undertaken by the principal to a monthly level)
- Geotechnical staff
- Surveyors
- Contract management personnel (including supervisors)

All mobile maintenance will be the responsibility of the contractor.

17. 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.

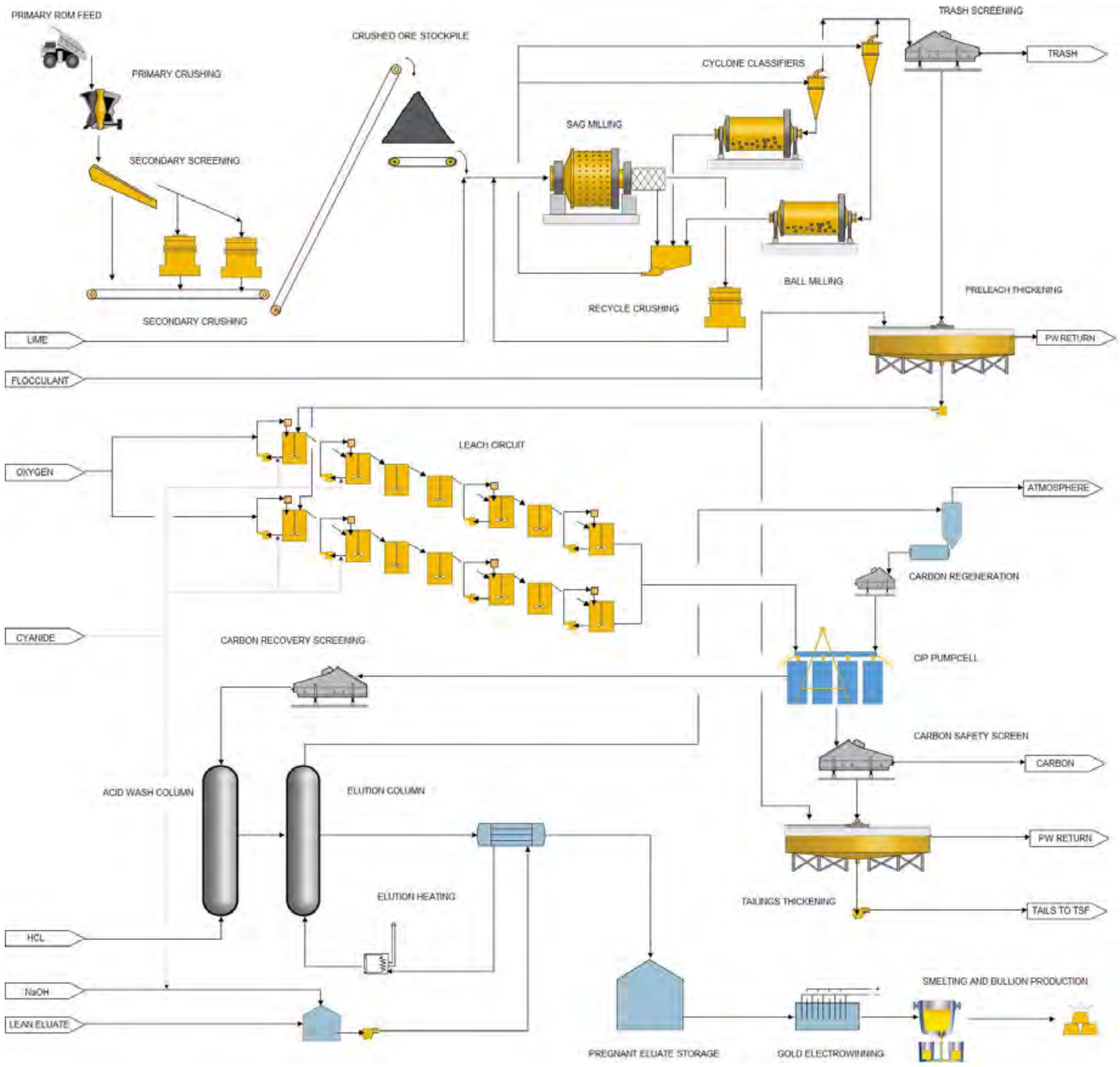
The key criteria for equipment selection are suitability for duty, reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements whilst maintaining a layout that will facilitate construction progress in multiple areas concurrently.

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, reputable western vendor supplied equipment and connection to an onsite LNG fired power station.
- Sufficient automated plant control to minimize the need for continuous operator interface but allow manual override and control if and when required.
- The treatment plant design incorporates the following unit process operations:
    - Primary and full secondary crushing using a gyratory crusher and two cone crushers to produce a crushed product size P_{80} of approximately 64mm. Due to the high competency of this ore, SAG milling is inherently energy inefficient and secondary crushing has been added to the flowsheet to improve this problem.
    - A crushed ore stockpile with a nominal live capacity of nominally 21,000 wet tonnes. Three reclaim apron feeders will deliver ore to the milling circuit via conveyor.
    - A grinding circuit configured as a two stage circuit with a SAG mill, two closed circuit ball mills and two recycle pebble crushers (SABC). The circuit will produce a P_{80} grind 75 µm. A single ball mill is not possible as this would exceed the maximum power that is currently achievable using a twin-pinion ball mill.
    - Pre-leach thickening to increase the slurry density feeding the leach and carbon in pulp (CIP) circuit to minimise tankage and reduce overall reagent consumption.
    - Leach circuit incorporating 14 leach tanks, arranged in two parallel trains of 7 each in series, to provide 36 hours leach residence time.
    - A Kemix Pumpcell CIP circuit 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 tonne split AARL elution circuit, electrowinning and gold smelting to recover gold from the loaded carbon to produce doré.
    - Tailings thickening to recover and recycle process water from the CIP tailings.
    - Tailings pumping to the tailings storage facility (TSF).

Figure 17.1 shows a simplified overall flow diagram of the circuit.

Figure 17-1 Overall Schematic Flow Diagram



The Site Plan and Plant Layout are included in Figure 17.2 and 17.3.

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 head grade of 1.26 g/t Au with overall gold recovery 92%. This reflects the highest yearly mine schedule grade, with design margin to cater for short term high grade batches.

Figure 17-2 Treatment Plant Site Facilities Plan

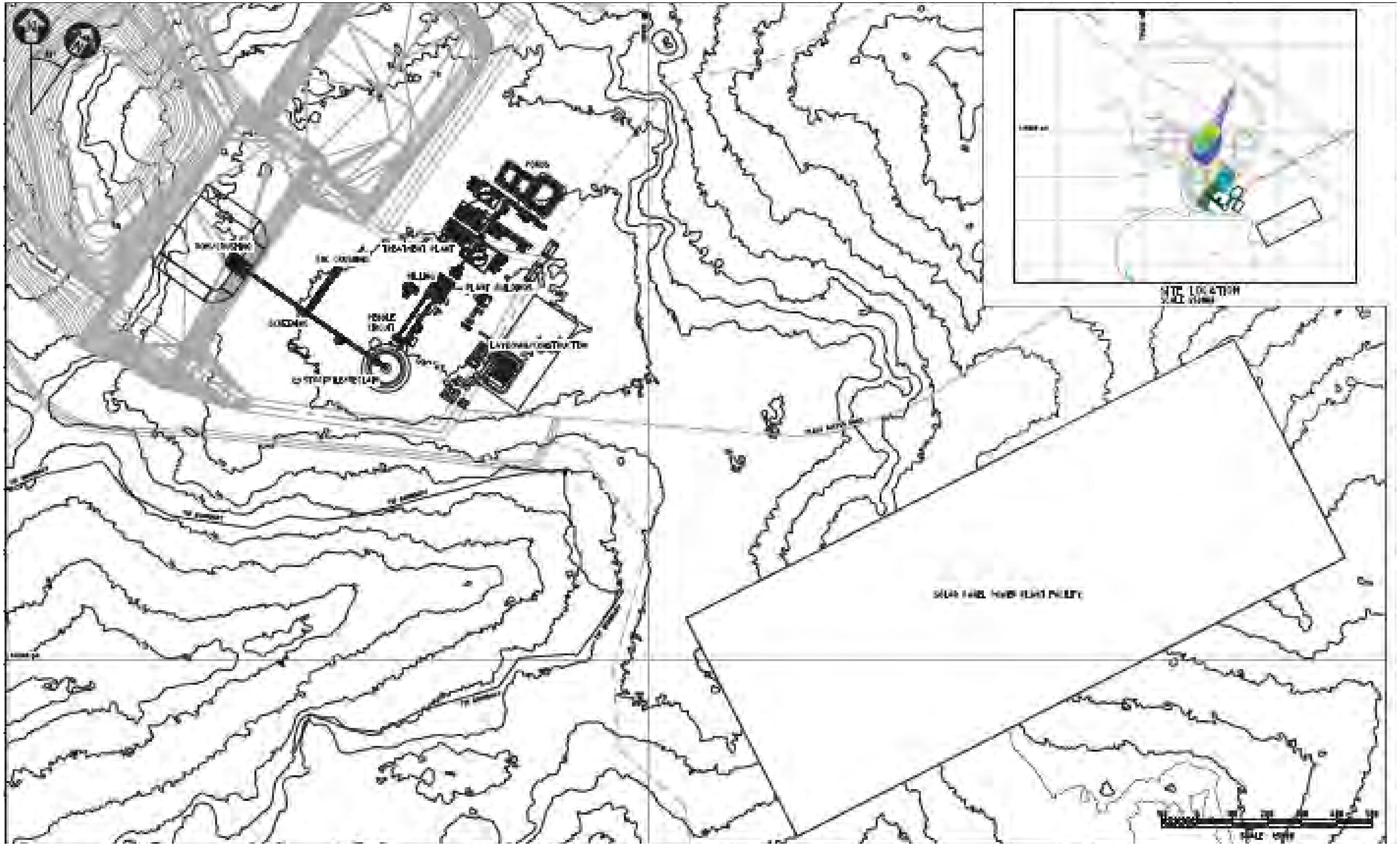
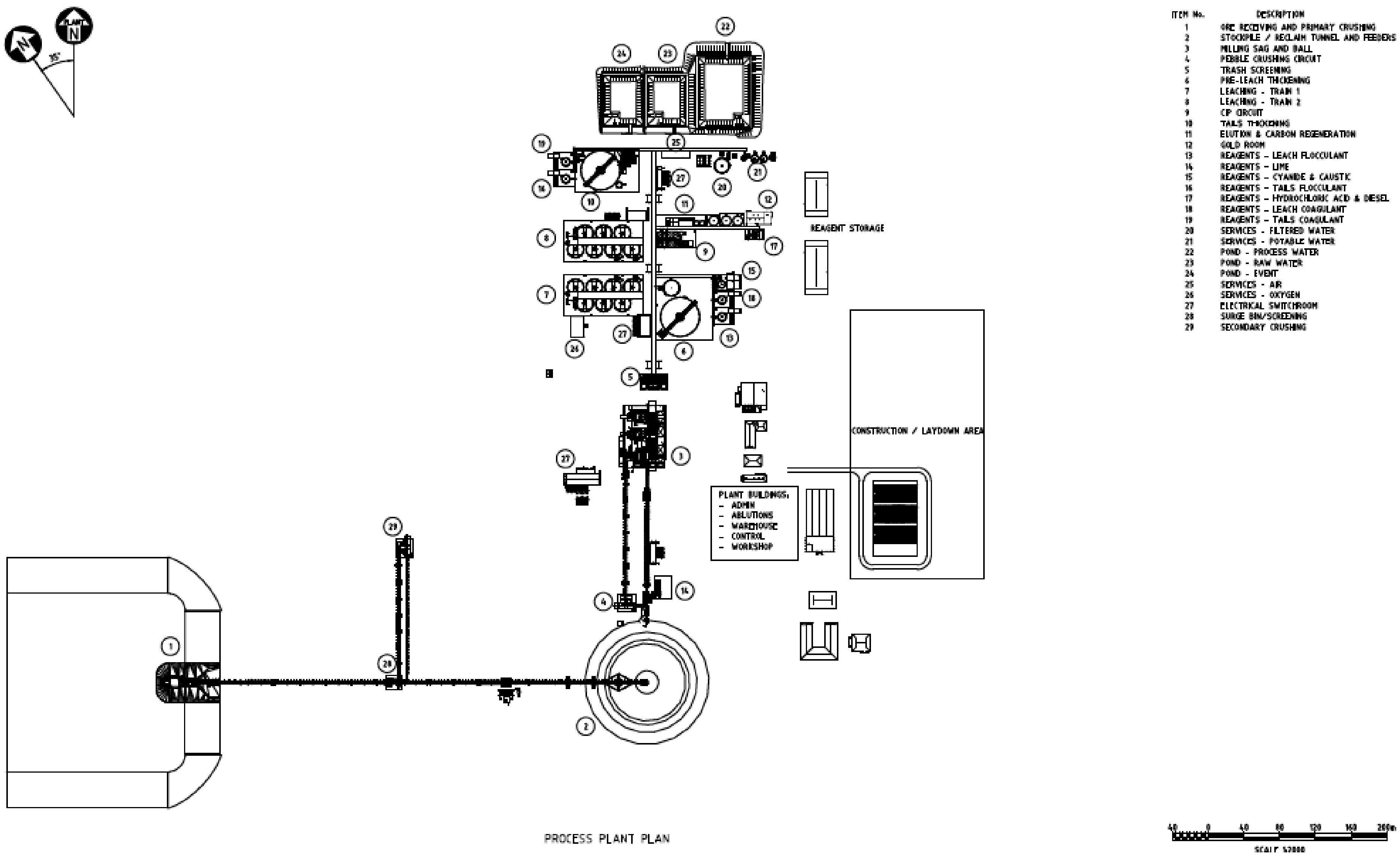


Figure 17-3 **Treatment Plant Layout**



17.2.2 ROM Pad and Crushing Circuit

The ROM pad will be used to provide a buffer between the mine and the plant. Separate stockpiles will be constructed to allow blending of different ore types to be carried out using the front end loader as required to ensure that a consistent grade and hardness will be delivered to the plant. Importantly, oxide ore will be limited to dry season only and to a maximum of 15% of the blend so that problems with crushing, screening and thickener operation do not occur.

Due to the high competency but relatively low hardness of the Koné ore, SAG milling is very inefficient. Partial secondary crushing and full secondary crushing were evaluated against a standard primary crush SABC circuit. Both the partial secondary crushing and full secondary crushing had much lower power draw and overall lower operating costs than the SABC circuit and therefore the NPV was significantly higher. Consequently, an open circuit secondary crushing circuit has been adopted ahead of the SAG mill.

17.2.3 Crushed Ore Stockpile

A 21,000 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, which will allow blending for optimal SAG mill feed size distribution.

17.2.4 Milling

Comminution circuit design has been based on achieving the required 11.0 Mtpa throughput rate and grind size for for the harder fresh ore which comprises the majority of the ore over the life of mine. A SAG and ball mill circuit, with duty / duty pebble crushers, has been selected to reduce crushed product to the nominal circuit P_{80} size of 75 µm. During periods where up to 15% oxide ore is treated, the grind size may be as fine as 56µm at the 11.0 Mtpa throughput rate. Conversely, a higher throughput rate can be achieved if the P_{80} is maintained at 75µm.

17.2.5 Classification

Cyclones of 650 mm in diameter have been selected for the classification duty to reduce the potential for spigot blockages occurring from coarse mill discharge material and miminise the overall size of the cyclone cluster. Design has been based on up to 318% 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.

17.2.6 Trash Screening

Vibrating trash screens have been selected to prevent oversize particles and rubbish from entering the downstream leach and adsorption circuit. Although minimal trash is expected, acceptable trash screen performance will be essential for good carbon management.

17.2.7 Pre-Leach Thickening

A high rate pre-leach thickener ahead of the leach circuit has been included to thicken trash screen undersize to between 52 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.

17.2.8 Leach and CIP Circuit

Head assays have indicated that the ore does not show preg-robbing characteristics and therefore a leach and CIP circuit can be considered for design.

Based on testwork, a leach time of 36 hours was selected requiring 14 off 5,000 m^3 leach tanks at 50% w/w solids density.

A Kemix Pumpcell CIP circuit consisting of six tanks was selected for recovery of gold onto carbon, to minimise carbon inventory, gold in circuit and operating costs.

17.2.9 Elution, Electrowinning and Gold Recovery

A split AARL elution circuit has been selected to remove absorbed gold and silver from carbon. The split AARL circuit can accommodate additional elution cycles if required due to the relatively short stripping time of approximately eight hours. The circuit will include an acid wash column to remove inorganic foulants from the carbon prior to elution, and a cold cyanide wash cycle has been included in the design in the event that ores with elevated copper are processed.

A 20 tonne batch size is required for the 11.0 Mtpa plant capacity at the design head grade and gold recovery.

Two parallel 33 cathode electrowinning cells are proposed for the gold room to provide a high pass efficiency and ensure a low gold tenor in the spent electrolyte returning to the strip solution tank.

A sludging cell design with in-tank washing of the cathodes has been adopted to simplify the cathode handling process. Electrowinning cell sludge will be filtered in a pressure filter prior to transfer to a drying oven before smelting to produce doré.

Barren carbon will be regenerated in a horizontal rotating kiln to remove any organic foulants.

17.2.10 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 tailings storage facility to be recovered for re-use in the leaching circuit.

The tailings thickener underflow will be pumped to the tailings storage facility.

17.3 Key Process Design Criteria

The key process design criteria listed in Table 17.1 form the basis of the process design criteria and mechanical equipment list. Inputs into the design criteria include metallurgical testwork, Montage advice, comminution modelling by Orway Mineral Consultants (OMC), and Lycopodium calculations and modelling and vendor advice.

Table 17.3-1 Key Process Design Criteria

Parameter	Units	Oxide Blend*	Fresh	Source
Plant Capacity	tpa	11,000,000	11,000,000	Montage
Gold Head Grade	g Au/t	1.09	1.26	Montage
Design Gold Recovery (Leach)	%	96.9	92.1	Testwork
Crushing Plant Utilization	%	80.0	80.0	OMC
Plant Availability	%	91.3	91.3	Lyco
Crushing Work Index (CWi)	kWh/t	N.A.	16.5	Testwork
A*b (SMC Test)		125	27.2	OMC / Testwork
Bond Ball Mill Work Index (BWi)	kWh/t	6.9	13.3	OMC / Testwork
Abrasion Index (Ai)		0.050	0.410	OMC / Testwork
Grind Size (P_{80})	µm	56	75	OMC / Montage
Leach Circuit Residence Time	hrs	36	36	Montage
Leach Slurry Density	% w/w	50	50	Montage
Number of Leach Tanks		14	14	Lyco
Number of Adsorption Tanks		6	6	Vendor

Parameter	Units	Oxide Blend*	Fresh	Source
Cyanide Addition – Plant	kg/t	0.35	0.38	Testwork / Lyco
Quicklime Addition – Plant	kg/t	1.04	0.24	Testwork / Lyco
Elution Circuit Type		Split AARL	Split AARL	Lyco
Elution Circuit Size	t	20	20	Lyco
Frequency of Elution	strips / wk	7	7	Lyco

* Up to 15% oxide treated in plant feed; Comminution parameters quoted reflect 100% oxide ore, benchmarked by OMC.

OMC Report No. 8197 Rev 0, Jan 2021, should be referenced for detail of the comminution modelling.

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

Haul trucks will deliver run-of-mine (ROM) ore from the pits to the ROM pad where it will be dumped in stockpiles arranged by ore head grade and ore type. A front end loader (FEL) will be used to reclaim and tram ore from the various stockpiles to the ROM bin.

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 cognisance of the potential clayey nature of the oxide ore, with total oxide ore content limited to 15% to minimise chute blockages and rheology issues in the crushing, milling, leach and CIP circuits.

17.4.2 Crushing and Ore Stockpile

ROM ore will be transferred from the stockpiles into the crusher ROM pocket by FEL. The facility to truck dump into both sides of the ROM pocket will also be provided. 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 static magnet on this belt will be provided to remove any tramp steel from the conveyor belt. A metal detector will also be provided to allow material to be diverted to the metal reject bunker ahead of the secondary screen feed bin to protect the secondary screen and crushers.

Material will be extracted from the secondary screen feed bin by the secondary screen apron feeder, which will feed the secondary screen. The secondary screen will be fitted with 100mm and 30mm 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.

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 30mm. Product from the secondary crushers will report to the secondary crusher discharge conveyor, which will be equipped with a weightometer.

Secondary screen undersize and crusher product will both report to the stockpile feed conveyor, which will be fitted with a weightometer. This conveyor will deliver ore, with a P_{80} of 36mm, 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. Three dust collectors will also be installed; at the primary crushing area, secondary screen area and the secondary crusher area, to ensure that dust generated at transfer points will be adequately captured. Dust collected from 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. The FEL driver will ensure feed is maintained to the crushing circuit and will communicate with the crusher control room using a two-way radio to supply information on crusher feed operation.

The stockpile feed conveyor will discharge ore onto the crushed ore stockpile. The stockpile will have a live capacity of approximately 13,000 t, which is 15 hr of mill feed at 11 Mtpa. Ore will be withdrawn from the stockpile by two of three variable speed apron feeders. These feeders will discharge ore onto the SAG mill feed conveyor.

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.

Quicklime, used for pH control in the leach circuit, will be added directly onto the SAG mill feed conveyor. Quicklime will be stored in a lime silo and will be metered onto the belt using a rotary valve.

Grinding media will be added directly onto the SAG mill feed conveyor via the SAG mill ball hopper using a FEL. This hopper will also be used to return any coarse spillage back into the circuit. Clean up around the circuit will be carried out using mobile equipment such as a FEL or skid-steer loader.

17.4.3 Grinding and Classification Circuit

Crushed ore will be fed directly into the SAG mill feed chute. Process water addition to the SAG mill feed will be in ratio to the ore feed rate in order to maintain a relatively constant mill feed slurry density to optimise grinding efficiency. Process water addition to the cyclone feed hopper will be automatically controlled to maintain a constant cyclone feed density.

The SAG mill will be a 9.75 m x 6.86 m EGL variable speed mill fitted with two 6.75 MW twin pinion drives. The SAG mill will be installed with discharge grates which will allow slurry and small pebbles to pass out of the mill. The SAG mill product will flow to a double deck vibrating screen for pebble separation.

Two ball mills will be installed, each with its own dedicated cyclone pack. The ball mills will be 6.71m x 11.58m EGL fixed speed mills, each fitted with two 4.85 MW twin pinion drives. Slurry from the cyclone underflow launder will be returned to the relevant 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 a dedicated ball mill ball charging kibbles and hoists.

Combined undersize product from the SAG mill discharge screen and ball mill trommels will gravitate to the cyclone feed hopper where it will be diluted with process water. Slurry will be pumped to the two hydrocyclone clusters for classification, by the duty / duty cyclone feed pumps. A bareshaft rotable spare cyclone feed pump will also be provided.

The combined cyclone overflow stream will gravitate through a metallurgical sampler which will be used as the plant feed sampler. Both solids and solution assays will be undertaken on this sample. An in-line particle size analyser (PSA) sampler will follow the cyclone overflow metallurgical sampler to provide feed for the PSA to provide real time sizing information for control of the milling circuit.

Four vibrating trash screens will be installed in a parallel configuration prior to the pre-leach thickener. Four 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 collected in trash bins.

The grinding area will be serviced by three 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.

17.4.4 Pebble Crushing

Oversize from the SAG mill discharge screen will be conveyed to the pebble crusher feed bin via a series of belt conveyors. A self-cleaning belt magnet will be positioned at the head chute of the first conveyor to remove any scrap metal and steel media which could potentially damage the pebble crusher. A pebble crusher feed weightometer will be installed prior to the pebble crusher feed bin.

Downstream of the cross-belt magnet, the pebbles will pass under a metal detector prior to discharging into the pebble crusher feed bin. The feed bin will provide surge capacity ahead of the pebble crushers and allow a controlled feed to be presented to the crusher. Should the pebble crusher not be operational, a diverter gate ahead of the pebble crusher feed bin will allow pebbles to bypass the pebble bin and crusher and feed directly onto the pebble crusher discharge conveyor. Similarly, should the metal detector detect tramp metal (not removed by the cross-belt magnet), the diverter gate ahead of the pebble crusher feed bin will automatically allow pebbles to bypass the pebble bin and crushers and feed directly to the pebble crusher discharge conveyor.

Two pebble crushers will be installed in a duty / duty configuration. The crushers will be HP4 or equivalent units with a closed side setting of 12mm. Pebbles will be withdrawn from the pebble crusher feed bin by two variable speed belt feeders operating in parallel. The pebble crushers will discharge crushed pebbles directly onto the pebble crusher discharge conveyor which will return the crushed pebbles to the SAG mill feed conveyor.

The pebble crusher discharge conveyor will be fitted with a weightometer for process control purposes.

17.4.5 Pre-Leach Thickening

Trash screens undersize will be thickened in a 44 m 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. The facility to add coagulant to the thickener feed for oxide blends will also be provided.

Thickener underflow at 52% solids w/w or higher will be pumped to leach circuit. Three thickener underflow pumps will be installed in a duty / duty / standby configuration. Thickener overflow will gravitate 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.

17.4.6 Leach Circuit

Due to the number of leach tanks required to achieve a 36 hour residence time, two trains of seven mechanically agitated tanks is 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 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 bypass facilities to allow any tank to be removed from service for agitator maintenance.

On-site generated oxygen will be added via oxygen addition devices and dedicated oxygen contactor slurry pumps, to four leach tanks in each train. In the first tank of each train, all new leach feed slurry along with recycled slurry from the tank, will be introduced to oxygen gas in the contactor, and sodium cyanide solution added to the suction side of the oxygen contactor slurry pump. In the remaining tanks, recycled slurry will be introduced to oxygen gas in the contactor and sodium cyanide solution added to the suction side of the oxygen contactor slurry pump. In tanks 1 and 2, three oxygen contactors will be used for each tank and in tanks 5 and 7, two oxygen contactors will be used for each tank. It is anticipated that 16 tonnes per day of oxygen will be required, most of which will be added at the head of the trains.

Slurry will flow by gravity through the tanks and report to the CIP circuit feed launder.

The leach circuit will be serviced by eight floor sump pumps. Sump pumps will return spillage to a nearby tank.

A cyanide analyser for on-line monitoring of the free cyanide concentration will allow the sodium cyanide dose rate to be optimised. A hydrogen cyanide (HCN) gas monitor will also be installed in the leach area.

17.4.7 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 six 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.

17.4.8 Tailings Thickening and Disposal

Carbon safety screens undersize will gravitate 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. The facility to add coagulant to the thickener for oxide blends will also be provided. 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 two 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 gravitate 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.

17.4.9 Elution, Carbon Regneration and Gold Room Operations

The following operations will be carried out in the elution and gold room areas:

- Acid washing of carbon.
- Optional cold cyanide wash to remove copper zinc from loaded carbon.
- Stripping of gold from loaded carbon using the split AARL method.
- Electrowinning of gold from pregnant solution.
- Filtration of electrowinning sludge.
- Drying of the filter cake.
- Smelting of filter cake to produce a gold doré.

The elution and gold room areas will operate seven days per week, with the majority of loaded carbon preparation and stripping occurring during day shift. The AARL stripping circuit will be automated and will contain separate acid wash and elution wash columns. The stripping circuit will be sized for a 20 tonne batch of carbon.

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 five bed volumes (BV) of solution pumped from the lean eluate tank, through the heat exchangers and elution column to the pregnant solution tank. Caustic will be added to the pregnant solution tank during this step to ensure that a high enough solution pH is attained for electrowinning.

Following this step, five BV of treated water or barren electrowinning solution will be pumped from the stripping water tank, through the heat exchangers and elution column and into the lean eluate tank to provide lean solution for the next stripping cycle. The temperature set point will be maintained at 130°C for this step.

The final step of the sequence will be a cool down of carbon where treated water will be used to cool the carbon down to approximately 80°C. Treated water exiting the column will be directed to the leach feed distribution box.

A vertical spindle sump pump will be provided in the elution column are 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 and barren solution will discharge to the duty pregnant solution tank. Barren solution from this pregnant solution tank will be returned to the leach circuit via the barren solution pump. To conserve water, the barren solution will be re-used as strip solution, with the option to direct this solution to leach feed when required.

Fume extraction will be provided to remove noxious gases from the cells. In addition to this, a number of gold room vent fans will be provided to ensure there is adequate ventilation inside the gold room.

Gold Room

Upon completion of electrowinning, precious metal sludge will be washed off the cathodes with a high pressure cathode washer. The gold bearing sludge will gravitate 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 SAG mill.

The gold room and electrowinning area will be serviced by a gold trap and dedicated gold room area sump pump. Any spillage within this area will be pumped back to the leach circuit.

Carbon Regeneration

After completion of the elution process, the barren carbon will be transferred from the elution column to the carbon dewatering screen to dewater the carbon prior to entering the feed hopper of the horizontal carbon regeneration kiln. In the kiln feed hopper any residual and interstitial water will be drained from the carbon before it enters the kiln. Kiln off-gases will also be used to dry the carbon prior to entering the kiln.

The carbon will be heated to 650 - 750°C and held at this temperature for 15 minutes to allow regeneration to occur. Regenerated carbon from the kiln will be quenched in the carbon quench vessel and pumped to the carbon sizing screen using the regen carbon transfer pump. New carbon will be added to the carbon quench vessel to ensure that the carbon is sized over the carbon sizing screen prior to entering the CIP circuit.

The screen oversize (regenerated, sized carbon) will report to the carbon transfer hopper and be returned to the CIP circuit using the barren carbon transfer pump. The quench water and fine carbon (carbon sizing screen undersize) will report to the carbon safety screen via the fine carbon hopper and pump.

The carbon regeneration kiln off-gases will be treated for mercury removal by venturi scrubbing and filtering via sulphur impregnated carbon columns.

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.

17.4.10 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 transfer hopper via the bag breaker. The lime 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 metered via a rotary valve directly onto the mill feed conveyor for leach circuit pH control.

Sodium Cyanide

Cyanide will be delivered as dry briquettes in one tonne bulk bags in boxes. Cyanide will be added to the mixing tank via a bag breaker and be dissolved in raw 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. The cyanide solution will be transferred to the cyanide storage tank on completion of a mix. Cyanide will be dosed to the leach circuit via a individual dosing pumps to the oxygen contactor pump suction.

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 tonne bulk bags of 'pearl' pellets. Caustic will be added to the mixing tank via a bag breaker and be dissolved in raw 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. Raw 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 pump 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 SAG mill via the SAG mill ball hopper and to the ball mill using a kibble and a fork lift with a hydraulic drum tipper attachment. Media will be added as required to achieve the target SAG and 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 raw 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 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 transfer any spillage to the tailings thickener.

Coagulant

Coagulant will be used when treating blends containing oxide ore, in order to 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 raw 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. A vertical spindle sump pump will be provided to transfer any spillage to the pre-leach thickener.

Coagulant will be distributed to the tailings thickener using the variable speed pre-leach thickener coagulant dosing pumps which will be installed in a duty / standby configuration. A vertical spindle sump pump will be provided to transfer any spillage to the tailings thickener.

Plant Diesel

Diesel will be delivered to the 15 m^3 plant diesel day tank by the mine diesel tanker. Diesel in the header 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 90 days stock of reagents will be stored on site to ensure that supply interruptions due to port, transport or weather delays do not restrict production.

17.4.11 Services

Raw Water

Raw water for the plant will be supplied from both the Marahoué River and mine dewatering pumps. The mine dewatering pumps will deliver water to the centralised mine dewatering transfer tank, from where it will be transferred to the raw water pond by the duty / standby mine dewatering transfer pumps.

At the Marahoué River, three pontoon mounted pumps will supply water to the river extraction water tank and be transferred to the site water storage facility by a series of river water extraction transfer and booster pumps, with intermediate break tanks as required. At the water storage facility, the water storage pumps and the water storage booster pumps will transfer water to the plant raw water pond.

The raw water pond will be a 7,500 m^3 lined pond with approximately nine hours of storage capacity. Water will overflow from the raw water pond into the process water pond for make-up. 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. Filtered and potable water treatment plant waste streams will also report to the process water pond. The process water pond will be a 30,000 m^3 lined pond which will have a nominal capacity of six hours.

Duty / standby process water pumps will be provided for the plant water supply. Anti-scalant will be added to the process water to reduce scaling of pipelines, spray nozzles and screen decks.

The process water system will be configured such that bulk water for the milling circuit such as SAG and 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 SAG mill discharge screen sprays and 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, gold room, 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

A separate borefield will be established for supply of potable water. The borefield pumps will supply water to a centralised potable water feed transfer tank. Water will be transferred to the plant potable water treatment plant by the potable water feed transfer pumps.

The water treatment facility will include ultra-violet sterilisation and chlorination. 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 potable water treatment plant will report to the process water pond.

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 receiver and the mill area instrument receiver.

Oxygen

Oxygen, for use within the leach circuit, will be supplied by two duty oxygen pressure swing adsorption (PSA) plants. A standby plant will also be provided. Oxygen will be generated to a supply pressure of 500 kPa(g). Oxygen will be distributed to the required leach tanks.

Diesel

Provision for site diesel storage and distribution has been made in the plant design. Site diesel tanks will be installed to provide bulk storage for the site. The plant diesel tank and ring main pumps will be located within the process plant to reticulate diesel to the elution heater, carbon regeneration kiln and smelting furnace.

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. PROJECT INFRASTRUCTURE

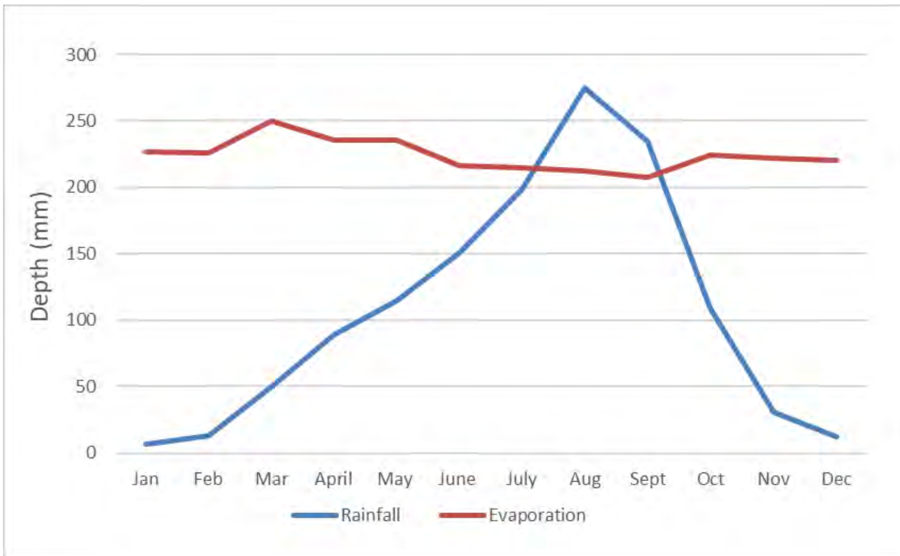
18.1 Water Supply

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. The majority of rainfall occurs within the months of June to October. This is locally referred to as two rainy seasons; the main period between June and August and a shorter rainy season from September to October. The runoff mainly occurs during this period, with peak flows occurring in August. See Figure 18-1 for climatic averages. Climatic data was used from Korhogo, approximately 140km to the North East of the Koné license. Evaporation is consistently high throughput the year.

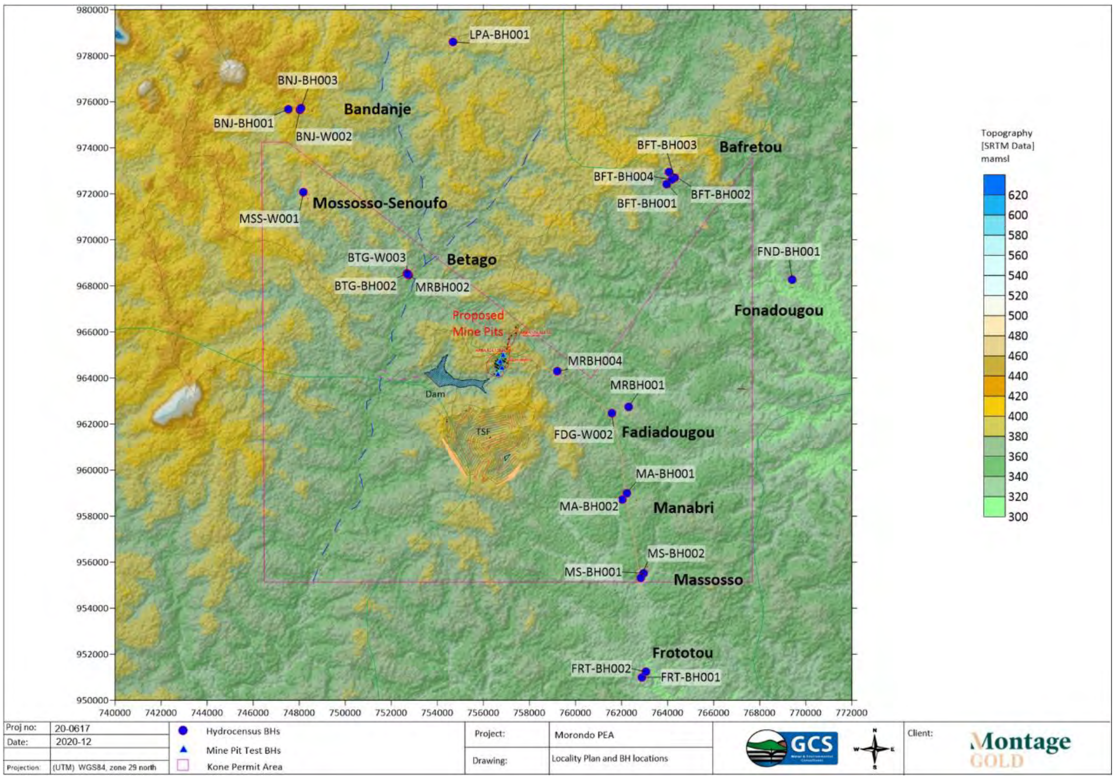
Figure 18-1: Monthly rainfall and evaporation for Koné



Hydro-Census

A hydrocensus was completed within a 15km radius of the Koné Exploration Site to determine the potential of the regional groundwater resource and existing community groundwater users Figure 18-2.

Figure 18-2: Koné area topographical setting and hydrocensus borehole locality map

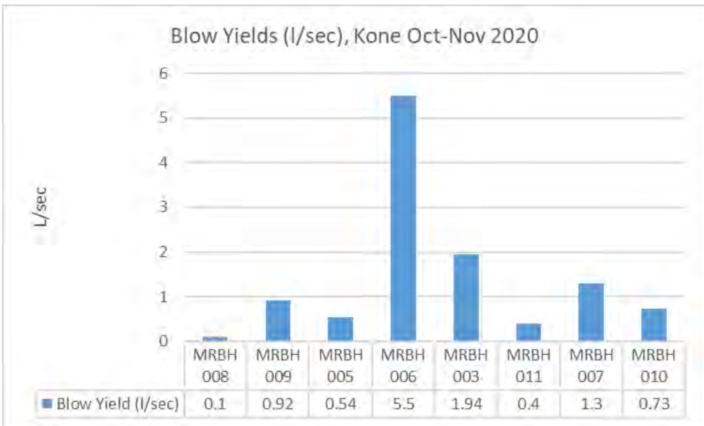


Hydrology

Three borehole pairs were drilled with 2 additional exploratory test/observation boreholes within the main proposed pit area. The following observations were made during the drilling of the boreholes:

- The first groundwater was intersected at the saprock transition zone at all the boreholes exceptt for MRBH008 and MRBH009 located in the northern part of the main pit area.
- The upper saprolite / laterite contain significant clay content and generally low to insignificant groundwater flow with low to very low permeability.
- Fractures were intersected along the dyke contact zone and deeper zones where groundwater flow observed. Generally, the airlift yields showed low groundwater flow with only borehole MRBH006 above 5l/sec and boreholes MRBH003 and MRBH007above 1 l/sec. The rest were below 1l/sec (refer to Figure 18-3).

Figure 18-3: Blow Yields Observed During Drilling

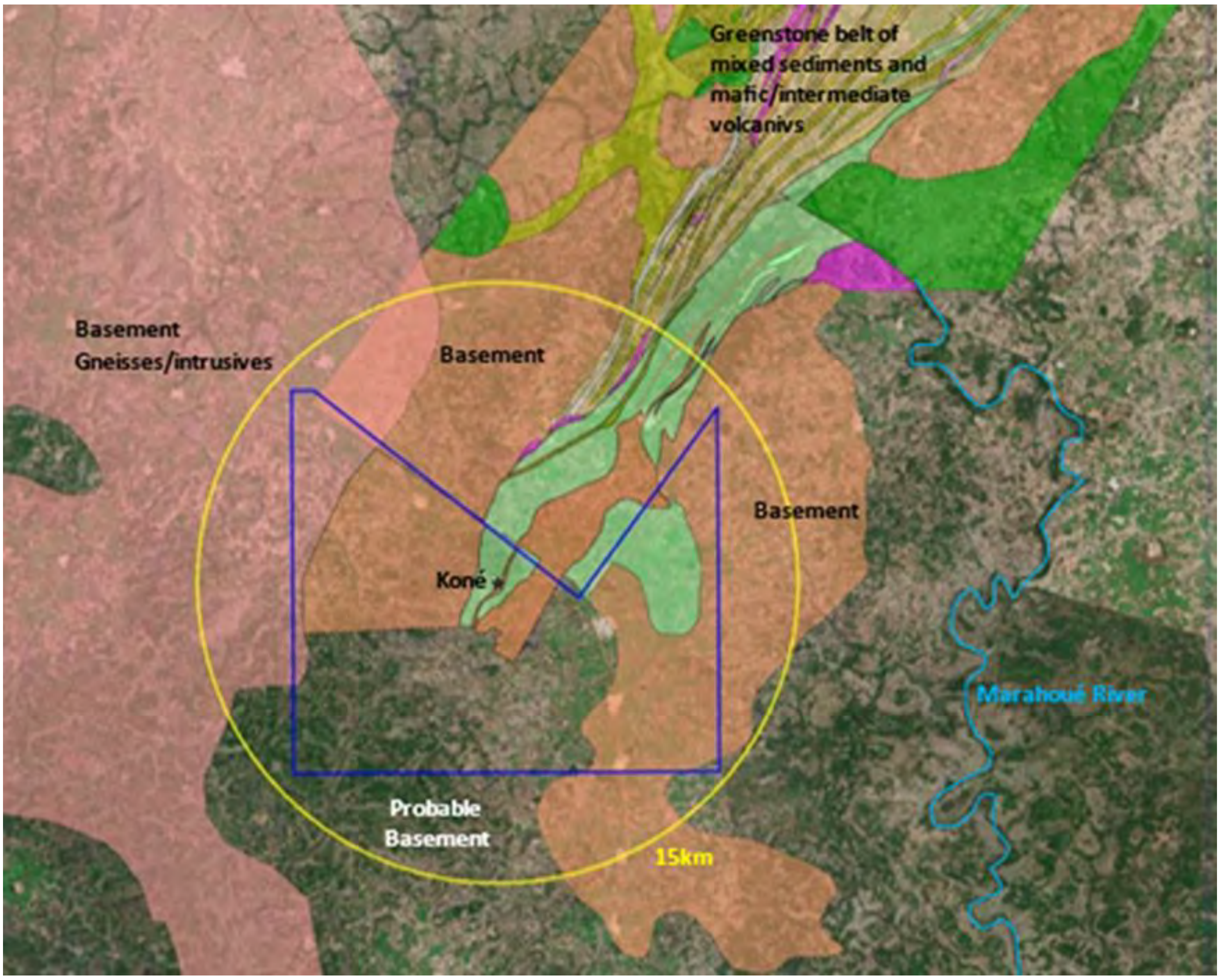


18.1.2 Preliminary Groundwater Assessment

Geology and proposed mining

The prospect is located 5-10 km south of the southern end of the Boundiali Basin, probably on the southern continuation of a merged Fonondara/Gbéou Shear Zone (Montage, 2018) Figure 18-4.

Figure 18-4: Simplified Geological Setting (Montage, 2020)

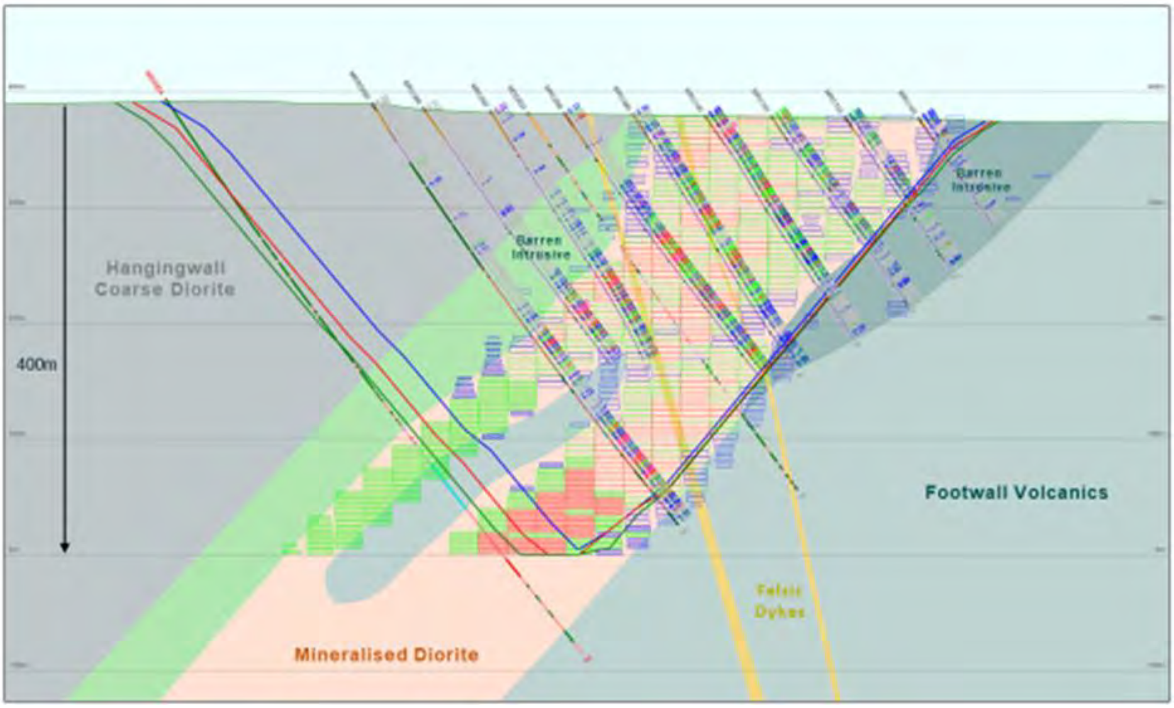


Drilling indicates that the mineralised zones strike NNE and dip approximately 40° towards the west. Drill holes plunge 50° towards an azimuth of 120°, perpendicular to the orientation of mineralised zones in the core of the prospect.

Gold is hosted within a sheared, foliated, lineated, and locally folded and heterogeneous crystalline unit whose composition varied between tonalite and quartz diorite. Anomalous gold grades are typically associated with stringer veinlets of pyrite, quartz, carbonate, chlorite, and chalcopyrite.

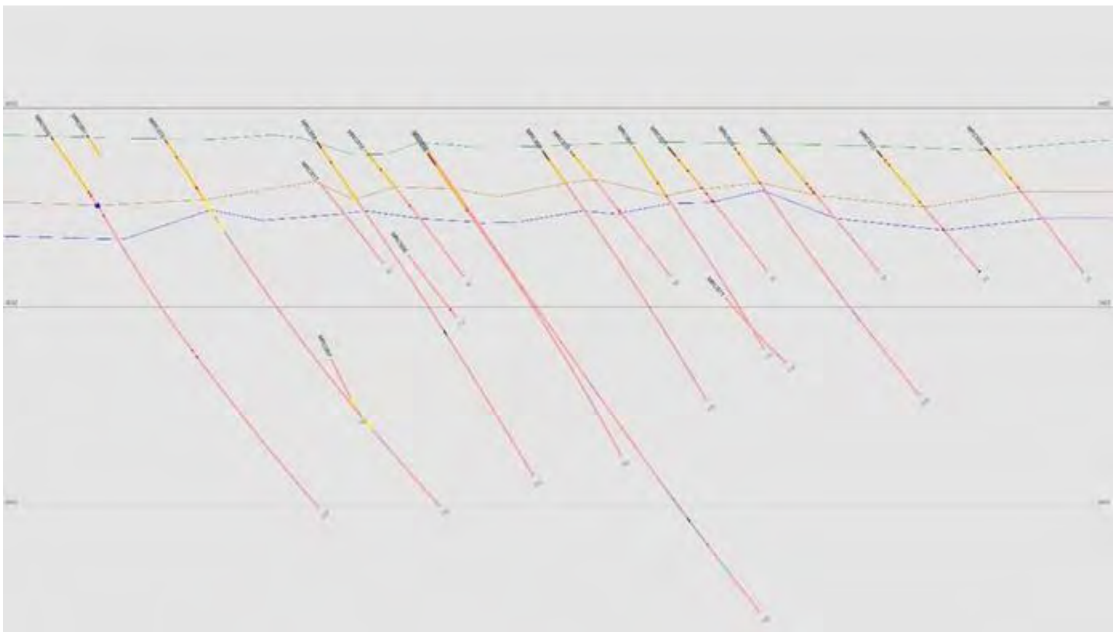
Pit depths will be in the order of 400m (Figure 18-5).

Figure 18-5: Preliminary example of pit dimensions at cross section (Montage, 2020)



The surface area is covered in a saprolitic regolith overburden (soil and clays) layer above the fresh rock which varies in depth. The saprolitic material is presented in Figure 18-6 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 and 25m below surface.

Figure 18-6: Example of saprolite thickness (Montage, 2018)



Aquifer Characteristics
Groundwater occurrence is related to both alluvial aquifers associated with drainage channels and upper weathered saprolite and deeper fractured rock aquifers associated with geological structures. These are referred to as the upper and lower aquifer systems extending across the proposed pit area and surroundings. The lower aquifer has generally higher exploitable potential when comparing to the shallow aquifer system.

Upper Aquifers
Alluvial aquifers are used as a water supply for the communities who reside along the river channels. Hand dug wells have been constructed in the sediments and are progressively deepened after the rainy seasons. The sediments contain sufficient clay to maintain the sidewalls of the well. Extraction is by means of bucket and rope.

Saprolitic weathering is present across most of the site and overlies the deeper fractured rock. This highly weathered zone is defined locally between approximately 0 and 25m and has limited aquifer properties but critical to understand the characteristics for pit wall stability and pore pressure assessments. It has, however, a significant impact on interflow and groundwater recharge mechanisms.

Groundwater flow is mainly associated with the medium weathered and fractured rock to fractured fresh rock at depths normally >40m, however significant flows may occur along the contact zones with fresh bedrock.

Deeper Aquifers
The deeper system is associated with fractured rock aquifers occurring within discrete shear zones created by faulting, fracturing, and contact metamorphism along intrusive dyke margins. These aquifers range from limited extent and interconnectivity to higher yielding more competent zones associated with the regional geological systems.

Intrusions include older sill type intrusive gabbro and pyroxenites to younger dolerite dykes. A north-south mafic dyke system occurs across the main pit and dips slightly to the east (Figure 18-12: Geotech Borehole Core with Dyke Contact). The dyke was intersected during the drilling of GT005 and MRBH006 and indicates higher groundwater flow when comparing to the other boreholes.

In more recent times, extensive weathering of rocks has occurred forming saprolite, accompanied by periods of laterite development. The secondary aquifers can be subdivided into two hydrogeological units (sub-aquifer types):

- Variably weathered intergranular and fractured aquifers, and
- Underlying fractured fresh rock aquifers.

Groundwater Levels
Groundwater levels were measured at 70 open exploration boreholes within the proposed pit area. The range of groundwater levels measured across the proposed pit can be viewed in

Figure 18-7, range between 4 and 42 mbgl and average approximately 20mbgl. The graph also provides an overview of weathering depths which vary between 1 and 33 mbgl and the average is around 11m.

The groundwater gradients have been investigated by plotting the topography against groundwater levels to determine of the groundwater gradients tend to mimic the topography under ambient conditions. Figure 18-8 and Figure 18-9 represents the correlation graphs and the following can be derived from these two graphs:

- Initial groundwater level data obtained from the exploration boreholes indicated a poor correlation between groundwater elevation and topography. This might be due to various affecting factors such as the construction, depth, and incline of the boreholes. A rather flat groundwater table is indicated by the data.
- A second graph where only the newly drilled boreholes and some hydro-census boreholes were applied indicated a better correlation of 90% (Figure 18-9).

Figure 18-10 and Figure 18-12: Geotech Borehole Core with Dyke Contact provide an overview of the preliminary Kriging interpolated groundwater piezometric head elevations and depth contours. Available data indicates that the north western and south eastern portion of the main pit area is characterised by deep weathering. In turn the norther portion of the pit shows deeper groundwater levels and the southern portion shallow groundwater levels. It is fair to assume that shallow groundwater levels with deeper weathering (southern end of the pit) may experience more issues with pit wall depressurisation implying that a portion of the upper soft weathered material could be saturated.

The rest and majority of the groundwater levels are mainly below (deeper) the weathering profile and within the fresh bedrock which implies that minimal saturated weathered material exist along the proposed pit shell.

Figure 18-7: Groundwater level data from exploration boreholes

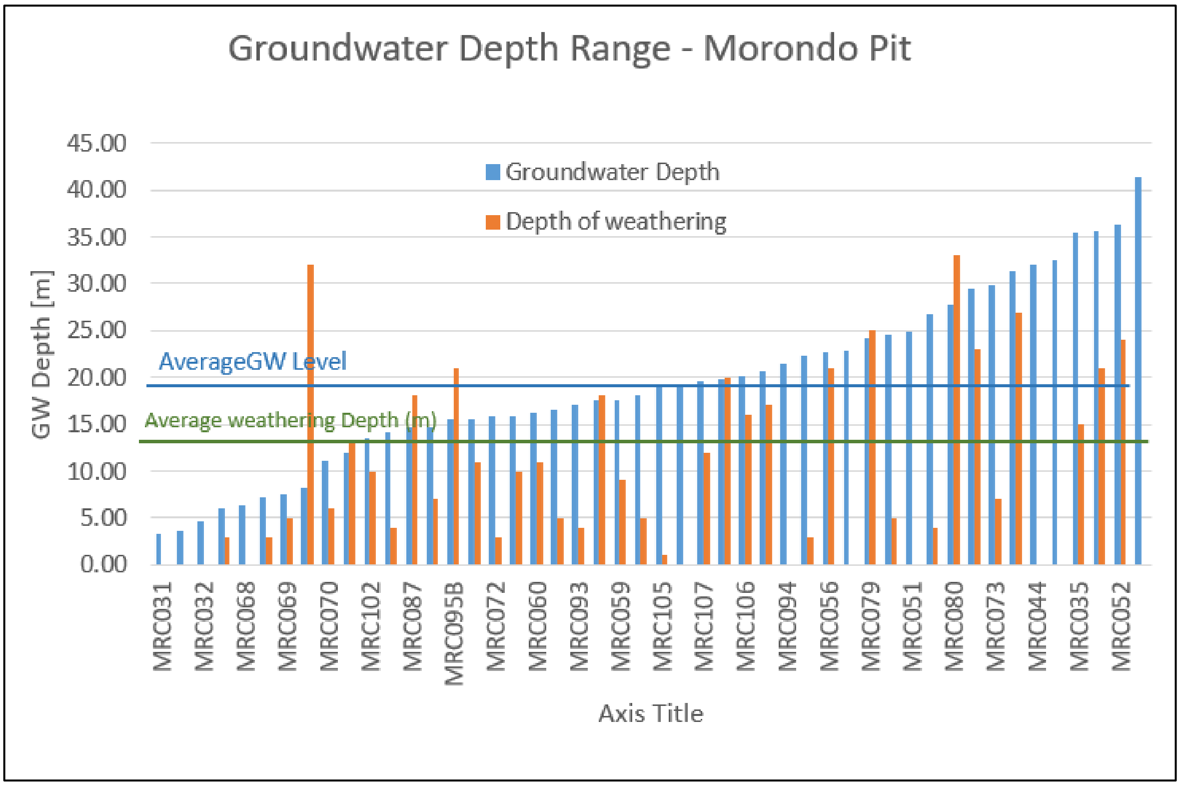


Figure 18-8: Correlation Plot: Groundater Elevation and Topography for Exploration Boreholes

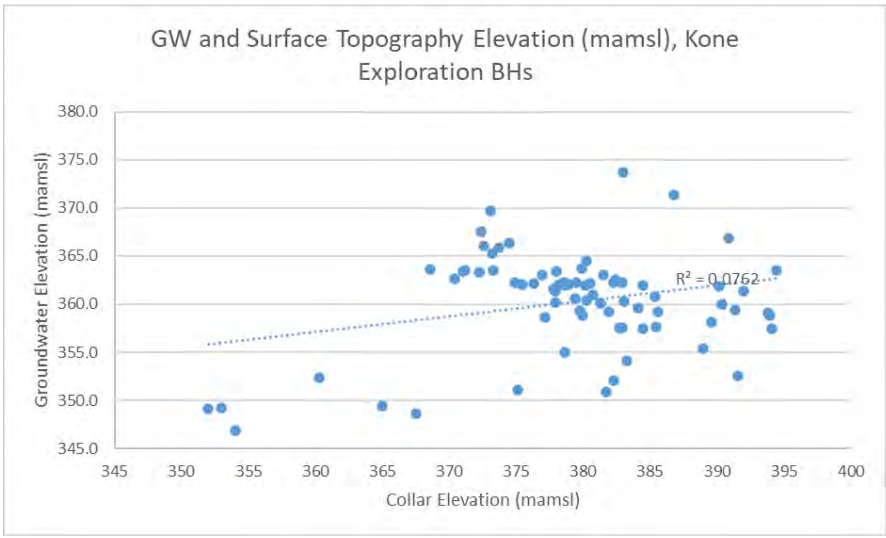
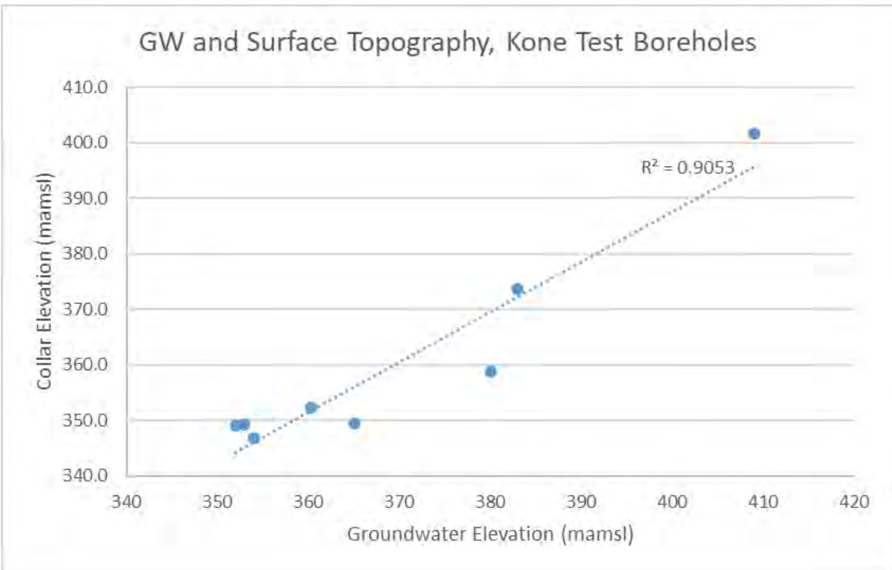


Figure 18-9: Correlation Plot: Groundwater Elevation and topography for Hydrocensus and Test Boreholes



Groundwater Occurrence

Generally, groundwater occurrence can be seen as low to medium within the Koné area and the likely yield of a production borehole drilled to a depth of 100 to 180m is between 0.2 and 2.5 l/sec.

There are five areas where higher groundwater flow was observed during drilling, these can be seen from Figure 18-10 and in Table 18-1 Hydrology Test Boreholes. These observations are associated with the interception of structural geology and more site-specific the felsic dyke intersections. The geotechnical borehole GT005 intersected the dyke at 48m where groundwater flow was observed, a photo of the core can be seen from Figure 18-12. It is fair to assume that if more detailed groundwater exploration methods are applied (i.e. precision geophysics and exploratory drilling) that groundwater yield in production boreholes can be >2.5 l/sec.

Table 18-1 supplies an overview of the water strike depths and the blow yields obtained during drilling.

Figure 18-10: Test and Observation Borehole Locations and Groundwater Piezometric Contours

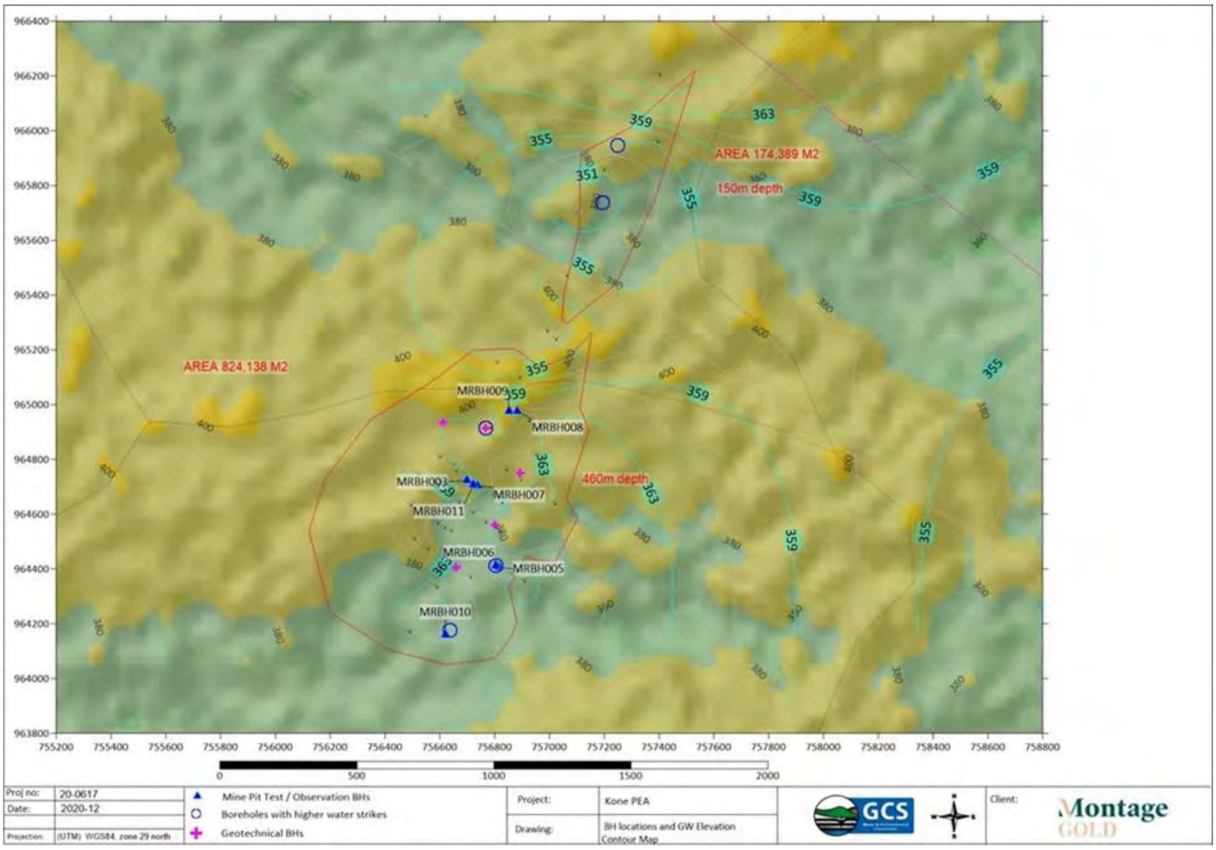


Figure 18-11: Test and Observation Borehole Locations and Groundwater Depth

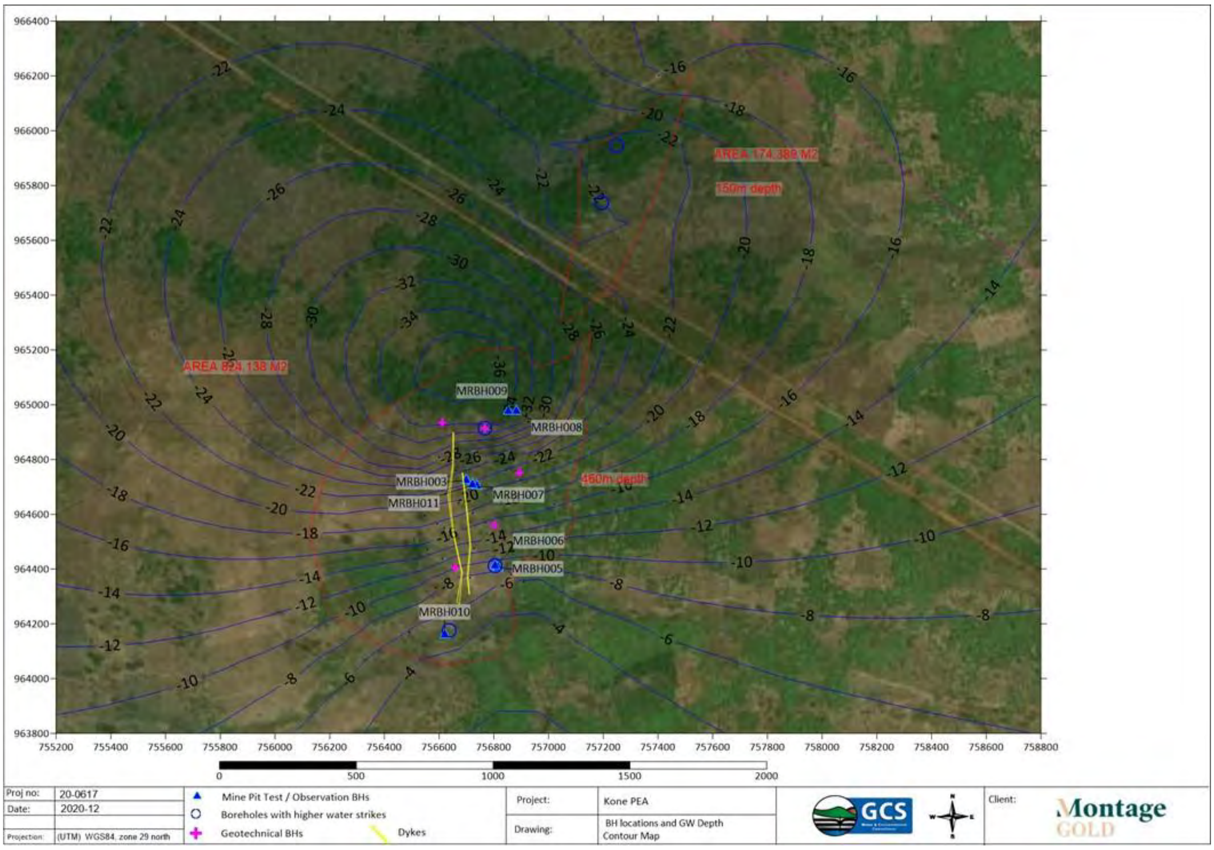
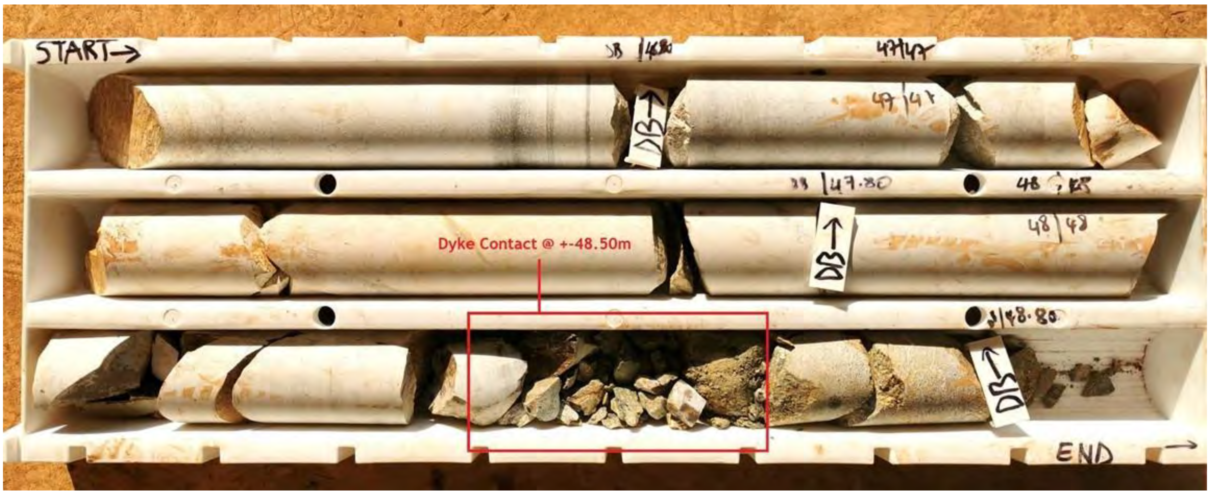


Table 18-1 Hydrology Test Boreholes

BHID	Borehole ID	X UTM - 29N	Y UTM - 29N	Z	Date drilled	Overburden depth (m)	Depth of BH (m)	Water Strike Depth (mbgl)	Blow Yield (l/sec)	Blow Yield (m3/hr)	Solid PVC (126/140) (m)	Screened uPVC (126/140) (m)	Gravel Pack (m)	Pump Tested
BH1_S	MRBH008	756878.50	964978.56	395.00	16/11/2020	32	42	seepage only	0.1	0.4	0.2	0.2-32		observation
BH1_D	MRBH009	756846.77	964974.97	394.06	17/11/2020	32	179.7	66, 99, 149, 180	0.92	3.3	0 - 99.7	99.7 - 179.7	179.7	Yes
BH2_S	MRBH005	756815.63	964405.57	373.55	05/10/2020	28	33	14, 28, 33	0.54	1.9	0 to 11	11 to 32		observation
BH2_D	MRBH006	756804.69	964411.93	373.92	15/10/2020		163.8	20, 86, 95, 120, 127, 156, 163.8	5.5	19.8	0 - 35, 152 - 163	35 - 152	163.8	Yes
	MRBH003	756699.00	964723.00	386.00	23/06/2020	25	80	33, 66, 69	1.94	7.0	0-32.6 35.5 - 65, 73.8 - 80	32.6 - 35.5 65- 73.8	11 - 80	No
BH3_S	MRBH011	756723.00	964707.00	389.00	25		77							observation
BH3_D	MRBH007	756736.89	964702.42	381.27	21/10/2020	27	180	40, 65.75, 86, 148.5, 180	1.3	4.7	0 - 50, 178 - 180	50 - 178	180	Yes
BH4	MRBH010	756628.52	964155.37	369.00	23/11/2020	26	156.22	45, 140, 160	0.73	2.6	0 - 6.5, 56.1 - 115.1	6.5 - 56.10, 115.1 - 156.2	160	Yes

Figure 18-12: Geotech Borehole Core with Dyke Contact



Source: Montage

18.1.3 Aquifer Test

Four aquifer hydraulic tests (conventional pump test) were conducted and the data collected used to calculate the hydraulic conductivity and interim mine pit inflow volumes Table 18-2.

Transmissivity (T) range between 7.4 and 1.5 m^2/day and hydraulic conductivity between 0.2 and 0.02 m/day if the total thickness of the borehole depth below the transition zone is considered. A wider range of K values will be adopted for the numerical groundwater model as per the following discussions.

Table 18-2 Pump Test Data

ID	UTM - 29N			Static Water Level		Pump Inlet	Step Test				Constant Rate Test				Recovery		Drawdown in Obs BH	T Value	K
	X	Y	Z				No. Steps	Duration	Yield	Drawdown Obtained	Duration	Yield	Available Drawdown	Drawdown Obtained					
				(mbGL)	(maMSL)	(mbGL)		(min)	(L/s)	(m)	(min)	(L/s)	(m)	(m)	(min)	(%)	(m)	(m2/day)	(m/day)
MRBH009	756815	964406	373.5	35.7	337.8	99.1	3	60	0.81	13.3	480	1.1	62	35.5	180	97	0.77	1.437	0.026
								60	1.05	17.3									
								60	1.32	30.3									
MRBH006	756805	964412	373.9	10.2	363.7	79.3	3	60	4.04	15.3	2880	4.8	66.6	46.2	240	63	MRBH10 = 0.19 MRBH05 = 0.63	7.395	0.134
								60	4.93	15.0									
								60	5.00	7.2									
MRBH007	756737	964702	381.3	28.5	352.7	99.2	3	60	1.10	10.7	480	2.1	70.7	44.8	180	97	MRBH003 = 2.11 FRCBH001 = 5.25	3.58	0.048
								60	1.53	7.6									
								60	2.02	28.9									
MRBH010	756628	964155	369.0	2.9	366.1	99.3	3	60	0.66	12.6	480	0.9	97.1	38.1	180	87		7.28	0.208
								60	1.01	31.6									
								15	1.54	49.3									

18.1.4 Hydrogeological Conceptual Model

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 3 to 8% of MAP. The mean annual precipitation (MAP) is in the order of 1100 mm/annum from the weather station data considered Table 18-3.

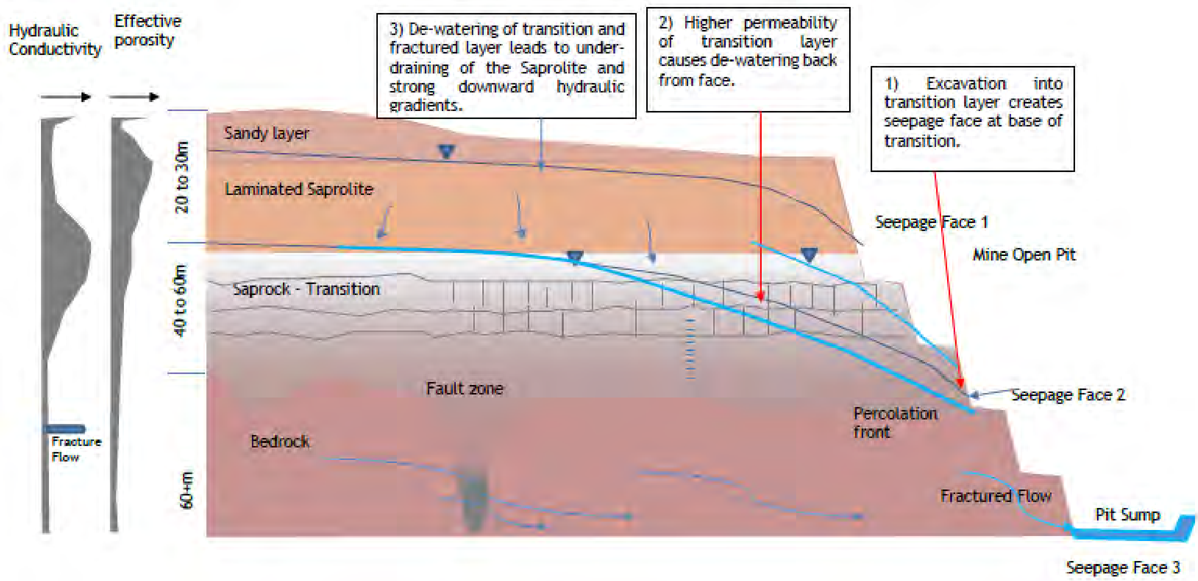
Table 18-3: Weather Station Data

Station	Period	Minimum	Average	Maximum	Type	Coef. Var.
Sarhala	1980-2010	569	1,124	1,623	128	0.11
Worofla	1980-2001	1,054	1,307	1,572	123	0.09
Séguéla	1980-2002	883	1,108	1,316	145	0.13
	2017-2018	1,145				

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 18-12).

Figure 18-13: Conceptual Pit Hydrgeology (Not to scale)



18.1.5 Mine Groundwater Make Prediction

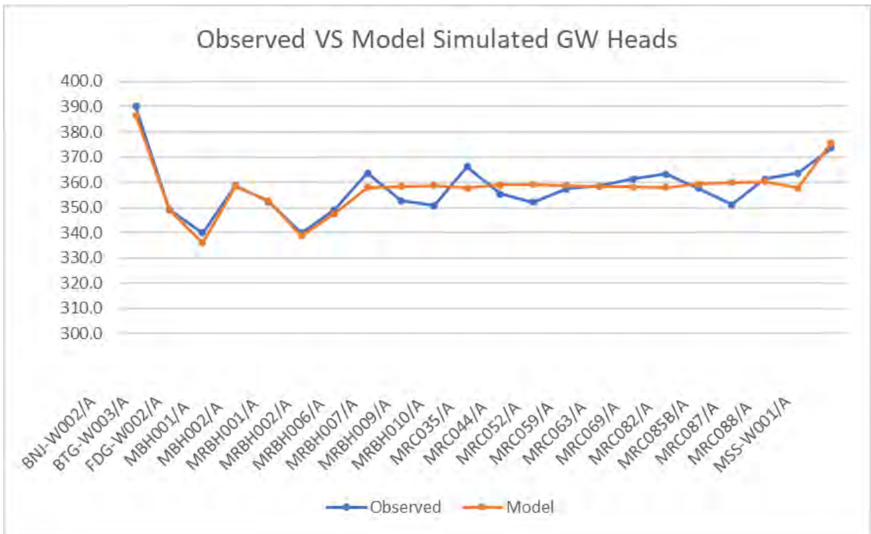
A simplified numerical groundwater model was developed to simulate and predict groundwater flow rates in terms of the main mine pit and the secondary northern pit. The numerical model was based on the conceptual model developed from the desktop and baseline investigations. The numerical model for the project was constructed using the classic version of Visual Modflow, Pro, Build 4.6.0.169 (2019).

Compilation of the finite difference grid using Visual Modflow graphic user interface facilitated the construction of a rectangular horizontal grid, as well as vertical geometry provided for each of the layers. The model grid was refined over the area of the mining pit and larger cells sizes further away. The model grid was vertically subdivided into 13 layers to allow for the inclusion of drain cells at different depths at the proposed mine pit.

Local hydraulic boundaries were identified for use as model boundaries. They were represented by local watershed boundaries, topographic highs, and general head boundaries. These hydraulic boundaries were selected to be far enough from the area of investigation to not to distort the numerical model behaviour.

The correlation between the field monitoring observations and the model output can be seen from Figure 18-14. The achieved correlation is reasonable and model simulations can now be completed.

Figure 18-14: Groundwater Elevations Model vs Observed



The initial model values were derived from the pump tests data. To achieve calibration, the hydraulic conductivity and other input model parameter values were varied until a reasonable match was obtained between the model-calculated groundwater levels and the observed groundwater levels. The calibration was done as a steady state calibration.

To simulate the predicted groundwater inflow rates, drain cells were applied to the proposed mine pits. The drains were simulated in the calibrated numerical groundwater model and assigned according to the mine level depth and proposed time schedule of the development of the level. The drains were kept active for the duration of mining, 14 years.

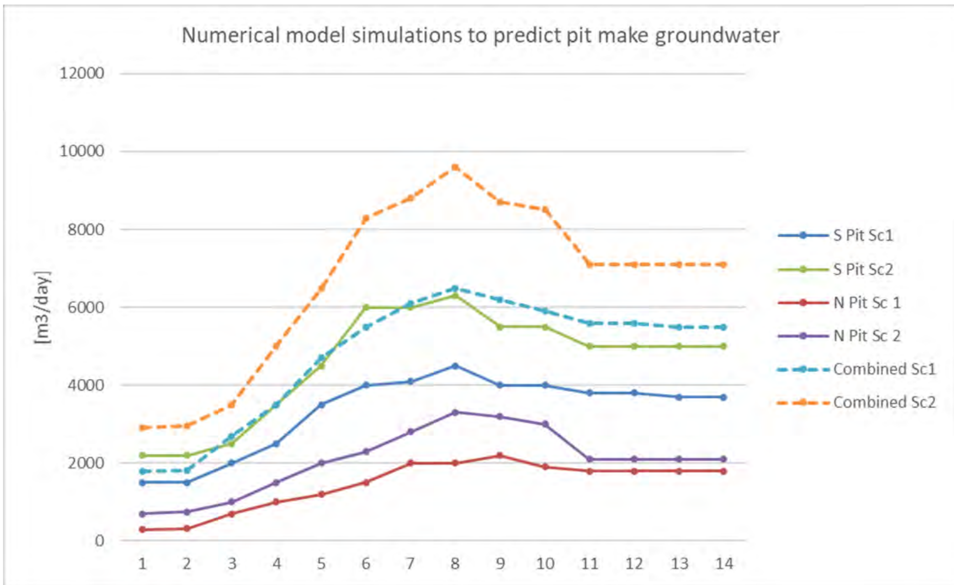
The groundwater balance depicted from the numerical groundwater flow model for the proposed mine workings were obtained from the model's Zone Budget function, the values were calculated for the main mining pit (S pit) and secondary northern pit (N pit).

Several model simulations were completed where the hydraulic conductivity was altered to demonstrate the sensitivity of small changes to hydraulic conductivity (or transmissivity) and to supply a range of probable inflows (Table 18-4 and Figure 18-15).

Table 18-4: Model Simulated Pit Inflow Predictions

Time [day]	Time (y)	S Pit Sc1	S Pit Sc2		N Pit Sc 1	N Pit Sc 2	Combined Sc1	Combined Sc2	Combined Sc1	Combined Sc2	In Pit De-watering (70%)	Out of Pit De-watering (30%)	Out of Pit De-watering (30% of maximum)	Total groundwater make if 8 BH drilled from start	Ave yield per BH (No of boreholes (8))	Ave yield per BH (No of boreholes (8))	In Pit De-watering (70%)	Out of Pit De-watering (30%)	Out of Pit De-watering (30% of maximum)	Ave yield per BH (No of boreholes (8))	Ave yield per BH (No of boreholes (8))
		(m^3/day)							(l/sec)		(m^3/day)					(l/sec)	(m^3/day)				(l/sec)
365	1	1500	2200		300	700	1800	2900	21	34	1260	540	1950	2580	243.75	3	2030	870	2800	350	4
730	2	1500	2200		320	750	1820	2950	21	34	1274	546	1950	2714	243.75	3	2065	885	2800	350	4
1095	3	2000	2500		700	1000	2700	3500	31	41	1890	810	1950	3273	243.75	3	2450	1050	2800	350	4
1460	4	2500	3500		1000	1500	3500	5000	41	58	2450	1050	1950	3910	243.75	3	3500	1500	2800	350	4
1825	5	3500	4500		1200	2000	4700	6500	54	75	3290	1410	1950	4911	243.75	3	4550	1950	2800	350	4
2190	6	4000	6000		1500	2300	5500	8300	64	96	3850	1650	1950	5415	243.75	3	5810	2490	2800	350	4
2555	7	4100	6000		2000	2800	6100	8800	71	102	4270	1830	1950	5793	243.75	3	6160	2640	2800	350	4
2920	8	4500	6300		2000	3300	6500	9600	75	111	4550	1950	1950	6500	243.75	3	6720	2880	2800	350	4
3285	9	4000	5500		2200	3200	6200	8700	72	101	4340	1860	1950	6290	243.75	3	6090	2610	2800	350	4
3650	10	4000	5500		1900	3000	5900	8500	68	98	4130	1770	1950	6080	243.75	3	5950	2550	2800	350	4
4015	11	3800	5000		1800	2100	5600	7100	65	82	3920	1680	1950	5870	243.75	3	4970	2130	2800	350	4
4380	12	3800	5000		1800	2100	5600	7100	65	82	3920	1680	1950	5870	243.75	3	4970	2130	2800	350	4
4745	13	3700	5000		1800	2100	5500	7100	64	82	3850	1650	1950	5800	243.75	3	4970	2130	2800	350	4
5110	14	3700	5000		1800	2100	5500	7100	64	82	3850	1650	1950	5800	243.75	3	4970	2130	2800	350	4

Figure 18-15: Range of Predicted Groundwater Ingress



It can be seen from the table and graph that:

- Scenario 1 (Sc1) demonstrates the probable groundwater flow towards the allocated drain cells with hydraulic conductivity values applied as per the pump test data obtained from the field data. This can be regarded as the "likely and conservative" scenario.
- Scenario 2 (Sc2) represents hydraulic conductivity values slightly increased with overall higher groundwater flow dynamics. This can be regarded as the "high and over conservative" scenario.
- Table 18-4 was further sub-divided to illustrate a 30:70 split for out of pit de-watering boreholes and in pit sump de-watering requirements. It is believed that Sc1 can be applied for PEA planning purposes.
- These simulated predictions do not include any direct rainfall or storm water flow during and after rain events.

18.1.6 Mine Pit De-Watering Design

De-watering of the proposed ~470m deep South Main Pit and the ~150m deep North Pit and the slope depressurization the following key aspects have been considered:

- The areas where the upper saprolite layer occur below the water table will pose stability issues.
- The lower fractured rock aquifer will pose stability issues along the mafic dyke system that was identified to crisscross the mineralized zone from north to south. This dyke dips slightly eastwards and where intersected by test and geotechnical boreholes showed higher hydraulic conductance (transmissivity).
- Combining in and out-of-pit de-watering is used to address the higher risk of flooding of the pits when mining reaches greater depths. At the ratio of in-pit to out-of-pit de-watering 70:30:
  - Approximately 1,200 to 4,500 m^3/day from in pit de-watering for (between years 1 to 8 from where it will stabilise) will be required, and
  - Approximately 1,800m^3/day from perimeter boreholes will be required.

- The challenge with perimeter boreholes is to ensure that zones of higher hydraulic conductivity and groundwater flow is intersected and that yields in the order of 4 l/sec be achieved to make the economic of a deep production boreholes feasible. A follow up groundwater assessment for the FS will evaluate the proposed locations. Eight boreholes will be required to yield between 1,500 and 2,200 m^3/day).
- Perimeter groundwater interception should be implemented on a small scale prior to mine development started.

18.1.7 Groundwater Quality

The groundwater samples obtained during the pump testing and hydrocensus were submitted to the analytical laboratory in Abidjan. A summary table is provided in Table 18-5:

- Three standards are displayed, the WHO Drinking Water Standards, IFC Mining Effluent guidelines and South African Drinking Water Standard. The latter was included as some constituents do not have a specified exceedance value in the former two standards.
- Arsenic concentrations occur slightly above the WHO guideline limit for drinking water quality of 0.01 mg/l. it is recommended that follow up groundwater samples be assessed to establish baseline and ambient conditions. If arsenic concentrations remains high in mine discharge water further/additional containment and/or treatment options will be included in the FS.

Table 18-5: Ground Water Quality Data

DETERMINANT	UNIT	SMOBH001 Monday, 14 December 2020	FDGBH001 Monday, 14 December 2020	MRBH010 Monday, 14 December 2020	MRBH007 Monday, 14 December 2020	BTGBH001 Monday, 14 December 2020	MRBH009 Monday, 14 December 2020	MRBH006 Monday, 14 December 2020	2017 WHO Guidelines for Drinking Water Quality	2015 SABS SANS 241-1 Standards	2007 IFC Guidelines for Mining Effluent
pH		6.5	6.6	6.1	6.3	6.3	6.8	6.5	n/s	5.0 - 9.7	n/s
Electrical Conductivity	uS/m	38	44.5	29.6	43.9	65.5	41.8	45.7	n/s	70	n/s
Hardness	mg/l	45.5	57.8	45.2	55.3	56.8	50.3	55.3	n/s	n/s	n/s
Alkalinity	mg/l								n/s	n/s	n/s
Turbidity	NTU	0.4	0.96	3.6	0.8	0.39	1.8	0.36	n/s	1	n/s
Calcium	mg/l	43.6	62.9	37.86	67.35	128.3	60.13	75.53	n/s	n/s	n/s
Magnesium	mg/l	22.64	18.43	10.55	13.47	14.05	15.46	12.51	n/s	n/s	n/s
Sodium	mg/l	15.6	13.3	15	17	16.7	16.3	14.9	200	200	n/s
Potasium	mg/l	1.942	7.298	3.779	5.887	3.534	8.64	5.6	n/s	n/s	n/s
Iron	mg/l	<0.05	<0.05	<0.05	0.2177	<0.05	<0.05	<0.05	0.30	0.30	2.00
Lead	mg/l	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	0.01	0.01	0.20
Bicarbonate	mg/l								n/s	n/s	n/s
Sulphate	mg/l	<6	<6	37.38	<6	144.1	<6	<6	250	250	n/s
Chloride	mg/l	<5	8.52	<5	<5	9.9	<5	<5	250	300	n/s
Manganase	mg/l	0.064	0.087	0.03654	0.08879	0.02632	0.05617	0.1484	0.4	0.1	n/s
Copper	mg/l	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	2	2	0.3
Amonium	mg/l								n/s	n/s	n/s
Cynide	mg/l								n/s	0.20	n/s
Arsenic	mg/l	<0.005	0.0078	0.006772	0.0108	0.01833	0.01307	0.006617	0.01	0.01	0.10
Nitrate	mg/l	<0.177	0.177	<0.177	<0.177	2.205	<0.177	<0.177	3.00	0.90	n/s
Fluorine	mg/l	0.25	0.3	0.3	0.1	0.8	0.1	0.15	1.50	1.50	n/s
Ortho Phosphate	mg/l								n/s	n/s	n/s
Total Dissolved Solids	mg/l	167	186	135	190	288	185	196	n/s	490	
Total Suspended Solids	mg/l	<5	<5	7	<5	<5	<5	<5	n/s		
Chlorine	mg/l	0.2	0.2	0	0	<0.02	<0.02	<0.02	5.0		
Nitrites	mg/l	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	3.0		
Zinc	mg/l	<0.05	<0.050	<0.05	<0.05	<0.05	0.3678	<0.05	0.1		
Chromium	mg/l	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	0.1		
Nickel	mg/l	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	0.1		
Cadmium	mg/l	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	0.0		
Antimony	mg/l	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	0.0		
Aluminium	mg/l	<0.005	<0.005	0.05898	<0.005	<0.005	<0.005	<0.005	0.2		
Molybdenum	mg/l	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	0.0		
Mercury	mg/l	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	0.0		
Total Hydrocarbons	mg/l	0.00723	0.00435						n/s		

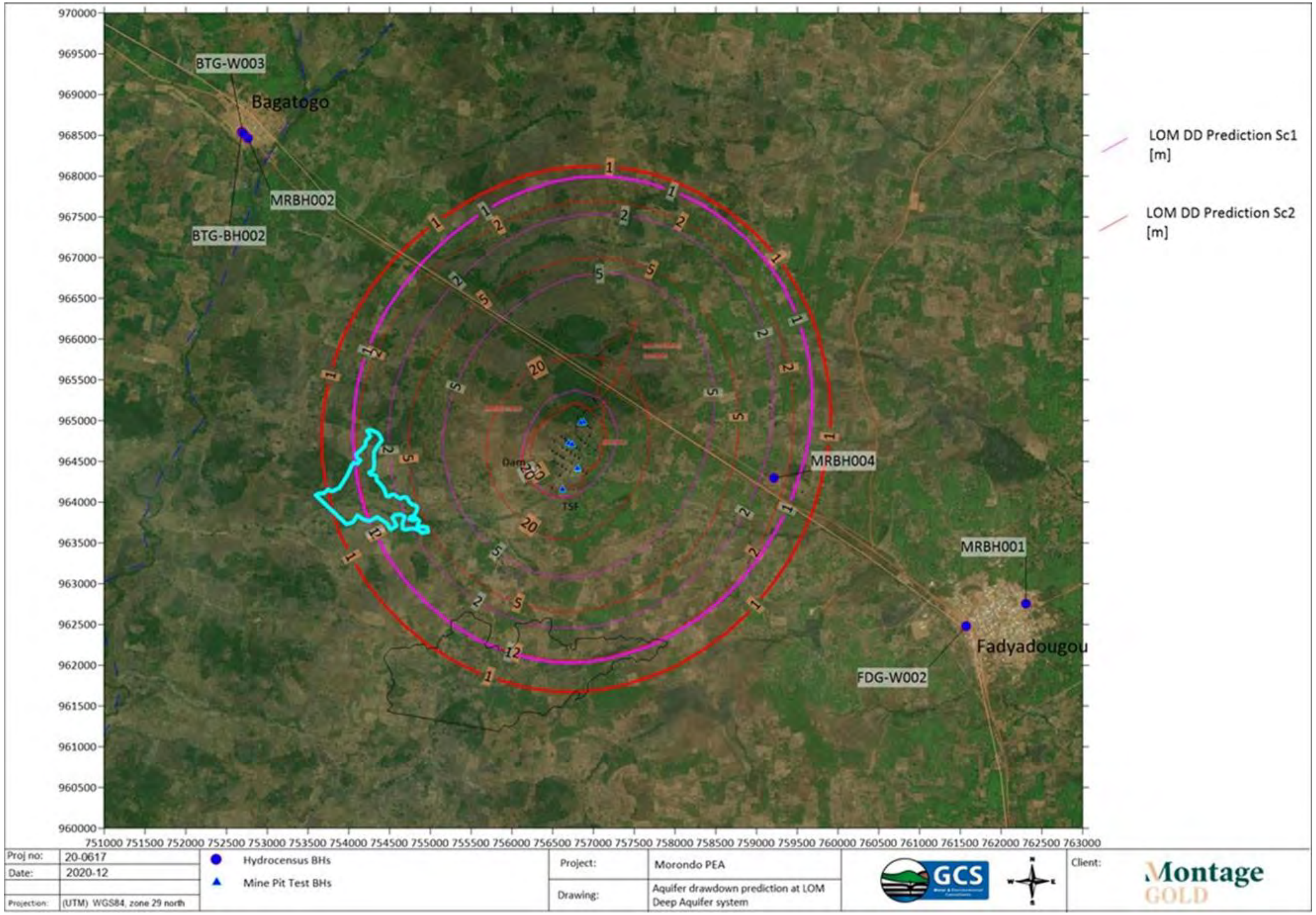
18.1.8 Environmental Impact of Pit De-Watering

The numerical groundwater model was applied to simulate the extent of the likely cone of depression or zone of influence developed by 14 years of constant pit de-watering activities. Figure 18-16: supplies an overview of the expected zone of influence by means of the 1m drawdown contour line. An area of approximately 6km in radius will be affected and no community supply boreholes occur in this zone. The only water supply borehole within the zone is the Montage borehole situated at the core shed (MRBH004). The environmental risk of mine de-watering will pose a low risk for regional groundwater supply activities.

No major rivers and streams occur within the zone of influence.

The presence of Arsenic in some samples need to be followed up in future groundwater sampling and analyses. If present under pre-mining conditions, it may increase once mining and oxidation processes commence. The presence of Arsenic in the ore and various waste rock types needs to be understood to determine the risk level.

Figure 18-16: Pit Dewatering - Predicted Zone of Influence



Preliminary Aquifer Calculations

Preliminary pit de-watering volumes were estimated based on typical aquifer parameters obtained from literature. Estimates were provided for in-pit de-watering and out-of-pit dewatering and presented in Table 18-3. Total de-watering volumes may range between 1,500 to 6,000 m^3/ day. A better understanding of potential pit de-watering requirements will be made after the first round of hydrogeological tests.

Table 18-3: Description of community boreholes drilled by Montage

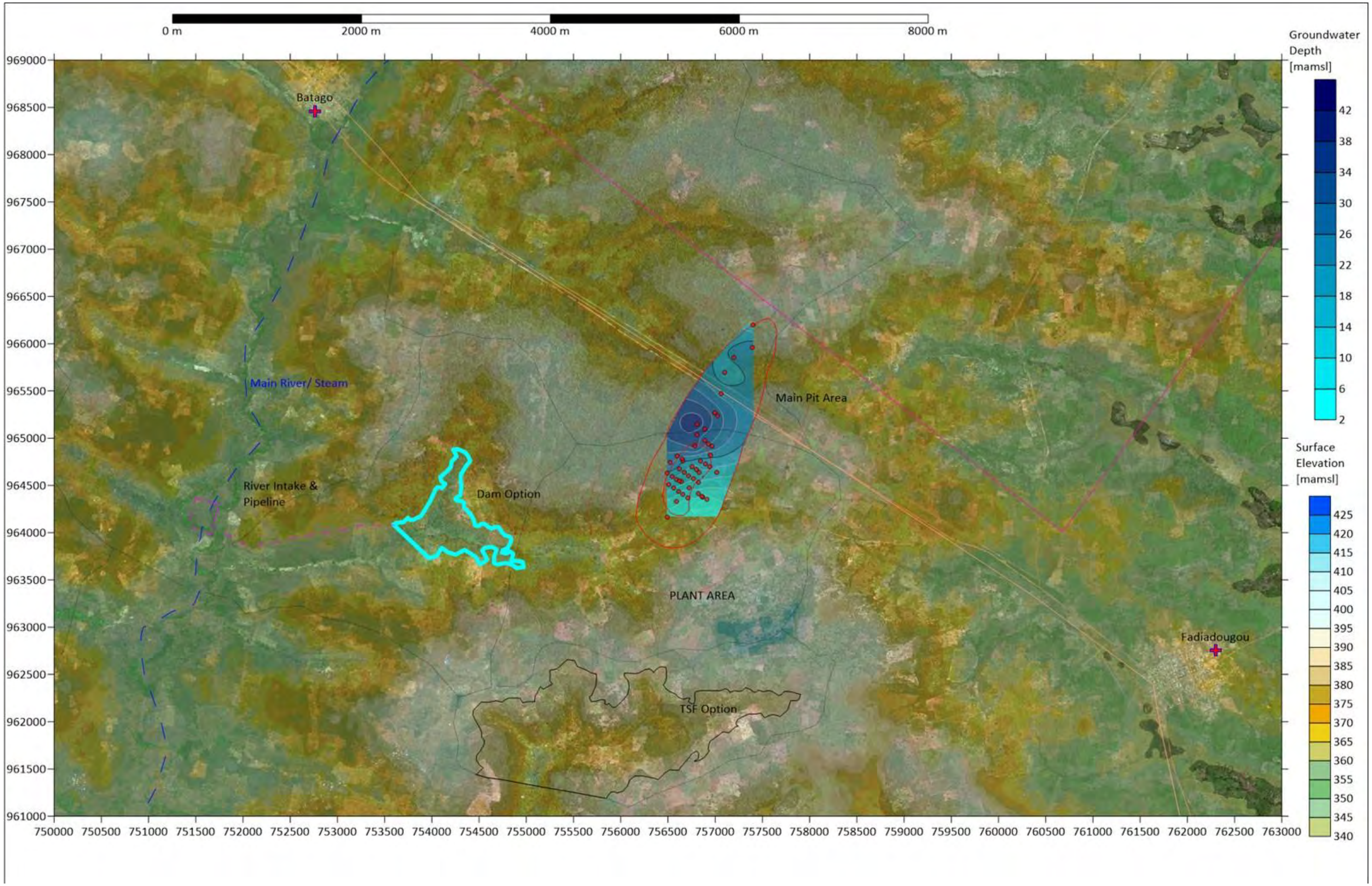
Year	Preliminary Groundwater Availability (m^3/day)		
	Pit Dewatering	Supplementary Borefield	Total
1	610	1,500	2,110
2	2,270	1,500	3,770
3	1,230	1,500	2,730
4	2,990	1,500	4,490
5	4,490	1,500	5,990
6	4,490	1,500	5,990
7	2,990	1,500	4,490
8	2,990	1,500	4,490
9	2,250	1,500	3,750

Groundwater and Pit Water Management

The following outlines the typical de-watering management system for similar open pits:

- Open pit dewatering is recommended using sub-horizontal gravity drainage boreholes (drains) drilled into the walls of the pit in areas where seepage is observed. The depth (length), sequence and potential localities can only be confirmed after preliminary hydrogeological assessments is completed.
- In the event that dewatering through the use of drains is not effective, or inflow rates are higher than expected, it is recommended that a number of dewatering boreholes are drilled along structural features which have been shown to contribute to inflow. This can reduce the groundwater inflow to the pits by intersecting the flows before they reach the walls.
- Groundwater extracted from the open pits can be discharged to the WSF. The extracted groundwater augments the process water supply to some degree, although it will not be sufficient to meet all of the process water requirements.
- Dewatering of the open pit will affect the groundwater quantity by lowering the piezometric water elevation and this influence requires monitoring so that there is minimal or no impact on nearby boreholes and surface water (such as community supply boreholes and base flow in streams).
- Groundwater quality should also be routinely monitored throughout the Life of Mine (LoM) to control any impact of mining activities such as leakage from the TSF, WSF, the discharged water, and the storage and handling of chemicals, such as reagents and fuels.

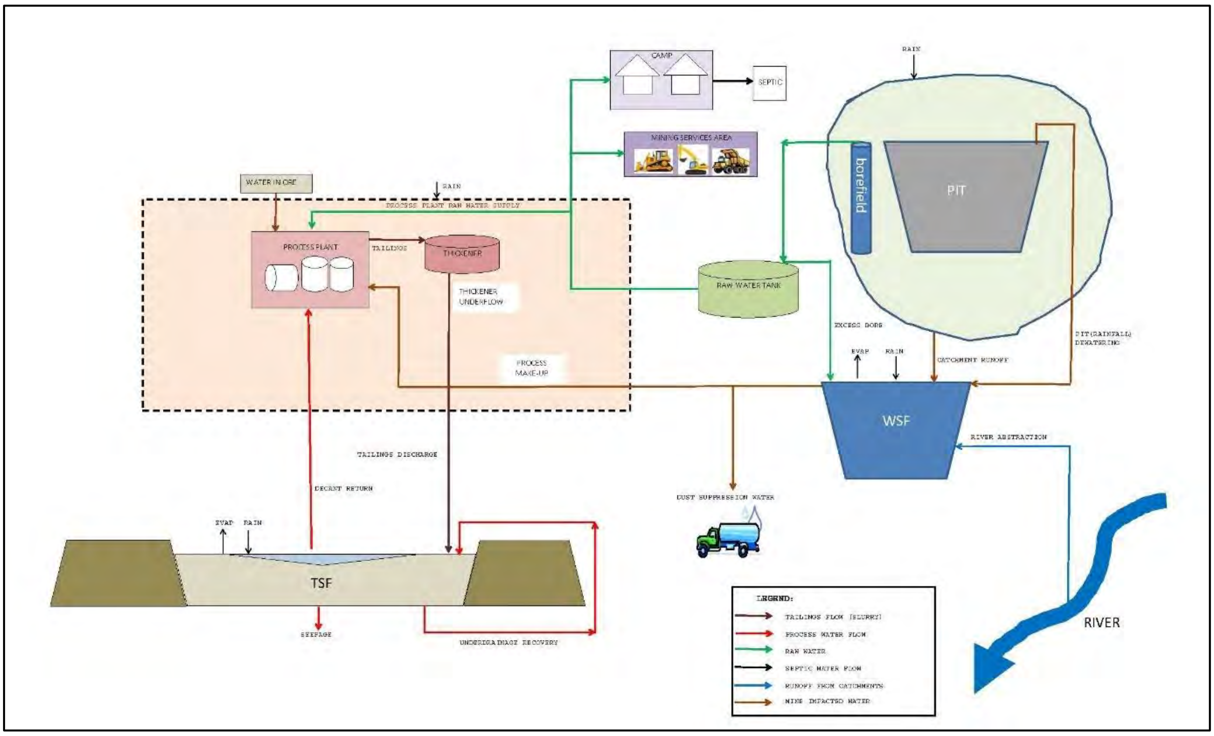
Figure 18-17: Site Layout concept, Groundwater Depth and Village Wells



18.1.9 Water Balance Modelling

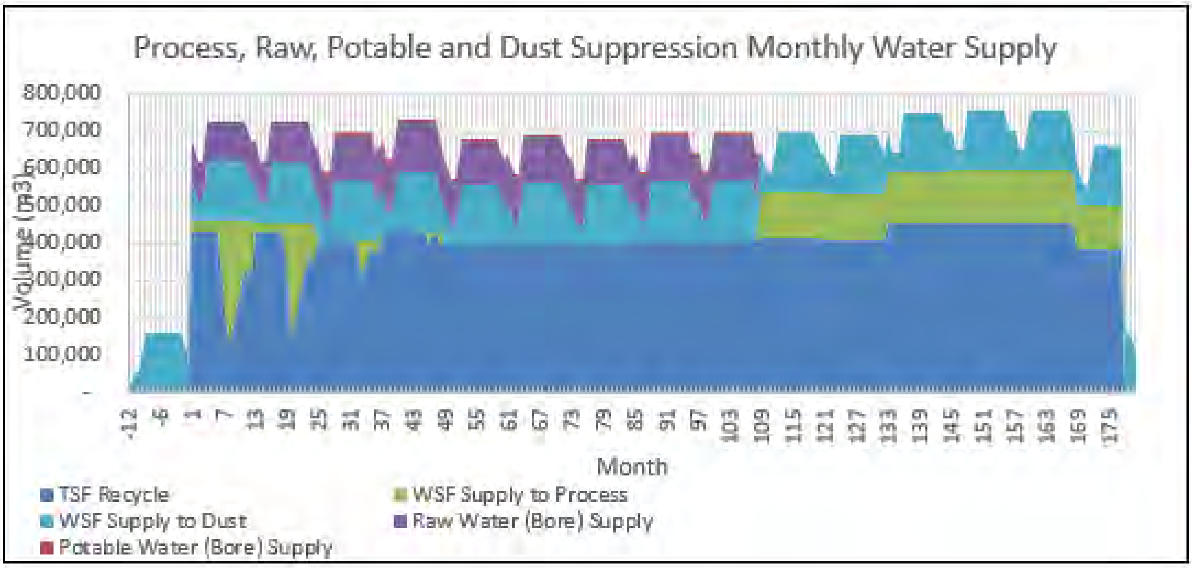
Water demand for the mineralisation processing, potable water and dust suppression demands will be supplied to the project from tailings storage facility recycle, river harvesting, pit dewatering, harvesting of runoff from the mining disturbance areas and a supplementary borefield for potable water supply. A preliminary water balance model has been established to estimate the water demands for the project, to assess the availability of water to meet the demands and to size the various component of the water management system. The schematic flow diagram for the model in shown in Figure 18.18.

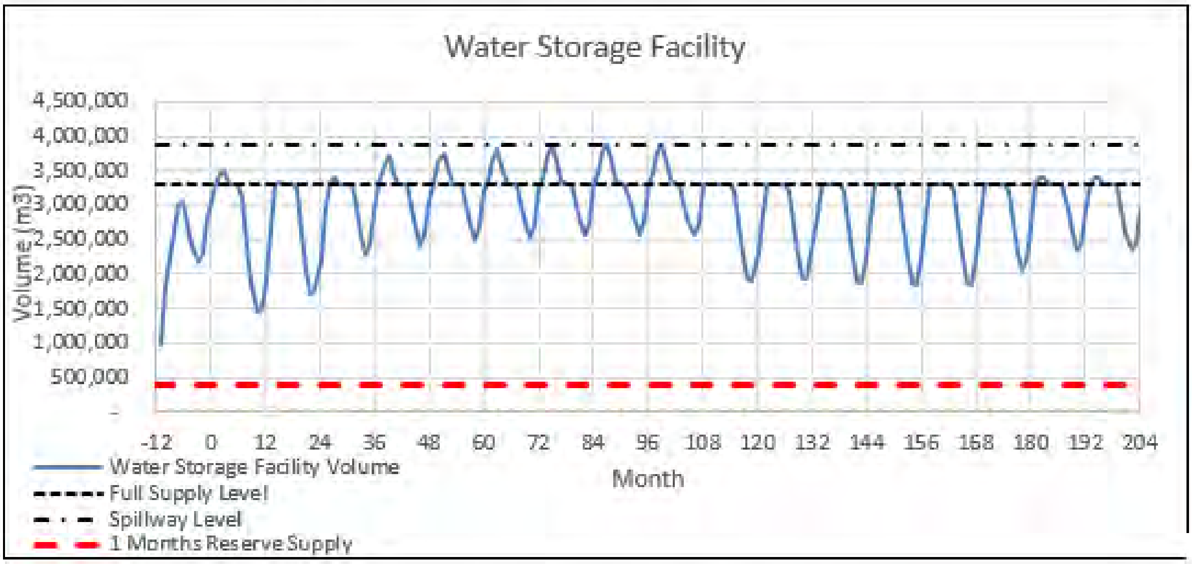
Figure 18.18 Waterbalance Schematic

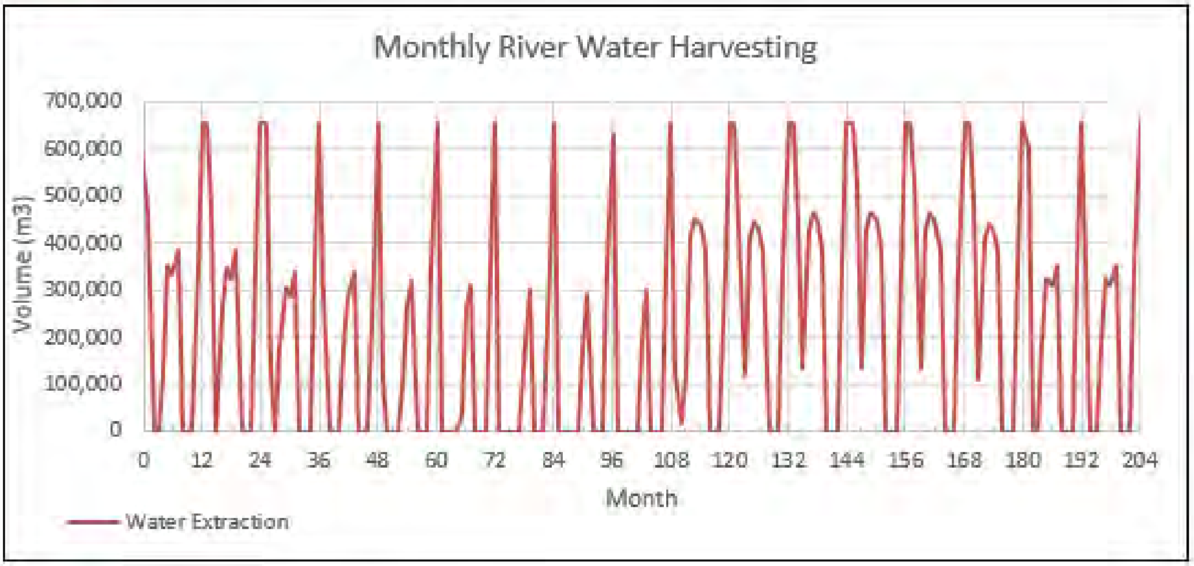


The model indicates that sufficient water is available to meet process, potable and dust suppression demand under average and extreme dry conditions, although the early years of operation are the most critical in terms of water supply and demand. The water balance results under average conditions are summarized in Figure 18.19.

Figure 18.19 Waterbalance Results (Average Conditions)







18.1.10 Water Storage facility

A water dam will be constructed downstream of the mining and processing area to act as the main water storage facility and sediment control dam. The facility will have a capacity of 3.9 Mm^3 (up to the spillway invert level) with a pond area of 110 Ha. The water storage facility embankment will be a maximum of 16 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.

18.1.11 Water Harvesting Facility

The river abstraction facility will be constructed adjacent to the Marahoué River at a location approximately 26 km east of the WSF. The facility will comprise a sump to capture and allow for harvesting of water. Water will be reclaimed from the facility by a pump mounted on a floating pontoon. A pipeline alignment has been nominated between the river abstraction location and WSF, with an access road located adjacent to the pipeline to allow for inspection and maintenance.

18.2 Power Supply

Electric power consumption for the Koné project is estimated to be:

- Connected load 59.3 MW
- Maximum Demand 47.4 MW
- Average annual demand 39.0 MW (at a load power factor of 0.95 lagging)
- Energy consumption 342 GWhr/yr
- LNG delivered price $10/GJ
- HFO delivered price $0.67/kg

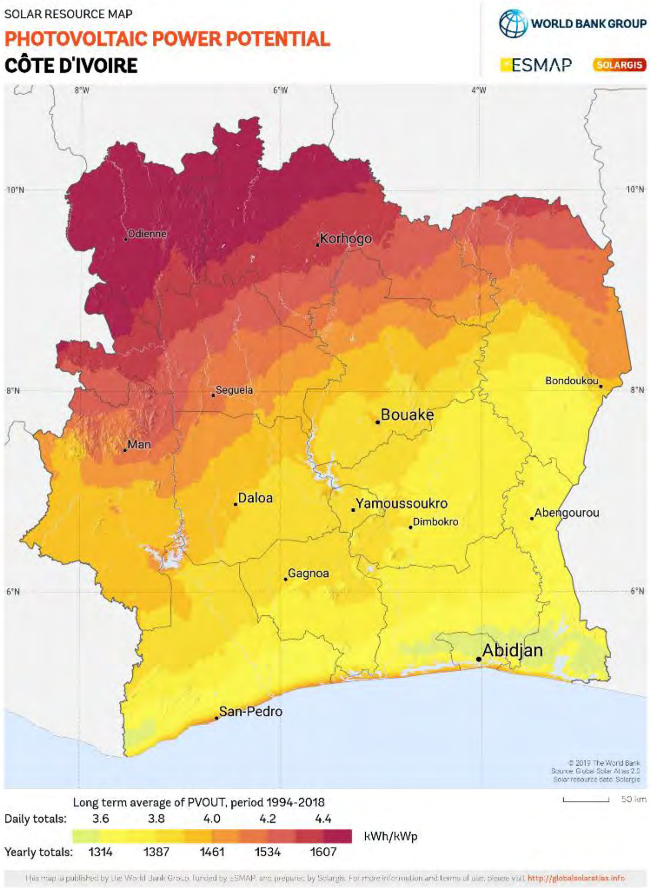
Options for the supply of electric power to the project are:

- Grid connection to the CIEnergie network
- Construction of an HFO Power Station (with Solar / BESS)
- Construction of a LNG Power Station (with Solar / BESS)

A comparison of the estimated capital cost and operating cost for a variety of options was undertaken by ECG and reported in their report “KON-G-RP-0002-B - Power Supply Options Review”.

The grid returned the lowest capital cost but due to the high unit cost was discarded. The capital costs for the HFO and LNG were not substantially different, however the LNG fuel returned the lowest LOM NPV. The addition if solar with BESS reduced the LOM cost further. The solar irradiance has been modelled based on the NASA surface meteorological data for the Ivory Coast Koné Gold Mine site, Figure 18-9

Figure 18.20 Côte d'Ivoire Solar Irradiance Map



A BOOT commercial model assumes that the supplier will be responsible for the design, construction, financing, ownership and operation of the asset until it is transferred to the owner at the conclusion of the contract period – which is assumed to be 10 years or other period by negotiation.

The supplier will be responsible for the ongoing operation and maintenance of the equipment and guarantee the power supply at a standardised cost per kWhr at the outgoing feeders for the life of the contract. This includes the following;

- Operate and maintain the power station and solar plant including meeting all key performance indicators – fuel, gas and solar yield guarantee
- Scheduling and performing routine maintenance of overall asset and equipment as per manufacturer's recommendations
- Engine overhauls as per manufacturer's recommendations
- Procurement and holding of onsite spares necessary to maintain the power station and solar plant availability
- Containment and handling of waste oil (incinerator or other)
- Provide and maintain at least two or more qualified operators and maintenance technicians onsite at any given time.

It has been assumed that he LNG supply will be procured from Prestea GCP in Prestea, Ghana. The feed gas for the Prestea GCP is natural gas sourced from Ghana's gas fields via a pipeline infrastructure in Ghana. LNG will then be trucked in bulk road vehicles (BRVs) to Koné for power generation.

LNG from Prestea can be trucked quickly and effectively using specialized LNG tank trucks of 23.67MT (typical industry capacity requirement for LNG tank trucks) by road covering a total distance of 725km. Seven trucks per day will be required to meet daily fuel consumption.

The LNG/Solar/BESS hybrid power station solution has been selected as the preferred power supply option for KGP. This option selected has a pre-production cost of $2.5M for earthworks etc, sustaing capital of $140.4M (10 year BOOT payments) and a unit price of $0.076/kWhr.

18.3 Tailings Storage Facility

The tailings management arrangement comprises two cells located approximately 3.2km southwest of the process plant. The two cells are separated by a natural ridgeline, with each cell confined by a cross valley embankment. Initially the northern cell (TSF A) will be constructed, with the southern cell (TSF B) constructed later in the mine life to reduce the capital expenditure in the early mine life. Towards the end of the mine life, the two cells will merge into a single facility, separated by a deposition causeway. Deposition will rotate cyclically between the cells of the merged facility with the confining embankments raised concurrently.

A consequence category assessment has been carried out in accordance with ANCOLD guidelines, which indicated the dam break severity level for the both TSFs is 'major', with TSF A classed as High C and TSF B as High A based on the estimated population at risk. The consequence category of the final combined facility would also be High B.

The embankment crest elevations and storage capacities for each facility are provided in Table 18.7.

Table 18-6: Key Tailings Storage Facility Parameters

Facility	Year	Stage	Tailings Storage (Cumulative) (Mt)	Tailings Storage (Cumulative) (Months)	TSF Embankment Elevation (m RL)	Embankment Volume (Cumulative) Mm^3
TSF A	-1	1	9.9	12	375.2	0.57
	1	2	20.9	24	378.6	0.89
	2	3	31.9	36	382.2	1.49
	3	4	42.9	48	385.1	2.07
	4	5	53.9	60	387.8	2.68
	5	6	64.9	72	390.2	3.26
TSF B	6	1	75.9	84	383.7	3.88
	7	2	97.9	108	390.2	4.56
Combined TSF	9	1	119.9	132	392.9	5.71
	11	2	141.9	156	394.9	7.73
	13	3	161.1	180	396.7	8.38

18.3.1 TSF A

The TSF A 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 the natural terrain.

The initial two stages of the embankment will be constructed by downstream construction techniques, followed by seven subsequent modified centreline raises. The initial two stages 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). Due to the difficulties associated with installing HDPE on centreline raised facilities, the subsequent stages exclude a liner and incorporate a two stage filter (Zones F1 and F2) between the upstream Zone A and downstream Zone C. At this time, the pond is located significant distance from the embankment.

Water from the decant pond will be recovered by a suction pump with floating intake located in a channel excavated adjacent to an access causeway. The normal operating pond extents within the basin area will be lined with HDPE, with a compacted soil liner provided elsewhere. An above liner underdrainage and embankment drainage system will report to recovery towers located immediately upstream of the embankment. A sub-liner seepage recovery drainage system will be installed to prevent groundwater build up below the liner in early stages of operation and to collect seepage during later stages of operation. The sub-liner seepage recovery system reports to a downstream collection sump.

18.3.2 TSF B

The TSF B embankment is located to the southeast west of TSF A and forms a cross valley storage orientated northeast to southwest. The embankment will have a length of 1.5 km at the final stage. Containment for the remainder of the facility will be provided by the natural terrain.

All four stages of the embankment will be constructed by downstream construction techniques. The upstream embankment batter 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).

Water from the decant pond will be recovered by a by a suction pump with floating intake located in a channel excavated adjacent to an access causeway. The normal operating pond extents within the basin area will be lined with HDPE, with a compacted soil liner provided elsewhere. The design includes an above liner underdrainage and embankment drainage system and below liner seepage recovery system. All three drainage systems will report to sumps located immediately upstream of the embankment, with riser pipes provided to allow submersible pumps to be installed and the water to be recovered.

The layout of the tailings storage facilities at the final stage is shown on Figure 18.21 with the proposed embankment sections shown of Figure 18.22

Figure 18-21 Tailings Storage Facility Layout

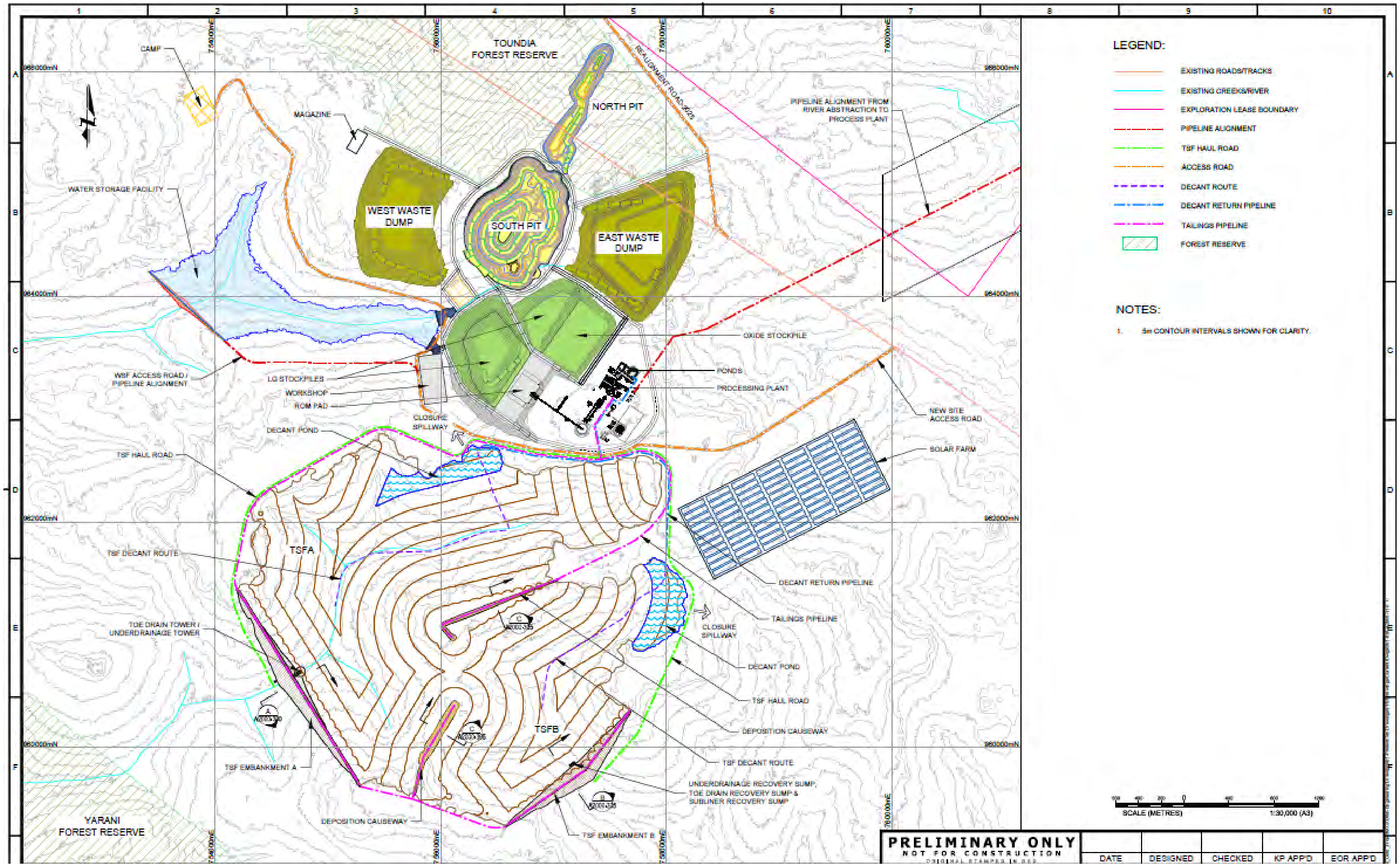
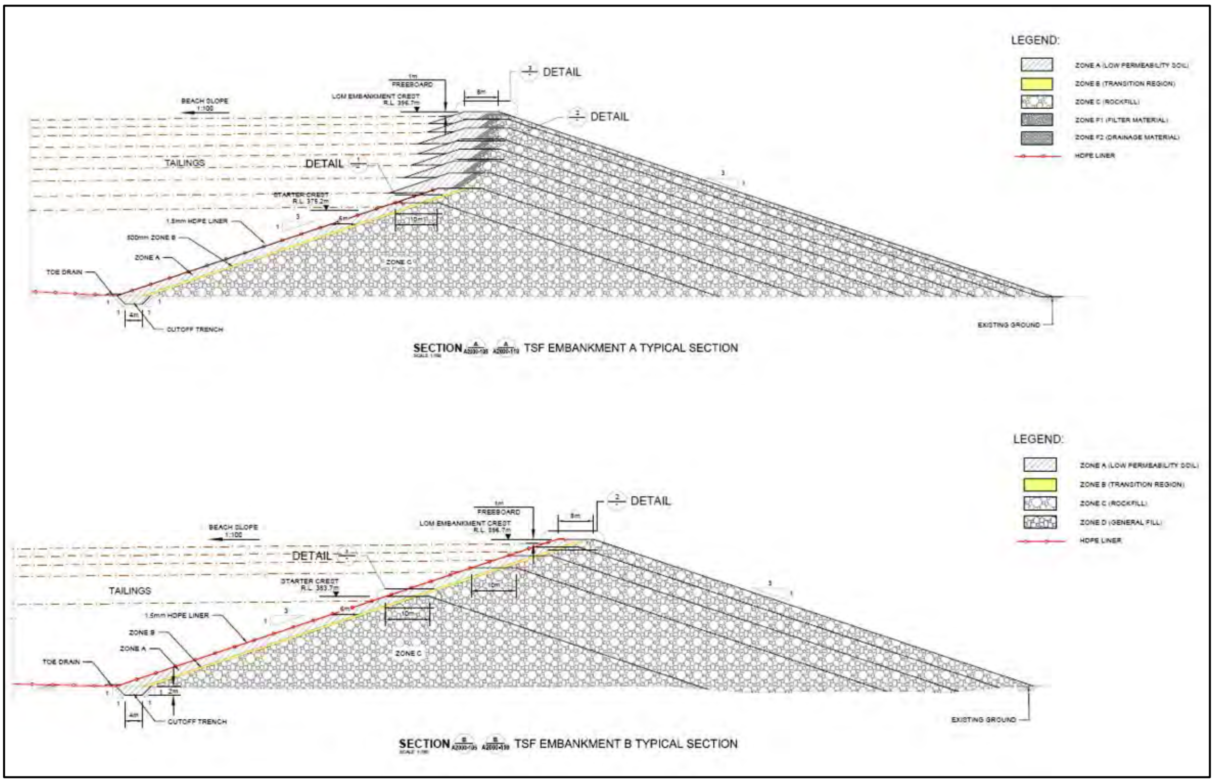


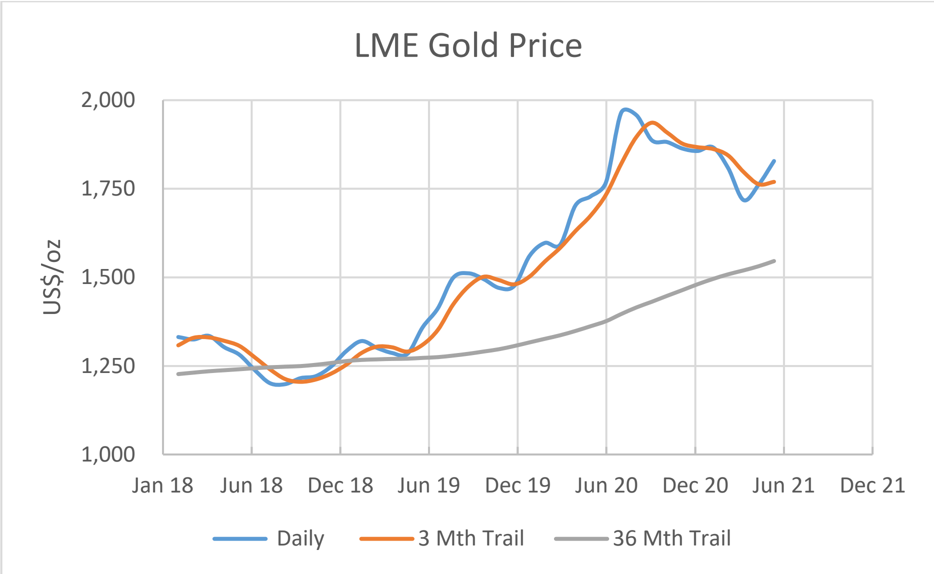
Figure 18-22 Tailings Storage Facility Embankment Design



19. MARKET STUDIES AND CONTRACTS

No Market Studies were carried out for this study. The final product of the Koné project will be gold doré bars. These can be sold in the current market at prevailing global gold prices. Gold bullion sells on several international markets, the most well-known being the London Metals Exchange or LME.

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 should the project progress.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

Mineesia Ltd, a UK based consultancy, has supported the environmental management of the Project activities of the Koné Gold Project, including supporting the development of the environmental impact assessment by CECAF, an Ivorian environmental consultancy. The primary environmental and social requirements include:

- Preliminary evaluation of the project's impact on the environment;
- Schedule of environmental and social other permitting requirements;
- Evaluate project setting for potentially significant environmental and social permitting constraints from site data;
- Collect and review available data from existing databases for environmental studies, assessments or audits;
- Regulatory inspections, waste handling practices; management plans; and all applicable laws and regulations; plans;
- Initiate baseline data gathering;
- Social, training, and health/safety programs identified;
- Draft EIS/EA initiated;
- Preparation of preliminary environmental and social management plans and monitoring programs, including sediment and erosion control plan, conceptual closure plan, assessment of acid rock drainage potential, and preliminary spill and emergency response plan;
- Outline of a site environmental monitoring plan; and
- Comprehensive overview and listing of required permits.

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 Code and the Forestry. Environmental issues are administered by the Ministry of Environment, Urban Sanitation and Sustainable Development and by the National Environmental Agency (Agence Nationale de L'Environnement (ANDE)).

The Environment Code requires that every project must be subject to an Environmental Impact Assessment prior to granting of any authorization. It applies to mining installations and includes the minimum environmental impact study requirements. Decree No. 96-894 (8 November 1996) details the relevant rules and procedures for environmental and social impact assessments for development projects. This decree specifies mining operations as Annexe 1 projects, which require Environmental Impact Assessment. The Mining Code requires that all mining title applicants (excluding artisanal) submit an Environmental and Social Impact Study (EIES, in French) to the General Directorate of Mines and Geology (DGMG) and ANDE and all other institutions as required by the Mining Decree. The Mining Code also includes provisions regarding mine closure. To ensure environmental protection, mining titleholders must open an escrow account in a leading Ivoirian financial institution at the beginning of mining operations, to be used to cover costs related to the environmental management and mine closure plans.

20.3 International Requirements and Guidelines

Côte d'Ivoire is a Member State of the West African Economic and Monetary Union (WAEMU), which enacted a mining code in 2003 (the WAEMU Mining Code). This Mining Code governs any mining operation related to prospecting, exploration, exploitation, detention, traffic, transport, treatment, trade and transformation of minerals within the WAEMU Member States' territories.

Côte d'Ivoire has been a member of the Extractive Industries Transparency Initiative (EITI) since 2008, when the government issued Presidential Decree 2008 25 establishing the EITI multi stakeholder group (known as the National Council (Comité National) for implementation of EITI Principles. Although the EITI Standard does not require or encourage disclosures regarding environmental management, EITI Principles emphasise that natural resource wealth should be an engine for sustainable economic growth. Côte d'Ivoire is one of the countries that have included information related to environment as part of their EITI reporting. The Mining Code also requires adhesion to good governance principles, including the Equator Principles and the Extractive Industries Transparency Initiative principles. As such, mining titleholders must issue EITI reports.

The Koné Project is classified as a Category A development in accordance with International Finance Corporation (IFC) Guidelines, due to the scale and type of operation. The IFC Sustainability Framework, as revised in 2012, with associated Performance Standards on Environmental and Social Sustainability, provide the basis for most impact assessments. Additional guidance is provided by the Equator Principles, which provide an approach to determine, assess and manage environmental and social risk in project financing.

20.4 Project Permitting

The development of the Project will be subject to receiving environmental approval of its design, environmental management programme and appropriate mitigation measures where required. Based on the provisions of the various legal requirements and sectoral laws as well as policies of different departments, the impacts of any proposed project will need to be assessed and appropriate mitigation measures recommended where appropriate.

Under the Mining Code, all applicants for an exploitation licence must submit an EIES to ANDE, which is the environmental authority in charge of supervising, validating and controlling environmental impact studies. The EIES will include an Environmental and Social Management Plan and a site rehabilitation plan. The Environment Code provides the minimum requirements for environmental impact studies, with the purpose of evaluating the environmental effects of an activity and proposing measures to eliminate, reduce or mitigate potential adverse environmental impacts. As a minimum, the EIES must include:

- A description of the proposed activity;
- Description of the environment likely to be affected, including the specific information needed to identify or assess the effects of the proposed activity on the environment;
- List of products used where appropriate;
- Description of the alternative solutions, if any;
- Assessment of the likely or potential effects of the proposed activity and other possible solutions on the environment, including direct, indirect, cumulative effects in the short,
- medium and long term;
- Identification and description of measures to mitigate the effects of the proposed activity and other possible solutions on the environment, and an assessment of these measures;

- 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;
- 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); and
- A financial estimate of the measures recommended to prevent, reduce or offset the negative effects of the project on the environment and regular monitoring and control measures of relevant environmental indicators.

The rehabilitation plan must take into account several aspects such as the cleaning of the site, the dismantling and removal of mining installations, the post-rehabilitation surveillance of the site, and suggestions on how the site could be reconverted. Such operations must start during the exploitation period and not only at the end of operations. After the closure of the mine, any exploitation permit holder remains liable under civil law for damages and accidents on the site that could be caused by the former installations during the five years following closure.

Some protected areas, such as classified forests, places of worship or cultural sites, cannot be subject to mining activities without the prior consent of the owners, occupants and concerned communities, as well as authorisation from the Minister of Mines.

In addition, the Code follows modern African mining legislation, which increasingly aims at protecting the rights of local populations. The Code guarantees a right to a fair indemnity for the land's occupants and legal owners in the event of occupation of the land. Such indemnity will be paid following the signing, under the supervision of the mining administration, of a memorandum of understanding by the exploitation companies, the occupants and the legal owners. The mining code requires setting aside 0.5% of revenue to fund a local mining development committee (CDLM) which is created formally (with Mining ministry, state Admin, etc.) at the local level (prefect, other state services, communities) to handle this fund for local development.

20.5 Project Permitting

The proposed Project layout is described in previous sections, Figure 20-1. In summary, the main components of the operation are anticipated to comprise the following:

- Open pit mines – two open pits to be developed using standard open pit mining techniques, treating approximately 11 Million tonnes/yr;
- Processing Plant – ore crushed prior to adsorption of gold onto activated carbon through carbon-in-leach (CIL) extraction methods. The process plant will be located near the Koné deposit;
- Tailings Storage Facility (TSF) – with capacity of 75 million tonnes and including tailings water drainage system, a recovery water basin and pipeline connecting the TSF to the plant;
- Waste rock dumps – for disposal of overburden and waste material from the open pits;
- Water supply and treatment – Pipe work will be required to supply water from the Marahoué River to a raw water storage facility on the mine site;
- Power will be sourced from an on site LNG/Solar Hybrid power station;

- Associated infrastructure – including haul roads, ROM pads, offices, workshops, domestic waste facility for non-mineral wastes, ablutions and sewage treatment systems, explosives storage and a minerals laboratory;
- A deviation of the national road passing between the 2 pits; and
- Accommodation camps for construction workers and mine employees

Figure 20-1: Site Layout

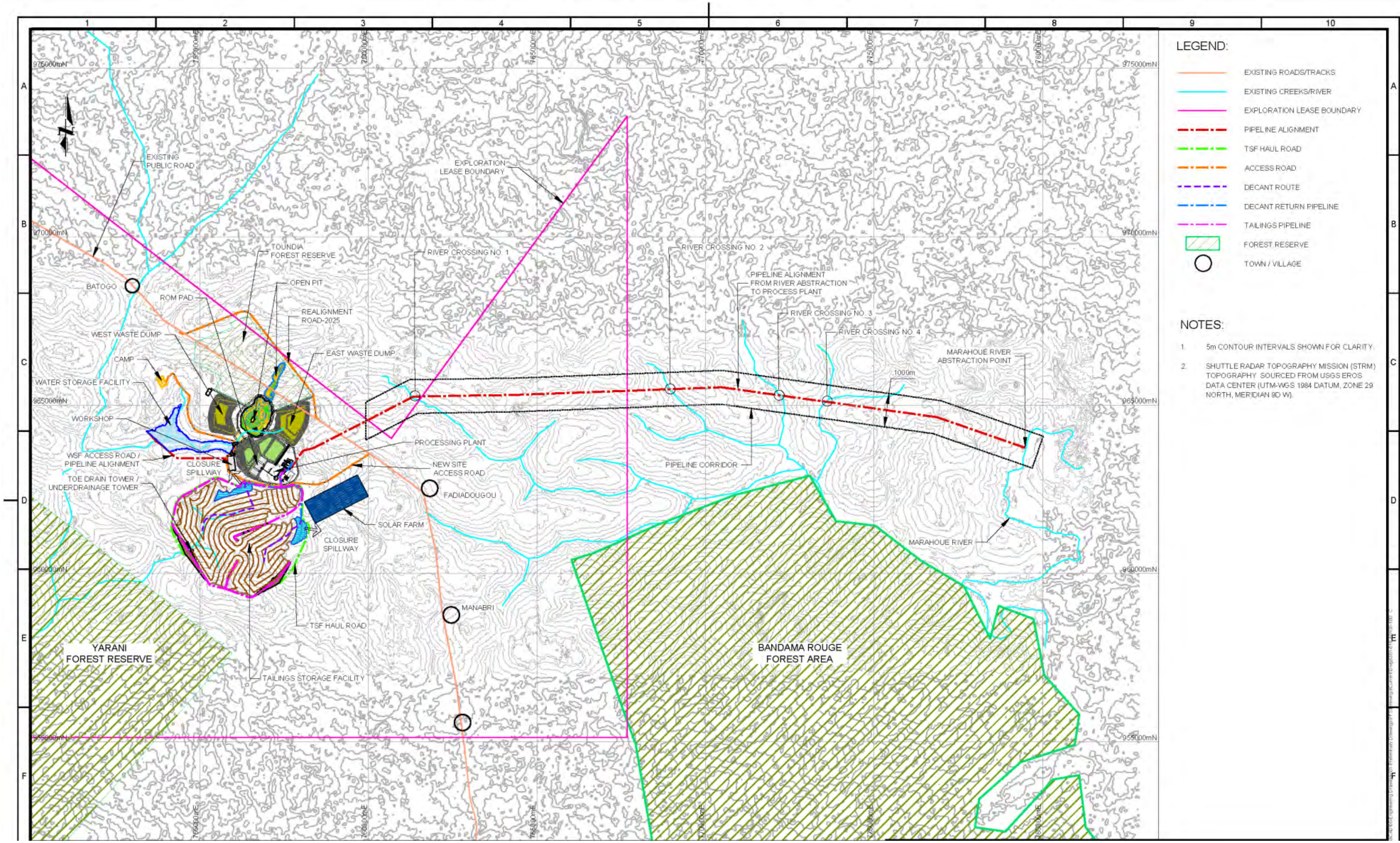
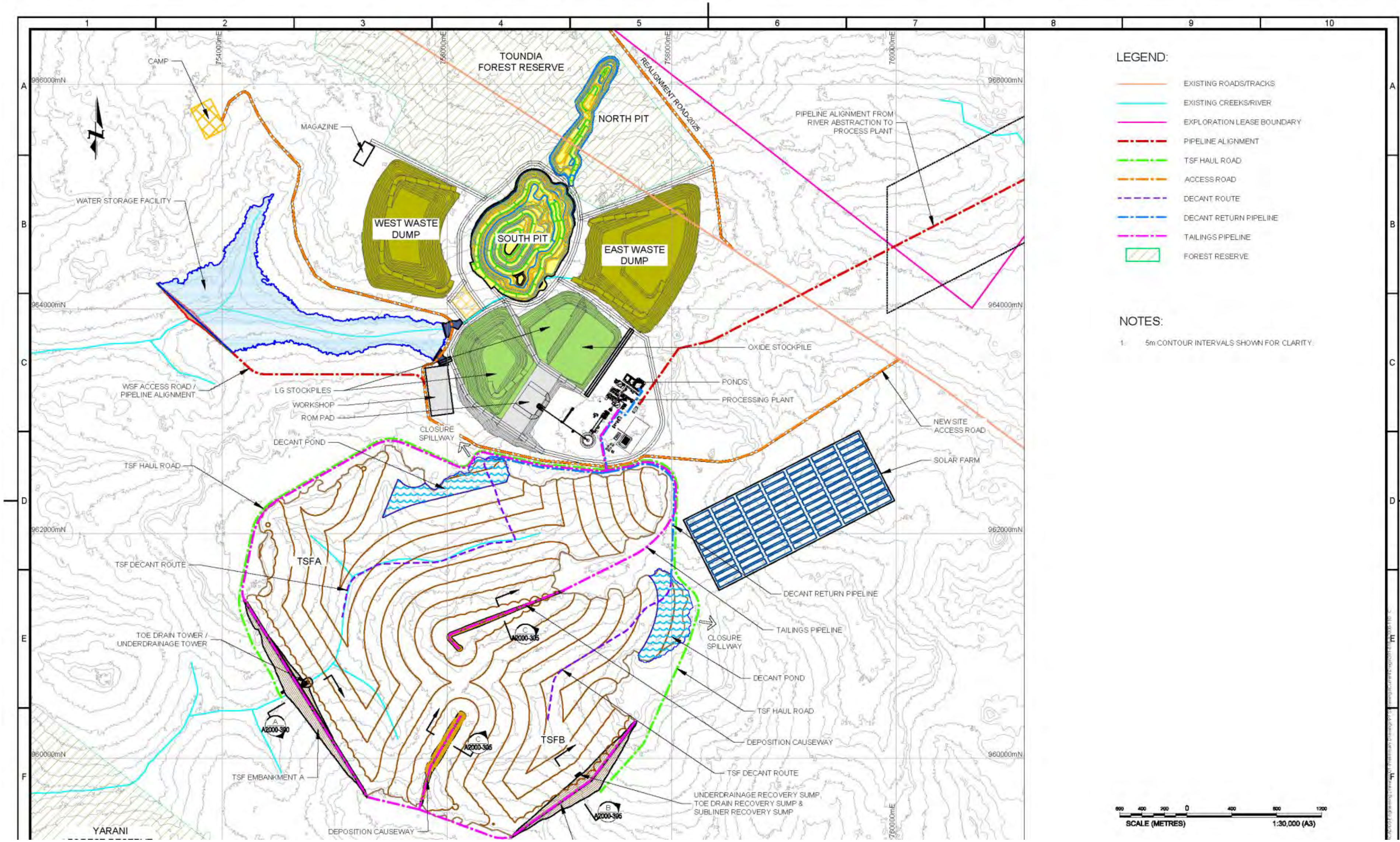


Figure 20-2: Central Site Layout



20.6 Baseline Environmental and Social Setting

Côte d'Ivoire is the most biodiverse country in West Africa, but unlike other countries of the region, its diversity isn't concentrated along the coast, but rather in the interior. More than 1,200 animal species and 4,700 plant species have been recorded.

20.6.1 Project Location

The Project is located midway between the villages of Batogo to the north-west and Fadiadougou to the south-east, along the Séguéla-Boundiali national road joining these 2 villages. A road diversion for this road is currently being assessed.

The smaller northern pit and the northern edge of the larger pit are located within the Toudian classified forest. Permission is being sought to facilitate mining activities in this forest.

20.6.2 Protected Areas

There are protected forest reserves affected by and adjacent to the Project. Forest reserves are portions of state lands where commercial harvesting of wood products is excluded in order to capture elements of biodiversity that can be missing from sustainably harvested sites.

In the northern part of the Koné project area, and covering parts of its extraction pits, is the Toudian Classified Forest Reserve (identified in the UNEP-WCMC database as Classified Forest Reserve Name Unknown (CIV) No. 14) with an area of 527ha. It is listed as an IUCN Category V protected area by UNEP-WCMC, where the interaction of people and nature over time has produced an area of distinct character with significant ecological, biological, cultural and scenic value (Dudley, 2008). In these areas, safeguarding the integrity of this interaction is vital to protecting and sustaining the area and its associated nature conservation and other values.

To the southwest and outside the project footprint is the Yarani Classified Forest Reserve. This is an IUCN Category IV protected area (UNEP-WCMC, 2019), aiming to protect particular species or habitats and management reflects this priority (Dudley, 2008). Many category IV protected areas need regular, active interventions to address the requirements of particular species or to maintain habitats, but this is not a requirement of the category.

To the east and outside project footprint is the Kani-Bandama-Rouge or Bandama Rouge (UNEP-WCMC, 2021) Classified Forest Reserve, with an area of 1,055km^2.

20.6.3 Baseline Environmental Setting

The climate of Ivory Coast is generally warm and humid, ranging from equatorial in the southern coasts to tropical in the middle and semiarid in the far north. There are three seasons: warm and dry (November to March), hot and dry (March to May), and hot and wet (June to October). Temperatures average between 25 and 32 °C (77.0 and 89.6 °F) and range from 10 to 40 °C (50 to 104 °F).

In the southern half of the country, rainfall is higher and the soils more productive, making it the center of production of most of the export crops, such as coffee and cacao. Palm, coconut trees, and rubber tree plantations also occur mostly in the southern and central parts of the country. In the northern half of Côte d'Ivoire, subsistence and cash crops such as cashew, cotton, sugar, starches, and rice have greatly increased, fragmenting large expanses of woodland and savannas.

The Project area occurs within the Guinean forest-savanna ecoregion of West Africa, a band of interlaced forest, savanna, and grassland running from western Senegal to eastern Nigeria and dividing the tropical moist forests near the coast from the West Sudanian savanna of the interior. Agricultural expansion is a key factor compromising forest cover. In general, farmers still use slash and burn techniques to clear land for agriculture. This practice is destructive, lays waste to large amounts of land, and undermines reforestation efforts.

Given the lack of industry within 10 km of the Project, there are no obvious sources of anthropogenic air emissions, noise and vibration. Intermittent use by road traffic, agricultural and forestry machinery are the most discernible sources of such. The nearest towns are Fadiadougou and Batogo. These towns have light industry supporting the communities, but no heavy industry. Road traffic generates dust which is captured by roadside vegetation. The majority (about 73%) of power in Côte d'Ivoire is from power stations fired by natural gas, with the remaining 27% sourced from hydropower. The Project has assessed the use of a hybrid system, using liquefied natural gas combined with solar as a power source.

Water samples have been collected from a number of exploration holes through the site and from the Marahoué River. Results have been compared to previous sampling to develop the database. While the wells have not been developed as monitoring wells, all the results indicate that water quality is good. From the samples collected, water is generally turbid, with elevated levels of dissolved iron, selenium and manganese, but these are aesthetic parameters rather than health related. Water from the well installed at Fadiadougou indicated that the water was of good quality.

Figure 20-3: Water Tanks at Fadiadougou

Source: Montage

As yet, no protected species have been identified within the Project footprint area. A flora and fauna inventory is scheduled for completion as part of the environmental impact study. The area is predominantly agricultural and comprises modified habitat. Trees have been cleared and habitat is fragmented (Figure 20-4,

Figure 20-5 and

Figure **20-6**). Monkeys and cattle have been observed in the early images captured by the passive cameras.

Figure 20-4: Panoramic of pit area, showing level terrain

Figure 20-5: Panoramic of area between TSF and WRD, in south of project

Figure 20-6: View from camera trap #2

20.6.4 Baseline Social Setting

The Koné project area is located relatively close to the communities of Batogo and Fadiadougou, but is sufficiently remote that environmental impacts on these communities are likely to be minor. Preliminary investigations indicate that the local community is overwhelmingly positive towards the company. Montage staff have engaged them regularly and the nearest communities have received specific support, such as beds for the clinic and construction materials for the school. The local community provides labour for construction. A water well has been provided in Fadiadougou, along with water tanks to ensure suitable supply.

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. Minimal physical resettlement is anticipated 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, but this will need to be verified through the EIES process. In the meantime, the Project has a chance finds procedure to mitigate against any impacts from their exploration activities. The Chance Finds Procedure defines a series of steps to minimize physical impacts to cultural heritage by providing a process for conducting archaeological look ahead-survey; monitoring of ground disturbing activities; and responding to any tangible cultural heritage encountered unexpectedly during exploration.

20.7 Potential Environmental Impacts

National environmental consultants will be used to gather data and produce a potential environmental and social impacts assessment, with support from international consultants as required. The consultant will commission and complete studies and work necessary to produce the ESIA in accordance with national regulatory provisions and IFC performance standards. They will also facilitate engagement with ANDE and other stakeholders relevant to the Project.

A request for ESIA terms of reference has been submitted to ANDE. Key components of these terms of reference are expected to include:

- Project description and context of the ESIA and the institutional framework of Côte D'Ivoire;
- Description of the baseline conditions of the project area;
- Identification of study area and area of influence of the Project;
- Description of the physical environment, such as climate, air quality, acoustic environment, geology, geomorphology, topography, pedology, hydrogeology, surface hydrology, etc;
- Description of the biological environment, such as fauna, floras, rare or endangered species, natural habitats and sensitive habitats, terrestrial and aquatic environment;
- Description of the social environment, such as administrative, socio-economic, land, cultural and archaeology, ecosystems services, involuntary displacement, nuisances and contributions of the mine to local development;
- Identification and assessment of impacts, and definition of mitigation measures;
- Specific chapters of the ESIA, such as:
  - Health, safety and emergency management;
  - Environmental and social management plan;
  - Conceptual framework for mine closure;
  - Resettlement policy framework; and
  - Public participation and consultation.

The Project is likely to give rise to a range of environmental and social impacts. However, these impacts are considered manageable and controllable through reasonable mitigation practices and therefore would enable effective environmental and social development, operation and closure of the Project

20.8 Environmental Management Plan

The Project is following Montage's environmental and social policies and has developed an Exploration Environmental and Social Management Programme (ESMP) to guide environmental and social management as well as stakeholder and community relations. The Project will aim to conform to the environmental and social requirements of the IFC International Finance Corporation Performance Standards, its associated Environmental Health and Safety guidelines, International Council of Metals and Mining and Equator Principles where they are relevant to the Project.

An Environmental Management Plan (EMP) has been developed for exploration work and this is designed to be developed through the life of the Project. The key priorities of this management are:

- Protect the health of workers, the public, flora and fauna,
- Manage all waste generated by exploration operations in a responsible manner, and
- Minimise emissions generated by exploration, particularly dust.

The Exploration Management Plan being implemented on site will continue to be developed to inform the Koné Project ESMP. The purpose of the ESMP is to ensure that appropriate control and monitoring measures are in place to deal with all significant impacts of the Project. The ESMP has been designed so that it can be regularly reviewed and updated according to company policies. The ESMP includes details of the area of impact, objectives to reduce negative or enhance positive impacts, specific targets adopted to achieve those objectives and definition of responsibilities for implementing the programme. It is a live document that can be reviewed and updated on a systematic basis, in-line with the principles of continual improvement.

Records are being maintained during exploration to monitor all activities and engagement. This includes interaction with local communities, observations of wildlife and environmental conditions and location of boreholes, including those to be abandoned. Procedures for monitoring baseline data have been developed. All exploration programs will be under the control and responsibility of a designated qualified representative of the company and audited to ensure that requirements are met.

20.9 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.

20.10 Monitoring

The Project has initiated an environmental and socially related baseline data collection programme to determine the current conditions of the potential exploitation area. Initial data collection will include collection and analysis of surface water and groundwater quality, installation of weather recording and air quality instrumentation, recording wildlife type and movements, and identification of important environmental and cultural sites in the Project area.

Montage has developed and implemented an environmental and social monitoring plan, including appropriate sampling procedures. Currently, baseline monitoring of weather data, water sampling and ecology are underway. Passive infrared camera traps are being used to capture wildlife, as the density of vegetation and abundance of both water and food, combined with the presence of farmers, means that wildlife is shy and difficult to count.

Groundwater levels are recorded on a quarterly basis, and water quality samples are to be collected and sent for analysis on a six-monthly basis. Interactions with local communities are recorded in a daily diary, along with wildlife observations and any other items of environmental interest. The Marahoué river water quality is being monitored monthly, with river levels monitored using dataloggers and manual verification.

The baseline assessment studies will be used to develop a more detailed environmental and social assessment and determine any additional further monitoring requirements and planning. The objective of the studies is to identify receptors of potential impacts that the Project may have on the surrounding environments (biophysical and social) and which should be examined and assessed in more detail as the Project develops. The EIES is a multi-disciplinary and iterative process, and these baseline studies provide the first stage of this process. Monitoring programmes will continue to inform important activities through the life of the Project to observe any changes in the environment.

20.11 Public Consultation

There are currently no objections to the development of the Project. As part of the environmental assessment, public consultation and disclosure is required. In order to ensure that the Project is developed and operated in an appropriate manner, Montage Gold will incorporate the concept that effective engagement with its stakeholders is an essential component of the assessment process and its ongoing "licence to operate". Montage is committed to a proactive program of communications with all relevant stakeholders.

The Project has few stakeholders, with the closest people being the towns of Fadiadougou and Batogo. Meetings have been held with the leaders of these communities already (

Figure **20-7** and

Source: Montage

Figure 20-8), and ongoing meetings are planned. A record of all meetings is being maintained, summarizing the numbers of people engaged with, their activities and any issues or concerns they may have with the Project.

Figure 20-7: Meeting with Fadiadougou chief and elders

Source: Montage

Figure 20-8: Meeting with Batogo chief and elders

Source: Montage

20.12 Mine Closure

Closure and rehabilitation of the mine site will commence once mining from is complete and a detailed Closure Plan will be developed and finalized prior to closure to guide these activities. Progressive reclamation will be carried out during normal mine operations where circumstances allow.

Post-closure management and maintenance objectives will be to ensure that the site achieves a sustainable and maintenance-free status. The proposed overall strategy for the decommissioning and closure of the Project is as follows:

- Decontaminate, dismantle and demolish, as far as practicable, all installations, structures and infrastructure not identified for retention and hand over to another entity;
- Safe disposal of all contaminated materials removed during decontamination, dismantling and demolition activities;
- Salvage for sale and/or allocation to other operations, all equipment, mechanical and electrical plant, identified in the asset register as having a residual value or useful life;
- Removal from the site as scrap (if economically viable) or dispose as solid waste of all equipment, plant and structures not deemed suitable for future refurbishment and/or re-use;
- Apply closure design options which are effective, practical and cost effective;
- Ensure the site is left in a safe condition;
- Where practical, undertake phased closure of the facilities making allowance in the implementation timeframe for retention of facilities required to support the closure process and subsequent post closure monitoring activities;
- Address any potential residual environmental impacts, where appropriate, resulting from Project activities; and
- Minimise residual impacts requiring on-going monitoring post closure of the facility.

Closure activities commence during the construction period with pre-stripping of topsoil and dumping onto topsoil stockpiles. Revegetation of the area is planned with predominantly indigenous species, establishing an on-site nursery and seed harvesting of local species. The waste rock and tailings are considered benign and non-hazardous; no acid rock drainage (ARD) or metal leaching is expected during operations or post-closure.

Initial closure activities will focus on the rehabilitation of the waste rock dumps (WRD) and tailings storage facility (TSF). Benches on the WRD will be cut and filled to produce a landform in keeping with the surrounding landscape. Where possible, the maximum slope angles of the WRD will be approximately 20°. Topsoil will be placed over the WRD to a depth of about 0.15 m. The combination of the shallow slope, compaction of the waste rock and revegetation will minimize the infiltration of precipitation into the WRD, and maximise water runoff and evapotranspiration.

Closure of the TSF will commence with the placement of approximately 0.5m of waste rock over the surface. Topsoiling and revegetation will follow the same methodology as the WRD. The placement and levelling of the waste rock will promote water runoff and minimize ponding.

All buildings and structures will be removed including the process plant, conveyors, workshops, offices and other ancillary structures. The building structures will be dismantled, and the materials removed from the site for sale, reuse, recycling, or disposal at a registered waste site. All oil, fuels, and processing chemicals will be drained from the equipment and disposed of at a licensed off-site disposal facility. The processing equipment and conveyor structures will be removed from site and sold or recycled. All the disturbed areas will be ripped or ploughed (to increase water infiltration and reduce the potential for surface erosion and instability), levelled and covered with about 0.15 m of topsoil (except the concrete structures). Revegetation will be as for the WRD. Concrete foundations will remain in situ and covered with about 0.40 m of topsoil, either from stockpiles or imported as necessary. The tailings and water supply pipelines will be removed and disposed of off-site. Any roads that will not be required for post-closure management will be decommissioned.

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.

21. CAPITAL AND OPERATING COSTS

21.1 Introduction

The overall study capital cost estimate was compiled by Lycopodium and is presented here in summary format. The various elements of the Project estimate have been subject to internal peer review by Lycopodium and have been reviewed with Montage for scope and accuracy.

The capital cost estimate was developed to an accuracy level range of +20/-15% to cover engineering, procurement, construction, and start-up of the mine and processing facilities, as well as the ongoing sustaining capital costs. The capital cost estimates were developed for a conventional open pit mine, CIP process plant and supporting infrastructure for an operation capable of treating 11.0 Mtpa of material. For the purpose of this PEA, provision of the power plant, the operations camp and the onsite laboratory via a third-party Build Own Operate Transfer arrangement together with a contract mining scenario have been assumed.

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 allowance for these costs has been made in the financial model.

Indirect costs associated with the design, construction and commissioning of the new facilities, owner's costs, and contingencies have also been estimated, based on percentages of the direct capital cost estimate. Risk amounts are specifically excluded from this estimate. A breakdown of the capital cost estimates is shown in Table 21.1.

All costs are estimated in United States dollars (US$) as at 2Q21.

21.2 Capital Cost Summary

The capital estimate is summarized in Table 21.1 and 21.2. The initial project capital cost is estimated at US$487.7M, including a contingency allowance of US$64.8M.

Table 21-1 Capital Estimate Summary (2Q21, ±20/10%)

Main Area	US$M
Mine	32.0
Treatment Plant	263.3
Power	2.5
TSF	53.7
Camp	1.5
EPCM	40.2
Owners Costs	31.6
Subtotal	**424.8**
Contingency	65.1
Grand Total	**489.9**

The total LOM cost is estimated at US$936.7M including sustaining capital costs of US$444.9M, as shown in Table 21.2. The LNG power plant and the camp will be financed under a Build Own Operate Transfer (BOOT) contract. The duration of the contract will be 10 and 5 years respectively.

Table 21-2 Sustaining Capital Estimate Summary (2Q21, ±20/-10%)

Main Area	US$M
Camp	4.5
TSF	205.8
Power	140.4
Process Plant	27.5
Closure	66.6
Grand Total	**444.9**

21.3 Direct Capital Costs – Mining

Due to the use of mining contractors, who will provide the mining fleet, the capital costs only include a single lump sum value of $0.4 million has been included in Year 1 to cover the Principal mine office.

21.4 Capital Cost Estimate – Process Plant and Infrastructure

To develop the process plant and infrastructure cost estimate a mechanical equipment list was compiled, based on the process requirements, and major equipment pricings were obtained from technology / equipment suppliers or from a database of similar size projects and factored as required for the project capacity.

Project infrastructure includes mine infrastructure as itemised in Section 18.6.2.

The EPCM estimate was factored based upon Lycopodium's recent experience with similar type and size of projects. Expenses such as catering and accommodation for the Engineer's site personnel, as well as site telecommunications costs, are included in the estimate.

A contingency allowance is included to make specific provision for uncertain elements of cost within the project scope. Contingencies do not include allowances for scope changes, escalation, or exchange rate fluctuations. Contingency has been applied to all parts of the process plant estimate.

21.5 Operating Costs – Mining

As the bulk of the mining costs would be related to a mining contract, the mine operating costs were derived from several existing mining contracts using similar equipment awarded in West Africa and in some instances were validated against equipment purchase and parts supply studies conducted in 2020 in Europe and the CIS.

Fixed mining costs were calculated for the assumed fleet and dependant on material type (Crusher Feed/Waste and Oxide + Transitional/Fresh) and destination. They included components for:

- Loading costs
- Fixed hauling costs
- Drill & Blast costs
- Ancillary costs
- Mine admin costs

Fixed mining costs do not include any time haul trucks spend travelling up or down in-pit ramps.

Table 21-3 Fixed Mining Costs

Ore/Waste	Feed	Feed	Feed	Waste	Waste	Waste
Material	Oxide	Transitional	Fresh	Oxide	Transitional	Fresh
Loading	0.32	0.34	0.36	0.32	0.34	0.36
Fixed Hauling	0.34	0.34	0.34	0.36	0.36	0.36
Drill & Blast	0.89	0.89	1.00	0.74	0.74	0.91
Ancillary	0.50	0.50	0.50	0.50	0.50	0.50
Mine Admin	0.14	0.14	0.14	0.14	0.14	0.14
Grade Control	0.26	0.26	0.26	0.00	0.00	0.00
Total Fixed Cost	2.45	2.47	2.60	2.06	2.08	2.28

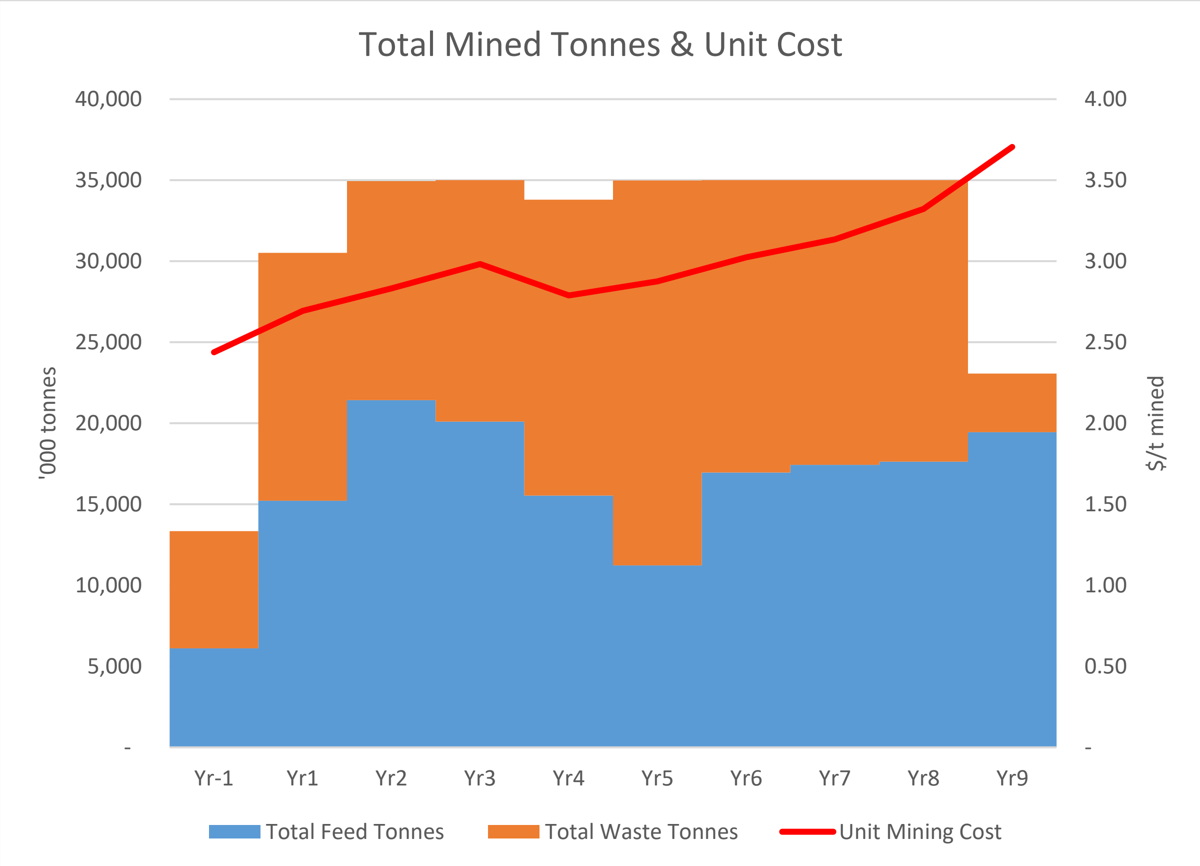
Incremental mining costs were determined for the fleet and included a fuel and non-fuel component. The non-fuel component covered costs such as operator salary, maintenance costs and other running costs associated with the time spent on ramps. The Incremental Mining Cost was determined to be $0.027/t/10m vertical lift based on a reference RL of 375m.

The fuel price used for mining optimisation calculations was $0.65/litre.

Rehandle costs were not included in the optimisation as it was anticipated that the amount of rehandle would be quite low. Rehandle was costed at $0.87/tonne in the financial model.

Figure 21-1 shows the unit mining cost for the life of the operation compared to the total tonnes mined in each period. The unit costs increase gradually as the depth of the pit increases.

Figure 21-1 Unit Mining Cost



21.6 Operating Cost – Plant and Infrastructure

The Operating Cost Estimate (OPEX) for the plant and infrastructure has been divided into multiple cost centres, with Fixed and Variable costs calculated for each cost centre for each different material type. The operating cost estimate is presented in Table 21.3 and is deemed to have an accuracy of +20/10% based on pricing as at 2Q21. The process operating cost includes all direct costs to produce gold bullion for the Project.

In general, costs have been built up from first principle estimates, with quotations obtained for major reagents and consumables and consumption rates based on metallurgical testwork, calculations or modelling. Minor reagents, laboratory, expatriate labour rates and a number of G&A costs have been sourced from the Lycopodium database. Power consumption has been calculated from the gross power required to achieve the desired grind size on each material type, based on OMC comminution modelling, plus the remaining installed power from the mechanical equipment list, with suitable drive efficiency and utilization applied and factored for the design throughput. The total power draw was used to calculate power costs based on a IPP power price of US$0.076/kWhr.

Table 21-4 Operating Cost per Material Type

COST CENTRE	**Total**			**LOM**		
	Oxide	**Transition**	**HW Fresh**	Fix	Variable	
	US$/t	US$/t	US$/t	US$'000/yr	US$/t	% Cost
Power (exc crush & grind)	0.80	0.80	0.80	8,787	0.80	10%
Power (crushing/ore storage)	0.12	0.12	0.12	1,289	0.12	2%
Power (grinding)	0.60	0.72	1.32	13,517	1.23	16%
Operating Consumables	3.16	2.62	3.61	38,737	3.52	46%
Maintenance	0.65	0.65	0.65	7,102	0.65	8%
Total Processing (Variable)	5.32	4.91	6.49	69,432	6.31	82%
Laboratory	0.14	0.14	0.14	1,499	0.14	2%
Process & Maintenance Labour	0.36	0.36	0.36	3,907	0.36	5%
Administration Labour	0.34	0.34	0.34	3,794	0.34	5%
General & Administration Costs	0.51	0.51	0.51	5,590	0.51	7%
Total G&A, Labour, Lab (Fix)	1.34	1.34	1.34	14,790	1.34	18%
TOTAL	**6.67**	**6.25**	**7.83**	**84,222**	**7.66**	**100%**

21.7 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.
- Subsidies to local communities.
- Licence fees.
- Royalties.

- 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.
- Gold refining costs.
- Mercury treatment / stabilisation costs after closure.
- Bullion transport costs.
- Bullion marketing costs.
- Bullion insurance in transit costs.
- Tailings storage costs, including future lifts and rehabilitation, which are included in sustaining capital.
- Tailings dust suppression costs.
- Any rehabilitation or closure costs

22. ECONOMIC ANALYSIS

22.1 Introduction

The economic analysis is based on Inferred Resources and mine schedule as per Table 22.4.

An economic analysis has been carried out for the project using a cash flow model. The model is constructed using annual cash flows by taking into account annual mined and processed tonnages and grades for the CIP feed, process recoveries, metal prices, operating costs and refining charges, royalties and capital expenditures (both initial and sustaining).

The financial assessment of the project is carried out on a “100% equity” basis and the debt and equity sources of capital funds are ignored. No provision has been made for the effects of inflation. Current Côte d’Ivoire tax regulations are applied to assess the tax liabilities. All amounts in this section are presented in US$. Discounting has been applied from the first year of operation.

The model reflects the base case and technical assumptions as described in the foregoing sections of this report.

22.2 Model Inputs and Assumptions

The model inputs and assumptions used in the economic analysis are summarized in Table 22.1 and unless otherwise stated, is used in the model.

Table 22-1 Model Inputs and Assumptions

Model Inputs	Unit / Value
Base Currency	US$
Base Date	2nd Quarter 2021
Côte d’Ivoire Royalty @$1,600/oz (charged against Revenue)	4.0%
Maverix Royalty (charged against Revenue)	2.0%
Community Royalty (charged against Revenue)	0.5%
Côte d’Ivoire Tax Rate	25%
NPV Discount Rate	5%
Metal Price – Fixed for LOM	US$1,600/oz
Refining Payability	99.5%
Refinery Charges & Shipping	US$4/oz
Assumptions	
Capex excludes Finance Charges & Fees	
Capex excludes Pre-production Investigations	
Capex Amortisation/Depreciation based on no salvage value	
Capex excludes Escalation	
Tax paid on an Annual mid year basis	

22.2.1 Capital Costs

Pre-production capital expenditures are defined in Table 22.2. Sustaining capital for the Plant, Mining and TSF expansion costs have been phased over the life of the project and detailed in Table 22.3.

Table 22-2 Pre-production Capital Expenditure

Item	Unit	Total	Year		
			-3	-2	-1
Mine	US$'000	32,036			
Process Plant	US$'000	262,043			
Power	US$'000	2,500			
TSF	US$'000	53,159			
Camp	US$'000	1,533			
EPCM	US$'000	40,041			
Owner	US$'000	31,617			
Construction Sub Total	**US$'000**	**422,928**			
Contingency	US$'000	64,799			
Construction Total	**US$'000**	**487,728**	**3,807**	**185,252**	**298,668**

Table 22-3 Sustaining Capital Expenditure

Item	Unit	LOM
Camp	US$'000	4,539
TSF	US$'000	206083
Power	US$'000	140,420
Plant	US$'000	27,500
Mine Closure	US$'000	66,596
Sustaining Total	US$'000	**445,139**

22.2.2 Revenue

Revenue has been calculated allowing for 0.5% refinery loss.

22.2.3 Royalties

Royalties at 6.5 % have been included for the LOM and will be charged against the revenue.

22.2.4 Cost of Sales

Cost of Sales includes freight and refining costs. A value of US$4.00 /oz gold recovered has been allowed for in the model.

22.2.5 Depreciation

Depreciation is calculated using the units of production method starting with first year of production and can be summarized as follows:

- Initial pre-production capex depreciated over the total LOM, based on units of production.
- Capitalised pre-production costs (i.e. cumulative exploration and PEA costs) to date depreciated over the total LOM, using estimated total capitalised pre-production costs of US$26.5M to December 30, 2020.
- The annual sustaining capital is assumed to be largely repairs and maintenance

Remaining sustaining capital items (i.e. TSF, Generators, Camp) each depreciated separately based on respective remaining LOM.

22.2.6 Inflation

Inflation has not been included in the cash flow analysis.

22.2.7 Operating Costs

Annual fixed and variable costs, as per Sections 21.5 and 21.6, are included in the cash flow.

22.2.8 Financial Model

Figure 22.1 shows the pre-tax and post –tax cumulative cash flow for the project over the LOM; the payback period corresponds to when the cumulative cash becomes positive for the pre-tax and the post-tax model. Figure 22.2 shows the annual and cumulative post-tax cash flow.

The pre-tax and post-tax financial results of the project are summaries in Table 22.4. On a pre-tax basis, the project has a Net Present Value (NPV) of US$54M at a discount rate of 5%, an Internal Rate of Return (IRR) of 8.4%; on a post-tax basis the NPV is US$21M at a discount rate of 5%, the IRR is 6.3% and the payback period is 7.0 years following commencement of production.

Figure 22-1 Cumulative Cash Flow

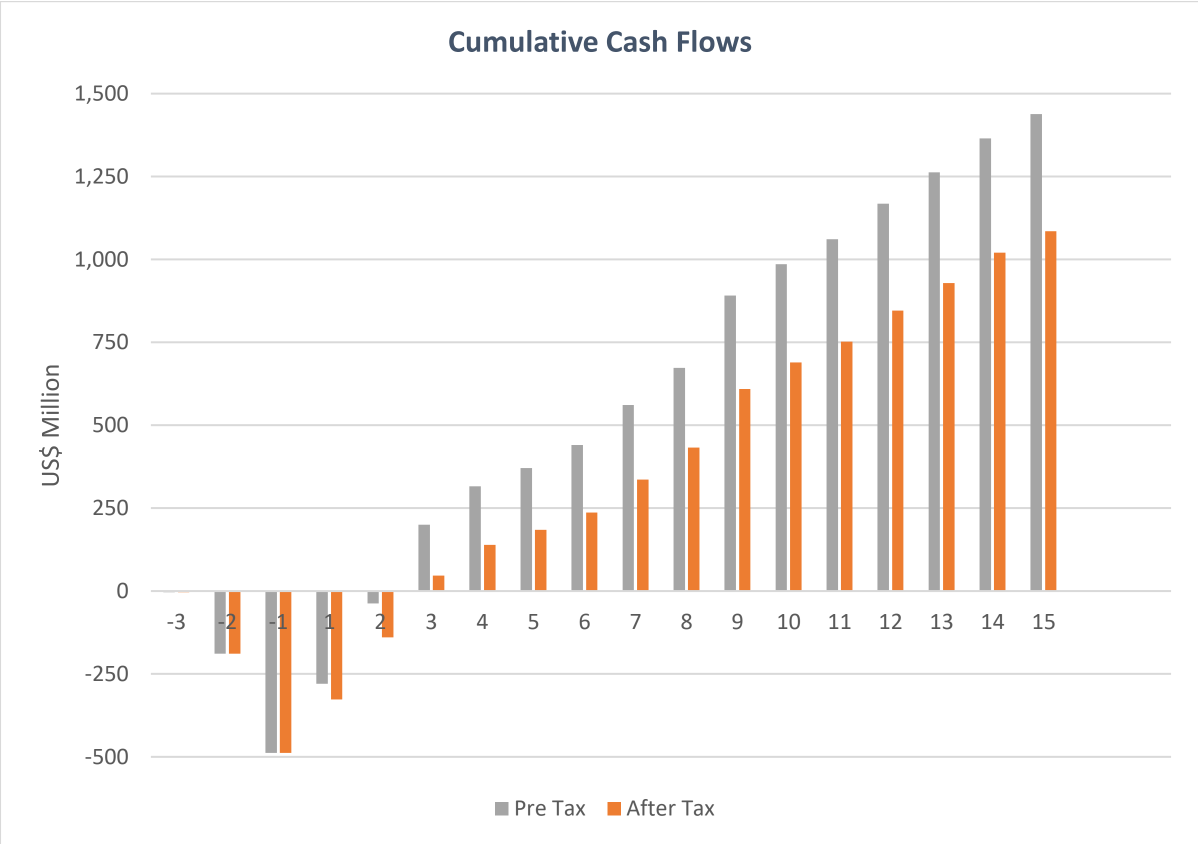


Table 22-4 Mine and Process Schedule

Kone - PFS Deswik vs 5f.xlsx

	Description	Unit	LOM Total	Yr-3	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15
Mining																					
	North Pit Tonnes	Mt	7.7							1.0	1.4	1.5	1.3	2.5	0.1						
	North Pit Grade	Au g/t	0.50							0.40	0.44	0.47	0.49	0.59	0.75						
	South Pit Tonnes	Mt	153.3			6.1	15.2	21.4	20.1	14.5	9.8	15.5	16.2	15.1	19.4						
	South Pit Grade	Au g/t	0.66			0.70	0.70	0.70	0.73	0.58	0.56	0.59	0.62	0.61	0.72						
	Total Tonnes	Mt	161.1			6.1	15.2	21.4	20.1	15.5	11.2	17.0	17.4	17.6	19.4						
	Total Grade	Au g/t	0.66			0.70	0.70	0.70	0.73	0.58	0.56	0.59	0.62	0.61	0.72						
	Total Waste Tonnes	Mt	149.6			7.2	15.3	13.5	14.9	18.3	23.7	18.0	17.6	17.4	3.6						
	Strip Ratio	W:O	0.93			1.18	1.01	0.63	0.74	1.18	2.11	1.06	1.01	0.99	0.19						
Processing																					
	Stockpile Rehandle	Mt	73.9				1.7	0.8	0.5	0.8	5.4	0.3	0.2	0.4	0.4	11.0	11.0	11.0	11.0	11.0	8.2
	Oxide Tonnes	Mt	12.4				1.5	1.1	0.5	1.0	0.1	0.3	0.1	0.4	0.4	0.4	0.4	1.6	1.7	1.7	1.2
	Oxide Grade	Au g/t	0.56				0.98	0.97	0.97	0.61	0.73	0.46	0.43	0.40	0.40	0.40	0.40	0.39	0.39	0.39	0.39
	Transition Tonnes	Mt	8.6				2.6	0.1		0.6	0.2		0.1			1.0					4.1
	Transition Grade	Au g/t	0.57				0.87	1.00		0.70	0.60		0.72			0.41					0.37
	Fresh Tonnes	Mt	140.0																		
	Fresh Grade	Au g/t	0.66																		
	Total Porcessed Tonnes	Mt	161.1				9.9	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	8.2
	Total Processed Grade	Au g/t	0.65				0.91	0.96	0.96	0.68	0.59	0.66	0.73	0.70	0.92	0.45	0.45	0.44	0.44	0.43	0.39
	Total Process Recoveries	%	89.4%				91.6%	91.1%	90.8%	89.7%	88.3%	89.1%	89.5%	89.4%	90.6%	86.9%	86.6%	87.1%	87.1%	87.0%	88.8%
	Total Recovered	000 ozs	3,012				265	308	308	217	184	209	231	220	296	138	139	137	136	133	91

22-5 Financial Model

Kone - PFS Deswik vs 5f.xlsx

Description	Unit	LOM Total	Yr-3	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16	Yr17
Pre-Production Capex	$M	(489.9)	(3.8)	(185.9)	(300.2)																	
Sustaining Capex	$M	(444.9)				(30.4)	(33.9)	(33.1)	(30.5)	(30.2)	(52.6)	(31.5)	(16.6)	(51.4)	(16.6)	(35.3)	(2.5)	(13.8)	(0.0)		(32.7)	(33.9)
Revenue	$M	4,794.5				422.5	490.1	490.9	345.3	292.5	333.2	368.2	350.1	470.7	219.5	220.6	217.7	216.7	211.0	145.6		
Selling Costs	$M	(12.0)				(1.1)	(1.2)	(1.2)	(0.9)	(0.7)	(0.8)	(0.9)	(0.9)	(1.2)	(0.6)	(0.6)	(0.5)	(0.5)	(0.5)	(0.4)		
Royalties	$M	(311.6)				(27.5)	(31.9)	(31.9)	(22.4)	(19.0)	(21.7)	(23.9)	(22.8)	(30.6)	(14.3)	(14.3)	(14.1)	(14.1)	(13.7)	(9.5)		
Op Cost Mining	$M	(868.9)			(31.6)	(79.9)	(96.4)	(101.5)	(91.1)	(96.8)	(102.1)	(105.9)	(111.8)	(83.3)								
Op Cost Process Fixed	$M	(172.6)				(11.7)	(11.7)	(11.7)	(11.7)	(11.7)	(11.7)	(11.7)	(11.7)	(11.7)	(11.7)	(11.7)	(11.7)	(11.7)	(11.7)	(8.7)		
Op Cost Process Variable	$M	(986.7)				(54.1)	(64.2)	(64.8)	(63.6)	(69.3)	(64.8)	(64.8)	(64.8)	(64.8)	(72.5)	(74.0)	(72.7)	(72.6)	(72.6)	(47.4)		
G&A	$M	(138.3)				(9.4)	(9.4)	(9.4)	(9.4)	(9.4)	(9.4)	(9.4)	(9.4)	(9.4)	(9.4)	(9.4)	(9.4)	(9.4)	(9.4)	(7.0)		
Operating Profit	$M	2,304.3				238.9	275.3	270.3	146.3	85.6	122.7	151.6	128.7	269.8	111.1	110.6	109.2	108.4	103.1	72.7		
Net Cash Flow before tax	$M	1,369.4				208.4	241.4	237.3	115.8	55.4	70.1	120.1	112.1	218.4	94.6	75.3	106.7	94.6	103.1	72.7	(32.7)	(33.9)
NPV Pre Tax	$M	928.7																				
IRR Pre Tax	$M	45.1%																				
Depreciation	$M	(867.2)				(48.2)	(59.7)	(63.7)	(47.6)	(43.0)	(55.1)	(65.2)	(64.4)	(100.2)	(49.2)	(56.6)	(55.9)	(59.9)	(58.3)	(40.2)		
Other Sustaining / Closure	$M	(94.1)						(2.5)	(2.5)	(2.5)	(2.5)	(2.5)	(2.5)	(2.5)	(2.5)	(2.5)	(2.5)	(2.5)			(32.7)	(33.9)
Taxable Profit	$M	1,342.9				190.7	215.6	204.2	96.2	40.1	65.1	83.8	61.8	167.1	59.4	51.5	50.8	46.0	44.8	32.4	(32.7)	(33.9)
Tax	$M	(352.4)				(47.7)	(53.9)	(51.0)	(24.0)	(10.0)	(16.3)	(21.0)	(15.4)	(41.8)	(14.9)	(12.9)	(12.7)	(11.5)	(11.2)	(8.1)		
Net Cash Flow after tax	$M	1,017.1				160.8	187.5	186.2	91.8	45.4	53.8	99.1	96.7	176.6	79.7	62.4	94.0	83.1	91.9	64.6	(32.7)	(33.9)
NPV Post Tax	$M	652.2																				
IRR After Tax		30.9%																				
Cash Cost	$/payable	827				697	715	918	1,128	1,007	937	1,008	679	786	794	793	795	814	797			
AISC	$/payable	975				830	845	1,082	1,315	1,282	1,096	1,106	876	929	1,072	834	920	837	819			
Payback Period	yrs	2.8																				

22.2.9 Financial Summary

The results of the financial model are summarized in Table 22.5.

Revenue generated per domain is shown in Figure 22.2.

A breakdown of the total cash costs is shown in Figure 22.3. Table 22.6 shows the breakdown of the LOM cash costs and unit costs per tonne processed.

Table 22-6 Financial Model Summary @ $1,600

Description	Units	LOM
Feed Tonnage	Mt	161.1
Waste Rock	Mt	149.6
Total Mined	Mt	310.6
Strip Ratio	W:O	0.93:1
Feed Grade Processed (average)	g/t	0.66
Gold Recovery (average)	%	89.4
Gold Production	'000 oz	3,012
Annual Gold Production (average)	'000 oz/y	205
Pre-production Capital Cost	US$M	(490)
Sustaining Capital Cost	US$M	445
Total Capital Cost	US$M	(935)
Net Revenue	US$M	1,498
Royalties	US$M	(312)
Total Operating Costs	US$M	(2,167)
EBITDA	US$M	2,304
Tax	US$M	(352)
Net Cash Flow After Tax	US$M	1,017
$NPV_{5\%}$ After Tax	US$M	652
Cash Cost	US$ /pay oz	827
AISC	US$ /pay oz	975

Figure 22-2 Revenue Generated per Material Type

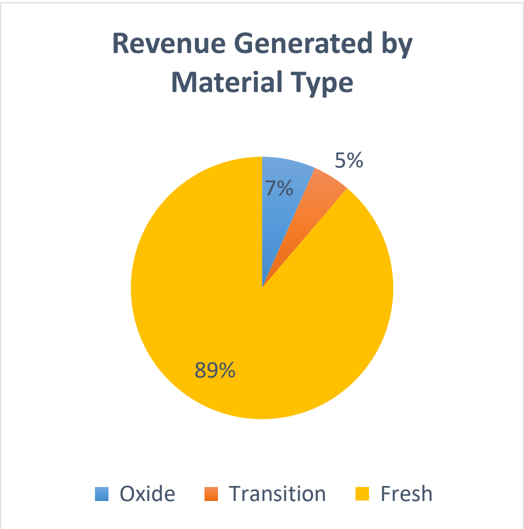


Figure 22-3 Operating Expense Split

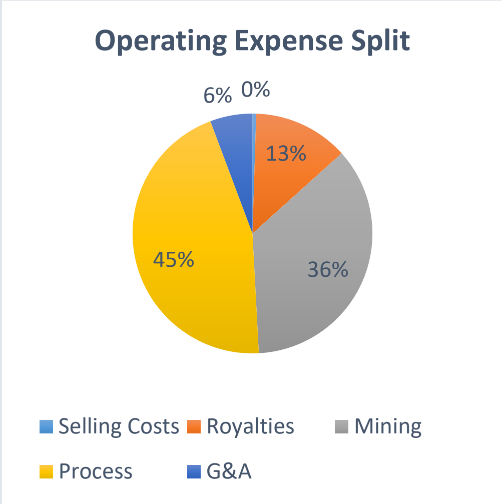


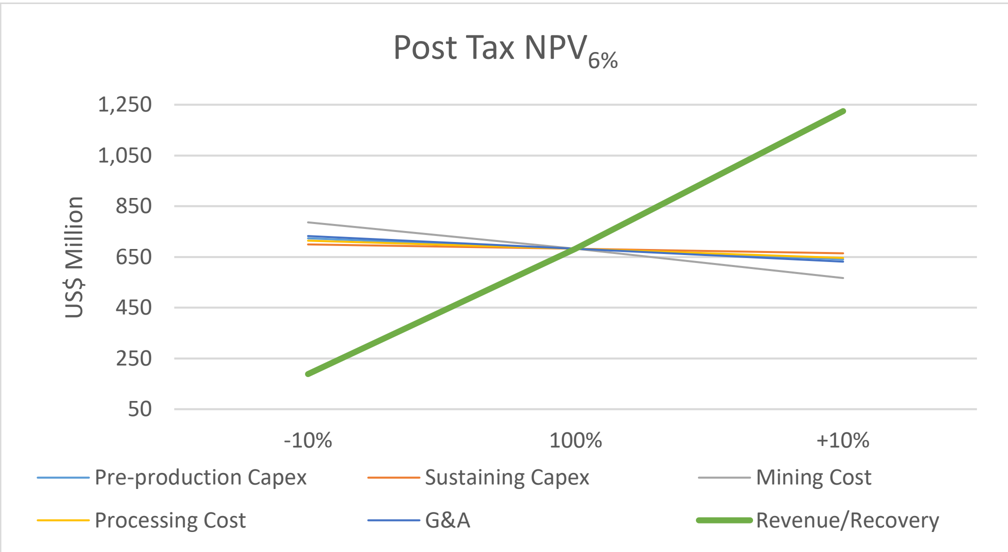
Table 22-7 Cash Cost and Unit Cost Summary

Description	LOM ($/oz)	LOM ($/t processed)
Mining	290	5.39
Processing	387	7.20
G&A	46	0,86
Royalties	104	1.93
Total Cash Cost	827	15.39
Sustaining Capital	126	2.35
Closure	22	0.41
All-in Sustaining Costs	975	18.15

22.2.10 Single Parameter Sensitivities

Figure 22.4 shows the changing post-tax $NPV_{7\%}$ and IRR for varying single parameter sensitivities for revenue, pre-production and sustaining capital costs, mining, plant and G&A operating costs and revenue / gold recovery. Figure 22.4 also shows the post-tax IRR sensitivity to parameters that the NPV is most sensitive revenue / recovery

Figure 22-4 NPV and IRR Sensitivity



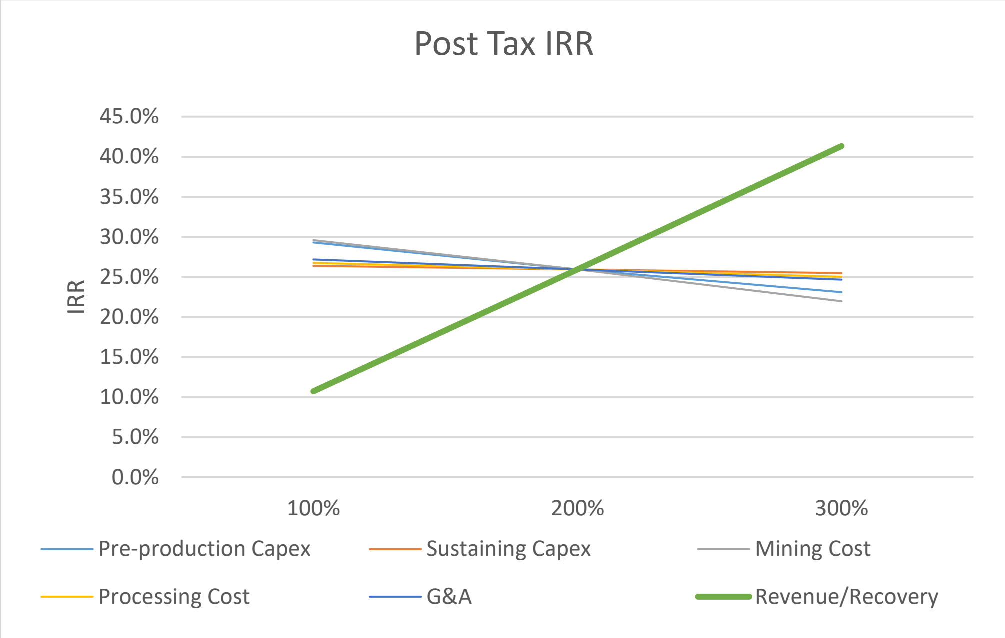


Table 22.8 shows the sensitivity of the NPV and IRR with gold price and discount rate.

Table 22-8 NPV and IRR Sensitivity

Gold Price	1,400	1,500	1,540*	**1,600**	1,700	1,850	2,000
NPV $_{5\%}$	337	495	558	**652**	781	1015	1249
IRR	18.5%	24.7%	27.2%	**30.9%**	36.1%	45.8%	55.9%
Cash Cost	814	821	823	**827**	851	862	873
AISC	962	969	972	**975**	999	1010	1021
Payback	4.9	3.2	3.0	**2.8**	2.5	2.2	2.0

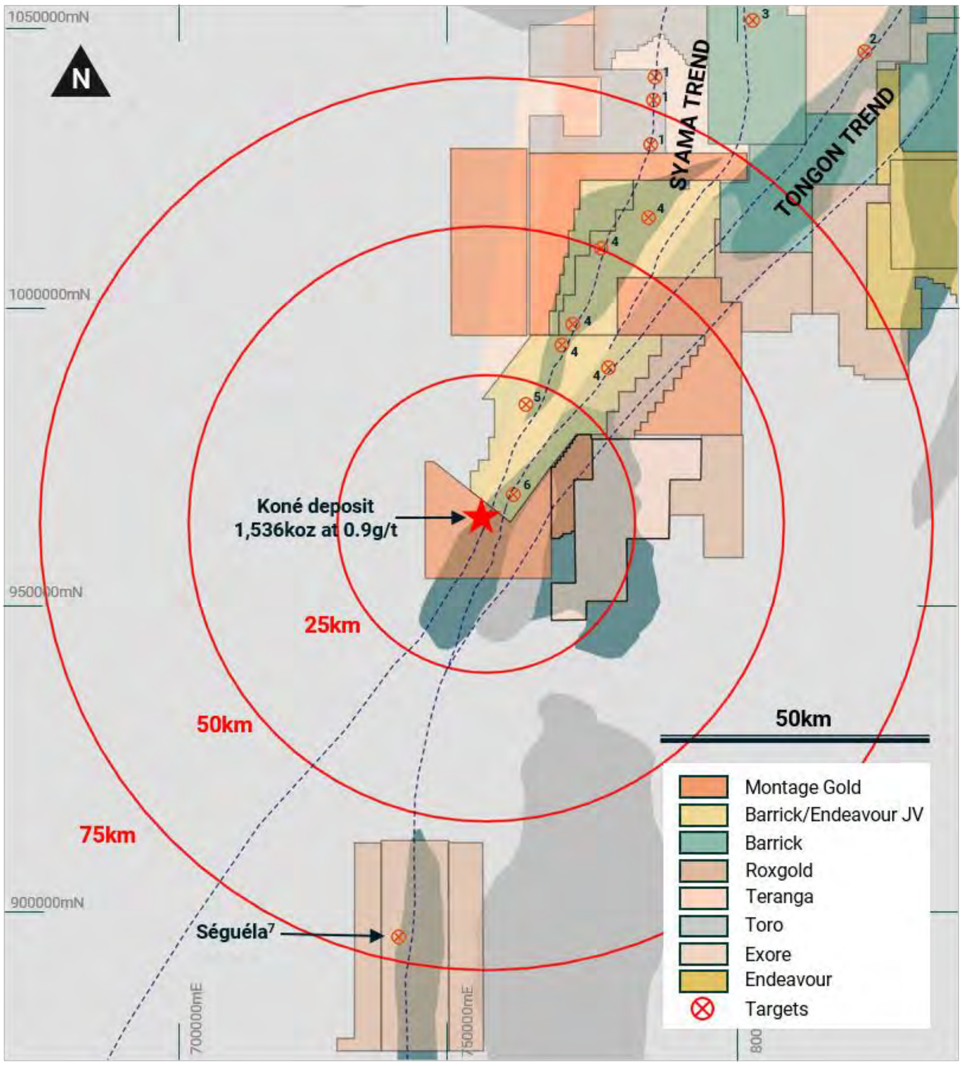
*** Three-year trailing average**

A diesel price of $0.75/l was used.

23. ADJACENT PROPERTIES

Figure 23-1 shows tenements held by other owners in the region of the Koné Gold Project. This figure is derived from the Côte d'Ivoire Ministry of mines and geology's mining cadastral (Côte d'Ivoire Ministry of mines, 2020). Immediately to the north of the Koné Exploration Permit lies the Mankono Joint Venture held by Barrick Gold and Endeavour Mining. To the east of the Koné Exploration Permit is the Dianra Exploration Permit that is held by Teranga Gold Corp.

Figure 23-1. Adjacent properties



Source: Montage August 2020

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24. OTHER RELEVANT DATA AND INFORMATION

There is no additional information or explanation required in order to make this report understandable and not misleading; all relevant information has been summarized in the Feasibility Report and Appendices.

25. INTERPRETATION AND CONCLUSIONS

25.1 Geological setting and assessment status

The Koné Exploration Permit lies within the Birimian Baoulé-Mossi domain which locally comprises metamorphosed sediments, volcaniclastics and volcanics flanked to the west by basement tonalite and diorite units. Local stratigraphy comprises a moderately dipping sequence of mafic volcanics, which is intruded by a 250 m wide package of diorite-quartz and diorite-monzonite Gold mineralization generally occurs within a wide zone of variably sheared and foliated intrusive units and is associated with finely disseminated pyrite and biotite alteration.

The Inferred Mineral Resource Estimate for the Koné deposit is based on 40,700m of drilling (25,545m of RC and 15,155m of core). The deposit has been tested by 100 m spaced traverses of generally 50 and rarely 25 m spaced holes with drilling on each traverse extending to vertical depths of between 60 m and 490 m.

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

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

The author considers that quality control measures adopted for sampling and assaying have established that the field sub-sampling, and assaying is representative and free of any biases or other factors that may materially impact the reliability of the sampling and analytical results. The author considers that the sample preparation, security and analytical procedures adopted for the Koné drilling provide an adequate basis for the Mineral Resource estimates.

25.2 Data verification and Mineral Resource estimation

Table 26.1 shows the Mineral Resource estimates at 0.4 g/t cut off subdivided by oxidation type. The figures in this tables are rounded to reflect the precision of the estimates and include rounding errors.

Table 25-1 Mineral Resource Estimates at 0.7 g/t cut off

Oxidation Zone	Mt	Au g/t	Au moz
Oxidized	7.0	0.81	0.18
Transition	4.7	0.80	0.12
Fresh	112	0.80	2.88
Total	**123**	**0.80**	**3.16**

There do not appear to be any other factors (including environmental, permitting, legal, title, taxation, socio economic, marketing or political) which could materially affect the exploration potential of the project as presented in this report.

25.3 Mineral Processing and Metallurgical Testing

A comprehensive testwork programme was carried out on 43 comminution and 39 leach optimisation and variability samples

Table 25-2 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 Comminution Testwork

Oxidation Zone	No Samples	Average 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

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

Forecast gold recoveries were estimated based on predicted residue grades for average feed grades, solution loss of 0.01g/t and carbon fines loss of 0.15%. Table 25-3 estimates the gold recoveries based on the average deposit grades, which are good considering the low head grades due to consistently low tailings residues grades being observed. Cyanide consumptions are all low to very low and lime consumptions are low for the predominant fresh zone (88%), but higher for the less dominant transition (5%) and oxide (7%) zones.

Table 25-3 Metallurgical Testwork Summary

Oxidation Zone	Average LOM Grade Au g/t	Forecast Recovery % Au	NaCN Consumption kg/t	Lime Consumption kg/t
Fresh	0.67	89.1	0.18	0.22
Transition	0.57	91.1	0.07	1.45
Oxide	0.63	94.8	0.15	1.99

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

25.4 Mining

Using available geotechnical information and a series of pit optimisations and mining schedules, the study has shown that the project can support a 5mtpa processing plant for a little over 9 years. The estimated crusher feed under this scenario is 43.0 million tonnes at a gold grade of 0.91 g/t. This is comprised entirely of Inferred material.

25.5 Hydrology

Subject to final approval by government authorities, water will be sourced from the nearby Marahoué river, from pit dewatering and a supplementary borefield. Hydrological assessment of the river catchment indicates that the river will have flow in excess of total water demand for 8 months of the year.

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

Eight hydrogeological exploration boreholes were drilled to determine the preliminary aquifer characteristics at the proposed Koné Gold Project and was focused on the main South pit. Four aquifer tests (pump tests) were conducted and interpreted to derive aquifer parameters for three auifer systems. The aquifer parameters obtained suggest overall low aquifer transmissivity with higher transmissivity associated with fracturing along geological structures.

- The upper saturated weathered and laterite/saprolite zone (between 5 and 40m below surface) - groundwater flow potential is low, and boreholes may have probable range between 0.1 and 0.5 l/sec.
- The transition zone aquifer (Between 25 to 60m) - borehole yields are likely to be between 0.1 and 2.5 l/sec.
- The deeper fractured rock aquifer - groundwater borehole yields are likely to be between 0.1 and >5 l/sec.

The preliminary numerical groundwater model simulations concluded that pit de-watering will require abstraction in the order of 2,000 to 6,000 m3/day (23 l/sec to 70 l/sec). The overall mine pit de-watering will be supplemented by perimeter de-watering boreholes that will increase the overall water-make from the mining activities slightly. It is not expected that mining will supply more than 15 to 25% of the total water balance.

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

Harvested river water, pit de-watering and supplementary borefield water will be pumped to an off-stream water storage facility (WSF), adjacent to the process plant. Surface runoff from the mining area, ROM pad and stockpiles will gravity flow to this WSF. The WSF will have a capacity of approximately 3.0Mm3 and will enable accumulation of water during the wet season and a gradual drawdown in the dry season. In addition, water will be recycled from the tailings storage facility to the process water pond.

The processing, potable and dust suppression water requirements will be in the order of 30,000 m3/day. The site water balance indicates that sufficient water will be available for the duration of the life of mine with the proposed WSF, river harvesting, pit de-watering and supplementary borefield.

25.6 Power

A Build Own Operate Transfer (BOOT) contract is the preferred commercial arrangement for the power station supply and an LNG/Solar Hybrid has been assessed as the preferred power station.

The Koné Plant is estimated to have a maximum demand of 47.4 MW, an average annual demand of 39 MW with an expected energy consumption of 342GWhr/yr.

The up-front capital cost estimate for this LNG/Solar hybrid power station estimated at US$2.5M with annual repayments of $14.57M over 10 years. The operating cost is estimated at $0.076/kWhr. The solar PV and Battery Energy Storage Systems integration is expected to save in the order of 50,000 tonnes/year of CO2 emissions compared to the stand-alone LNG power plant.

25.7 Environment and Permitting

There are currently no objections to the development of the Project. The Project has commenced baseline data collection, to inform environment management plans. This will be an ongoing process, considering that Côte d'Ivoire is the most biodiverse country in West Africa. The Toudian Classified Forest Reserve is a protected forest reserve affected by and adjacent to the Project. To the southwest of Koné is the Yarani Forest Classified Reserve, and to the east is Kani-Bandama Rouge Classified Forest Reserve; neither are directly impacted by the Project footprint. The protection criteria of each of these forests will be assessed during the impact assessment process.

The Project is located relatively close to the communities of Batogo and Fadiadougou, but is sufficiently remote that environmental impacts on these communities are likely to be minor. Montage Gold provides support to local communities and exploration geologists engage frequently with the local people. Preliminary investigations indicate that the local community is positive towards the company. The company records all contact with local communities through monthly records, including support provided. An environmental management plan has been developed for exploration work, which is designed to be developed through the life of the Project and used to inform the impact assessment and subsequent Environmental and Social Management Plan (ESMP). The ESMP will include details of the area of impact, objectives to reduce negative or enhance positive impacts, specific targets adopted to achieve those objectives and definition of responsibilities for implementing the programme. Records shall be accurately maintained during exploration to monitor all activities and engagement. All exploration programs will be under the control and responsibility of a designated qualified representative of the company and audited to ensure that requirements are met.

Montage Gold is committed to managing the impacts of its operations, in conformance with recognised international best practice. The Project aims to conform to the environmental and social requirements of the IFC International Finance Corporation Performance Standards, its associated Environmental Health and Safety guidelines, International Council of Metals and Mining and Equator Principles where they are relevant.

26. RECOMMENDATIONS

In addition to exploratory drilling within the greater Koné project area, the work program includes infill and extensional/close off drilling at Koné designed to improve confidence in estimates for the current resource area and improve definition of mineralization extents.

The proposed budget and work program detailed in Table 26.1 enables a Feasibility Study to be completed by the end of 2021.

Table 26-1 Work Program Budget

Item	US$ M
Exploration	10.5
Met Test Work	0.2
Geotech	0.2
CHydrology	0.4
Tailings and Water Storage Facilities	0.4
PEA Study	0.5
Total	**12.2**

26.2 Geology

Future resource and definition drilling programs at the Koné Gold Project, consistent with Montage's planned work program. The program should reflect the following:

- Koné mineralization is open at depth and along strike and, in the author's opinion, additional drilling is warranted to define the limits of potentially economic mineralization to allow the Inferred Mineral Resource estimate to be updated. The drilling required to complete this program is estimated to be in the region of 10,000m of core drilling.
- The current resource area drilling is comparatively broadly spaced. Additional infill drilling will be required to form the basis of an Indicated Mineral Resource estimate for inclusion in the proposed FS. The drilling required to complete this program is estimated to be around 40,000m of combined RC and diamond core drilling.

Available information suggests the available sampling information drilling is sufficiently reliable for the current Mineral Resource estimates. The author's recommendations to further investigate the reliability of sampling data during future drilling programs are outlined below.

- Future drill programs aimed at higher confidence resource estimates should include diamond cored holes twinning representative RC holes.
- Selected, representative pulp samples from drilling to date and future programs should be routinely submitted to a second laboratory for third party check assaying.

26.3 Mining

Proceed with a budget tender exercise for the mining contract to confirm the assumptions in mining costs.

Review geotechnical information and complete additional geotechnical investigations (including drilling) in order to confirm pit wall slopes.

Complete hydrogeological study in conjunction with geotechnical work to confirm the effects of groundwater on both wall angles and operating costs due to dewatering.

Complete waste dump and haul road design to allow for more accurate estimate of haulage requirements.

26.4 Metallurgical Testwork

The next phase of testwork will require the following flowsheet development activities:

- Additional comminution testing to increase the variability sampling particularly on oxide mineralisation.
- Further variability testing, specifically on low grade samples to verify metallurgical response at the lower grades at site ambient temperature and design DO conditions
- Further metallurgical testing to evaluate the potential for the addition of a gravity stage.

26.5 Process

The flowsheet incorporates primary crushing followed by open circuit full secondary crushing prior to a SAG, Ball Mill, pebble crushing circuit. It is recommended that, in the next study phase, sensitivity analysis be conducted on SAG mill pebble extraction rates to determine the impact on the pebble crushing circuit.

The next study phase should consider alternative options for feeding oxide ore when a better definition of the split between saprolite, saprock and transition like oxide material is available to ensure the best flowsheet option is incorporated.

26.6 Infrastructure

26.6.1 Water

The following work requirements will be required for the FS:

- Test boreholes should continue to be monitored for groundwater level monthly and groundwater quality on a quarterly basis
- Three additional deep test boreholes should be drilled, two at the north pit and one at the south pit to a depth of at least 180m to 250m. A 4th intermediate depth borehole should be drilled east of GT005 to a depth of 100m. The locations are selected based on the identification of areas where groundwater was intersected by exploration RC drilling and diamond core geotechnical drilling. The boreholes will be pump tested to derive aquifer parameters and yield potential.
- The numerical groundwater model will be updated, and the de-watering system revised for project infrastructure and outlay.

26.6.2 Tailings Storage Facilities and Water Management

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

- Expanding topography to include areas potentially impacted by a dam break.
- Sterilisation of infrastructure footprints.
- Site inspection visit by KP project manager, COVID-19 permitting.
- Expansion of geotechnical investigation to include TSF areas, WSF and river abstraction location.

- River flow monitoring at proposed abstraction point.
- Stream gauging in WSF and TSF drainage lines to verify runoff co-efficients.
- Further groundwater assessment and verification of pit dewatering volumes.
- Probabilistic water balance model.
- Update of the design based on the findings of the above investigations.

26.6.3 Electric Power Supply

Further optimisation of the hybrid LNG/Solar/BESS power station is recommended during the next phase to minimise the overall cost of energy over the life of mine. Further options for the LNG supply chain are also to be explored in the next phase, including gas storage and back-up power options.

26.7 Environmental

By initiating the impact assessment process early, results will be used to improve the design, as well as maximise the benefits of the study without incurring excessive costs. In accordance to continual improvement processes, there are several strategies that are being 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;
- Recording of community engagement, including information sharing as well as support initiatives and infrastructure development;
- Examining the potential for energy efficiency, including the potential for renewable power as technology continues to improve; and
- Maintaining a grievance procedure to identify and pre-empt potential issues.

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28. DATE AND SIGNATURE PAGE

I **Geoffrey Alexander Duckworth** hereby state:-

1. I am employed as a Senior Consultant – Process, with the firm Lycopodium Minerals Pty Ltd, Level 2, 60 Liechhardt Street, Spring Hill, Queensland 4000 Australia, specialising in minerals processing engineering consulting, contracting for a wide variety of clients, and have been so employed since 2008.
2. This certificate applies to the technical report with an effective date of 25th May 2021, and titled "PRELIMINARY ECONOMIC ASSESSMENT FOR THE KONÉ GOLD DEPOSIT, CÔTE D'IVOIRE".
3. I am a practising Metallurgical Engineer and registered Fellow of the Australasian Institute of Mining and Metallurgy.
4. I am a graduate of the Royal Melbourne Institute of Technology with a Bachelor of Engineering (Chemical) 1974 and post graduate of the University of Queensland Australia with a M Eng Sc degree (1979) and PhD (1982), both in Mining and Metallurgy. I have practiced my profession continuously since 1974.
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 1.9, 1.10.2, 1.13, 17, 18.2, 21, 22 (overview), 25.6 and 26.5.3.
8. I am independent of the Issuer pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Montage or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with 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 7th day of June 2021

Geoffrey Alexander Duckworth

I **Jonathon Robert Abbott** hereby state:

1. I am a Consulting Geologist, with the firm of MPR Geological Consultants Pty Ltd, 19/123A Colin Street, West Perth, WA 6005, Australia.
2. This certificate applies to the technical report with an effective date of 25th May 2021, and titled "Preliminary economic assessment for the Koné gold deposit, Côte d'Ivoire".
3. I am a practising a practising Geologist and registered Member of the Australian Institute of Geoscientists.
4. I graduated with a Bachelor of Applied Science in Applied Geology from the University of South Australia in 1990. I am a member of the Australian Institute of Geoscientists. I have worked as a geologist for a total of 30 years since my graduation from university. My experience includes mine geology and resource estimation for a range of commodities and mineralization styles. I have been involved in preparation and reporting of resource estimates in accordance with JORC guidelines for 25 years, and NI43 101 guidelines for approximately 17 years
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I have been involved with the Morondo Gold Project since July 2018 and visited the project site on the 23rd and 24th August 2018.
7. I am responsible for Sections 1.7, 12 and 14 of the Technical Report.
8. I am independent of the Issuer pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Montage or any associate or affiliate of such company.
10. I have had prior involvement with the Morondo Gold Project. Between August and November 2018, I prepared Mineral Resource estimates for Orca Gold and authored a Technical Report titled "Mineral Resource Estimation for the Koné gold deposit Morondo Gold Project Côte d'Ivoire NI 43-101 Technical Report with an effective date of the 3rd of October 2018. During August and September 2019, I was co-author of an updated Technical Report titled "NI 43-101 Technical Report for the Morondo Gold Project, Côte d'Ivoire" with an effective date of the 17th of September 2020. In January 2021 I was co-author of a Technical Report titled "NI 43-101 Technical Report for the Morondo Gold Project, Côte d'Ivoire" with an effective date of the 27th of January 2021.
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 7th day of June 2021

____

Jonathon Robert Abbott

I **Michael Peter Hallewell** hereby state:-

1. I am a consulting Metallurgist, with the firm of MPH Minerals Consultancy Ltd, 8 The Gluyas, Falmouth, Cornwall, TR11 4SE.
2. This certificate applies to the technical report with an effective date of 25th May 2021, and titled "PRELIMINARY ECONOMIC ASSESSMENT FOR THE KONÉ GOLD DEPOSIT, 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. I have 40 years practical experience in Minerals Processing as Plant Manager, Consulting or Senior Metallurgist in precious metals, base metals and ferrous metals industry.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I have not visited the project site.
7. I am responsible for Sections 1.6, 1.15.4, 13, 25.3 and 26.4 of the report.
8. I am independent of the issuer as described in section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Montage or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with 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 7th day of June 2021

____

Michael Peter Hallewell

I **Pieter Ferdinandus Labuschagne** hereby state:-

1. I am a consulting Hydrogeologist, with the firm of GCS (Pty) Ltd and situated in the Durban office in South Africa (4A Old Main Road, Kloof, 3610).
2. This certificate applies to the technical report with an effective date of 25th May 2021, and titled "PRELIMINARY ECONOMIC ASSESSMENT FOR THE KONÉ GOLD DEPOSIT, 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.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I Have not visited the Project site.
7. I am responsible for sections 1.10.1, 1.15.5, 18.1, 25.5 and 26.5.1.
8. I am independent of the Issuer pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Montage or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with 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 7th day of June 2021

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 25th May 2021, titled "PRELIMINARY ECONOMIC ASSESSMENT FOR THE KONÉ GOLD DEPOSIT, CÔTE D'IVOIRE".
3. I am a practising Environmental Consultant and registered Member of the Institute of Materials, Minerals and Mining.
4. I am a graduate of Imperial College, London, UK with a Masters in Environmental Diagnosis, with a Bachelor of Science (Honours) degree in Biodiversity Conservation and Environmental Management. I have practiced my profession continuously since 2005, and have 12 years practical experience in Environmental Impact Assessments for mining projects.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I visited the Koné Gold Project between 13th March and 18th March 2021. The purpose of the visit was to review the baseline conditions and establish priorities for environmental management for the project.
7. I am responsible for sections 1.12, 1.15.1, 20, 25.7 and 26.6.
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 not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with 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 7th day of June 2021

Carl Steven Nicholas

I **Chris Reardon** hereby state:-

1. I am the Country Manager with the Orca Gold Inc and situated in Khartoum, Sudan.
2. This certificate applies to the technical report with an effective date of 25th May 2021, and titled "PRELIMINARY ECONOMIC ASSESSMENT FOR THE KONÉ GOLD DEPOSIT, CÔTE D'IVOIRE".
3. I am a practising Mining Engineer with over 25 years of relevant experience in open pit mining operations, 17 of which have been in open pit gold mines.
4. I am a graduate of the University of Queensland, Australia with a Bachelor of Science degree in Geology (1994). I have practised my profession continuously since 1995.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I have not visited the Project site.
7. I am responsible for sections 1.8, 1.15.3, 16, 21.3, 21.5, 25.4 and 26.3.
8. I am independent of the issuer pursuant to Section 1.5 of NI 43-101.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 7th day of June 2021

C.PRead

Chris Reardon

I **Ed Tuplin** hereby state:-

1. I am employed as Senior Project Engineer, with the firm Knight Piésold Pty Limited Level 1, 36 Cordelia Street, Brisbane, QLD 4101, AUSTRALIA.
2. This certificate applies to the technical report with an effective date of 25th May 2021, and titled "PRELIMINARY ECONOMIC ASSESSMENT FOR THE KONÉ GOLD DEPOSIT, CÔTE D'IVOIRE".
3. I am a current member of the Australian Institute of Mining and Metallurgy (No 3222279) and have been awarded the status of Chartered Professional (CP) in the field of Environmental Engineering.
4. I am a Registered Professional Engineer of Queensland (RPEQ) and in good standing with the Board of Professional Engineers of Queensland, Australia (N 17816)
5. I graduated from the University of Leeds with a Bachelor of Science in Environmental Geology in 2003 and Master of Science in Engineering Geology in 2004.
6. I have practised my profession continuously since 2004.
7. 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.
8. I have not visited the project site.
9. I am responsible for sections 1.10.3, 1.15.5, 18.1.9, 18.1.10, 18.1.11, 18.3 and 25.5.2.
10. I am independent of the Issuer pursuant to Section 1.5 of NI 43-101.
11. I do not beneficially own, directly or indirectly, any securities of Montage or any associate or affiliate of such company.
12. I have not had prior involvement with the property that is the subject of the Technical Report.
13. 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.
14. 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 7th day of June 2021

Ed Tuplin