NI 43-101 TECHNICAL REPORT
BOTO PROJECT FEASIBILITY STUDY – SENEGAL

Prepared by Lycopodium Minerals Canada Ltd in accordance with the requirements of National Instrument 43-101, "Standards of Disclosure for Mineral Project", of the Canadian Securities Administrators

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1.0 SUMMARY

The Boto Project (the Project) is located in the southeast corner of Senegal within the Daorala-Boto Exploration Permit. The exploration permit is made up of two non-contiguous properties, i.e. the Boto property (the Property) and the Daorala property to which no mention is made in this report.

IAMGOLD is an intermediate gold producer with four operating gold mines and several exploration projects located in North and South America, and Africa. AGEM Senegal SUARL Exploration (AGEM), a wholly owned subsidiary of IAMGOLD, controls the Property.

In 2017, IAMGOLD engaged several leading consulting firms to undertake a Feasibility Study (FS) for the Project. The principal contributors are listed in Table 1.1.

Table 1.1 Study Contributors

Contributor	Scope
Lycopodium Minerals Canada Limited (Lycopodium)	Metallurgical testwork, process plant, project infrastructure, project development plan, compile capital and operating cost estimates, coordination and compiling of report
AGP Mining Consultants Inc (AGP)	Geology, mineral resources, mining reserve statement
Knight Piésold (KP)	Tailings management facility, water storage facility, water management, geotechnical
Absolute Geotechnics (AG)	Geomechanical
IAMGOLD	Environmental and permitting, financial modelling

1.1 Property Location and Description

The Daorala-Boto Project is located in the Kédougou Region (Saraya Department) in the southeast of Senegal and is situated along the triple border junction of Senegal-Mali-Guinee, bounded by the Balinko and Falémé Rivers. AGEM holds the mineral rights to two exploration permits consisting of the Daorala-Boto and Boto West projects. The Daorala-Boto exploration permit covers a total area of 236 km².

The Daorala-Boto exploration permit consists of two non-contiguous areas, the Daorala and the Boto, along the southeast border with Mali. These two areas are separated by the Bamabji exploration permit, situated between these two areas, and is also held by AGEM which is part of an agreement with Randgold Resources Ltd. (Randgold), who is the operator on the permit.

The exploration permit for the Daorala-Boto permit was renewed on 8 August, 2017, by the Senegalese government, and has an expiry date of 4 March 2019.

1.2 Accessibility, Climate, Local Resources, Infrastructure and Physiology

Access to the Project from Dakar is either by paved road from the capital, Dakar, to the town of Saraya (approximately 760 km by road) and then by gravel/laterite road to the village of Noumoufoukha (approximately 80 km). The Boto exploration camp is situated 12 km from the village of Guémédji. There are no regular scheduled flights to Kédougou, situated 135 km by road from the Project, but there are aircraft that are available for charter from Dakar.

The Project is located in a subtropical continental climate zone and is characterized by two seasons: a rainy (wet) season from June to October, and a dry season from October to May. Exploration activities may be conducted all year round. However, during the wet season, the Kolia Kabe River, situated 14 km by road to the northwest of Boto Exploration Camp, floods and cuts off the road access at the Saroudia Bridge.

There is minimal infrastructure at the Project site. Electricity is provided by diesel generators at the Exploration Camp. Water is supplied by a well with a water treatment plant. There is some cellular telephone coverage which is supplied to this area of Senegal, and the Project, by Senegal-based cellular towers and by cellular towers in neighbouring Mali. All equipment, supplies, and fuel are transported by road to the project site. Most supplies, consumables, and fuel are sourced either from Kédougou or Dakar depending on availability. The village of Guémédji, and some surrounding villages, are a source of unskilled workers and fresh produce. Skilled and professional workers are sourced from Dakar.

The Property lies between 100 m and 300 m above sea level with generally low to moderate relief consisting of broad lateritic plateaus and eroded valleys. The vegetation is typical of a tropical forested savannah, with scattered trees (including baobab), scrub brush, elephant grass, and bamboo. Trees are more abundant along rivers and creeks as gallery forests and around lateritic plateaus that have been broken down by erosion.

1.3 History

Prior to 1994, there is no known or recorded systematic mineral exploration carried on the Property. From 1994 to 1996, the first exploration activities were carried out by Anmercosa Exploration (Anmercosa), a subsidiary of Anglo American. From 1997 to 1998, Ashanti Goldfields Corporation (Ashanti Goldfields), completed further exploration activities in a joint venture with AGEM. From 1999 to present, AGEM has conducted all succeeding exploration activities on the Property.

1.4 Geology and Mineralization

The Project is located in the West African Craton, in the south-eastern part of the Early Proterozoic formation of the Kédougou-Kéniéba inlier, which covers the eastern part of Senegal and western Mali. In the southern part of the craton, Lower Proterozoic Greenstone Lands are described as Birimian based on the Kits (1928) in the Birim River Valley of Ghana. The Kédougou-Kéniéba inlier, where the Project is located, is the exposure in the far west of the Birimian. The Kédougou-Kéniéba inlier is bounded on the west side by the Hercynian Mauritanide belt; and on all other sides, it is unconformably overlain by the underformed upper Proterozoic sediments and the Early Phanerozoic rock of the Taoudeni, Tindouf, and Volta basins

At Boto, the material near the surface consists of a layer of regolith which is varying in thickness and includes lateritic plateaus. Few rocky outcroppings are visible in the property; the banks of streams and rivers serve as the main source for geological observations. Only drilling can provide a detailed knowledge of the geology below surface. Boto can be divided into three north trending litho-structural domains (020˚ N) that are well delineated in both induced polarization (IP) and magnetic surveys. From west to east, the three domains are: Western Flyschoid Domain, Central Deformation Corridor, and the Eastern Siliciclastic Domain.

The western domain is dominated by a sequence of flyschoid turbidites, black shales (or graphitic pelite), carbonate rocks, minor volcanics (mainly basalt with subordinate rhyolite and pyroclastic breccia or agglomerate), and dioritic intrusions. The Boto 5 deposit is located along the contact between this domain and the central deformation corridor.

The eastern domain is dominated by a detrital assemblage composed of greywacke and sandstone (± quartzite), called Guémédji sandstone. It is thought that these sandstones/wackes are part of the Kofi Series which is very present in the Malian portion of the inlier.

Between the west and east domains is a highly deformed north-trending domain (020˚ N) that is well defined in magnetic geophysical data. It is likely this highly deformed domain corresponds to a regional scale structural corridor that branches from the SMSZ. Lithologically, it is composed of fine schistose sediments that are carbonaceous in places, locally referred to as the "Pelite Unit", and fine laminated sediments (± carbonates) that subtly grade into an impure marble, locally called the "Cipolin Unit".

The Project consists of four (4) deposits, Malikoundi/Boto 2, Boto 5, Boto 4 and Boto 6, all of the late orogenic type. The late orogenic gold mineralization is typically associated with brittle-ductile deformation and is characterized by the association of Au, B, W, As, Sb, Se, Te, Bi, Mo, with traces of Cu, Pb, Zn. Mineralization at Malikoundi/Boto 2, Boto 4 and Boto 6 is associated mainly with chlorite-albite alteration. Gold commonly occurs as native gold or as fine inclusions within the base-metal sulphides or the gangue that consists of quartz, albite, carbonate, muscovite, pyrite, and tourmaline. Mineralization at Boto 5 is associated with a phase or quartz tourmaline veining as well as pyrite and related bleaching. The mineralizing event was accompanied by biotite alteration and pyrite mineralization, and a small proportion of chalcopyrite, covellite, and chalcocite. The presence of arsenopyrite appears to be confirmed by recent XRF measurements.

1.5 Exploration and Drilling

The Project has been subject to exploration and development by AGEM since 1999 to present. Early exploration consisted of geochemical soil, lag, rock and termite mound sampling; pit and trench sampling; geophysical surveys; and drilling. Exploration to date has defined the Malikoundi/Boto 2, Boto 5, Boto 6 and Boto 4 deposits. Additional exploration activities around the known deposit have resulted in several other targets for further exploration.

Drilling at the Project has been conducted in various campaigns from 2000 to present. As of March 2018, a total of 126,429 m have been completed from 784 drill holes. Of the 784 drill holes in the drill hole database, 496 drillholes intercept the interpreted mineralized zones in Malikoundi/Boto 2, Boto 5, Boto 6, and Boto 4 deposits.

1.6 Metallurgy

Extensive metallurgical testwork on the Boto ore deposit has been conducted from 2013 to 2018. The testwork results were analyzed and used in flowsheet development and inputs into the process design criteria.

The comminution parameters determined based on lithology weighted average per weathering type are as follows:

- 85th percentile BWi of 10.8 kWh/t, 11.2 kWh/t, and 20.6 kWh/t for saprolite, saprock, and fresh rock, respectively.
- 85th percentile CWi of 16.4 kWh/t for the fresh rock.
- 50th percentile Ai of 0.033 g, 0.043 g, and 0.542 g for saprolite, saprock and fresh rock, respectively.

The Boto fresh rock is classified as hard ore while the Boto saprolite and saprock are classified as softer ore when compared to the A.R. MacPherson Grinding Specialist database.

Other key results from the metallurgical testwork include:

- Gold extraction increased with decreasing grind size, however, a grind size of 80% passing (P_{80}) of 75 µm was determined to be optimum for the Project.
- Preg-robbing assessment on the Malikoundi/Boto 2 and Boto 5 samples showed no evidence of preg-robbing activity.
- Gravity separation tests (E-GRG tests), and whole ore leach tests showed limited benefits from inclusion of a gravity circuit in the flowsheet. The majority of the GRG amount found in MC-2 of the 2018 testwork was very fine in nature; hence, recovery with gravity at full scale would be difficult.
- Synergistic effects from lead nitrate and oxygen addition during pre-treatment, and oxygen addition during leaching provided faster leach kinetics, significant reduction in cyanide consumption, and gold extraction benefits.
- Cyanide consumption was low with a consumption rate of 0.13 kg/t ore expected at the design ore blend (approximately, 90% fresh rock, 10% saprolite/saprock).
- Lime consumption was moderate with a consumption rate of 1.64 kg/t ore expected at the design ore blend.

1.7 Mineral Resources

The mineral resources for the Project were updated for this report. The mineral resources were prepared and disclosed in accordance with the CIM Standards and Definitions for Mineral Resources and Mineral Reserves

(2014). The QP responsible for these resource estimates is Mr. Paul Daigle, P.Geo., Associate Senior Geologist for AGP. The effective date of this mineral resource is May 8, 2018.

The resource estimate has been prepared using interpreted mineralized veins (domains) for four deposits that comprise the Project; in order of priority: Malikoundi/Boto 2, Boto 5, Boto 6, and Boto 4. The resource estimates were conducted using Geovia GEMS™ 6.8.1 resource estimation software. The blocks models were estimated using inverse distance cubed.

Table 1.2 presents a summary of the Mineral Resources for the Project.

Table 1.2 Summary of Mineral Resources for the Boto Project; effective date 8 May, 2018

Classification	Tonnes (,000 t)	Grade (g/t Au)	Contained Metal (,000 oz Au)
Indicated	48,045	1.61	2,487
Inferred	2,483	1.80	144

***Notes**:*

- *Mineral resources are reported within an optimized constraining shells using MineSight 3D software.*
- *Summation errors may occur due to rounding.*
- *Cut-off grades vary between 0.37 g/t Au and 0.51 g/t Au, depending on the deposit and the alteration type of material.*
- *Mineral resources were estimated based on a gold price of $US 1,500/ oz.*
- *Capping of grades varied between 1.71 g/t Au and 42.02 g/t Au on raw assays by mineralized zone or sub-domain.*
- *The density varies between 1.70 g/cm^3 and 2.76 g/cm^3 depending on alteration zone.*

The mineral resources for the Project include the Malikoundi/Boto 2, Boto 5, and Boto 6 deposits. The Boto 4 deposit is currently not classified as mineral resources because the deposit is situated within a 500 m exclusion zone of the Balinko River (the border of Senegal and Mali) and underneath the village of Guémédji. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.8 Mineral Reserves

The reserves for the Project are based on the conversion of the Indicated resources within the current Technical Report mine plan. No Measured resources are currently part of the model. Indicated resources are converted directly to Probable Reserves. The total reserves for the Project are shown in Table 1.3.

Table 1.3 Proven and Probable Reserves – Boto Gold Project

	Proven			Probable			Total		
Ore Type	Tonnes (kt)	Grade (g/t)	Gold (oz)	Tonnes (kt)	Grade (g/t)	Gold (oz)	Tonnes (kt)	Grade (g/t)	Gold (oz)
Saprolite	-	-	-	2,910	1.85	173,000	2,910	1.85	173,000
Transition	-	-	-	2,160	2.01	139,000	2,160	2.01	139,000
Fresh Rock	-	-	-	29,990	1.67	1,614,000	29,990	1.67	1,614,000
Total	**-**	**-**	**-**	**35,060**	**1.71**	**1,926,000**	**35,060**	**1.71**	**1,926,000**

Note: This mineral reserve estimate is as of Aug 30, 2018 and is based on the new mineral resource estimate dated May 8, 2018 for Malikoundi and Boto 5 by AGP. The mineral reserve calculation was completed under the supervision of Gordon Zurowski, P.Eng of AGP., who is a Qualified Person as defined under NI 43-101. Mineral reserves are stated within the final design pit based on a $1,044/oz gold price pit shell with a $1,200/oz gold price for revenue for Malikoundi, $960/oz for Malikoundi North and $900/ounce for Boto 5. The cut-off grade varied by material type from 0.46 g/t Au in saprolite, 0.50 g/t Au in transition and 0.63 g/t Au in fresh rock for the Malikoundi and Malikoundi North pit areas. The cut-off was 0.48 g/t Au in saprolite, 0.49 g/t Au in transition and 0.59 g/t in fresh rock for the Boto 5 area. The mining cost varied by rock type and area but averaged $2.13/t, processing costs vary by rock type but averaged $15.04/t milled and G&A was $4.22/t milled. The process recovery averaged 89.5%. The Technical Report scope only considers the Malikoundi, Malikoundi North and Boto 5 open pit mineralized zones.

The reserves are based solely on the Malikoundi, Malikoundi North and Boto 5 areas.

The QP has not identified any known legal, political, environmental, or other risks that would materially affect the potential development of the Mineral Reserves. The risk of not being able to secure the necessary permits from the government for development and operation of the project exist but the QP is not aware of any issues that would prevent those permits from being withheld per the normal permitting process.

1.9 Mining Methods

The Project is amenable to extraction by open pit methods. Costs were developed from base principles and with current equipment quotations from local vendors.

All design work is based on the Malikoundi and Boto 5 models generated by AGP with an effective date of May 8, 2018. Only Indicated Resources were used for the FS and all inferred resources are considered as waste. No measured resources exist in the current models.

A geotechnical study was completed on the Malikoundi and Boto 5 deposits by AG. The study provided detailed slope recommendations by alteration zone, material type and orientation. These recommendations were incorporated in the pit optimizations completed for the FS with allowances made for ramps in the slopes to determine an overall angle to use. The recommendations were also incorporated in the detailed mine design.

Pit optimizations utilizing the Lerch-Grossman routine were examined for Malikoundi, Malikoundi North and Boto 5. A base price of $1,200/oz was used. A series of nested shells were generated using a revenue factor (rf). These were varied between a gold price of $300/oz (rf=0.25) and $1,440/oz (rf=1.2) to examine the deposit sensitivity to gold prices and outline the higher value areas. This information was graphed, and the various phases and final shell determined based on a net revenue curve. The final pits are based on the $1,044/oz gold price shell for Malikoundi, $960/oz gold price shell for Malikoundi North and $900/oz shell for Boto 5.

The geologic models used for the FS are whole block fully diluted models. Contact dilution was also believed to be a consideration and was estimated based on 0.5 m per block side if the material was below cutoff grade. The calculation for dilution for the Malikoundi pits resulted in 4.5% more tonnes and 4.1% lower gold grade than insitu. Boto 5 had slightly higher dilution results of 7.8% more tonnes and 6.7% lower gold grade than insitu. The diluted tonnes and grade were reported in the detailed pit designs.

The detailed pit design utilized the pit shells developed to provide guidance on the phasing and final pit. Wall slopes for the inter-ramp were per the geotechnical recommendations.

Equipment sizing for ramps and working benches is based on the use of 95 t rigid frame haul trucks. The operating width used for the truck is 6.9 m meaning single lane access is 21.4 m (2x operating width plus berm and ditch) and double lane widths are 28.3 m (3x operating width plus berm and ditch). Ramp gradients are 10% in the pit for uphill gradients and 8% uphill on the dump access roads.

The Malikoundi pit is designed as 4 phases within the main pit. Phase 0, as the initial pit is called, is a subset of Phase 1 to drive quickly to fresh rock for tailings dam construction purposes. Malikoundi North is a single-phase pit as is Boto 5.

The mine schedule delivers 35.1 Mt of ore grading 1.71 g/t Au to the process plant over a mine life of 12.8 years. Waste tonnage totalling 204.3 Mt will be placed into waste rock management facilities. The overall strip ratio is 5.83:1 life of mine. The Malikoundi cut-offs used are 0.46 g/t Au for saprolite, 0.50 g/t Au for transition material, and 0.63 g/t for fresh rock. In Boto 5 the calculated cut-offs are 0.48 g/t Au in saprolite, 0.49 g/t Au for transition material and 0.59 g/t Au for fresh rock. 1 g/t Au cut-off was used for all materials to separate low grade material and high-grade material.

The current mine life includes two years of pre-stripping and 12.8 years of mining. Plant throughput is 2.75 Mtpa. The final year will complete the pit and clear the stockpiled ore. The stockpiled ore, together with pit phasing will be utilized to ensure enough mill feed is available in the rainy season. This will also be coupled with in-pit sumps and surface ditches around the pits. Phases will be advanced quickly in the dry season to develop low spots that will provide temporary water storage after a rainfall event that pumping will remove in the wet season.

Laterite and saprolite overlay a zone of transition which in turn is underlain by fresh rock. Laterite for Malikoundi and Boto 5 is typically comprised of 3-5 m of ferricrete with another 3-5 m of agglomerate. Malikoundi material mined is primarily fresh rock while the Boto 5 pit only mines the saprolite and transition material. No blasting is required in Boto 5 and the laterite and saprolite in Malikoundi require minimal blasting as this material will be used for construction purposes.

Ore control for mining is designed using a reverse circulation drilling program in advance of mining. Ore samples will be collected each metre in the drill hole and used in the ore control model to guide mine planning and operations.

The mine schedule is based on mining the four phases in Malikoundi, the single phase in Malikoundi North and the single phase in Boto 5. Boto 5 has been designed as a contract mining pit that utilizes a maximum truck size of 40 t with no drilling and blasting.

Mining commences in Year -2 which is a three-month period and continues in Year -1 which is a full 12-month period. This pre-production stripping is required to assist in the construction of various infrastructure items and prepare sufficient material in stockpile prior to mill commissioning. The infrastructure items include mine roads to the tailings management facility (TMF), fresh water pond (FWP), process plant, various waste management facilities and the TMF itself.

A total of 2.3 Mt of material will be moved in Year -2 and an additional 13.7 Mt moved in Year -1. This includes the development of a 2.2 Mt ore stockpile grading 2.25 g/t Au. Malikoundi Phase 0 and Phase 1 will be the only active phases in the preproduction period. Phase 0 provides fresh rock for construction needs.

Year 1 production assumes that the plant will require 3 months to achieve full production levels. The first month the plant will be capable of 60% of capacity, the second month 80% and the third month 90%. Subsequent months will be at 100% of nameplate capacity in the mill. Ore mining will be from stockpile, Boto 5 and Malikoundi Phase 0, 1 and 2. Mining will be initiated in Malikoundi Phase 2, Malikoundi North and Boto 5.

Mining occurs from Year -1 until part way through Year 12. The remainder of Year 12 and Year 13 is stockpile reclaim to feed the mill. Peak mining requirements are in Year 2 with Phase 3 initiating. This peak is 33.4 Mt of total material movement. Years 3 to 6 average 28 Mt of total material movement then the mine production starts to drop off steadily before finishing in Year 12. The saprolite feed to the mill is held to 10% of mill feed from Year 1 to 10 then only the remainder of the saprolite stockpile is mined in Year 11.

Mining in Boto 5 occurs from Year 1 to 3. Malikoundi North is mined from Year 1 until Year 5. Malikoundi is mined continuously throughout the Project.

The life of mine (LOM) schedule by year is shown in Table 1.4.

Table 1.4 Life of Mine Schedule

Period	Ore to Plant kt	g/t Au	Direct to Mill kt	To Stockpile kt	From Stockpile kt	Waste kt	Total Material Mined kt
Pre-production (Yr-2)	-	-	-	-	-	2,296	2,296
Pre-production (Yr-1)	-	-	-	2,245	-	11,460	13,705
Year 1	2,590	2.34	1,409	799	1,181	21,876	24,084
Year 2	2,750	2.27	2,163	1,312	587	29,952	33,426
Year 3	2,750	1.67	2,475	789	275	26,360	29,624
Year 4	2,750	1.93	2,022	841	728	24,686	27,550
Year 5	2,750	1.81	2,475	557	275	24,548	27,580
Year 6	2,750	2.14	2,375	1,626	376	23,358	27,359
Year 7	2,750	1.58	2,431	290	320	15,536	18,257
Year 8	2,750	1.39	2,475	269	275	10,567	13,311
Year 9	2,750	1.79	2,475	387	275	7,302	10,164
Year 10	2,750	1.74	2,475	260	275	4,152	6,887
Year 11	2,750	1.59	2,221	-	529	1,916	4,137
Year 12	2,750	1.08	692	-	2,058	326	1,017
Year 13	2,222	0.72	-	-	2,222	-	-
Total	**35,062**	**1.71**	**25,686**	**9,376**	**9,376**	**204,335**	**239,397**

Various rock types are present in the material mined within the final pits. They include the weathering profile of laterite, saprolite, transition and hard rock. The percentages vary by pit and phase. Ferricrete is present in some areas and will be utilized for construction material and roads. All material types will be co-mingled in the waste management facilities. Certain portions of the material will be directed to the TMF for the embankment construction. In addition, there will be four waste storage areas. These are shown in Figure 1.2.

The waste dump northwest (WDNW) of the plant has a design capacity of 40.8 Mm^3 but only 36.5 Mm^3 is utilized. The waste dump northeast (WDNE) of the plant has a design capacity of 53.8 Mm3 which is fully utilized. The FWP requires 1.3 Mm^3 from the mine for construction. The Boto 5 facility is designed at 9 Mm^3 and is fully utilized.

Drainage from each of these facilities is diverted to sedimentation ponds to ensure sediment washed down from the facilities is captured before it leaves the mine property. Annual cleaning of the sediment ponds or more frequently if required is planned to ensure that storage capacity is not lost.

1.10 Recovery Methods

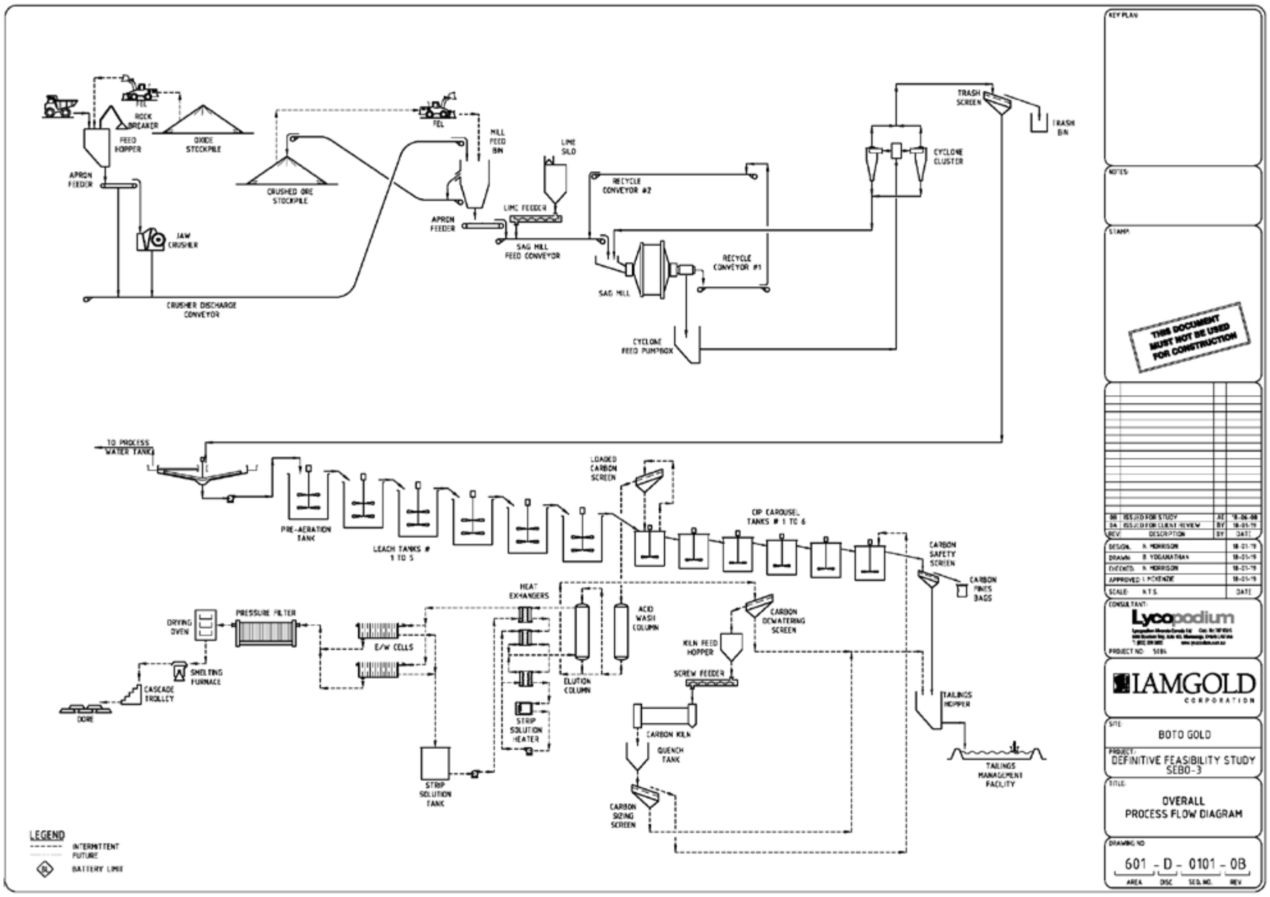
The process plant design for the Project is based on extensive metallurgical testing, Lycopodium's experience and industry standards. The flowsheet configuration and unit operations are well proven in the gold processing industry.

The plant has been designed with a nominal throughput of 2.75 Mtpa ore, crushing circuit availability of 75% and a mill utilization of 92%. The plant design incorporates the following unit process operations:

- Single stage primary crushing with a jaw crusher to produce a crushed product size of P_{80} of 138 mm.
- Mill feed surge/overflow bin that overflows to a stockpile.
- The grinding circuit is a single stage semi-autogenous mill (SSAG) type, which consists of a closed circuit single stage semi-autogenous (SAG) mill, producing a P_{80} grind size of 75 µm.
- Hydrocyclones are operated to achieve an overflow slurry density of 28.1% w/w solids to promote better particle size separation efficiency. The overflow stream passes through a trash screen to remove foreign materials prior to downstream processing. Subsequently, a pre-leach thickener is included to increase slurry density to the leach circuit, minimize leach tank volume requirements and reduce overall reagent consumption.
- Leach circuit with five tanks to achieve the required 34.4 hours of residence time at nominal plant throughput. A pre-oxidation step is included ahead of leaching to minimize cyanide consumption and improve downstream leach kinetics.
- Carbon-in-pulp (CIP) carousel circuit consisting of six stages for recovery of gold dissolved in the leaching circuit.
- Pressure Zadra elution circuit with gold recovery to doré. The circuit includes an acid wash column to remove inorganic foulants from the carbon with hydrochloric acid, followed by an elution column.
- Carbon regeneration kiln to remove organic foulants from the carbon and reactivate the adsorption sites on the activated carbon with heat.

Figure 1.1 shows an overall flow diagram of the Process Plant.

Figure 1.1 Overall Process Flow Diagram

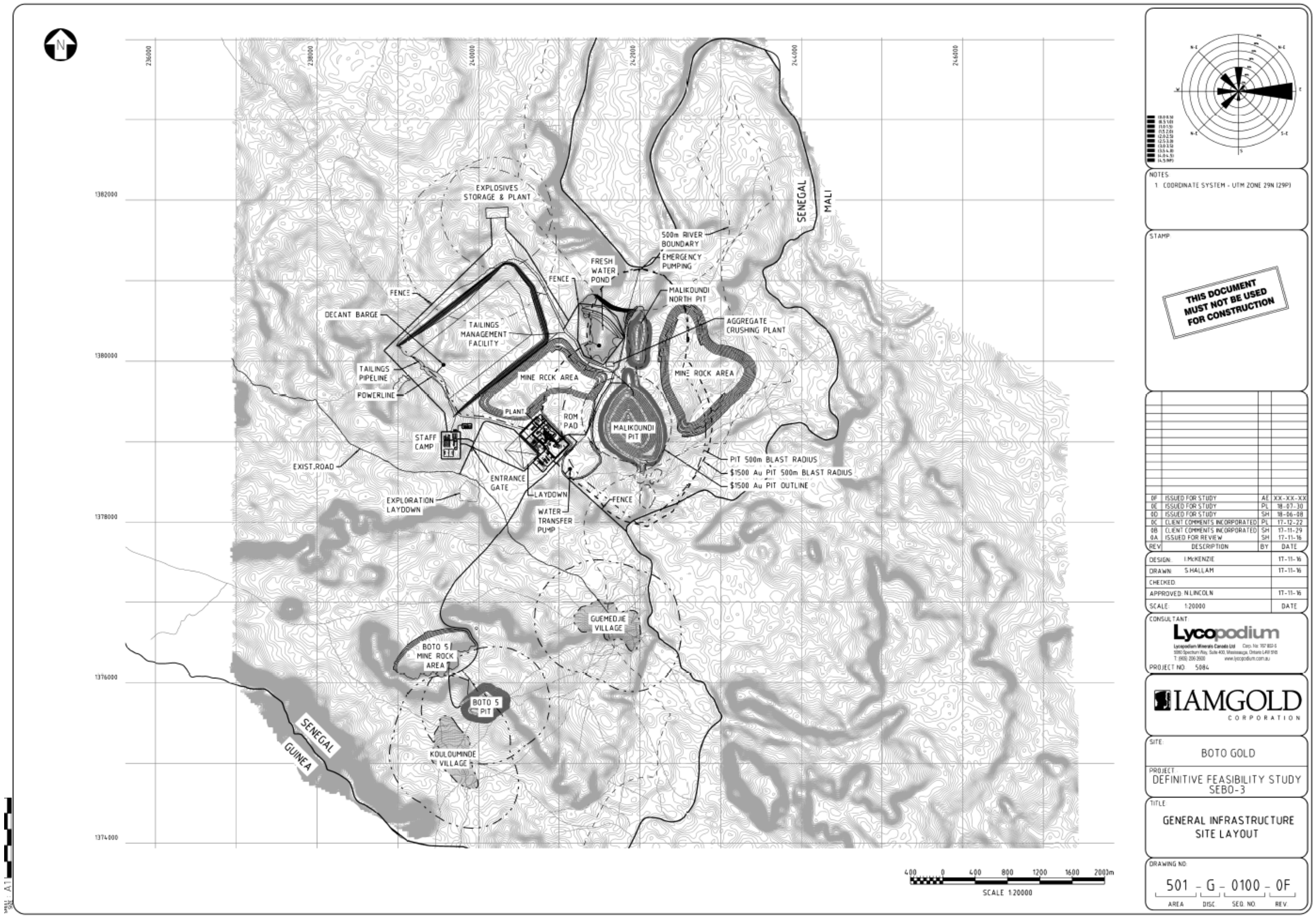


1.11 Infrastructure

The overall site plan for the Project (refer to Figure 1.2) includes the main facilities such as the open pit mines (Malikoundi, Malikoundi North and Boto 5), waste dumps, process plant, TMF, FWP, staff camp and main access road. An onsite power plant and bulk fuel storage not shown on Figure 1.2 is located at the process plant.

The process plant, associated buildings, onsite power plant and bulk fuel storage are located west of the Malikoundi mine. The TMF is located north east of the process plant. The staff camp is located near the main access road and west of the process plant for ease of personnel access. The main access road approaches the site from the west.

Figure 1.2 Overall Site Plan



1.12 Environmental Studies, Permitting, Social and Community Impact

A summary analysis of the initial environmental and social status of the exploration permit was carried out in 2014 by TROPICA Environmental Consultants and was completed during the preliminary study by field investigations.

In order to comply with these legal and regulatory requirements as well as the World Bank Group Guidelines, an environmental and social impact study process was launched in June 2015 and was completed in 2018.

To properly understand the project's human, physical and biological context, baseline environmental studies on social sanitation conditions, public health, fauna, flora and biodiversity, surface water and ground water quality, the water regime, and the cultural heritage were advanced in 2015, in the first half of 2016, and completed in the second half of 2017. Tailings and waste geochemical characterization studies were also conducted during these periods.

The upstream public consultation process took place in 2016, and a public inquiry was made in May and June 2016, at the request of the Kédougou region Governor.

The complete Environmental and Social Impact Study (ESIA) report, including the ESMP and the closure and reclamation framework, were submitted to the authorities in 2016, on the basis of the project as developed as part of the original prefeasibility study. At the request of IAMGOLD, the impact study validation procedure was suspended due to the continuation of technical studies.

Following the publication of the optimized prefeasibility study and the launch of the feasibility study, the ESIA report was updated with new data at the end of the first half of 2018 and submitted to the Ministry of Environment for instruction and validation. The report was reviewed in April 2018 by the technical committee, representing all key and administrative stakeholders, and additional information was requested. The amended report taking into consideration this feedback was submitted to the Ministry of Environment in May 2018.

The downstream public consultation procedure will take place once the DEEC authorities have validated the ESIA report. The ESIA update is not expected to have major changes – the Optimization team made general positive changes on environmental and social aspects.

The highlights of the baseline environmental studies and the impact study are provided in Section 20 and are from the ESIA filed in 2018.

1.13 Capital and Operating Costs

The overall capital cost estimate was compiled by Lycopodium and is presented in Table 1.5.

All costs are expressed in United States Dollars unless otherwise stated and are based on Q2 2018 pricing and deemed to have an overall accuracy of ±15%. The capital cost estimate conforms to AACEI (Association for the Advancement of Cost Engineering International) Class 3 estimate standards as prescribed in recommended practice 47R11.

The capital cost estimate was based on an engineering, procurement and construction management ("EPCM") implementation approach and typical construction contract packaging. Equipment pricing was based on quotations and actual equipment costs from recent similar Lycopodium projects considered representative of the Project.

Table 1.5 Capital Estimate Summary (Q2 2018, ±15%)

Area	M$ (Excluding Duties and Taxes)
Direct Costs	
Mining	$62.1
Infrastructure	$19.0
Ore Handling & Processing	$57.4
Tailings & Water Management	$16.6
Sub-Total Direct Costs	**$155.1**
Indirect Costs	
Construction In-directs	$49.0
Owner's Costs	$19.2
Contingency	$24.3
Sub-Total Indirect Costs	**$92.5**
Sub-Total Initial Capital Costs	**$247.6**
Additional Indirect Costs	$6.8
Total Initial Capital Cost	**$254.4**
Sustaining Capital Cost	$66.0
Total Project Capital Cost	**$320.4**

The estimated life of mine operating cost per tonne of ore processed is summarized in Table 1.6.

Table 1.6 Life of Mine Operating Costs per Tonne (Q2 2018)

	Total Cost ($M) from first gold pour	$/t Processed
Mining	$456	$13.01/t
Processing	$528	$15.04/t
G&A	$148	$4.22/t
Total Cash Cost	**$1,132**	**$32.27/t**

1.14 Economic Analysis

An economic assessment of the Project was completed using a pre and after-tax cash flow model prepared by IAMGOLD. The model was structured using an EXCEL workbook which presents annual cash flows during the expected life of mine of the project. Parameters affecting the project cash flow are: production schedule, revenues, royalties, sustaining and initial capital requirements, operational costs, working capital, financing costs, mine closure costs and Senegalese fiscal regime.

The after-tax financial analysis and results are based on a NPV, the IRR, the payback period which is initiated at project approval (except for payback which refers to an amount of time after commercial production is declared) and the all-in sustaining costs (AISC) which is a gold industry standard in benchmarking costs per ounce of gold.

The costs were evaluated in United States Dollars. All amounts are in constant 2018 dollars, no provision is made for inflation nor increase in gold price. All cash flows are estimated on the project base solely and are excluding debt financing and a discount rate of 6% was used for the calculation of the NPV.

At the award of the ''permis d'exploitation'', a mining company will be created where the Government of Senegal will hold a 10% free carrying interest. IAMGOLD will hold the other 90% remaining interests. Hence the Senegalese government will receive 10% of all declared divided produced by the Boto mining company.

Input data was provided from a variety of sources, including the various consultants' contributions to this report, pricing obtained from external suppliers and contractors, and exchange rates and project specific financial data such as the expected project taxation regime.

The life of mine capital cost for the project is estimated at $320.5M, with an initial capital expenditure of $254.4M. Table 1.7 presents a summary of the production information on which the financial model is based, while the summary of the financial results is presented in Table 1.8.

Table 1.7 Production Summary

	Value
Ore milled	35.1 Mt
Total tonnes mined	239.4 Mt
Average head grade	1.71 g/t Au
Contained gold in material	1 926 Moz
Total gold produced	1 724 Moz
Average gold recovery	89.5%
Production life (processing)	12.8 years
Nominal annual processing rate	2.75 Mtpa

Table 1.8 After-tax Financial Results

AISC	$753/oz Au
IRR	23.0 %
NPV (6%)	$260.9 M
Payback	3.4 years

1.15 Conclusions and Recommendations

Based on the work undertaken, as summarized in this Technical Report, and the individual Qualified Persons conclusions listed in Section 25, the FS has identified the Project as a viable and attractive development opportunity.

Following board approval, it is recommended that IAMGOLD commence implementation of the Project in line with the preliminary implementation plan and schedule developed during the FS, and committing to the capital expenditure presented in Section 21.

2.0 INTRODUCTION

This technical report was prepared by Lycopodium for IAMGOLD to summarize the results of a FS of the Project located in eastern Senegal. This report was prepared in compliance with the disclosure requirements of the Canadian National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1.

IAMGOLD is an intermediate gold producer with four operating gold mines and several exploration projects located in North and South America, and Africa. AGEM, a wholly owned subsidiary of IAMGOLD, controls the Property.

The Property is located in the southeast corner of Senegal within the Daorala-Boto Exploration Permit. The exploration permit is made up of two non-contiguous properties, the first one is Boto and the second property is the Daorala property to which no mention is made in this report. The Project was the subject of a prefeasibility study in 2017.

AGP, KP, AG and IAMGOLD provided input to the report and the individuals presented in Table 2.1, by virtue of their education, experience and professional association are considered Qualified Persons (QPs) as defined in NI 43-101 for this report. The QPs meet the requirement of independence as defined in NI 43-101.

2.1 Units

All the units of measurement used in this document are in metric units and all currencies are expressed in United States dollars, unless otherwise stated. The gold metal content is expressed in Troy ounces ("oz"), where 1 ounce = 31,1035 g. All material tonnes are expressed as dry tonnes unless stated otherwise.

2.2 Qualified Persons

The list of qualified persons responsible for the preparation of this technical report and the sections under their responsibility are provided in Table 2.1.

Table 2.1 Qualified Persons

Qualified Persons Responsible for the Preparation of this Technical Report						
Qualified Person	**Position**	**Employer**	**Independent of IAMGOLD**	**Date of Last Site Visit**	**Professional Designation**	**Report Sections**
Neil Lincoln	VP of Business Development & Studies	Lycopodium Minerals Canada Ltd	Yes	24 October 2017	P.Eng	1.6, 1.10-1.11, 1.13, 1.15, 2, 3, 13, 17, 18.1-18.9, 19, 21 (except 21.2, 21.4.2), 24, 25.3, 26.4,-26.5, 27
Rob Thomas	Engineering Geologist	Absolute Geotechnics	Yes	11-15 December 2017	MAusIMM (CP)	16.3
Gordon Zurowski	Principal Mining Engineer	AGP Mining Consultants	Yes	12-13 December 2017	P.Eng	1.8, 1.9, 15, 16 (except 16.3), 21.2, 21.4.2, 25.2, 26.3
Paul Daigle	Senior Geologist	AGP Mining Consultants	Yes	11-15 December 2017	P.Geo.	1.4, 1.5, 1.7, 5, 7-12, 14, 25.1, 26.1
Reagan McIsaac	Senior Engineer	Knight Piésold	Yes	Did not visit site	P.Eng	18.10-18.12, 26.2
Martin Lanctôt	Project Manager	IAMGOLD	No	December 10-15, 2017 February 15-16 and July 5, 2018	ing	1.1-1.3, 1.12, 11.14, 4, 6, 20, 22, 23

2.3 Visits to the Site

Neil Lincoln visited the site on 24 October 2017, where he observed the drill core at the core storage area, and walked the site of the proposed mine, process plant, infrastructure and access road.

Rob Thomas visited the site from 11 to 16 December 2017 to inspect the site, review a selection of available drill core, undertake QA/QC of geotechnical data collection and observe drilling practices.

Gordon Zurowski visited the site 12-13 December 2017, where he observed the drill core at the core storage area, and walked the site of the proposed mine, process plant and infrastructure.

Paul Daigle visited the site 11-15 December 2017, where he inspected the core logging, sampling and drill core storage facilities, checked coordinates for drill hole collars to survey data; and reviewed drill core logs against selected drill core.

Reagan McIsaac did not visit the site.

Martin Lanctôt visited the site on December 10-15, 2017, February 15-16 and July 5, 2018 and inspected the proposed open pits, process plants, TMF, infrastructure and access road.

3.0 RELIANCE ON OTHER EXPERTS

The information, conclusions, opinions and estimations contained in this document are based on:

- The information provided to IAMGOLD and the authors as of the date of this report.
- The assumptions, conditions and qualifications stated in this report.
- The data, reports and any other information provided by third parties to IAMGOLD.

The authors verified the available data and technical reports and considered that the quality of data was satisfactory and met the requirements for the production of a technical report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Property is located in the southeast corner of Senegal within the Daorala-Boto Exploration Permit. The exploration permit is made up of two non-contiguous properties, i.e. the Boto Property and the Daorala property to which no mention is made in this report.

The Property is located:

- On 1:200,000 Mapsheet Saraya ND-29-I.
- At approximately 12°28' North and 11°23' West.
- At approximately 241,000 E; 1,378,000 N, Zone 29P (WGS 84 datum) Universal Transverse Mercator (UTM) coordinates.
- At approximately 700 km south southeast of Dakar, the nation's capital; and approximately 835 km by road.
- At approximately 90 km south-southeast of Kédougou; and approximately 135 km by road.
- At the Region of Kédougou.
- In the Department of Saraya (Senegal Government website, 2018).
- In the Arrondissement of Fongolembi (Senegal Government website, 2018).
- In the Commune of Madina Bafé (Senegal Government website, 2018).
- Adjacent to the Senegal-Mali-Guinea borders.
- Adjacent to the west of the Falémé River (left bank).

Figure 4.1 and Figure 4.2 show the Property location in Senegal.

Figure 4.1 Location Map

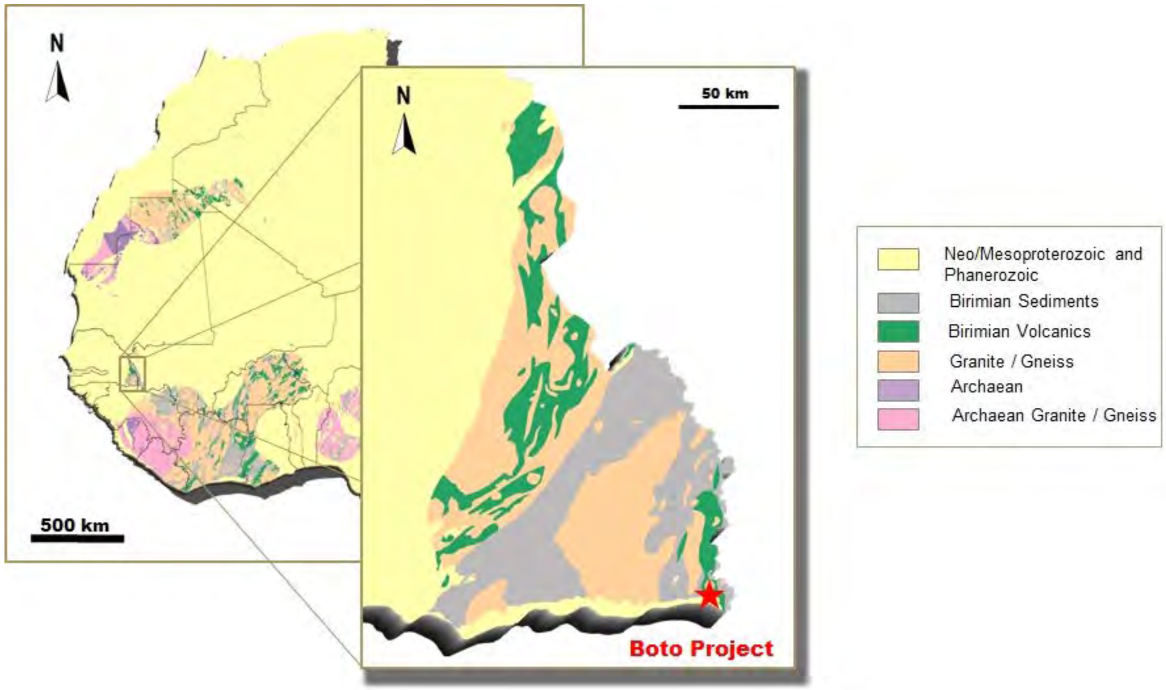
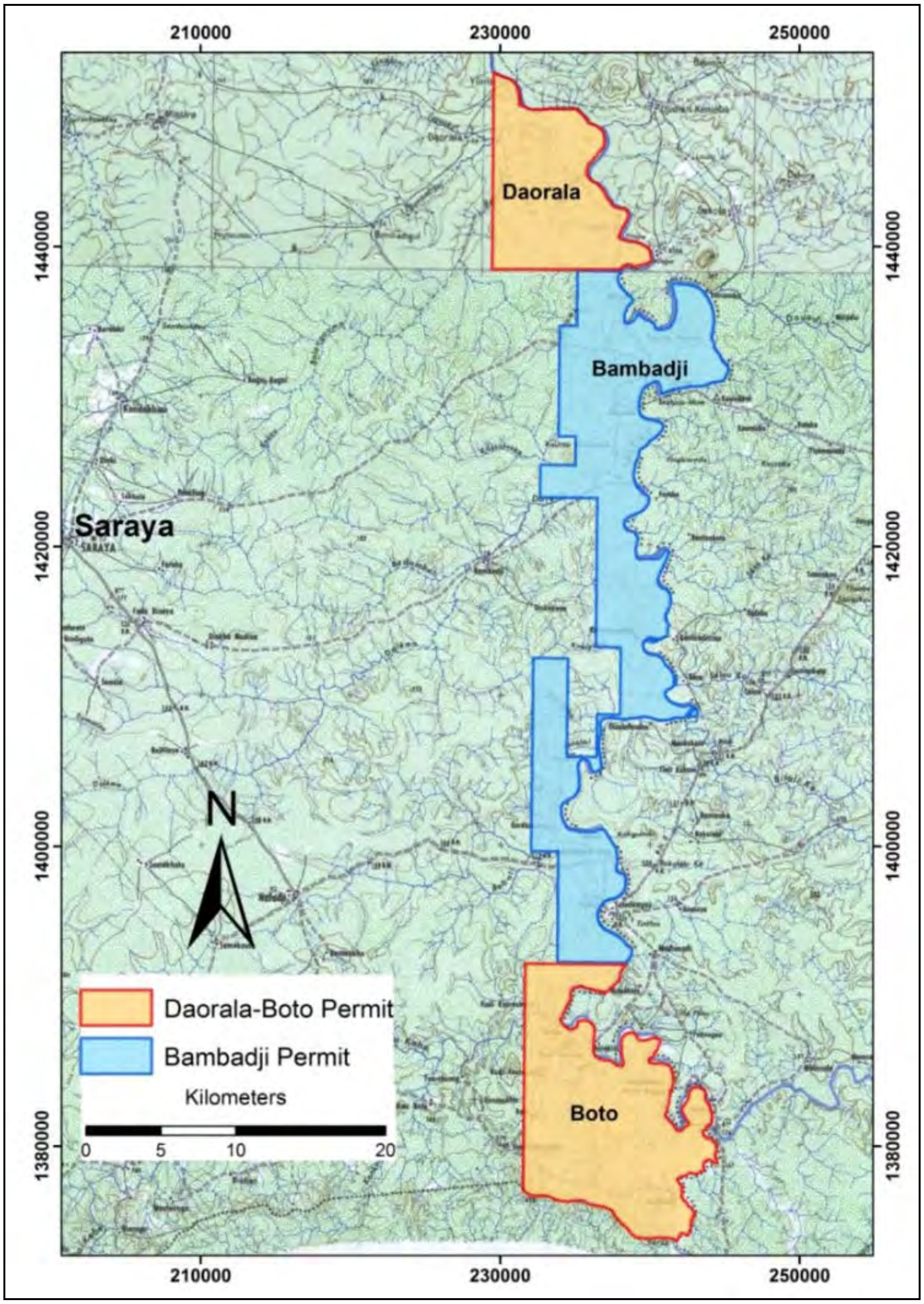


Figure 4.2 Property Location Map



4.2 Property Description

The Property is the southern sector of the Daorala-Boto exploration permit. The Exploration permit (Arrêté 13984 MMI/DMGrs is held by AGEM. The Daorala-Boto permit covers a total area of approximately 236 km^2 between the Boto sector and the Daorala sector. The two sectors are bounded to the east by the Falémé and Balinko Rivers which form the border with Mali and Guinea.

On 8 August 2017, the government of Senegal granted the renewal of the Daorala-Boto exploration permit for a period of two years.

Table 14.1 summarizes the details of the exploration permit and Table 14.2 lists the coordinates for each sector of the exploration permit. Figure 4.3 and Figure 4.4 show the boundaries of the Boto and Daorala sectors, respectively.

Table 4.1 Summary of the Daorala-Boto Property

Exploration Permit	Sector	Area (km^2)	Expiry Date
Arrêté 13984 MMI/DMGrs	Boto	148	4 March 2019
	Daorala	88	

Table 4.2 List of Coordinates for the Daorala-Boto Property

Points	Longitude (W)	Latitude (N)
Boto		
AF	11°28’11’’	12°35’00’’
Z	Senegal-Mali border	12°35’00’’
AD	Triple Point border Sénégal-Mali-Guinée	Triple Point border Sénégal-Mali-Guinée
AE	11°28’11’’	Senegal- Guinea border
Daorala		
AE	11°29’39’’	13°00’00’’
AD	Senegal-Mali border	13°07’07’’
Y	Senegal-Mali border	13°00’00’’

Figure 4.3 Boto Property Location Map

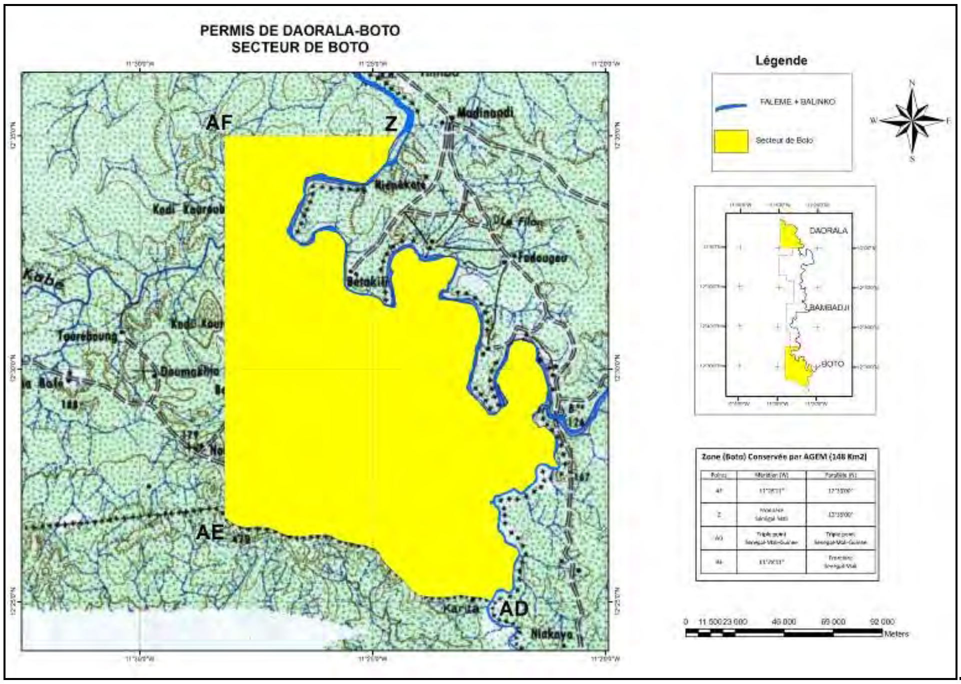
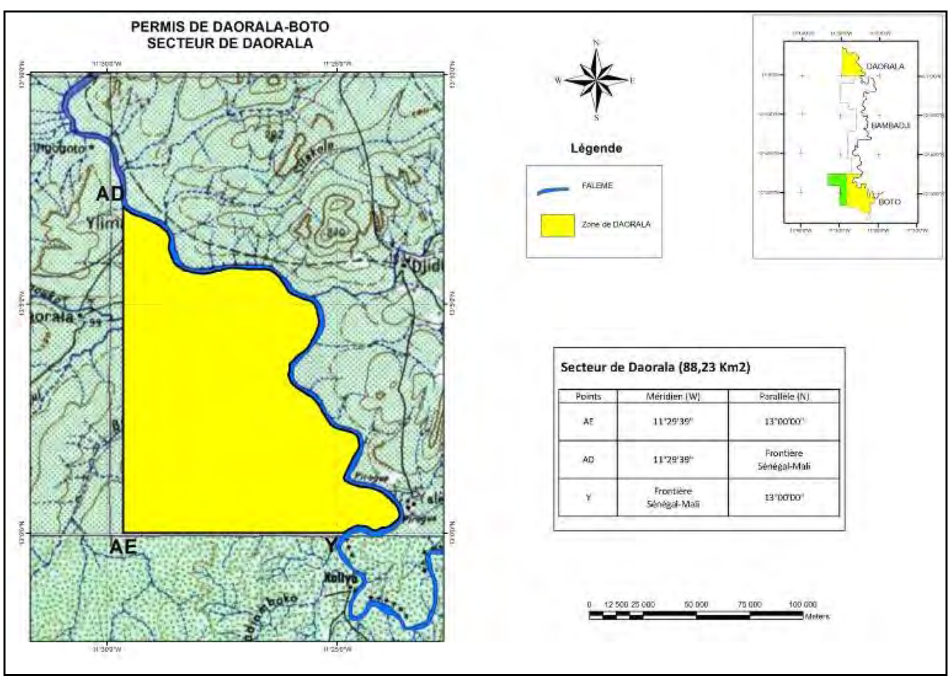


Figure 4.4 Daorala Property Map



4.3 Bamadji Joint-Venture (IAMGOLD/Randgold)

Between the Daorala-Boto Exploration Permit is the Bamadji property (see Figure 4.2). The mineral rights to this property are also held by Randgold Senegal. The Bamadji property is not subject to this report and is only described below for completeness.

The Bamadji property is a joint venture between AGEM (IAMGOLD) and Randgold Resources Ltd. (Randgold), an Isle of Jersey, UK, based mining company. Randgold holds the mineral rights to the Bamadji property which consists of a single permit, 343 km^2. The Bamadji property is situated between the Daorala and the Boto areas in southeast Senegal and is situated adjacent to the west of the Loulo gold mine in Mali and operated by Randgold. The distribution of the share capital is 65% Randgold and 35% AGEM.

As of 3 March 2017, Randgold became the owner of the Exploration Permit by maintaining the rights of AGEM in accordance with the joint venture agreement.

4.4 Stratex Option Agreement (IAMGOLD/Stratex)

On March 1, 2018, Stratex International PLC (Stratex) and IAMGOLD announced that an option agreement was signed for the Dalafin gold project in Senegal. The option agreement was approved by the government on 26 March, 2018. Stratex, is an AIM-listed exploration and development company focused on gold projects in Turkey and Africa.

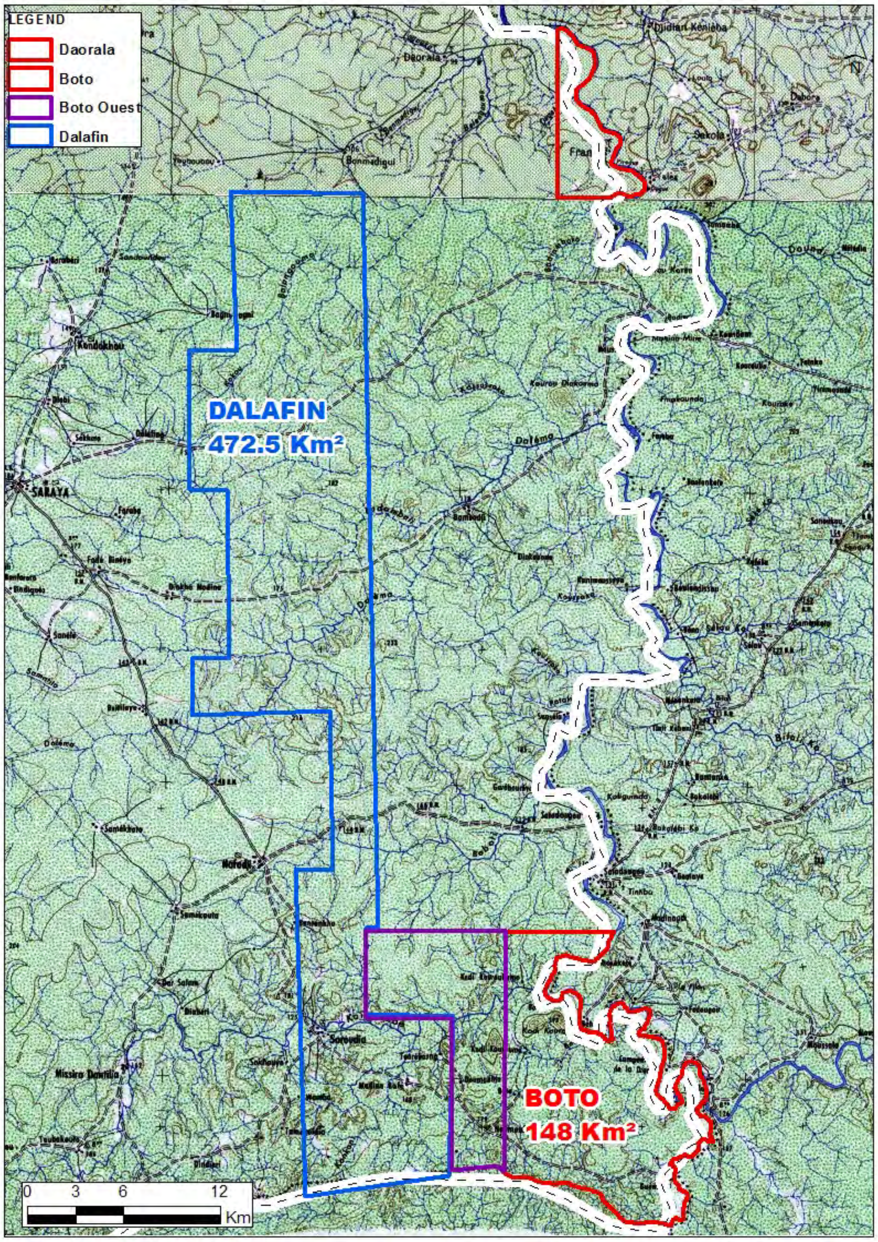
The Dalafin project consists of a single exploration permit, 472.5 km^2 is located west and adjacent to the Boto West permit, in the south, and extends approximately 70 km north, east of the town of Saraya (Figure 4.5).

The option agreement was signed between AGEM, IAMGOLD's Senegalese subsidiary, and Stratex EMC SA (Stratex EMC), Stratex' 85% owned Senegalese subsidiary. Under the terms of option agreement:

- AGEM will have the right to acquire an initial 51% interest in Dalafin by expending $4 M over 4 years at the project (the 'First Option').
- Subject to the First Option being exercised by AGEM, AGEM and Stratex EMC may agree to form a joint-venture ('JV') company for the management of Dalafin.
- AGEM has the option to increase its interest by a further 19%, to 70%, by expending a further $4 M at the project over the subsequent 2 years (the 'Second Option').
- Thereafter, AGEM and Stratex EMC will be required to contribute on a pro rata basis towards the Project, or will be diluted. Should either party be diluted below 10%, their interest will convert to a 2% Net Smelter Returns ('NSR') royalty (the 'Royalty') on production from Dalafin, of which AGEM will retain the right to buy-back 0.5% of the Royalty for consideration of $0.5 M (thereby reducing the Stratex EMC royalty to 1.5%).
- Subject to governmental approval.

IAMGOLD will focus exploration work initially on the Madina Bafé prospect, in the south, which is contiguous with IAMGOLD's Boto gold project.

Figure 4.5 Dalafin Property Map



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Property is most easily accessed from Dakar by paved highway via Tambacounda-Kédougou-Saraya to eastern Senegal and from there, maintained dirt roads to the Project. The Property may be accessed by road from Dakar via:

- Highway N1 east for approximately 470 km to Tambacounda to join.
- Highway N7 southeast for approximately 230 km to Kédougou to join.
- The Kédougou-Saraya Road for approximately 60 km to Saraya to join.
- A secondary dirt road for approximately 80 km to the village of Noumoufoukha and the Project camp.

The drive from Kédougou to the project camp is typically 2.5 hrs. During the site visit in December, it was noted that several sections of the dirt road from Saraya to camp were undergoing construction; installing fords, culverts, and small bridges across creek beds.

The nearest asphalt airfield to the Property is in Kédougou with a 1,800 m airstrip. This airfield is not serviced by regular scheduled flights, however, there are private airline companies based in Dakar that may be chartered to this airfield. In February 2016, the government of Senegal certified an 800 m laterite airstrip roughly 3 km southwest from the Boto Exploration camp. The airstrip is currently unusable due to rain damage during the wet season and will require a renewal of the certification.

5.2 Climate

The Property is situated in the climate region of the Sudan-Sahel of Africa and is classified as a subtropical continental climate (Csa Köppen classification). This climate zone is characterized by two seasons: a rainy season from June to October, and a dry season from October to May.

The weather is generally hot and dry from February to June (daytime temperatures 35°C to 45°C), hot and wet from June to November (daytime temperatures 30°C to 40°C); and mild and dry from December to February (daytime temperatures 20°C to 25°C). The Harmattan is a seasonal hot dry wind that blows from northeast or east from the Sahara desert during the dry season. This wind usually carries large amounts of dust from the Sahara out over the Atlantic Ocean.

Exploration activities may be conducted all year round. However, during the wet season, the Kolia Kabe River situated 14 km by road to the northwest of Boto Exploration Camp, floods and cuts off the road access at the Saroudia Bridge.

5.3 Infrastructure and Local Resources

There is very little infrastructure on, or to, this property. The Boto Exploration Camp consists of permanent brick structures for rooms, toilets, kitchen, and offices. Drill core storage and garages are sheltered open air structures. At the New Camp, approximately 3 km west of Malikoundi/Boto 2 deposit, there is also sheltered open air core storage facilities. Both the Boto Exploration Camp and New Camp are fenced and have 24-hour security.

There is no electricity from the national grid to this area of the country. Electricity is supplied to the Boto Exploration Camp by diesel generators on site well with a water treatment plant. There is some cellular telephone coverage which is supplied to this area of Senegal, and on the Project, by Senegal-based cellular towers and by cellular towers in neighbouring Mali.

All equipment, supplies, and fuel are transported by road to the project site. Most supplies, consumables, and fuel are sourced either from Kédougou or Dakar depending on availability.

The village of Guémédji, and some surrounding villages, are a source of unskilled workers and fresh produce. Skilled and professional workers are sourced from Dakar.

5.4 Physiography

The southeast of Senegal is situated in the foothills of the Fouta Djallon, a mountainous region in west central Guinea, and situated to the south of the project area. The Property lies between 100 m and 300 m above sea level with generally low to moderate relief consisting of broad lateritic plateaus and eroded valleys. The Falémé iron deposits are visible as prominent hills to the south of the Property.

The project area is situated in the south of the Senegal River watershed which drains northwest and west to the Atlantic Ocean. The Boto deposits are located in proximity to the west of the Falémé River and the Balinko River.

The vegetation is typical of a tropical forested savannah, with scattered trees (including baobab), scrub brush, elephant grass, and bamboo. Trees are more abundant along rivers and creeks as gallery forests and around lateritic plateaus that have been broken down by erosion.

6.0 HISTORY

Prior to 1994, there is no known or recorded systematic mineral exploration carried on the Property.

The first exploration activities were carried out by Anmercosa from 1994 to 1996. From 1997 to 1998, Ashanti Goldfields, completed further exploration activities in a joint venture with AGEM. From 1999 to present, AGEM has conducted all succeeding exploration activities on the Property. This work is described in Section 9 of this report.

6.1 Anmercosa Exploration, 1994-1996

From 1994 to 1996, Anmercosa, conducted regional exploration activities including what is now the Project. These activities included airborne geophysical surveys along with regional and local geochemistry. Table 6.1 summarizes the exploration activities carried out by Anmercosa.

Table 6.1 Summary of Exploration Activities by Anmercosa

	Exploration Activities	Details
Anmercosa Exploration de 1994 - 1996	Airborne geophysical surveys (magnetic, radiometric and VLF)	S.O.
	Regional geochemistry	7,591 soil samples 22,740 termite mound samples 406 stream sediment samples
	Detailed geochemistry	7,469 soil samples 3 rock samples

6.2 Ashanti Goldfields Corporation: 1997- 1998

After acquiring the property from Anglo American, Ashanti Goldfields continued to focus on the acquisition of the geochemical data and conducted some preliminary trenching in 1997 and 1998. Table 6.2 summarizes the work performed by Ashanti Goldfields.

Table 6.2 Summary of Exploration Activities by Ashanti Goldfields

Company and Period	Exploration Activity	Details
Ashanti Goldfields 1997 - 1998	Detailed Geochemistry	1,941 soil samples 998 termite mound samples 8 stream sediment samples 79 rock samples
	Trenches	2 trenches

There have been no known previous mineral resource estimates or production on the Property based on historical exploration completed between 1994 and 1998.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

The following information is taken from Lycopodium (2017) and RPA (2013).

7.1 Regional Geology

The Project is located in the WAC, in the southeastern part of the Early Proterozoic formation of the Kédougou-Kéniéba inlier, which covers the eastern part of Senegal and western Mali.

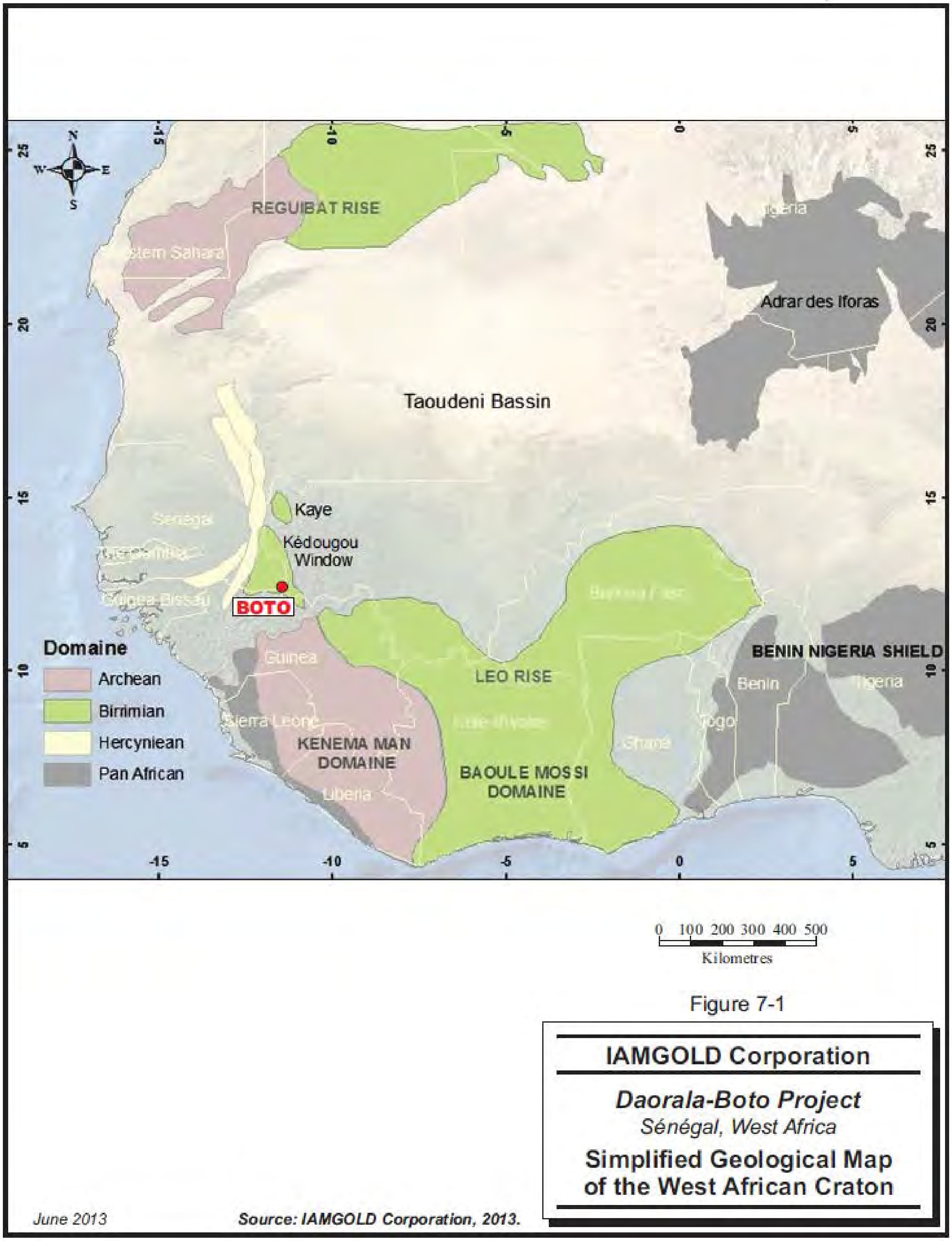
The bedrock of the WAC is exposed inside the Léo-Mann shield, the Reguibat Shield, and the inliers of a Kédoug ou-Kéniéba and Kaye. It comprises an Archean nucleus (3.0-2.7 Ga, Camil et al. 1983) covered by Lower Proterozoic rock (2.1 Ga, Abouchami et al. 1990; Hirdes et al., 1996) (Figure 7.1).

In the southern part of the craton, Lower Proterozoic Greenstone Lands are described as Birimian based on the Kits (1928) in the Birim River Valley of Ghana. These terranes have undergone the effects of Eburneen Orogeny (a major tectonic event to the 2.1 Ga) and are found throughout the inlier of Kédougou-Kéniéba and the Leo-Man Shield, except in the extreme western parts where Archean terranes outcrop.

Birimian terranes include linear volcanic belts and alternating sedimentary basins in a northeasterly direction that are separated by granite intrusions and past gneiss. Rocks are generally metamorphosed in green shale facies, although amphibolite facies are locally observed in metamorphic granitic intrusions (Boher et al., 1992).

The Kédougou-Kéniéba inlier, where the Project is located, is the exposure in the far west of the Birimian. The Kédougou-Kéniéba inlier is bounded on the west side by the Hercynian Mauritanide belt; and on all other sides, it is unconformably overlain by the underformed upper Proterozoic sediments and the Early Phanerozoic rock of the Taoudeni, Tindouf, and Volta basins (Boher et al., 1992; Villeneuve and Cornea, 1994).

Figure 7.1 Simplified Diagram of the Geology of the West African Craton



Lithostratigraphic Subdivisions

The Birimian terranes of the Kédougou-Kéniéba inlier were first divided into three groups facing north to northeast, spread from west to east: Mako, Dialé, and Daléma. On the basis of their similar lithology, the Dialé and Daléma groups were later combined into the Dialé-Daléma Group (Bassot, 1966, 1987).

Mako Group

The Mako group is a volcano-plutonic belt composed primarily of volcanic rocks with some sub-volcanic intrusions and granitoids, and minor sedimentary rocks. It consists predominantly of tholeiitic and calc-alkaline volcanic rocks with interbedded volcanoclastic sedimentary rocks and intercalations of fluvio-deltaic sedimentary rocks (Kéniebandi Formation) equivalent to what Tarkwaian described in Ghana (Davis et al., 1994). Typical lithologies include pillowed basalts with minor intercalated volcanoclastic rocks, high-Mg basalts, pyroxenites, sub-volcanic intrusions, and granitoids. The volcanic assemblage is dated between 2,160 Ma and 2,197 Ma. In the eastern portion, calc-alkaline series and detrital sediments are associated with volcano-sedimentary rocks (Boher, 1992; Dia et al., 1997, Bassot, 1987; Dia et al., 1997; Dioh et al., 2006). To the east of the Mako Group is the Daléma, an especially sedimentary group, which is separated from the Mako Group by a regional scale lineament called the "Main Transcurrent Zone" (MTZ).

Dialé—Daléma Group

The Dialé-Daléma Group consists mainly of sedimentary rocks with subordinate volcanic rocks. Typical lithologies include folded sandstones and siltstones, interbedded with calc-alkaline, and ash-and-lapilli tuffs (Bassot, 1987; Hirdes and Davis, 2002). This group is sub-divided into two series distinguished by their relative proportion of chemical and detrital sedimentary rocks.

The Dialé-Daléma Group has a higher proportion of chemical sedimentary rocks; typical lithologies are, from the base to the top, crystalline limestone and dolomitic marbles, greywacke, arenite sandstone, and schist (Milési et al., 1989). According to Schwartz and Melcher (2004), the Dialé-Daléma Group has the most extensive occurrences of carbonate in the Birimian. The Dialé-Daléma group was intruded by coalescing biotite-bearing granitic plutons. This sequence is overlain by distal turbidites, partially tourmalinized in the upper part, and carbonate-bearing fine-grained sedimentary rocks.

The Senegalese and Malian sides of the inlier use different terminologies for the same geological formations; since both sides of the inlier are mentioned in this section, Table 7.1 is provided for purposes of simplification.

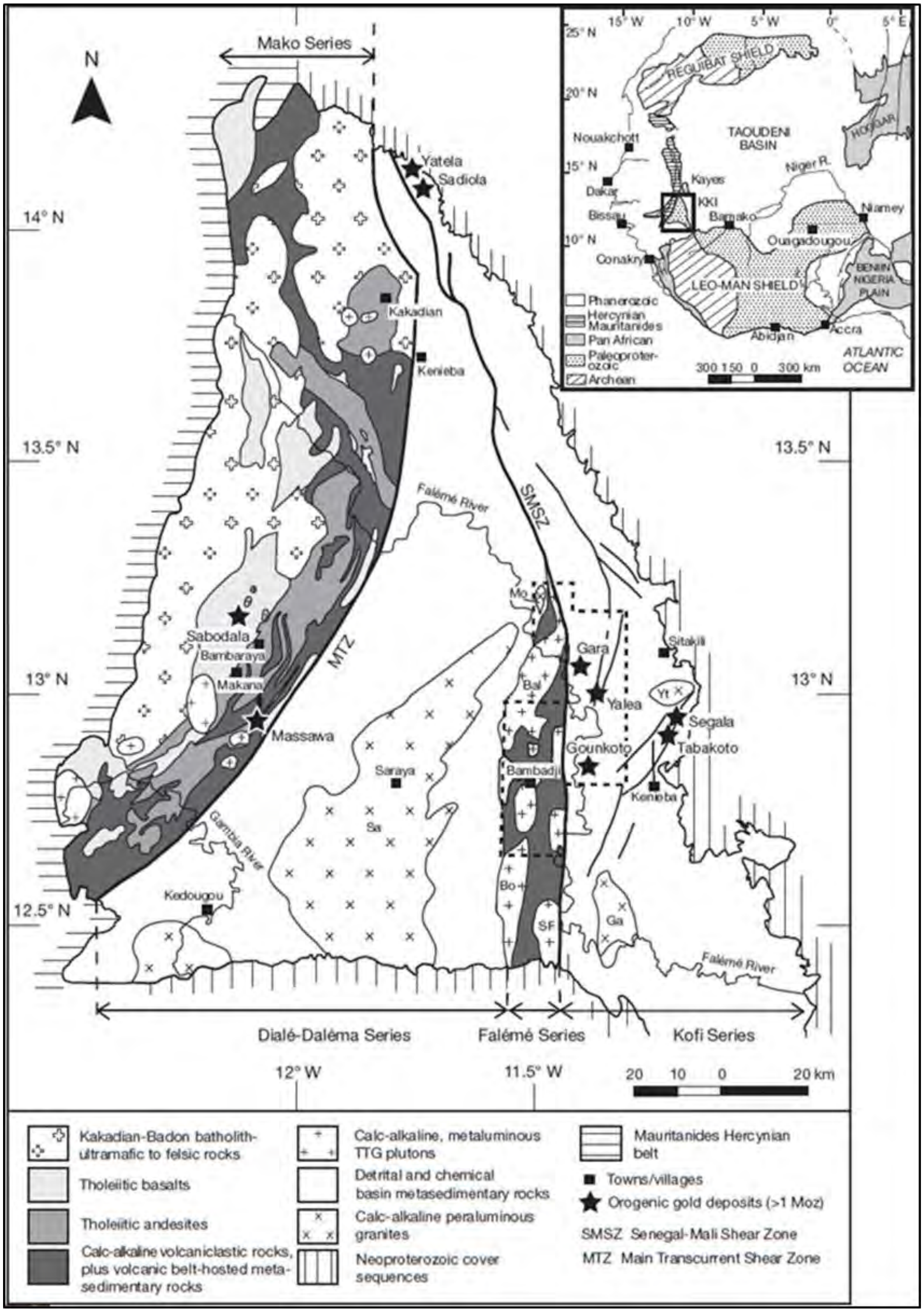
Table 7.1 Senegalese and Malian Terminology for the Birimian Formation

Basses (1966)	Western Mali (1989)	Lithologies
Mako Group	Saboussiré Training	Mafic Volcanics, volcano-sedimentary and sedimentary rocks
Dialé Group	Kéniebandi Training	Mainly sedimentary rocks, a bit of volcanics Possibly a late cup Mali
Daléma Group	Kofi Training	Mainly sediments, some volcanics Faleme Calcoalcalin Complex

One of the important features of the Dialé-Daléma Group is the north-south oriented lineament, known as the Senegal-Mali Shear Zone (SMSZ), located in the eastern part of the inlier (Figure 7.2). Early geological interpretations extended the Dialé-Daléma Group over this regional lineament to the east in the Malian part of the inlier. Based on new observations however, a new volcano-plutonic belt has been outlined in the southeast portion of the inlier. It is called the Falémé Series and separates the Dialé-Daléma Group in the west from the Kofi Series in the east. The Kofi Series is restricted to the east of SMSZ and consists of sandstones, argillites, and platform carbonates intruded by syntectonic S-type peraluminous biotite-bearing granites. The detrital sedimentary rocks at the Loulo Deposit found in the Kofi Series, were dated between 2,093 ±7 Ma and 2,125 ±27 Ma (Boher et al., 1992).

In summary, the inlier can be structurally described as consisting of two volcano-plutonic belts oriented north to northeast (the Mako Series and the Falémé Series), and two intervening sedimentary basins called The Dialé-Daléma Group and the Kofi Series (Figure 7.2).

Figure 7.2 Litho-structural Diagram of the Inlier Kédougou-Kéniéba



Tectonic Setting

The Birimian rocks of the Kédougou-Kéniéba inlier have been affected by a polycyclic deformation and metamorphic history related to the Eburneen Orogeny (2.2 Ga to 2.0 Ga). Three major deformation phases were identified: a collisional phase (D1) associated with the initial accretion of the Birimian, and two transcurrent phases (D2-D3) associated with the formation regional-scale north-south shear zones.

At the scale of the Kédougou-Kéniéba inlier, the D2-D3 deformation is clearly related to the two regional transcurrent ductile structures (i.e.: the north-east trending MTSZ, located between Mako and Dialé-Daléma, and the SMSZ located in the eastern part of the inlier - Ledru et al., 1991; Gueye et al., 2007), as well as with subsidiary structures (Bassot and Dommanget et al., 1986; Ledru et al., 1991; Milési and Al., 1989, 1992; Dabo and Aife, 2010).

D1 features include a penetrative cleavage (S1) that has transposed bedding (S0), a stretching lineation (L1) and an isoclinal syn-foliation folding (Figure 7.1) with various trends (north-south, northeast-southwest to east-west, or northwest–southeast).

The characteristics of D2 include upright or slightly overturned folding F2 and S2 cleavage, to the southeast which is parallel to the F2 axial plane (Figure 7.2), and usually marked by dissolution planes, a stretching lineation (L2) marked by stretched conglomerate clasts, and/or metamorphic mineral lineation. The D2 phase is associated with lateral left-shifting strike-slip faults trending north-south to northwest–southeast and major granite emplacement (Pons et al., 1992).

The S2 fabric, which typically transposes and overprints bedding (S0) and S1 structures, is the most obvious deformational feature of the region (Ledru et al., 1991; Pons et al., 1992). It is generally steep with statistical trends close to N30E, although it is overturned to become north-south near the SMSZ. D2 is also associated with the emplacement of the Kakadian (2,199 ±68 Ma) and Saraya (1,973 ±33 Ma) granitic batholiths (Pons et al., 1992; Gueye et al., 2007).

D3 is marked by northeast–southwest strike-slip faults with associated folding (Pons et al., 1992; Feybesse and Milési, 1994).

The tectonic history of the region can be summarized as follows:

- Early Proterozoic:
  - deposits of clastic, pelitic, greywacke, carbonate, and volcano-sedimentary units
- Eburnean Orogeny:
  - metamorphism (greenschist facies) of sediments to form quartzites, schists, marbles, etc. (Birimian D1, D2, D3)

- Late Proterozoic:
  - uplift, erosion, and peneplanation of Birimian rocks
- Late Proterozoic to Carboniferous:
  - deposit of clastic sediments (mostly sandstones) of the Taoudeni Basin

7.2 Local Geology

The Boto-Daorala and Bambadji concessions lie mainly within the Falémé Series, a volcanic-plutonic belt that is wedged between The Dialé-Daléma Group and the Kofi Series, and separated from the latter by the SMSZ. It can be chronologically correlated with the Mako Series. The most eastern part of the Property is in the Kofi Series.

Typical lithologies of the Falémé "volcanic belt" include carbonate rich sedimentary rocks, a small amount of basalt and andesite, rare rhyolites, and syn-tectonic granitoids. A series of calc-alkaline dominated granitoids occur within this "granite-volcanite belt", including the Balagouma, Bambadji, Boboti, and Falémé granitoids. The Boboti and South Falémé granitoids have emplacement ages of 2,080 ±1 and 2,082 ±1 Ma, respectively (Ndiaye et al., 1997; Hirdes and Davis, 2002). According to Lawrence et al. (2013), the Kofi Series comprises a sequence of shelf carbonates, limestone clastic rocks, turbidites, and impure sandstones of tourmaline quartz, feldspathic sandstones and greywackes with argillite interlayer.

The granites of Balagouma and Boboti are spatially associated with the Falémé iron deposits believed to be of skarn type. According to Schwartz and Melcher (2004), the iron deposits are genetically linked to the metasomatism related to the emplacement of these granite plutons. They were described as endoskarns and exoskarns hosted in calcitic and hematite-bearing bodies. The topographic peaks of these iron hills are the most apparent landmarks of the Falémé Volcanic Belt.

7.3 Property Geology

At Boto, the material near the surface consists of a layer of regolith which is varying in thickness and includes lateritic plateaus. Few rocky outcroppings are visible in the property; the banks of streams and rivers serve as the main source for geological observations. Only drilling can provide a detailed knowledge of the geology below surface. Drilling data and geological interpretation were used to create a regional representation of Boto geology.

Boto can be divided into three north trending litho-structural domains (020˚ N) that are well delineated in both induced polarization (IP) and magnetic surveys. From west to east, the three domains are:

- Western Flyschoid Domain.
- Central Deformation Corridor.
- Eastern Siliciclastic Domain.

The western domain is dominated by a sequence of flyschoid turbidites, black shales (or graphitic pelite), carbonate rocks, minor volcanics (mainly basalt with subordinate rhyolite and pyroclastic breccia or agglomerate), and dioritic intrusions. The Boto 5 deposit is located along the contact between this domain and the central deformation corridor.

The eastern domain is dominated by a detrital assemblage composed of greywacke and sandstone (± quartzite), called Guémédji sandstone. It is thought that these sandstones/wackes are part of the Kofi Series which is very present in the Malian portion of the inlier.

Between the west and east domains is a highly deformed north-trending domain (020˚ N) that is well defined in magnetic geophysical data. It is likely this highly deformed domain corresponds to a regional scale structural corridor that branches from the SMSZ. Lithologically, it is composed of fine schistose sediments that are carbonaceous in places, locally referred to as the "Pelite Unit", and fine laminated sediments (± carbonates) that subtly grade into an impure marble, locally called the "Cipolin Unit".

Generally, the geological units in the three domains strike 020˚ N with various dips. In the western domain, the dip is usually between 70˚ W and sub-vertical, while in the eastern domain, lithological units usually dip less than 60˚ W. Intrusive rocks are found within all three domains and include diorite, dolerite, granite, and granodiorite. Various volcanic rocks have also been observed in drill core, including andesitic vesicular lava, basalt, andesite, and rhyolite. Pyroclastic rocks including lapilli tuffs, ash tuffs, and agglomerates have also been documented. It should be noted however, that the local term agglomerate facies could have a tectonic origin rather than a pyroclastic one. Figure 7.3 demonstrates examples of some of the characteristic lithologies at Boto.

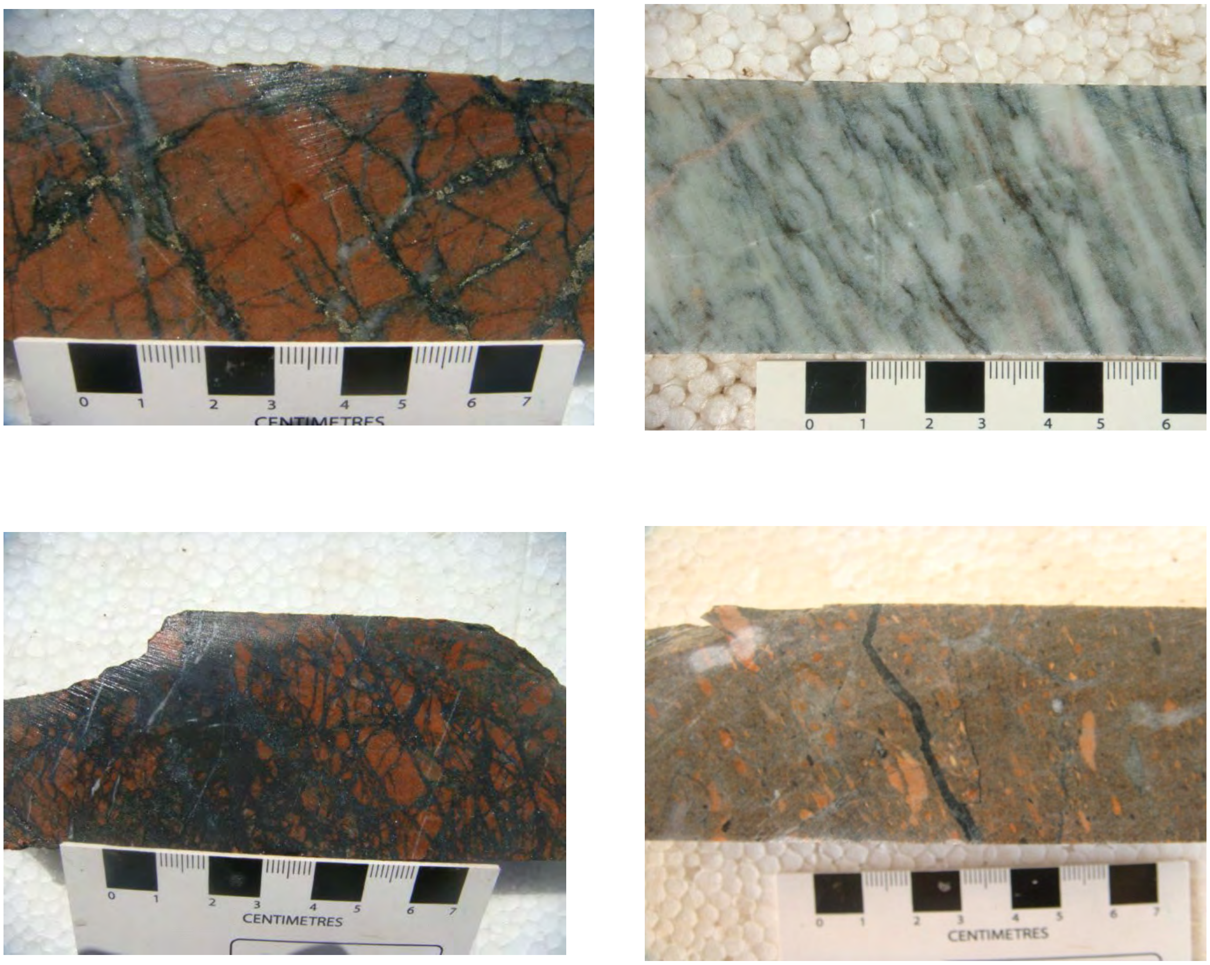
The known gold deposits at the Project occur on the margins of the central deformation corridor (Figure 7.4). Boto 5 lies along the western boundary (contact with carbonaceous turbidites) while Malikoundi/Boto 2, Boto 4, and Boto 6 lie along the eastern boundary (contact with the Guémédji sandstone).

Geochemical anomalies at Boto are strongly correlated with the structural trends described above. The Lelou and Guémédji geochemical domains correspond with the western flyschoid domain and the central deformation corridor, respectively.

The Lelou trend, which encompasses two surface geochemical anomalies - Boto 1 and Boto 3, remains a prospective exploration target and is yet to be tested. The Guémédji trend hosts Malikoundi/Boto 2, Boto 4, Boto 6 and Boto 5 (Figure 7.5).

Figure 7.3 Characteristic Lithologies of the Boto Project

A B



C D

A) and C) Cracks in altered albite quartzite (Guémédji sandstone), quartz-tourmaline-filled fractures-chlorite and pyrite ± magnetite; B) Unclean striated Marble; D) Agglomerate with stretched fragments.

Figure 7.4 Litho-structural Map of the Boto Project

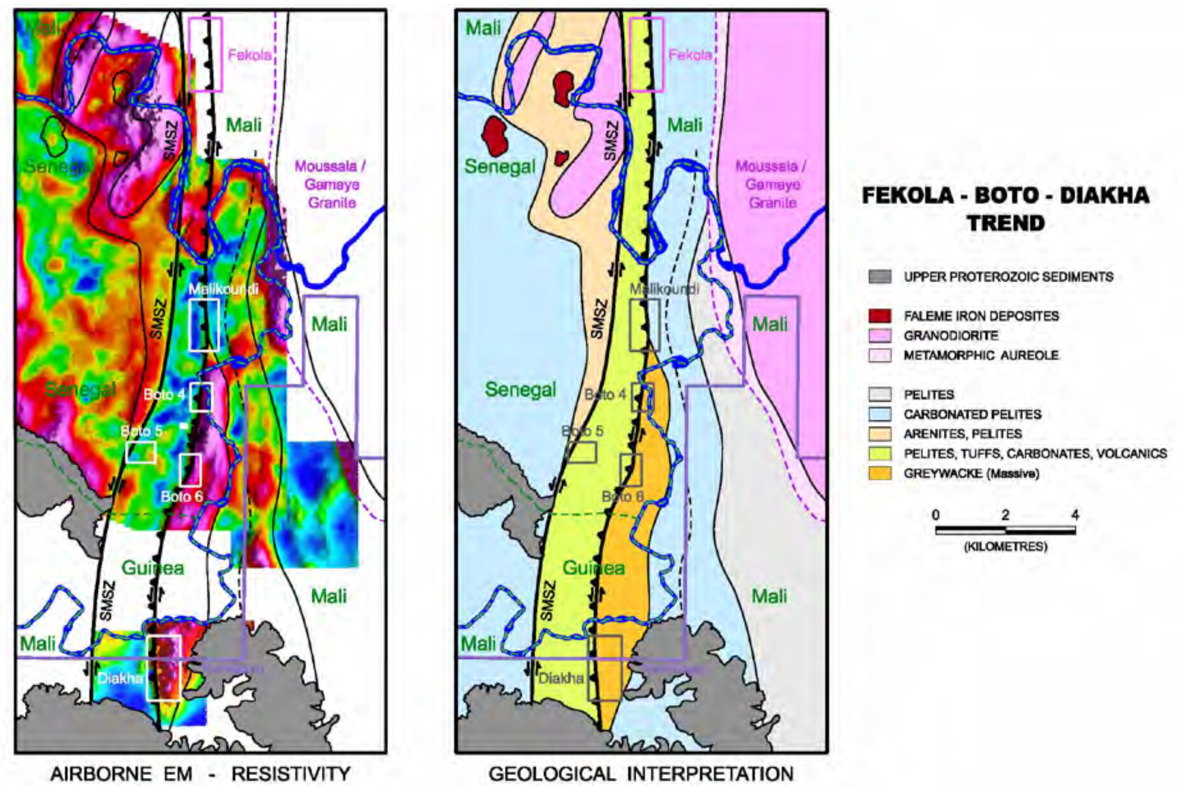
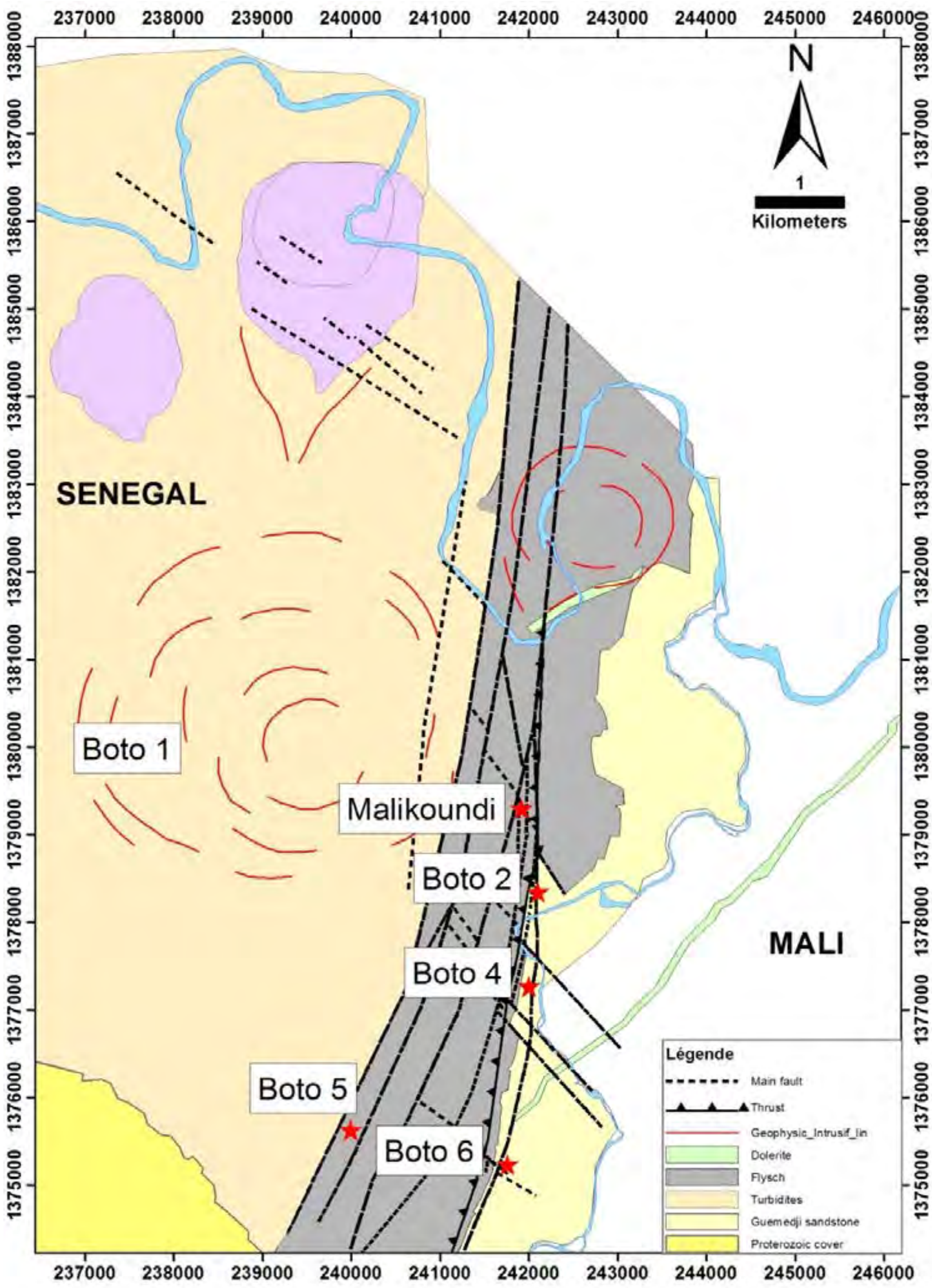


Figure 7.5 Structural Interpretation of the Malikoundi/Boto 2, Boto 4, and Boto 6 Deposits



Malikoundi/Boto2, Boto 4, and Boto 6

At Malikoundi/Boto 2, Boto 4 and Boto 6, the regolith is composed of pedolith (soil, ferricrete, and laterite), saprolite, and transition weathering profiles (saprock) that average 8 m, 20 m, and 10 m in thickness, respectively. The detailed study of regolith has made it possible to distinguish between transported and in-situ regolith. The assay results from up-dip expressions of mineralized zones, confirm in-situ mineralized regolith. Mineralization in fresh rock is mainly associated with pervasive albite alteration and pyrite.

Interpretation of structural data collected from oriented drill core has shown differences between Malikoundi/Boto 2, Boto 4, and Boto 6. Boto 6 is characterized by a bedding strike of 025˚ N, whereas Malikoundi/Boto 2 appears to have two bedding strike directions of 015˚ N and 030˚ N. The two bedding strike directions observed at Malikoundi/Boto 2 may result from ductile deformation within the impure marbles and laminated detrital sediments. Contrary to other parts of the structural corridor, a significant rotation of bedding strike to 147˚ N is noted in the drill core at Boto 4. The relationship between bedding and shear structures at Malikoundi/Boto 2, Boto 4, and Boto 6 relative to the central deformation corridor is shown in Figure 7.5.

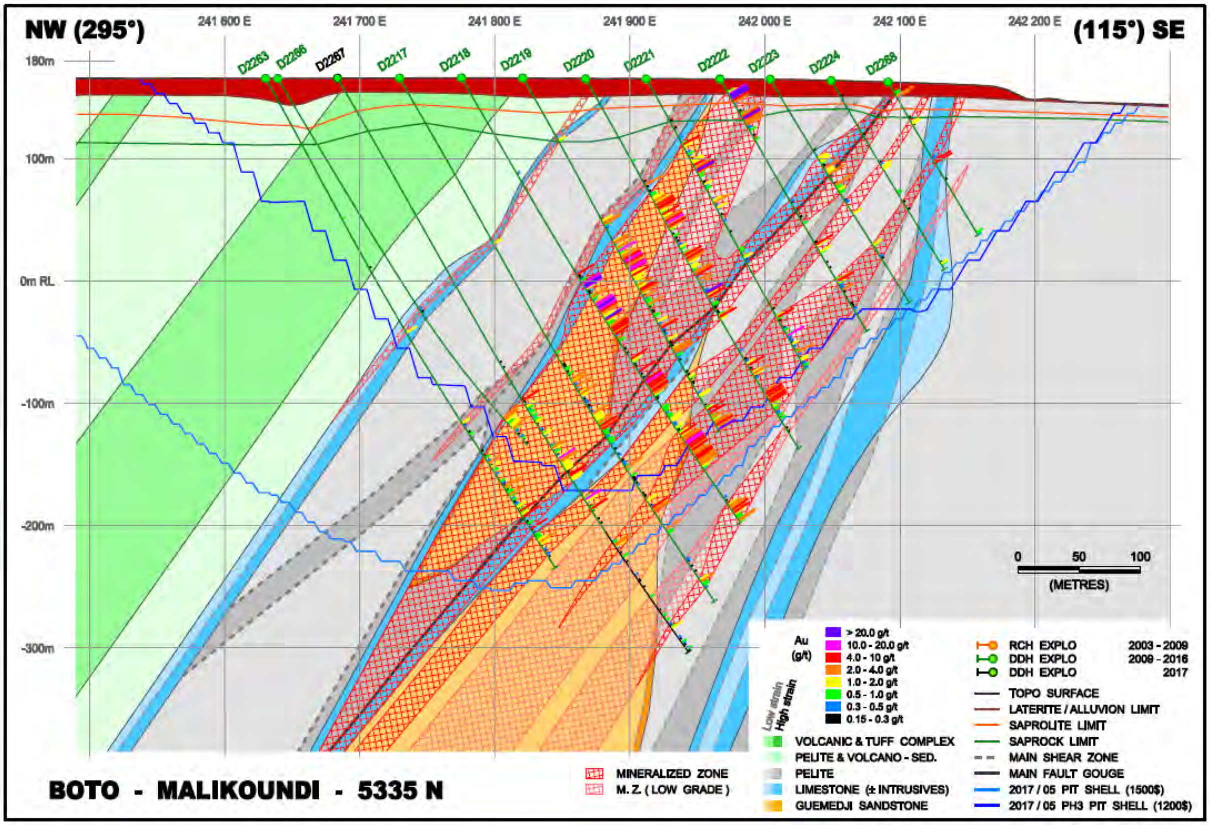
At Malikoundi/Boto 2, a 30° to 60° westward-dipping thrust fault has been observed in drill core at the contact between the Guémédji sandstone and the sequence of marble/laminated sediments. Of particular interest is a large lens of Guémédji sandstone that lies above the fault (Figure 7.6). This over-riding block of sandstone from the north end, cut out and moved from the Guémédji sandstone unit, is the main host of mineralization in this prospect. As a result of these movements, this lenticular block was severely fractured against adjacent rocks, and this fracturing was the conduit through which the gold-carrying fluids circulated and the mineralization was deposited. Thus, the fracturing associated with the sandstone lens is the principal carrier of mineralization in facies as diverse as sandstone, pelites, agglomerates, cipolin (unclean marble), or sometimes even syntectonic diorite. This overlap/shear fault was also identified further south in Boto 4 and Boto 6, further north of the Falémé River to the Fekola Gold Mine in Mali (called Medinandi permit), owned and operated by B2Gold; as well as further south to the Tammy permit (Mali). Several different units of cipolin were observed and these units played the role of deformation trends as well as permeability barriers to mineralizing fluids. At Malikoundi North, one of the units of cipolin corresponds to the mineralization zone having accommodated the deformation related to the circulation of mineralizing fluids.

Cipolin units can be subdivided into:

- Stratigraphic Cipolin In-situ: these marbles are distorted but remain in their stratigraphic place and are generally thick.

- Cipolin of Re-crystallized Deformation: these marbles are very distorted and re-crystallized and have been spread along shear structures by deformation. They no longer correspond to the stratigraphic orientation, but to a structural orientation usually making the junction between two different stratigraphic cipolins that were thus accommodated; their thickness is generally low, with an average thickness between 2-3 m and only rarely surpassing 10-15 m.

Figure 7.6 Representative Cross Section (5335N) of Malikoundi; looking 025°Az Northeast

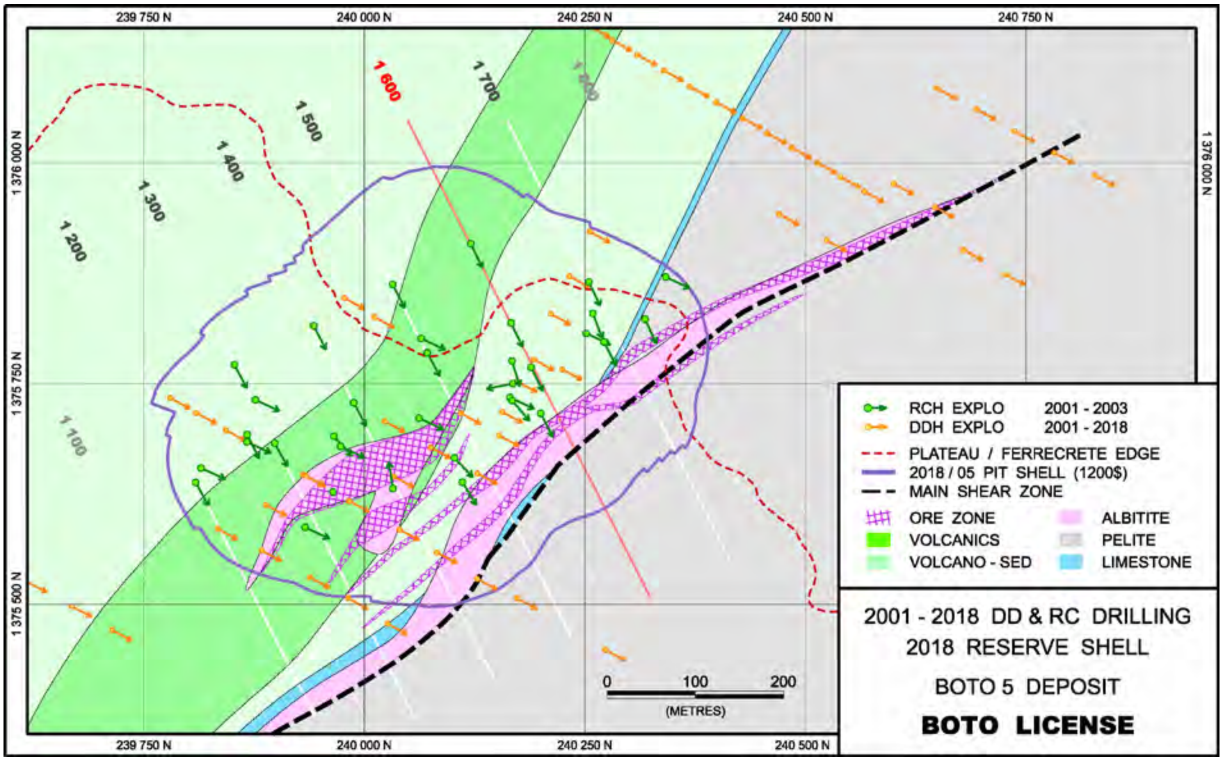


Boto 5

The weathering profile of Boto 5 is considerably deeper than that of Malikoundi/Boto 2, Boto 4 and Boto 6. Boto 5 is covered with a layer of pedolith 10 m to 40 m thick under which the saprolite layer can reach up to 80 m thick. The transitional layer under the saprolite is between 10 m to 40 m thick.

The lithological units at Boto 5 strike 015°-020° and include shale, carbonaceous sediment, and basalt. An albite-altered diorite dike that hosts the mineralization at Boto 5 cross-cuts the stratigraphy, striking 045° N dipping between 45° W and 60° W towards the west. Dieng (2005) described this dike as being discordant, approximately 30 m wide, and containing fragments of host rock in places. According to Dieng (2005), four deformation phases occurred at Boto 5. An early phase of brittle-ductile deformation led to the emplacement of barren tourmaline veins. This was followed by reverse brittle-ductile faulting overprinted and reactivated on the northeast trending structures. Gold-bearing quartz-tourmaline veins were formed during this phase. The D2 structures were subsequently covered by a third ductile deformation phase. The latest deformation event is characterized by north-northwest and northeast trending brittle faults that offset the mineralization into blocks (Figure 7.7).

Figure 7.7 Plan View of the Geology of Boto 5



Alteration

Based on core and thin section observations, it is believed the rocks that host the Boto deposits underwent five phases of alteration, which are strongly linked to the coeval structural and lithological hydrothermal events. The five main alteration phases observed are (Figure 7.8):

- Albite-sodic pervasive alteration that has turned the rock pink.
- Chlorite-calcite-magnetite fractures and wallrock alteration.
- Quartz-tourmaline-pyrite veining, with very limited wallrock alteration.
- Pyrite-hematite-calcite veining.
- Gypsum-anhydrite veining- with fissural hematization of the host rock.

The first alteration event is linked to intense fracturing and pervasive pink albitization. The second phase is related to a chlorite-calcite-magnetite alteration that developed along the fracture network as infill and locally overprinting the albitized wall rock. These two phases produced the crackle-breccia that characterizes most of the mineralized rock along the Guémédji trend.

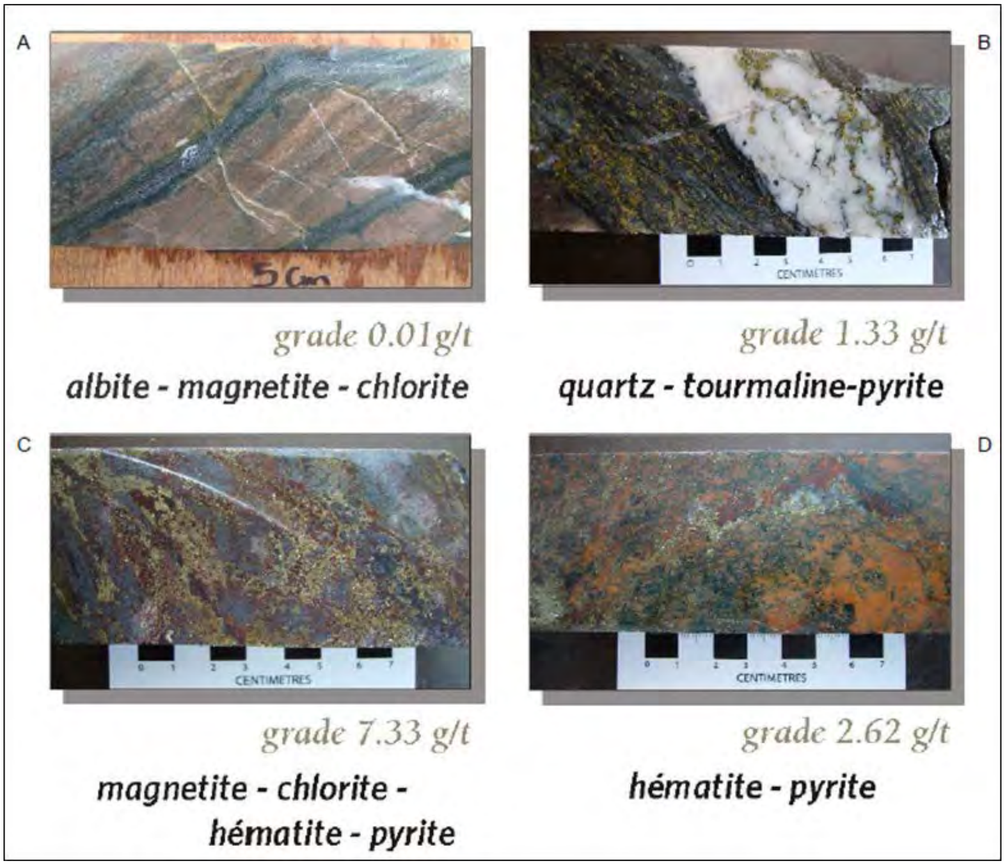
The third phase of alteration is characterized by a wallrock alteration associated with the emplacement of quartz-tourmaline-pyrite veins. Finally, the initial phase of chlorite-magnetite alteration is locally overprinted by pyrite-hematite-calcite alteration.

The last event (the gypsum-anhydrite phase) is essentially corresponding to the recent supergene alteration, destroying sulfuric acid and hematite. The acid was then combined with carbonates scattered in the rock, or from the marbles/cipolins, with a neoformation of minerals. This strong and recent supergene alteration is clearly related to the well-recognized thrust/shear fault of the core by the presence of fault gouge affecting either the cipolin or andesite. This is nevertheless a phase of hydrothermal alteration because the presence of anhydrite suggests a certain pressure and heat.

With the exception of Boto 5, Boto gold mineralization is mainly associated with crackle breccia. Brittle-ductile veins with a thickness varying from 0.5 cm to 2 cm have developed along pre-existing fractures filled with permutations of quartz, carbonate (calcite and ankerite), tourmaline, magnetite, chlorite, hematite, and pyrite.

At the scale of the prospect, it appears gold mineralization may have been favoured by the intersection of the north-northeast and north-northwest faults.

Figure 7.8 Alteration and Mineralization at Malikoundi/Boto 2, Boto 4, and Boto 6



7.4 Mineralization

Primary gold mineralization within the Early Proterozoic Birimian terrane was subdivided by Milési et al. (1989, 1992) into pre-orogenic, syn-orogenic, and late-orogenic. Mineralization of the five deposits in Boto are classified as late orogenic.

Late orogenic mineralization is usually associated with brittle-ductile deformation and is characterized by the association of Au, B, W, As, Sb, Se, Te, Bi, Mo, with traces of Cu, Pb, and Zn. Gold commonly occurs as native gold or as fine inclusions within the base metal sulphides or the gangue consisting of quartz, albite, carbonate, muscovite, pyrite, and tourmaline. In this category, there are two types of mineralization which, in some instances, may have been superimposed on each other locally:

- Disseminated gold-arsenopyrite and gold bearing quartz veins:
  - occur within northeast-southwest striking tectonic corridors

  - commonly hosted by metasediments

- Gold-quartz vein deposits with rare polymetallic sulphides (Pb, Cu, Zn):

  - associated with the final deformation phases of the Eburnean Orogeny

  - hosted in various lithological sequences

Boto 5

The Boto 5 prospect is located near the SMSZ. The gold is mainly hosted by an east-northeast striking, intensely albitized diorite intrusion, that penetrates and covers a northeast striking sequence of turbiditic sediments and limestone. Many diorite intrusions have also been observed. The mineralization is located mainly in albitite, either in a sill or in a casting. Mineralization in the host rock was not observed either in sediments or pyroclastic units. The intrusion of albitic lava resulted in a strong albitization when it intersected older diorite intrusions.

Gold mineralization follows a phase of quartz tourmaline veining as well as pyrite and related bleaching. The mineralizing event was accompanied by biotite alteration and pyrite mineralization, and a small proportion of chalcopyrite, covellite, and chalcocite. The presence of arsenopyrite appears to be confirmed by recent XRF measurements. Mineralization appears to be truncated against an east-northeast striking fault and is locally offset by a series of north-south striking faults.

Malikoundi/Boto 2, Boto 4, and Boto 6

The majority of the gold mineralization at the Malikoundi/Boto 2, Boto 4, and Boto 6 is hosted in the upper part of the Guémédji sandstone, near a structurally modified contact with the overlying, finer grained sedimentary sequence. The three main phases of alteration and mineralization were the subject of macroscopic observations, with gold mineralization interpreted as part of the last two events:

- Chlorite-albite alteration and magnetite-hematite-chlorite veining; calcite-tremolite alteration in distal settings.

- Quartz-tourmaline-pyrite alteration and veins.

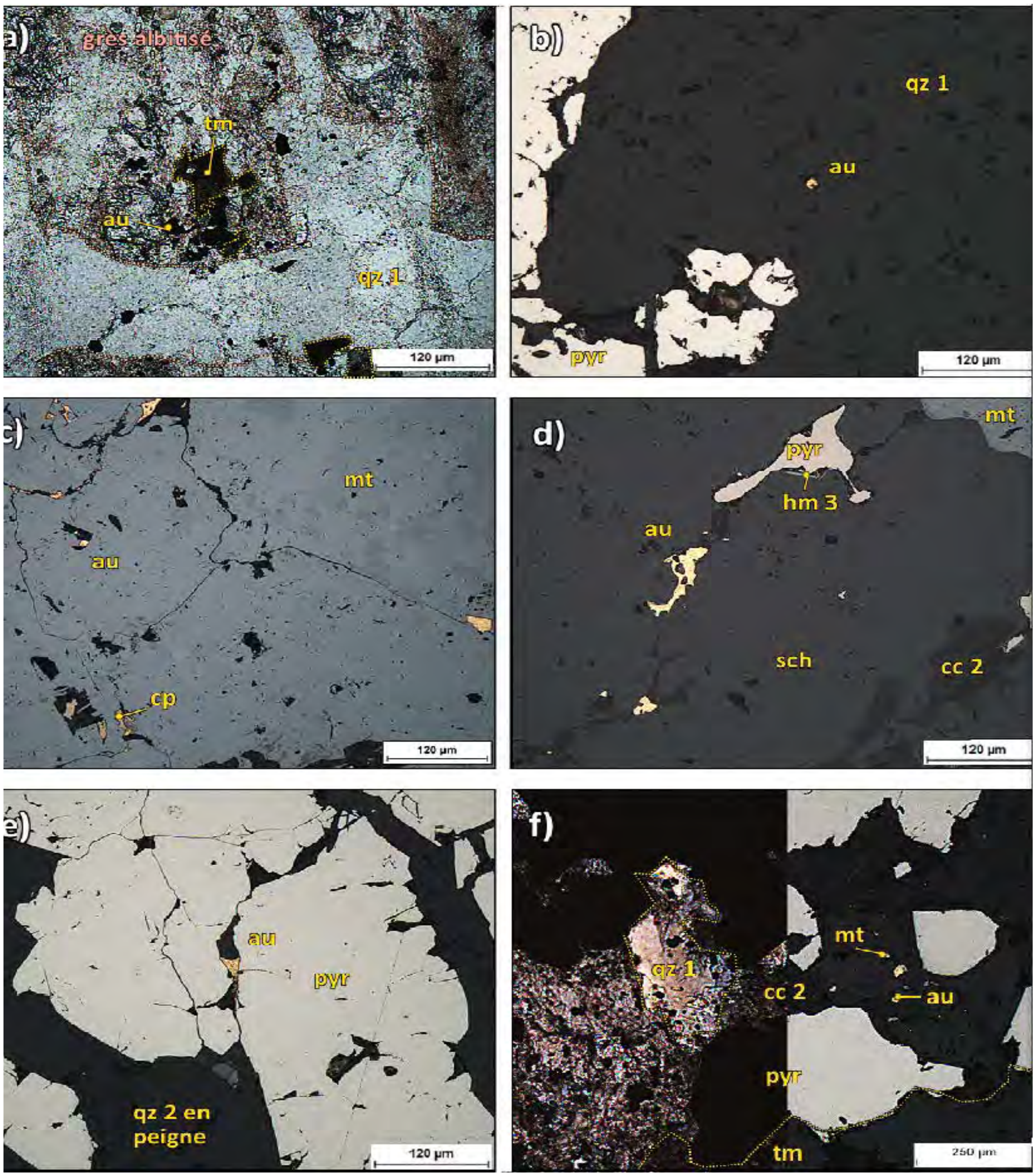
- Hematite-calcite-pyrite alteration and veins.

The chlorite-albite alteration is cited as associated with mineralization even if the fluids of this alteration did not carry gold due to the presence of magnetite which facilitated the precipitation of the gold; associated sulphides (pyrite) arrived during the phases of subsequent alteration. There is a strong relationship between the presence of magnetite and the gold associated with this precipitation. Microscopically, gold was observed in only two influxes at the end of the quartz-tourmaline phase. Macroscopically, gold is rarely observed in small sub-millimetre points on the core.

The size of the gold particles varies from <10 µm to 100 µm and have an average of approximately 20 µm. A review of gold mineralization and hydrothermal alteration at Boto identified six modes of emplacement (Gatinel, 2012) (Figure 7.9):

- Free gold in quartz-tourmaline veins.
- Free gold grains in quartz.
- Gold grains in fractures associated with chlorite-magnetite-pyrite ± quartz-calcite.
- Gold in fractures within the scheelite.
- Free gold in pyrite.
- Free gold in calcite.

Figure 7.9 Modes of Gold Emplacement in the Guémédji Trend



a) Free gold in quartz-tourmaline veins.
***b)** Free gold grains in quartz.*
***c)** Gold grains in fractures associated with chlorite-magnetite-pyrite ± quartz-calcite.*
***d)** Gold in fractures inside the scheelite.*
***e)** Free gold in pyrite.*
***f)** Free gold in calcite. (Gatinel, 2012)*

8.0 DEPOSIT TYPES

Similar to the majority of the deposits found in the Kédougou-Kéniéba inlier, gold mineralization at Boto is considered to be of the orogenic type. The orogenic gold deposits in the Birimian Province have been classified into three groups (Pre-, Syn-, and Post-orogenic). The characteristics of Boto mineralization are more similar to those of the post orogenic class.

As mentioned previously, the Malikoundi/Boto 2, Boto 4 and Boto 6 deposits are hosted by a turbiditic sedimentary sequence, with mineralization concentrating along the contacts of the litho-structural domains. The association of orogenic deposits with turbiditic sequences is well documented by Poulsen et al. (2000). Turbidite-hosted gold deposits within the eastern Kédougou-Kéniéba inlier are controlled by north-northeast trending structures linked to the SMSZ and, occur within the vicinity of intersecting north-northeast and north-northwest structures. At the Malikoundi/Boto 2, Boto 4 and Boto 6 deposits, gold is typically associated with pyrite, which is either disseminated along fractures (crackle-breccia hosted type) or along brittle-ductile veins.

Alteration assemblages observed at Boto 5 differ from those observed at Malikoundi/Boto 2, Boto 4, and Boto 6. The Boto 5 deposit is hosted in a diorite dike that contains abundant endogenic albite or has been pervasively altered to albite. The host rock at Boto 5 is highly deformed and contains a stockwork of quartz-tourmaline-pyrite veins. Although differing in appearance, this style of brittle-ductile deformation and veining is consistent with an orogenic gold mineralization model.

9.0 EXPLORATION

9.1 Exploration by AGEM, 1999-2012

AGEM has carried out exploration activities on the Project since 1999, with the majority of this work being done from 2007 and is currently on going. Between 1999 and 2007, AGEM compiled the results of the work carried out by Anmercosa and Ashanti Goldfields and carried out a number of geophysical surveys, such as gradient IP, radiometric, very low frequency (VLF) and HeliTEM (Helicopter Electromagnetics) surveys. Early drilling program centred upon the discovery and delineation of Boto 5, as well as the initial drilling fences at the Boto 2-4-6 anomalies. After 2007, the Boto 2-4-6 targets were the object of infill drilling as well as high resolution IP gradient surveys. The 2012 campaign led to the discovery of Malikoundi to the north of Boto 2.

Table 9.1 summarizes the work carried out by AGEM between 1999 and 2012:

Table 9.1 Summary of Exploration Activity by AGEM, 1999-2012

Exploration Activity	Details
Analysis of soil samples from the area collected but not analyzed by Anmercosa.	4,069 soil samples (one sample out of two was not analyzed by Anmercosa).
Detailed geochemical sampling	3,938 soil samples 14,851 termite mound samples 914 lag samples 549 rock samples
Exploration pits	821 pits
Trenches	29 trenches totaling 1,720 m
Augers	212 mechanical auger holes totaling 2,095 m
Airborne geophysical surveys	Magnetic and radiometric geophysical surveys
Detailed geophysical surveys	Gradient (2000) Magnetic and VLF (2000) Induced polarization Gradient IP (2000) Magnetic (2002) VLF (2002) Induced polarization IP (2006-2009) High resolution induced polarization (2008 to 2009)
Diamond Drilling (DD) and reverse circulation (RC)	13,097.5 m

9.2 Exploration by AGEM, 2012 to 2015

Following the discovery of Malikoundi in 2012, exploration activities focused on the development of Malikoundi with some follow-up exploration on Boto 5 and Boto 6. Table 9.2 summarizes the exploration activities completed on the Project from 2013 to 2015.

Table 9.2 Summary of Exploration Activity by AGEM, 2013-2015

Type of work	Details
Airborne geophysical Surveys	Electromagnetic (EM) surveys totaling 1,970 km
Forage Air Core (AC)	5,585 m in 475 holes
Diamond Drilling (DD)	43,564.5 m in 170 holes
Pre-feasibility Study	41 Pits for the study of infrastructure foundations
	2 Water sampling and analysis campaigns for the state 0 of groundwater
	9 Drill holes to study and analyze groundwater near infrastructure
	1 LIDAR Campaign
	2 Fauna and floristic inventory campaigns
	1 Survey of the panning
	1 Population Health Survey
	1 Public survey with the surrounding populations
	1 Survey of the social and environmental context before the start of the project
	1 Study on housing for miners
	1 Metallurgical study on gold recovery
	Installation of a weather station
	1 Economic study with financial model
	1 Geotechnical study Malikoundi pit
	1 Assessment of deposit considering various scenarios
	Water balance of the project

9.3 Exploration by AGEM, 2016

The 2016 exploration program consisted mainly of a diamond drill campaign and various technical studies. The 2016 diamond drilling campaign included:

- Exploration drilling of 4,813 m including four deep drill holes totalling 2,341 m, 22 short drill holes in Malikoundi totaling 1,952 m, and deepening of several drill holes that were stopped within mineralization totalling 492 m. This drilling defined the extension of the mineralization in Malikoundi to the north and at depth.

- Geotechnical drilling including four drill holes and the extension of three previous drill holes totalling 330 m. These drill holes were used to study the slopes on the east side of an open pit envisaged at the Malikoundi and were also used in the definition of mineralization.

- Definition drilling for Malikoundi/Boto 2 open pit including 607 m of drilling to define the northern extent of mineralization and 440 m from three drill holes to define a southeast extent of mineralization.

9.4 Exploration by AGEM, 2017 – March 2018

Exploration activities from 2017 to March 2018 were mainly focused on drilling with the following purposes:

- To improve definition of mineralization at Malikoundi/Boto 2 and Boto 5.

- To cover the gap in drill information between Malikoundi and Malikoundi North areas.

- To improve geotechnical characterization for the foundations of infrastructure.

- To install piezometers and carry out tests for hydrogeological testing at Malikoundi/Boto 2 and Boto5.

- To deepen geo-mechanical and hydrogeological knowledge for pits at Malikoundi/Boto 2 and Boto 5, as part of the feasibility study.

- To define mineralization at Boto 6 on a 50 m x 50 m grid.

- To further explore new targets in vicinity of Malikoundi, more specifically located to the East, West and Southeast.

Table 9.3 below summarizes the exploration activities from 2016 to March 2018.

Table 9.3 Summary of Exploration Activity by AGEM, 2016-March 2018

Type of work	Details
Diamond Drilling (DD)	23414 m in 132 holes
RC Drilling	11808 m in 119 holes

9.5 Exploration Potential

The Project is underlain by prospective Birimian age rocks and located in the southern part of Kédougou-Kéniéba inlier. The region is well endowed and estimated at 52 Moz gold with large deposits such as Sadiola and Loulo. Recent discoveries have been made and include nearby Fekola deposit (4.2 Moz) and Diakha (1.2 Moz).

The Guémédji geochemical trend, which hosts the Malikoundi/Boto 2 deposit, extends approximately 8 km in length from the Falémé River (north of Malikoundi) to the Balinko River (south of Boto 6). The area is covered by a thick lateritic cover, which makes traditional geochemical sampling ineffective. Termite mounts sampling was proven to be effective in identifying geochemical anomalies over Property. These anomalies are often well expressed and led to delineation of Boto 2, Boto 5 and Boto 6.

The Lelou trend, which hosts the Boto 5 deposit, has been poorly explored to the northeast, where the lateritic cover thick and ranges from 3 m to over 10 m. The area has not been tested by any sub-surface probing methods.

West of Malikoundi/Boto 2 and Boto 5 deposits, there is another structural trend which hosts the Boto 1 and Boto 3 targets, which are relatively underexplored. The Boto 3 deposit was only tested by pit sampling, where Boto 1 was scarcely drill-tested (Figure 9.1).

In June 2017, was organized a targeting workshop where various dataset were re-assessed and some fifteen new exploration targets were defined. These targets are mainly located east and west of the Malikoundi pit and were assigned priority. During Q4 2017, four of these targets were drill tested by RC, totalling 3,996 m and Rotary Air Blast (RAB) drilling, totalling 5,488 m.

In February 2018, a new campaign of RC and DDH drilling has been established to follow up on the positive results from Q4 2017. This program is still ongoing to the East, West and Southwest of Malikoundi.

10.0 DRILLING

AGEM, has completed several drilling campaigns on the Project since 2000. The following is a summary of all drilling completed from 2000 to March 2018.

10.1 AGEM, 2000-Present

The drill campaigns completed on the Project have been mainly focused on the Malikoundi/Boto2, Boto 5, Boto 6, and Boto 4 deposits. The drill hole database also includes: geotechnical/geomechanical drilling, metallurgical drill holes, shallow (<40m) RAB drill holes, and exploration drill holes on other exploration targets on the Project.

Table 10.1 summarizes all drilling on the Project in the drill hole database, up to and including March 2018.

Table 10.1 Summary of Drilling for the Project, 2000 – March 2018

Year	DD		RC		Total	
	Metres	Number	Metres	Number	Metres	Number
2000	1117	8	177	2	1294	10
2001	2057	13	2080	23	4137	36
2002			1593	24	1593	24
2003			3292	52	3292	52
2007	2639	11	10687	107	13326	118
2008	3721	18			3721	18
2009	3880	17	7618	73	11498	90
2011	284	1			284	1
2012	13322	50			13322	50
2013	13130	52			13130	52
2014	16223	60			16223	60
2015	14856	58			14856	58
2016	6139	38			6139	38
2017	11853	69	3997	41	15850	110
2018	2452	14	5312	53	7764	67
Total	**91673**	**409**	**34756**	**375**	**126429**	**784**

Of the 784 drill holes in the drill hole database, 496 drill holes intercept the interpreted mineralized zones in Malikoundi/Boto2, Boto 5, Boto 6, and Boto 4 deposits.

Figure 10.1 to Figure 10.4 present drill hole location maps for Malikoundi/Boto2, Boto 5, Boto 6 and Boto 4 deposits, respectively.

Figure 10.1 Drill Hole Location Map; Malikoundi/Boto 2

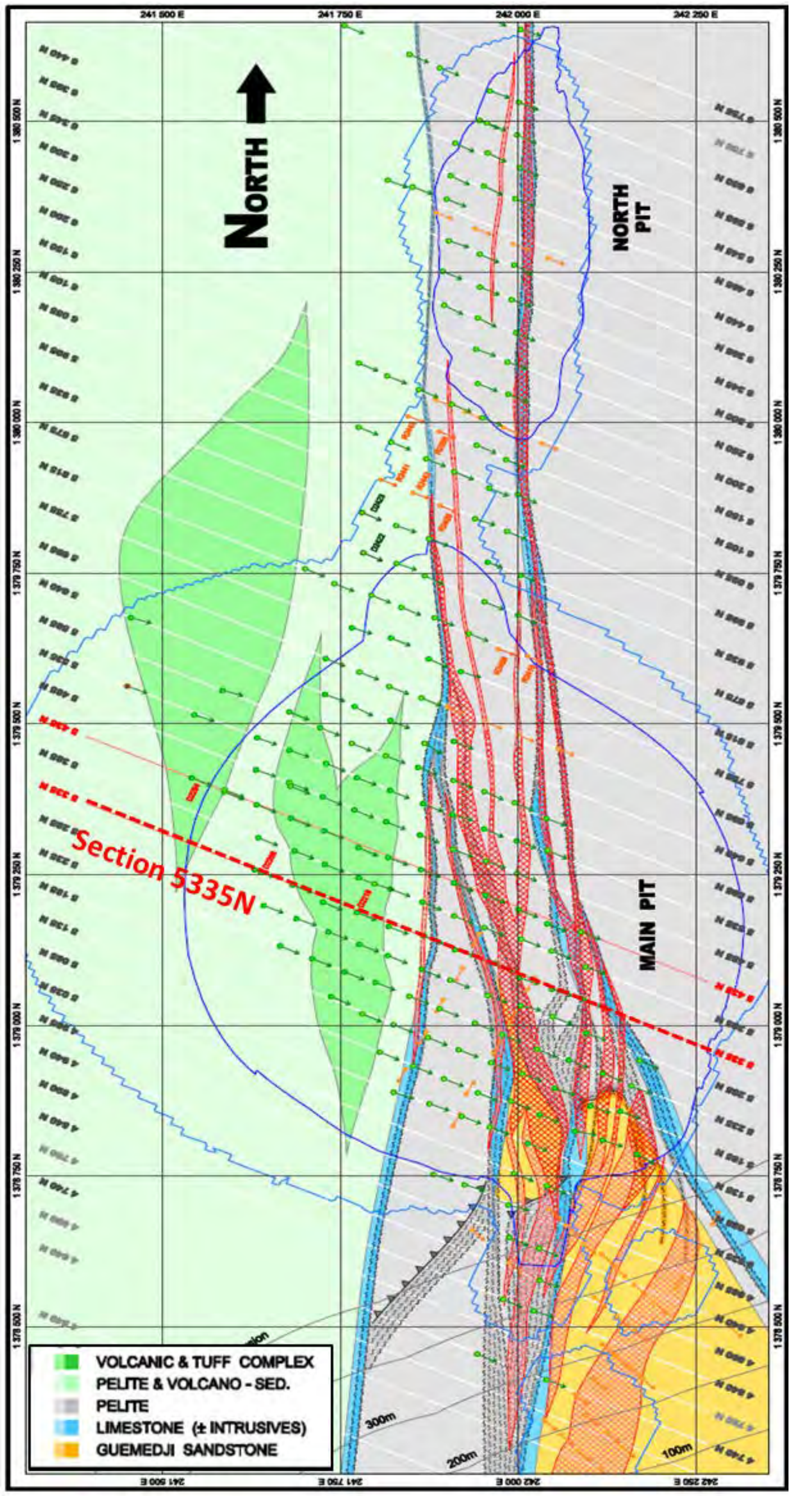


Figure 10.2 Drill Hole Location Map; Boto 5

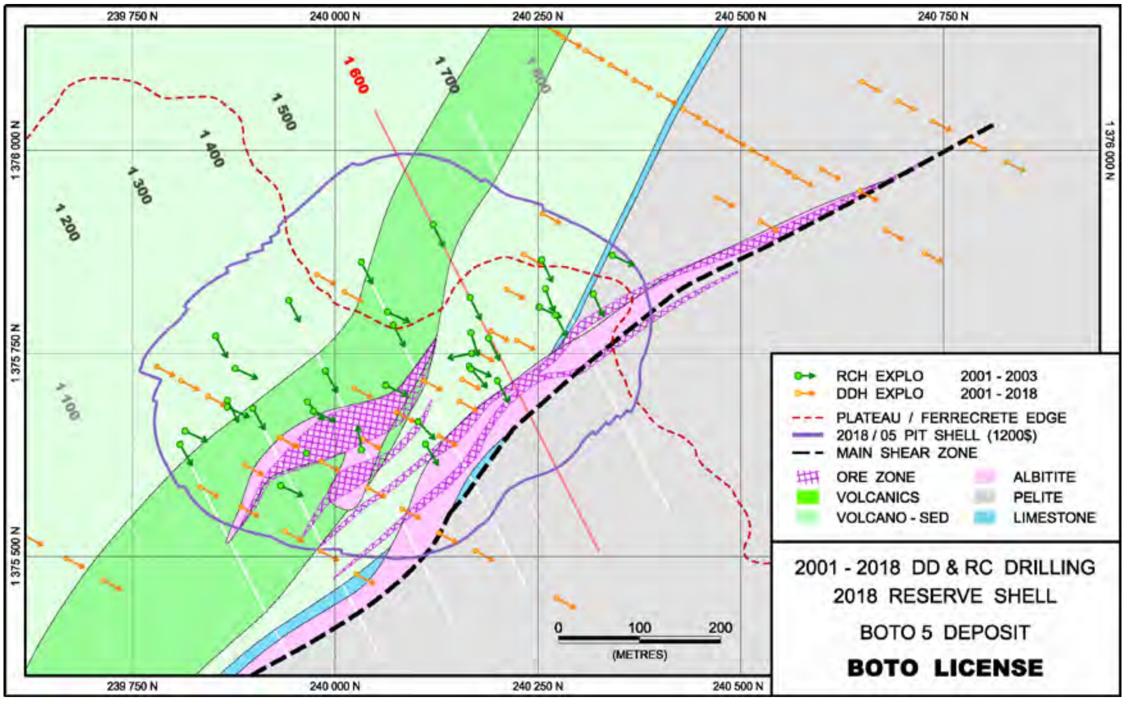


Figure 10.3 Drill Hole Location Map; Boto 6

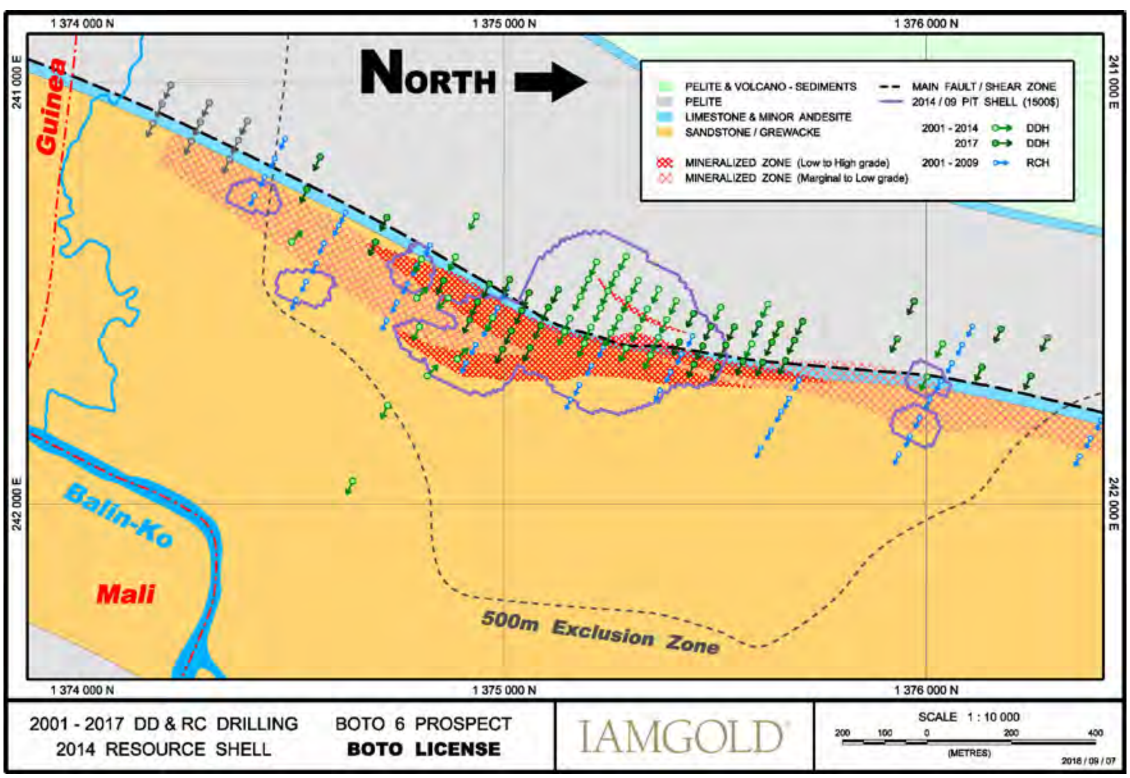
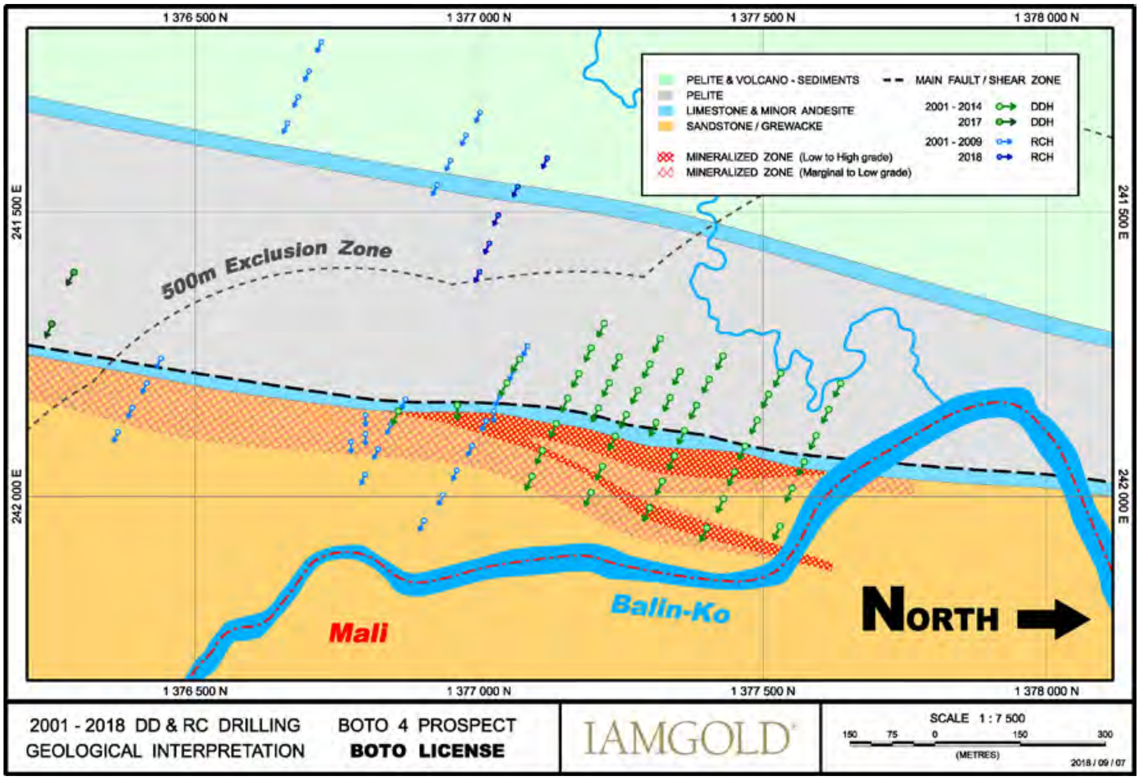


Figure 10.4 Drill Hole Location Map; Boto 4



10.2 Drilling Procedures

Drill pads are prepared to approximately 15 m by 8 m, and the positions of the planned drilling is located using a handheld GPS unit. A piece of wood with flagging tape that states the technical parameters for the holes to be drilled (i.e. drill hole number, Azimuth, dip and planned depth) is placed firmly in the ground.

For diamond core drilling, two pieces of wood, for a front sight and back sight, are placed in front of the hole to be drilled at 15 m and 25 m on the same line to facilitate alignment of the drill on the appropriate azimuth. For RC and RAB drilling, a line is drawn on the ground and the drill is aligned parallel to this line.

From the 2009 drilling campaign, drill hole collar surveys were carried out by Differential Ground Positioning System (DGPA). Drill hole collars, prior to 2009 were re-surveyed by DGPS.

10.2.1 Diamond Core Drilling

Generally, the DD holes were drilled using HQ size core within lateritic overburden and weathered material (saprolite and saprock), and then reduced to NQ size core in fresh rock. Since 2003, oriented core drilling has been employed. To mark the bottom of oriented core holes, two methods were used: a “down-hole spear” (used before 2010) and an ACE apparatus (more recently). For both methods, the downhole tools were handled by the driller and markings were made every three metres. DD holes were surveyed downhole with a reflex instrument. Downhole surveys were performed every 100 m, at the point where HQ was reduced to NQ, and at the end of the hole.

The drill rig is set up by drillers under the supervision of a geologist, who checks the planned azimuth and dip before the drilling starts. Since 2009, the drillers have been allowed to align the rig with the marks pre-made by a geologist or technician. Geologists align the drill rig with a compass and a clinometer. Core trays are transported from the drilling site to the camp by the technician at the end of each shift. Upon arrival in the camp, the subsequent operations are carried out under the direct supervision of the geologists.

At the camp, core trays are aligned on logging tables according to their depth, so the geologists can review the core for orientation, recovery, and rock quality designation (RQD). Core recovery and RQD measurements are then documented in detail by a trained technician under the supervision of the geologists who are usually logging the hole at the same time. The core is logged by geologists for lithology, alteration, structure, veining, mineralization (sulphide content), and weathering/oxidation.

For structural logging, alpha and beta angles for each type of structure are measured and recorded. Observations are usually made every metre. Commonly logged structures include bedding, schistosity, veining, shear bands, fractures, and fault markers. Vein characteristics such as size, infill material, alteration minerals, and sulphides are also recorded. After logging is complete, samples are taken for density measurements. The core trays are then transferred to the sawing area. Since 2012, 10 cm long pieces of core have been collected every 25 m for density measurements using the plastic wrapped water immersion method.

The core is sawn with a diamond saw blade and placed in bags. The saw is washed between samples. Where core recovery is poor, and no sufficient sample is available to prepare a sample, two or three metres are combined to make a composite sample.

The following activities take place in the core sawing area:

- Pictures are taken of core in the tray, three trays at a time. Core is split into two halves, with one half to be sent for assay and the other half kept for reference. Soft rocks such as saprolite are usually cut with a machete.

- Half of each one metre long core is broken with a hammer and placed in a 24 cm by 40 cm plastic bag. A pre-prepared sample tag is added, and the bag is wrapped and stapled at the top.

- Sample preparation starts immediately after all core in the core tray is cut.

- A sampling sheet is provided to the technician for each hole to be sampled.

10.2.2 Reverse Circulation and Rotary Air Blast Drilling

Samples are taken every one metre down the hole and the entire hole is sampled. Samples are collected at the exit of the drill cyclone using 50 cm X 80 cm plastic bags, resulting in 25 kg to 35 kg sample weights when the recovery is good. The cyclone is blown clean by the drill operator between each sample. The Hole ID and the sample depth are written on the plastic bag with a permanent marker. After collecting the sample, a sample tag, which includes the sample number as well as an aluminium-made tag that includes both the sample number and Hole ID, is put inside the bag. All these operations are under the supervision of a geologist, who is also in charge of logging the geology immediately after a sample is collected. Tags and sample bags are prepared and marked in advance.

After the rig has moved to another hole, another crew will start splitting the samples. Each sample is split with a high capacity splitter until a two to three-kilogram sample for assay and a duplicate are obtained, with both samples being bagged and numbered. Control samples are introduced approximately every 20 samples: a duplicate sample and a blank sample are alternatively inserted within the sampling sequence.

Prior to 2003, the bulk samples from the cyclone were transferred to the camp where they were weighed before splitting. Geology was being logged twice, with a quick log done by a site geologist to monitor geology while drilling and a more detailed logging was completed in the camp by another geologist, who would use a chipboard as a lithological reference tool.

Two-metre composites were usually submitted to the assay laboratory. After the 2003 campaign, samples have been logged and prepared in the field as outlined above. The chipboard reference tool has been replaced by a chip tray that can be brought into the field. RC and RAB holes are logged in 1 m increments and information captured in the logs is the same as core logging with the exception of structural information.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Historical Sample Preparation and Analysis

Prior to 1999, exploration was carried out by Ashanti Goldfield and Anmercosa. The only known sampling types conducted during this period were surface geochemical sampling and grab sampling.

From 1999 to 2004, sample preparation was carried out at Karakaena Camp for both the Bambadji and Daorala-Boto permits. The preparation included crushing, pulverization, and splitting of 100 g pulp which as shipped to the laboratory for analysis.

Quality Assurance and Quality Control (QA/QC) from 1999 to 2004 consisted of the insertion of duplicate samples, blank samples (blanks), and standard samples as follows:

- A duplicate was inserted in every batch of 10 samples.
- A certified blank was inserted every 20th sample.
- A certified standard was inserted every 40th sample.

During this period, preliminary preparation was carried out at the AGEM field laboratory before being submitted to a commercial laboratory. This field lab was under the supervision of an experienced technician.

Diamond drill cores were split with a core saw, half of which were used as a sample and the other half retained for future needs. The RC and RAB samples, samples were collected meter by meter, the whole meter is dried before splitting down all 1 m and 20 kg mean weight, were completely dried before being divided into 2 kg samples each using a riffle splitter. The representative sample was then crushed and pulverized.

Once each sample was processed, the material was cleaned using compressed air. The pulverizer was cleaned by pulverizing barren material (quartz sand) between samples.

The entire 2 kg sample was crushed using a standard or hand-held mechanical crusher to achieve a maximum particle size of 2 mm. Portion of crushed sample was then pulverized to have 80% of pulverization passing 120-mesh.

The entire sample was then split using a riffle splitter and 200g extracted; 100g of which was split and send to the laboratory and pulverized to less than 200-mesh size material before assaying 30 g at the request of IAMGOLD.

From 2004 to 2007, for certain periods only, duplicates and blanks were used to do the QAQC for RC, RAB, trench, and termite mound samples. Since 2004 no preparation has been made at the camp, other than splitting of the RC and RAB samples. Samples of core, exploration pits, trenches, and mound were packaged and numbered prior to being sent to the laboratory.

The insertion rates of QA/QC samples at this time were:

- A duplicate inserted in each batch of 10 samples.
- A local blank inserted every 20th sample.
- No certified standard was used.

Table 11.1 shows the method of analysis and the laboratories used from 1999 - 2008.

Table 11.1 Analysis Method and Laboratory, 1999 - 2008

Year	Laboratory	Titration Method (Gold)	Notes
1999	Chimitec in Val-d'or	FA30g	Reanalysis of Anglo soil samples of 1995
2000	Chimitec in Val-d'or	FA30g	
2001	Chimitec in Val-d'or	FA30g	
2002	Chimitec in Val-d'or and Abilab Bamako	FA30g	
2003	Abilab Bamako	FA50g	
2004	Abilab Bamako	FA50g	
2005	Abilab Bamako	FA50g	
2006	Abilab Bamako	FA50g	Abilab acquired by ALS Chemex
2007	ALS Chemex Bamako (ex-Abilab)	FA50g	
2008	ALS Chemex Bamako (ex-Abilab)	FA50g	

In 2007 and 2008, the QA/QC procedure was reviewed and new procedures were put in place to ensure an adequate degree of confidence in the sample preparation and assay results. An internal validation of the samples pre-2007 was carried out by IAMGOLD in 2007 and did not detect any significant sampling issues. The new QA/QC methods were applied to previous data from 1999 to 2007 and approximately 10% of the samples were re-analysed in batches that include certified standard to comply with the new procedures. From that point on, validation procedure was systematically applied.

11.2 Current QA/QC Procedures

Since 2009, all AGEM sampling campaigns have been using certified standards and blanks, in addition to taking duplicates and check assay samples.

During this period, AGEM used two types of blanks. One of the blanks is sourced from a Late Proterozoic sandstone near the border of Guinea (blank R) and the other is sourced from a termite mound known to have no gold (blank S). The first is usually inserted among the fresh rock samples and the second among the saprolite samples. Samples of certified standard materials were purchased from Rocklabs and with standard values covering the grade ranges observed at Boto.

For DD holes, a certified standard sample is inserted every 20 samples, alternating with blanks, which are also inserted every 20 samples. The same protocol applies to RC and RAB drilling.

QA/QC results are monitored in each drilling program. Standard and blank samples are plotted against their theoretical value and scatter diagrams are created for duplicates and check assays. An assay batch is considered validated if the value received for the certified reference is within a range of ±15% of the mean certified value for that standard. The entire batch will be re-assayed if any certified standard does not meet this requirement. For blanks, any assay value greater than 10 ppb signifies a batch failure and the entire batch is then re-assayed.

Boto maintains detailed records of each sample including the date of collection by the laboratory, the date of arrival in the laboratory, the assay results and the name and date of the file containing the results. Boto keeps detailed records to monitor the performance of blanks, certified standards, duplicate and check assays using the previously mentioned control charts. Boto also tracks the performances of the internal laboratory standards and blanks using the same type of control chart for its own data.

Until December 2013, all samples from Boto were being analyzed at the ALS Chemex Laboratory in Bamako. Upon reception in the laboratory, samples are removed from the sample bags and checked against the chain of custody form. Each sample is weighed and assigned a bar code number and a unique file number. The information of the sample is entered into the ALS system under an ALS file number.

The sample is placed in a drying tray and a label, with a unique sample and file number, is placed in the sample tray with the specimen. Samples dry for 24 hours. The following procedures are applied to the following sample types:

DD and RC drilling samples are coarse crushed to 75% passing 2 mm. The jaw crusher is cleaned using compressed air after each sample. Every five samples, barren rock is passed through the crusher for cleaning. ALS performs a sieve size analysis after every 70 samples, to ensure crushing is adequately performed. A 1,000 g split of the crushed material is pulverized in a “ring and puck” grinding mill to 80% passing 200-mesh. As a pulverization QAQC, ALS also performs a sieve size analysis after every 20 samples. After grinding, a 50 g pulp sample is split and used for analysis.

The analysis of the core and RC drilling samples is carried out by fire assay with an atomic absorption finish method (ALS code Au-AA24) on pulverized 50 g, with a lower detection limit of 5 ppb and an upper detection limit of 10ppm. Any results are greater than 10 ppm, are subsequently re-assayed using a gravimetric finish (ALS code Au-GRA22).

ALS Chemex inserts two internally certified standards and two blanks in each batch of 24 samples. Duplicates are also analyzed on a regular basis. An internal laboratory QA/QC assessment for each batch of samples is carried out. The results of the control samples are evaluated to ensure they meet the standards established by the precision and accuracy requirements of the method. In the event that any reference material or duplicate results are outside the established control limits, an error report is automatically generated and triggers a re-assay of the batch.

From December 2013, all Boto samples were processed in the Veritas laboratory. The staff of the Véritas laboratory is contacted when at least 800 samples are ready to be shipped. By the time the Véritas vehicle picks up the samples from the camp, the number has usually risen to approximately one thousand samples. The vehicle then carries the samples to the Kédougou preparation laboratory. Samples are then sorted by batches of 200 samples and a name given. Since 2016, Veritas stopped preparations in Kédougou and samples are currently been prepared at the Véritas laboratory in Bamako, Mali.

All samples are then dried and weighed. The drying temperature is between 60°C to 105°C; the drying time depends on humidity. The sample is fully crushed to have 70% passing 2 mm, size of the particles are check on regular basis. The sample is divided and homogenized to obtain a representative sub-sample before pulverization to 85% passing 75 µm.

Pulps are then sent to the Véritas laboratory in Abidjan, Ivory Coast, for assay. 50 g of pulp of the sample is weighed and mixed with a known mass of fondant, consisting of a mixture of lead oxide, sodium carbonate, borax, silica, silver, and other chemicals if necessary, to obtain a good lead-acid. This leaded fondant is then transformed into silver aggregate by cupellation. The silver pellet is dissolved with 1 ml of nitric acid and 1 ml of hydrochloric acid and digestion takes place in a water bath. The solution obtained from digestion is cooled, diluted with distilled water to a final volume of 10 ml, and analyzed by atomic absorption spectrometry to obtain the gold content.

The DD and RC samples are assayed using fire assay with atomic absorption finish on 50 g of pulp (Véritas Code FA450), with a lower detection limit of 10 ppb. Samples of core that are analyzed using fire assay with atomic absorption finish and return a result greater than 10 ppm are re-assayed using a gravimetric finish (FA550 Véritas Code).

The duplicate is automatically generated by the system for core and rock samples. The normal frequency is usually a duplicate per every 50 samples. The duplicated sample is obtained by splitting the sample after pulverization. The duplicated sample is treated like all other normal samples as soon as it is produced.

Repeats of analysis are systematically and randomly produced when preparing the fusion racks. In each fusion rack of 50 samples two blanks, two standards, and two duplicates are added. This number may vary depending on the quality of the results.

11.3 Sample Security

The samples were transferred from the field to the camp only in the presence of a qualified and experienced technician. Drill core cutting, sample packaging, and storage were carried out under the supervision of Boto geologists and technicians.

The core halves and the RC and RAB samples were packaged in sealed, plastic, sample bags. A sample tag is placed in each bag of samples taken. The samples are then picked up by laboratory personnel and transported to ALS Chemex in Bamako, or Véritas in Kédougou or Bamako, depending on the period of dispatch.

11.4 QP Opinion

It is the opinion of the QP that the sample preparation and analyses are adequate for this type of the deposit and that the sample handling and chain of custody are satisfactory and meet industry standards. The data is considered representative for the level of study presented in this report. The QP concludes that the exploration, sampling practices, and resulting data are suitable for the estimation of a NI 43-101 Mineral Resource Estimate.

12.0 DATA VERIFICATION

12.1 Data Verification

The database was verified by the QP. All assay analyses in the GEMS database since 2000 were extracted and approximately 12% of the assay values were randomly selected and verified against the official electronic copies of laboratory certificates. No errors were found.

In addition to these checks, the QP checked for abnormally high values, missing intervals or sample numbers, interval lengths, and zero levels. Any high values above detection limit, often > 10 g/t Au, were re-analysed. Any values below detection limit, often < 0.005 g/t, Au were assigned the value of half the detection limit. Drill holes were checked visually for deviations in the down hole survey. No errors were found.

12.2 Site Inspection, December 2017

A site inspection was conducted from December 11 to 15, 2017. There were no drilling activities in progress at the time the site visit. Table 12.1 lists the personnel included on the December 2017 site inspection

Table 12.1 Site Inspection Personnel, December 2017

Name	Company	Position
Martin Lanctot, P.Eng., MPM	IAMGOLD	Project Manager/Director Project Boto
Ian McKenzie,	Lycopodium	Engineering Mechanical Principal
Gordon Zurowski, P.Eng.	AGP	Principal Mining Engineer
Paul Daigle, P. Geo	AGP	Resource Geologist
Rob Thomas, MAusIMM CP(Geotech)	Absolute Geotechnics	Geotechnical Engineer

12.2.1 Geology Site Inspection

For the site inspection, Mr. Daigle was accompanied by M. Benoit Michel, Project Geologist and M. Yaouba Thiam, Project Geologist, both employees of IAMGOLD.

The site visit included inspection of core logging, sampling and drill core storage facilities, checking coordinates for drill hole collars, and reviewing drill core logs against selected drill core.

Drill Core Logging and Sampling and Storage Facilities

Drill core for the Project is logged, sampled, and stored in two locations: at the Boto Exploration Camp, situated approximately 12 km due west of Malikoundi, and at the New Camp, situated approximately 1.5 km west of the Malikoundi deposit.

AGP inspected the two facilities and found the core storage sheds at both sites to be clean and orderly. Core boxes are stacked by drill hole number. The aluminium core boxes are stacked one on top of the other with enough space between the rows of stacks to access any drill hole easily. The core boxes do not have lids but are easily stackable and only the top box is open to the elements. The stacks vary in height from less than 10 boxes to as many as 30 boxes high.

Figure 12.1 Shows the Core Storage and Logging Facilities at the New Camp

Figure 12.2 Shows the Core Logging (foreground) and Storage Facilities (background) at the Boto Camp

Drill Hole Collar Locations

AGP located 55 drill hole collars at three of the four deposit sites; Malikoundi, Boto 5, and Boto 6. At Boto 4, there was no evidence of drill holes found during the site visit as most of the drill holes were in the village of Guémédji and any drillhole markers have been removed.

Drill hole collars are typically marked by a cement cast around a 4" PVC pipe in the collar. The cement cast is inscribed with drill hole number, Azimuth, dip, and depth of drill hole. Many of these cement casts are showing signs of wear and, in some cases breakage, but most are still legible. Since the long grass is often burnt by the end of the rainy season, many of the PVC pipes are melted (Figure 12.3).

Figure 12.3 Drill Hole Collar for DBDD-2298 (Malikoundi)

The locations of diamond and RC drill hole collars were measured in the field using a hand held Global Positioning System (GPS) device (Garmin GPSmap 62s) using WGS 84 datum, the same datum used by IAMGOLD at the Project. A total of 55 waypoints were collected on the Malikoundi/Boto 2, Boto 5, and Boto 6. The majority of coordinates measured by AGP fell within a 5 m tolerance of those reported by IAMGOLD. In only one instance was the Easting off greater than 5 m (at 9 m). This is not considered a significant error given the accuracy of the handheld GPS (± 10m). It is the QP's opinion that the coordinates are acceptable, given the accuracy of the handheld GPS used to review the drill hole collar locations.

Table 12.1 to Table 12.3 present the comparison of the AGP and IAMGOLD drill hole coordinates for the Malikoundi, Boto 5, and Boto 6 deposits, respectively.

Table 12.2 Comparison of Collar Location Coordinates for Malikoundi

Drill Hole	AGP Easting (UTMm)	AGP Northing (UTMm)	IMG Easting (UTMm)	IMG Northing (UTMm)	Δ Easting (UTMm)	Δ Northing (UTMm)
DBDD-2299	241965	1380492	241956	1380497	-9	5
DBDD-2325	241947	1380503	241946	1380501	-1	-2
DBDD-2298	241956	1380554	241960	1380551	4	-3
DBDD-2326	242001	1380533	242001	1380532	0	-1
DBDD-2300	241998	1380478	241999	1380477	1	-1
DBDD-2301	241958	1380448	241957	1380447	-1	-1
DBDD-2303	241952	1380391	241952	1380393	0	2
DBDD-2304	241998	1380370	241996	1380372	-2	2
DBDD-2305	241946	1380279	241946	1380279	0	0
DBDD-2308	241991	1380202	241991	1380204	0	2
DBRC-2235	241983	1379991	241980	1379992	-3	1
DBDD-2335	241939	1379841	241940	1379841	1	0
DBDD-2216	241929	1379651	241931	1379650	2	-1
DBDD-2288	241648	1379198	241648	1379196	0	-2
DBDD-2258	241585	1379390	241587	1379387	2	-3
DBDD-2294	241541	1379413	241542	1379411	1	-2
DBDD-2410	241587	1379391	241588	1379391	1	0
DBDD-2217	241725	1379202	241726	1379204	1	2
DBDD-2120	241775	1379248	241776	1379245	1	-3
DBDD-2409	241799	1379238	241798	1379235	-1	-3
DBDD-2118	241816	1379117	241817	1379116	1	-1
DBDD-2117	241863	1379097	241865	1379093	2	-4
DBDD-2072	241958	1379053	241957	1379050	-1	-3
DBDD-2130	242004	1379031	242003	1379033	-1	2
DBDD-2073	242049	1379014	242049	1379014	0	0
DBRC-2254	241988	1379001	241988	1379001	-1	0
DBDD-2109	241979	1378984	241978	1378985	-1	1
DBDD-2077	241963	1378937	241962	1378938	-1	1
DBDD-2108	241938	1379006	241939	1379003	1	-3
DBDD-2076	241914	1378962	241917	1378959	3	-3
DBDD-2283	241922	1379022	241925	1379022	3	0
DBDD-2281	242014	1378805	242014	1378804	0	-1
DBDD-2385	241986	1378704	241986	1378702	0	-2
DBDD-2382	242029	1378579	242030	1378577	1	-2
DBDD-2381	242021	1378526	242022	1378526	1	0

Table 12.3 Comparison of Collar Location Coordinates for Boto 5

Drill Hole	AGP Easting (UTMm)	AGP Northing (UTMm)	IMG Northing (UTMm)	IMG Easting (UTMm)	Δ Easting (UTMm)	Δ Northing (UTMm)
DBDD-2363	241572	1375655	241574	1375654	2	-1
DBDD-2364	241611	1375689	241612	1375688	1	-1
DBDD-2361	241616	1375634	241616	1375634	0	0
DBDD-2357	241573	1375604	241573	1375604	0	0
DBDD-2355	241662	1375568	241662	1375566	0	-2
DBDD-2352	241624	1375469	241625	1375471	1	2
DBDD-2351	241616	1375416	241616	1375415	0	-1
DBDD-2166	241539	1375338	241538	1375339	-1	1
DBDD-2043	241558	1375270	241558	1375273	0	3
DBRC-2269	241597	1375242	241596	1375241	-1	-1
DBRC-2146	241719	1375176	241717	1375178	-2	2

Table 12.4 Comparison of Collar Location Coordinates for Boto 6

Drill Hole	AGP Easting (UTMm)	AGP Northing (UTMm)	IMG Easting (UTMm)	IMG Northing (UTMm)	Δ Easting (UTMm)	Δ Northing (UTMm)
DBDD-2389	240319	1375823	240318	1375823	-1	0
DBDD-2394	240274	1375796	240272	1375797	-2	1
DBDD-2396	240187	1375769	240189	1375769	2	0
DBDD-2392	240165	1375774	240167	1375776	2	2
DBDD-2393	240201	1375718	240200	1375717	-1	-1
DBDD-2391	240110	1375635	240111	1375638	1	3
DBDD-2022	240061	1375711	240062	1375711	1	0
DBRC-2012	240023	1375707	240023	1375707	0	0
DBRC-2022	240084	1375560	240082	1375559	-2	-1

Drill Core Review

The site visit also included a review of the drill core logs and comparison to selected drill core intervals. The lithology descriptions and sample intervals in the drill logs were consistent with the drill core intervals reviewed. Table 12.5 lists the selected drill core intervals examined during the site visit.

Table 12.5 Selected Drill Core Intervals Examined

Deposit	Drill Hole	From (m)	To (m)
Malikoundi/Boto 2	DBDD-2190	80	160
	DBDD-2302	0	65
	DBDD-2301	60	80
	DBDD-2312	20	40
	DBDD-2078	0	100
	DBDD-2075	20	220
	DBDD-2270	250	350
	DBDD-2116	75	200
	DBDD-2259	325	570
	DBDD-2228	25	125
	DBDD-2126	0	300
	DBDD-2208	150	275
	DBDD-2241	100	150
	DBDD-2242	20	60
Boto 5	DBDD-2012	40	100
	DBDD-2013	150	210
	DBDD-2018	0	100
	DBDD-2022	0	120
Boto 6	DBDD-2046	178	211
	DBDD-2099	139	172
Boto 4	DBDD-2090	137	182
	DBDD-2138	176	202

Density Measurements

Density measurements were collected from diamond drill holes within the Malikoundi/Boto 2, Boto 6 and Boto 4 deposit areas between 2012 and 2018. The density measurements were collected nominally at 25 m intervals along selected drill holes. The core samples were naturally dried and density was measured on site using the water immersion method. Regolith and more porous samples were wrapped in plastic film before immersion.

12.3 QP Opinion

The QP is of the opinion that the database is adequate and representative to support a resource estimate of the Boto deposit for the level of study presented in this report. The QP is also of the opinion that the core descriptions, sampling procedures, and data entries were conducted in accordance with industry standards.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

IAMGOLD has conducted extensive metallurgical testwork on the Boto ore deposit since 2013. The testwork results were analysed and used in flowsheet development and inputs into the process design criteria.

The testwork programs investigated the following main topics:

- Head analyses and mineralogy.
- Comminution.
- Pre-concentration (gravity separation, flotation).
- Cyanidation (whole ore CIL, gravity concentrate intensive cyanidation, gravity tails CIL).
- Environmental tests.

The metallurgical reports issued to date in chronological order include:

- Hendry, Lesley. "An Investigation into the Recovery of Gold from the Boto Project", Project 14037-001, August 2013. SGS Mineral Services.
- Delaney, Vivien. "An Investigation into the Recovery of Gold from the Boto Project", Project 14573-001, December 2014. SGS Mineral Services.
- Chaisson, Guillaume. "An Investigation into the Grindability Characteristics of Forty-Two Samples from the Boto Project", Project 15080-001, November 2015. SGS Mineral Services.
- Desharnais, Guy. "Sample Selection Report for Boto Deposit – Senegal", June 2015. SGS Geostat.
- Zhou, Huyun, et al. "An Investigation into the Mineralogical Characteristics and Gold Deportment of One Leach Residue from the Boto Project", Project 15080-001, June 2016. SGS Mineral Services.
- Jackman, Rene. "An Investigation into the recovery of Gold from the Boto Project Samples", Project 15080-001, November 2016. SGS Mineral Services.
- Halliday, Matthew. "Sample Selection Report for Boto Deposit – Senegal", June 2017. SGS Geostat.
- Zhou, Huyun, et al. "An Investigation into Gold Deportment of Two Composite Samples from the Boto Project", Project 15080-003, April 2018. SGS Mineral Services.
- Zhou, Huyun, et al. "An Investigation into Gold Deportment of Four Samples from the Boto Project", Project 15080-003, June 2018. SGS Mineral Services.
- MacDonald, James. "An investigation into Recovery of Gold from Boto Project Samples", Project 15080-003, July 2018. SGS Mineral Services.

13.1 Review of Previous Metallurgical Tests

The sections to follow provide a high-level summary of the metallurgical findings from each of the previous testwork program. For more details, refer to the individual metallurgical report.

13.1.1 SGS 2013 Testwork Program

The testwork conducted in 2013 was a scoping level metallurgical test program supervised by Pierre Pelletier of IMG. The scope of work for the program included head analyses, and tests on the Bond work index, gravity separation, whole ore carbon-in-leach (CIL), gravity tailings CIL, preg-robbing, and ore acid generating potential.

The location and rock type of the samples used in this program are shown in Table 13.1.

Table 13.1 SGS 2013 Sample Area Location and Rock Type

Sample ID	Area	Rock Type
Met #1	Boto 2 North	Pelite
Met #2	Boto 2 North	Cipolin
Met #3	Boto 2 North	Sandstone
Met #4	Boto 2	Sandstone
Met #5	Boto 4	Sandstone
Met #6	Boto 6	Sandstone
Met #7	Boto 5	Albitite - Saprolite
Met #8	Boto 5	Albitite - Saprock
Met #9	Boto 5	Albitite – Fresh Rock

Head Analyses

Selected head analysis results are shown in Table 13.2. The gold grade was analyzed using the screened metallic protocol, while other metals were analysed by ICP scan.

Table 13.2 SGS 2013 Selected Head Analysis Results

Element	Met #1	Met #2	Met #3	Met #4	Met #5	Met #6	Met #7	Met #8	Met #9
g/t Au	10.7	2.43	2.85	2.68	5.81	2.15	1.48	1.01	4.16
g/t Au	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
g/t Al	55100	69200	47500	56700	42000	47700	153000	167000	64100
g/t Ca	45600	51700	25600	23600	22700	16000	303	332	5420
g/t Cu	22.1	4.1	12.2	8.3	12.2	7.7	6.1	406	141
g/t Fe	85600	36900	54600	56700	62400	33800	52400	9130	27400
g/t K	10900	9290	7830	3370	3210	5220	175	818	529
g/t Mg	23800	36900	14800	19600	11600	8900	252	1900	13600
g/t Mn	729	489	348	300	328	323	5.5	31.1	230
g/t Na	25200	30200	23700	36400	28500	29300	625	810	43500
g/t Ni	41	65	32	35	43	22	100	188	146
g/t P	513	666	268	532	230	240	97	106	612
g/t Sr	60.3	163	43.3	66.5	77.4	68.1	11.6	28.4	82.1

Element	Met #1	Met #2	Met #3	Met #4	Met #5	Met #6	Met #7	Met #8	Met #9
g/t Ti	2140	2880	1690	2490	1310	1680	4010	6410	2110
g/t V	66	126	48	82	48	47	15	101	31
g/t Y	10.5	15.6	7.6	9.2	7.3	7.3	13.2	10.8	12.1
g/t Zn	19	12	10	12	57	7	33	14	18

Gold grades calculated from screened metallic analysis ranged from 1.01 to 10.7 Au g/t, with 3% to 72% of gold reporting to the coarse fraction. Silver grades were below the detection limit of 0.5 g/t Au for all samples except for Met #6 which was at 0.9 g/t Au.

Met #7 and 8 appear to contain different chemical make-up from the other samples as they were both higher in Al, Ni and Ti, and lower in Ca, K, Mg, Mn, Na, P and Sr in comparison to the other samples.

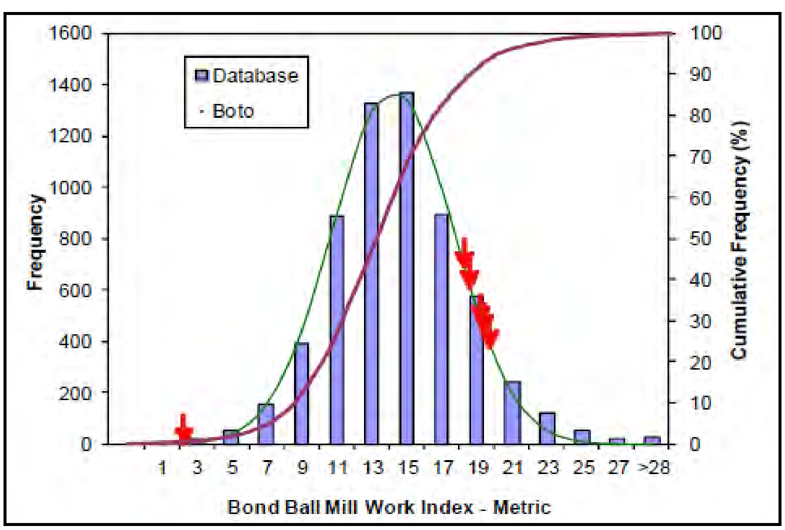
Grindability Testing

Met #1, 3, 4, 5, 6, and 8 were subjected to Bond ball mill grindability test at a grind size of 106 µm (150 Mesh). A summary of the results are shown in Table 13.3, with a comparison to the A. R. MacPherson Grinding Specialist database also shown in Figure 13.1.

Table 13.3 SGS 2013 Bond Ball Mill Grindability Test Summary

Sample ID	Grind Mesh	Feed Size, F_{80} (µm)	Product Size, P_{80} (µm)	Work Index, BWi (kWh/t)	SGS Hardness Percentile
Met #1	150	2,499	82	19.7	91
Met #3	150	2,457	84	19.2	90
Met #4	150	2,464	83	18.6	86
Met #5	150	2,541	86	18.3	85
Met #6	150	2,422	86	19.4	90
Met #8	150	2,211	62	2.2	0

Figure 13.1 SGS 2013 BWi Results Compared to BWi Database



At the time of this testwork, 3232 Bond ball mill work indices were available in the database. Five of the samples tested were considered as very hard, while one of them (Met #8) was considered as extremely soft.

Gravity Separation Testwork

Gravity separation tests were conducted for Met #1, 3, 4, 5, 6 and 8 at a targeted grind P_{80} of 150 µm. A summary of the results are shown in Table 13.4.

Table 13.4 SGS 2013 Gravity Separation Testwork Results

Sample	Test No.	Feed Size P_{80}, µm	Gravity Conc.		Gravity Tail Assay* g/t Au	Gravity Au Recovery %	Head Calculated g/t Au
			Wt. %	Assay, Au g/t*			
Met #1	G-1	171	0.091	3130	5.34	34.9	8.20
Met #3	G-2	151	0.087	140	3.16	3.7	3.28
Met #4	G-3	154	0.106	442	2.47	15.9	2.93
Met #5	G-4	135	0.088	1518	5.01	21.2	6.34
Met #6	G-5	164	0.061	245	3.72	3.9	3.86
Met #8	G-6	59	0.033	685	0.86	20.8	1.09
	G-7	51	0.013	2283	0.76	27.6	1.05
	G-8	54	0.031	626	0.87	18.3	1.06

** Knelson + Mozley Tailing = the weighted average calculated head grade from the test(s) completed on that product.*

Gold recovery for the six samples ranged from 4% to 35% with the concentrate gold grade ranging from 140 to 3,130 g/t Au.

Whole Ore CIL Testwork

Nine of the samples were subjected to whole ore CIL testwork (bottle roll) at a targeted grind P_{80} of 90 µm. The bottle roll test parameters included pulp density of 40% solids, leaching at 48-hours, addition of activated carbon at a concentration of 10 g/L, cyanide concentration maintained at 0.5 g/L NaCN, and pH of 10.5 to 11. The results are summarized in Table 13.5.

Table 13.5 SGS 2013 Whole Ore CIL Test Results

Sample ID	CN Test No.	Grind Size P_{80} (µm)	Reagent Consumption (kg/t)		Au Extraction %	Residue, Au g/t	Head Grade, g/t Au	
			NaCN	CaO			Calc'd	Direct
Met #1	CN-27	91	0.91	0.76	88.2	1.32	11.2	10.7
	CN-28	96	0.59	0.61	88.6	1.32	11.5	
Met #2	CN-19	82	0.93	0.76	91.8	0.19	2.26	2.43
Met #3	CN-29	89	0.94	0.71	82.0	0.45	2.50	2.85
Met #4	CN-30	92	0.73	0.71	85.7	0.39	2.70	2.68
Met #5	CN-31	90	0.71	0.53	91.8	0.40	4.89	5.81
Met #6	CN-32	92	0.86	0.58	88.2	0.19	1.57	2.15
Met #7	CN-20	67	0.65	1.8	25.0	1.92	2.55	1.48
Met #9	CN-21	82	0.74	0.76	95.6	0.15	3.39	4.16

The majority of the whole ore CIL results showed gold extraction in the range of 82 to 96% with only one exception being sample Met #7 at 25%. This sample requires further investigation into its poor leach performance. Oxygen sparging was used instead of air for Met #1 Test No. CN-28 and the results showed a noticeable decrease in the cyanide and lime consumptions.

Gravity Tailings CIL Testwork

The gravity tailings from Met #1, 3, 4, 5, 6 and 8 were also subjected to CIL testwork to study the impact of grind size on gold extraction. The test parameters were similar to the whole ore CIL tests, except that the sample grind sizes were at incremental targeted P_{80}'s of 90, 75, and 53 µm.

As seen in Figure 13.2 and Table 13.6, the overall gold extraction from gravity concentration and gravity tailings CIL ranged from 85 to 95%. All the samples exhibited a positive correlation between the finess of the grind and the extraction achieved.

Figure 13.2 Gold Extraction % vs. Grind Size

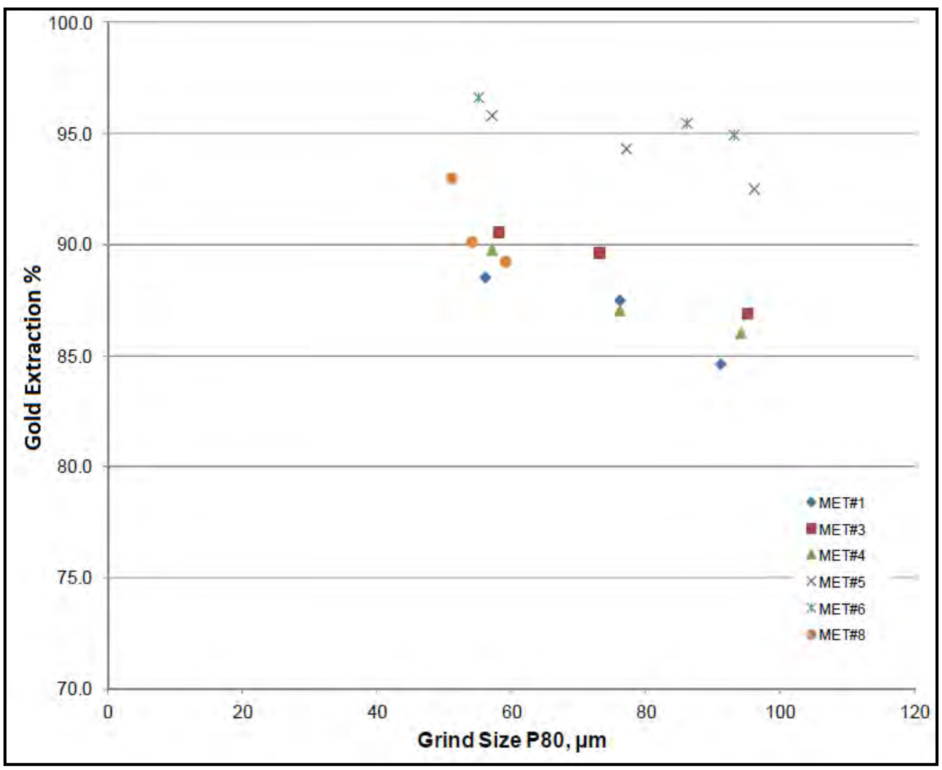


Table 13.6 SGS 2013 Gravity Tailings CIL Test Results

Sample	Feed from Test No.	CN Test No.	CN Feed Size P_{80} (µm)	Reagent Cons (kg/t)		Au Extraction %							Residue Au, g/t	Head Au, g/t			
				NaCN	CaO	CN 48 h	Normalised[1] (to avg calc CN feed grade) CN	Overall Grav	Overall CN	Overall Grav + CN	Normalised[1] (to avg calc CN feed grade) Grav + CN	Normalised[2] (to direct feed grade) Grav + CN		CN Calc	Avg Calc CN Feed Grade	Grav + CN	Direct (SM)
		CN-1	91	0.75	0.89	73.8	76.4	34.9	48.1	82.9	**84.6**	**88.3**	1.26	4.81			
Met #1	G-1	CN-2	76	0.81	0.90	88.3	80.8	34.9	57.5	92.3	**87.5**	**90.5**	1.03	8.71	5.34	8.20	10.74
		CN-3	56	1.02	0.87	62.5	82.4	34.9	40.7	75.6	**88.5**	**91.2**	0.94	2.51			
		CN-4	95	0.86	0.74	82.5	86.4	3.7	79.4	83.2	**86.9**	**84.9**	0.43	2.46			
Met #3	G-2	CN-5	73	0.87	0.78	86.5	89.3	3.7	83.3	87.0	**89.7**	**88.1**	0.34	2.52	3.16	3.28	2.85
		CN-6	58	1.01	0.79	93.1	90.2	3.7	89.7	93.4	**90.6**	**89.1**	0.31	4.51			
		CN-7	94	0.70	0.78	84.3	83.4	15.9	70.9	86.8	**86.0**	**84.7**	0.41	2.61			
Met #4	G-3	CN-8	76	0.73	0.71	83.5	84.6	15.9	70.2	86.2	**87.1**	**85.8**	0.38	2.31	2.47	2.93	2.68
		CN-9	57	1.05	0.72	87.9	87.9	15.9	73.9	89.8	**89.8**	**88.8**	0.30	2.48			
		CN-10	96	0.69	0.68	94.1	90.5	21.2	74.2	95.4	**92.5**	**91.8**	0.48	8.06			
Met #5	G-4	CN-11	77	0.83	0.66	91.7	92.8	21.2	72.3	93.4	**94.3**	**93.8**	0.36	4.32	5.01	6.34	5.81
		CN-12	57	0.92	0.62	90.0	94.7	21.2	71.0	92.1	**95.8**	**95.4**	0.27	2.64			
		CN-13	93	0.82	0.65	86.6	94.8	3.9	83.2	87.1	**95.0**	**90.9**	0.20	1.46			
Met #6	G-5	CN-14	86	0.82	0.63	88.0	95.3	3.9	84.6	88.5	**95.5**	**91.9**	0.18	1.46	3.72	3.86	2.15
		CN-15	55	0.95	0.66	98.4	96.5	3.9	94.6	98.5	**96.6**	**94.0**	0.13	8.23			
		CN-16	59	1.35	1.07	87.2	86.7	19.1	70.6	89.6	**89.3**	**89.1**	0.11	0.86		1.09	
Met #8*	G-6	CN-17	51	1.35	1.44	89.5	90.3	27.6	64.8	92.4	**93.0**	**92.1**	0.08	0.76	0.83	1.05	1.01
		CN-18	54	1.26	1.48	88.5	87.9	18.3	72.3	90.6	**90.1**	**90.1**	0.10	0.87		1.06	

Notes:

**Sample is clay - difficult to get proper size analysis - mixed feed sample of Met #8 with vigorous agitation for 24 h and performed Malvern - D80 = 19 µm*

*[1]Normalized Au Extraction = 100-(residue Au assay/average calculated CN feed grade)*100*

*[2]Normalized Au Extraction = 100-(residue Au assay/direct head grade (SM))*100*

SM = Screened Metallics

Standard preg-robbing tests were also conducted on Met #1, 3, 4, 5 and 6 and the results are shown in Table 13.7. After 24 hours of slurrying with synthetic gold stock solution, the pregnant solution of all five samples increased in gold tenor, indicating that there is no potential for preg-robbing. However, it is recommended that future testwork be conducted with and without carbon to provide a more definitive result.

Table 13.7 SGS 2013 Preg-robbing Test Results

CN Test #	Sample	Initial (mg/L Au)	Solution Assay, mg/L Au			
			1 hr	3.5 hr	6 hr	24 hr
PR-1	Met 1	8.44	8.69	9.96	10.3	10.3
PR-2	Met 3	8.44	7.85	9.19	9.56	10.6
PR-3	Met 4	8.44	9.28	9.48	9.52	10.6
PR-4	Met 5	8.44	9.28	8.89	9.59	11.6
PR-5	Met 6	8.44	10.6	10	9.72	11.1

Environmental Testwork

Modified acid-base accounting (ABA) tests were conducted on all nine samples to assist in determining the ore's acid generating potential. The results are shown in Table 13.8.

Table 13.8 SGS 2013 Modified Acid-Base Accounting Test Results

Parameter	Unit	Met #1	Met #2	Met #3	Met #4	Met #5	Met #6	Met #7	Met #8	Met #9
Paste pH	units	8.62	9.02	8.79	8.93	9.29	9.35	4.57	5.03	7.30
Fizz Rate	—	4	4	3	4	4	3	1	1	1
Sample weight	g	2.12	1.99	2.05	2.07	2.00	2.12	2.03	2.03	2.14
HCl added	mL	85.2	126.3	69.3	40.0	40.0	20.0	20.0	20.0	20.0
HCl	Normality	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
NaOH	Normality	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
NaOH to pH=8.3	mL	22.12	44.53	30.82	18.19	17.78	6.24	20.33	19.98	17.82
Final pH	units	1.88	1.62	1.55	1.59	1.52	1.82	1.01	1.06	1.19
NP	t $CaCO_3$/1000 t	149	206	94	53	56	32	-0.8	0.0	5.1
AP	t $CaCO_3$/1000 t	110	10.9	78.8	36.9	35.6	26.9	185	21.9	36.9
Net NP	t $CaCO_3$/1000 t	38.8	195	15.0	15.8	19.9	5.62	-185	-21.9	-31.8
NP/AP	ratio	1.35	18.8	1.19	1.43	1.56	1.21	0.00	0.00	0.14
S	%	4.06	0.42	2.8	1.29	1.28	1.06	5.76	0.898	1.31
Acid Leachable SO_4	%	0.54	0.07	0.28	0.11	0.14	0.20	< 0.01	0.20	0.13
Sulphide	%	3.52	0.35	2.52	1.18	1.14	0.86	5.91	0.70	1.18
C	%	1.94	2.6	1.32	0.618	0.674	0.402	0.131	0.075	0.091
CO_3	%	7.39	12.2	4.85	2.48	2.67	1.46	0.01	0.015	0.055

NP = Neutralization Potential
AP = Acid Generating Potential
PAG = Potential for Acid Generation

Met #1, 3, 4, 5 and 6 showed NP/AP ratios between 1 and 3 indicating the potential for acid generation (PAG), therefore, long term kinetic humidity cell testing should be conducted for these samples in the future.

Met #7, 8 and 9 showed negative NP values and high sulphide contents, indicating that these samples are more than likely acid generating.

Met #2 showed NP greater than 20 and NP/AP ratio greater than 3, indicating that the sample is potentially acid neutralising (PAN).

13.1.2 SGS 2014 Testwork Program

The testwork conducted in 2014 was also supervised by Mr. Pierre Pelletier, and was a continuation of the previous scoping-level metallurgical test program in 2013. The scope of work for the program included head analyses, flotation optimization tests, whole ore cyanidation tests, and ore acid generating potential tests.

The location and rock type of the samples used in this test program are shown in Table 13.9. Most of the samples were half NQ cores with only some HQ cores.

Table 13.9 SGS 2014 Sample Area Location and Rock Type

Composite ID	Area	Rock Type
Comp A	Malikoundi	Sandstone
Comp B	Malikoundi	Carbonate/Cipolin
Comp C	Malikoundi	Pelite

Head Analyses

Selected head analysis results are shown in Table 13.10.

Table 13.10 SGS 2014 Selected Head Analysis Results

Element	Comp A	Comp B	Comp C
g/t Au	1.81 (SM)	1.70 (SM)	1.01 (SM)
g/t Ag	<2	<2	<2
g/t Al	63800	72300	36600
g/t Ba	224	106	124
g/t Be	0.68	0.89	0.5
g/t Ca	26800	43600	137000
g/t Co	18	14	26
g/t Cr	95	100	34
g/t Cu	32.7	25.3	141
g/t Fe	57100	40500	52400
g/t K	4870	8680	7290
g/t Li	22	18	8
g/t Mg	20800	24300	59900
g/t Mn	444	476	1350
g/t Mo	<5	<5	10
g/t Na	37900	41400	17700
g/t Ni	31	42	48
g/t P	447	651	328
g/t Sr	61.3	80.8	102
g/t Ti	2440	2910	1430
g/t V	64	79	51
g/t Y	10.3	13.9	11.9
g/t Zn	12	26	17

**Results under detection limit have not been shown.*
SM = Screened Metallics

The three composites had similar chemical make-up with the only noticeable difference in the calcium content of Comp C where it is 3 to 5 times higher than the other two composites.

Grindability

SAG Mill Comminution (SMC) tests were performed for the three composites at three size fractions. The SMC test results are summarized in Table 13.11.

Table 13.11 SGS 2014 SMC Test Results

Sample Name	A	b	A x b	Hardness Percentile	t_a[1]	DWI (kWh/m^3)	M_{ia} (kWh/t)	M_{ih} (kWh/t)	M_{ic} (kWh/t)	Relative Density
Comp A (-31.5+26.5mm)	91.6	0.36	33.0	78	0.31	8.4	22.7	17.6	9.1	2.77
Comp A (-22.4+19.0mm)	91.6	0.31	28.4	88	0.26	10.0	25.9	20.7	10.7	2.79
Comp A (-16.0+13.2mm)	91.6	0.36	33.0	78	0.31	8.2	22.7	17.5	9.1	2.73
Comp A - Overall	**91.6**	**0.34**	**31.1**	**82**	**0.29**	**8.9**	**23.8**	**18.6**	**9.6**	**2.76**
Comp B (-31.5+26.5mm)	80.5	0.44	35.4	72	0.33	7.8	21.6	16.4	8.5	2.74
Comp B (-22.4+19.0mm)	80.5	0.37	29.8	85	0.29	9.1	24.7	19.4	10.0	2.71
Comp B (-16.0+13.2mm)	80.5	0.44	35.4	72	0.33	7.7	21.6	16.4	8.5	2.73
Comp B - Overall	**80.5**	**0.42**	**33.8**	**76**	**0.32**	**8.2**	**22.6**	**17.4**	**9.0**	**2.73**
Comp C (-31.5+26.5mm)	81.6	0.47	38.4	65	0.35	7.3	20.0	15.0	7.8	2.82
Comp C (-22.4+19.0mm)	81.6	0.38	31.0	82	0.28	9.2	24.0	18.9	9.8	2.82
Comp C (-16.0+13.2mm)	81.6	0.47	38.4	65	0.35	7.3	20.0	15.0	7.8	2.82
Comp C - Overall	**81.6**	**0.44**	**35.9**	**71**	**0.33**	**7.9**	**21.3**	**16.3**	**8.4**	**2.82**

[1] The t_a value reported as part of the SMC procedure is an estimate

The three composites were classified as hard with respect to resistance to impact breakage (A x b), with Comp C being the softest of the three. The average relative densities varied from 2.73 to 2.82.

Bond ball mill work index and Bond abrasion tests were also performed for the three composites. The results are presented in Table 13.12, with a comparison to the A. R. MacPherson Grinding Specialist database shown in Figure 13.3 and Figure 13.4.

Table 13.12 SGS 2014 Bond Ball Mill Grindability Test and Abrasion Test Results

Sample ID	Grind Mesh	Feed Size, F_{80} (µm)	Product Size, P_{80} (µm)	Work Index, BWi (kWh/t)	Hardness Percentile	Ai (g)	Percentile of Abrasivity
Comp A	150	2,497	82	20.1	93	0.706	91
Comp B	150	2,546	78	21.1	95	0.612	88
Comp C	150	2,400	75	14.9	57	0.202	42

Figure 13.3 SGS 2014 BWi Results Compared to BWi Database

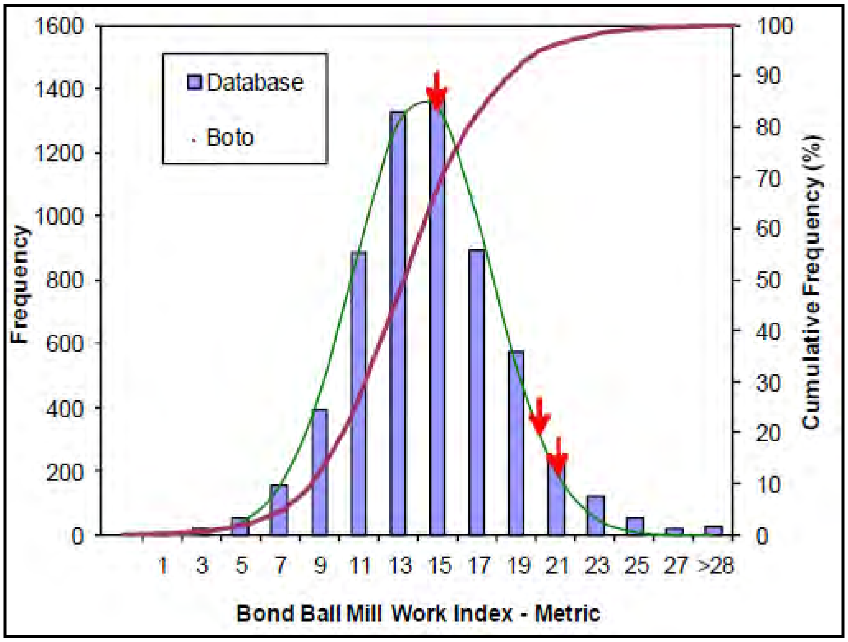
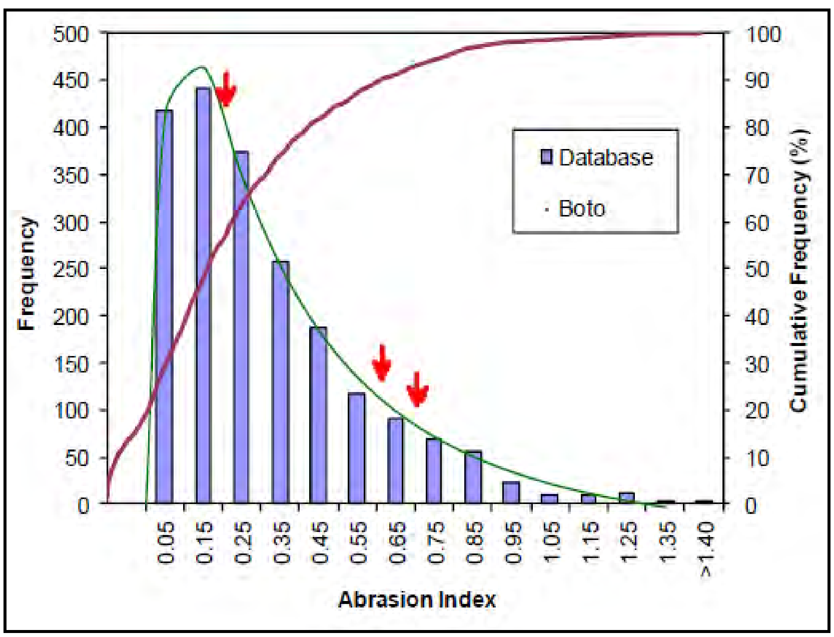


Figure 13.4 SGS 2014 Bond Abrasion Results Compared to Abrasion Database



Gravity Separation Testwork

The three composites were subjected to gravity separation testwork and the results are shown in Table 13.13.

Table 13.13 SGS 2014 Gravity Separation Testwork Results

Composite	Test No.	Feed Size P_{80}, µm	Gravity Concentrate		Gravity Tail Calc'd Assay Au g/t	Gravity Au Recovery %	Calc'd Head Grade g/t Au
			Wt. %	Grade, g/t Au			
Comp A	G-1	52	0.06	434	1.6	14.7	1.83
Comp B	G-2	50	0.12	240	1.1	21.0	1.38
Comp C	G-3	47	0.09	373	0.8	29.5	1.09

The gravity separation result indicated gravity recoverable gold of 15 % to 30%. The recommendation at the time of this testwork suggested that gravity could be implemented in a scoping-level flowsheet to decrease complexity and size of downstream cyanidation circuit.

Flotation Testwork

Whole ore flotation tests were conducted using 2 kg charges at different grind sizes for Comp A, B, and C. The results are presented in Figure 13.5, Figure 13.6, and Figure 13.7.

Figure 13.5 SGS 2014 Comp A Flotation Gold Extraction vs. Mass Pull

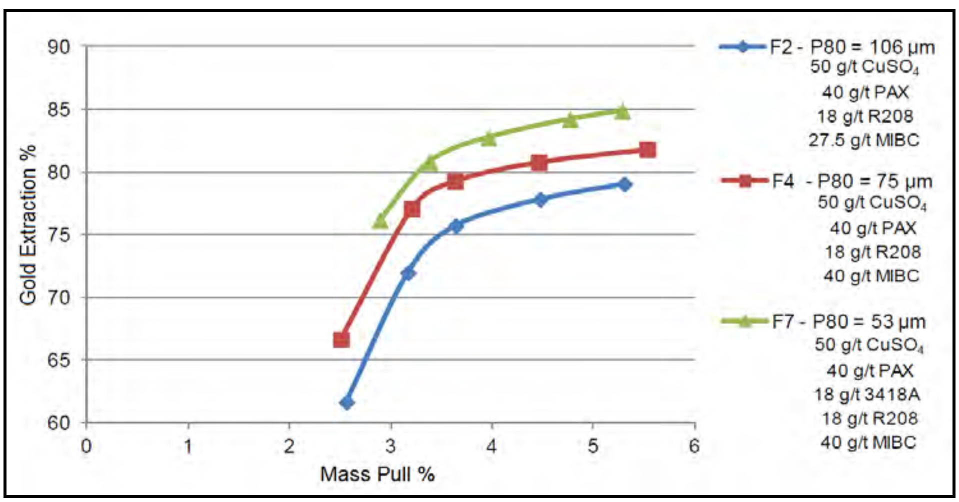


Figure 13.6 SGS 2014 Comp B Flotation Gold Extraction vs. Mass Pull

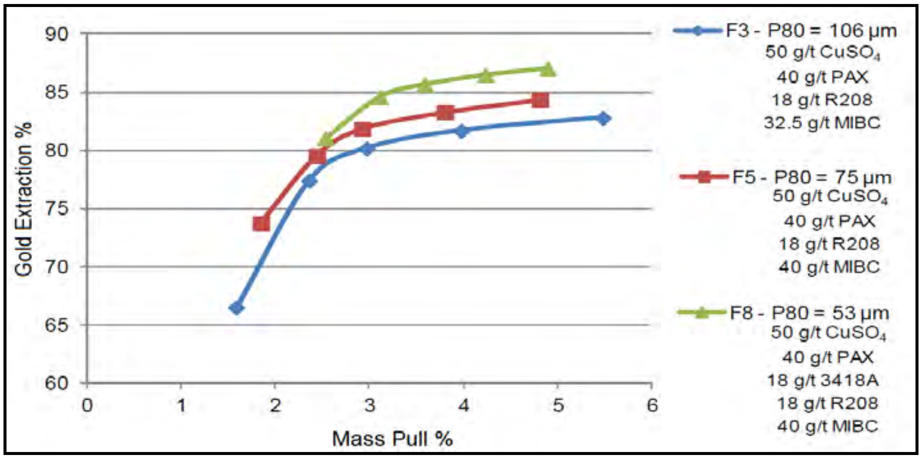
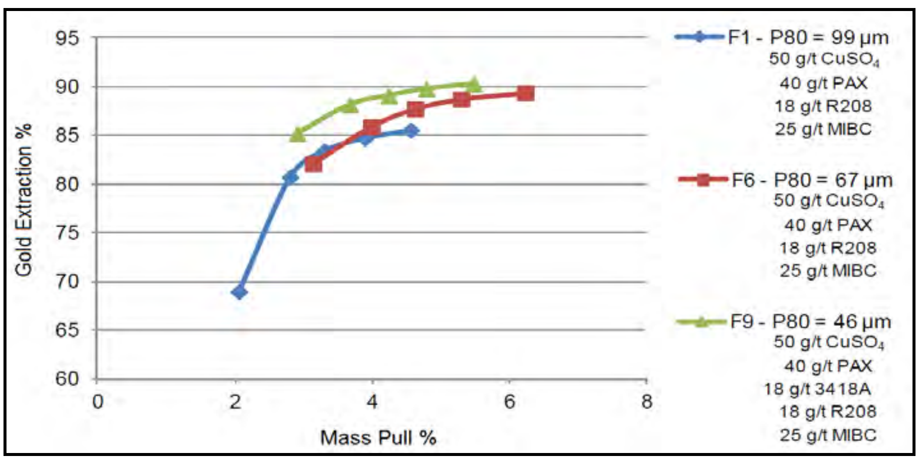


Figure 13.7 SGS 2014 Comp C Flotation Gold Extraction vs. Mass Pull



The three composites responded well to flotation treatment with gold extraction increasing with decreasing grind size as expected. Comp A had gold extraction ranging from 79% at a P_{80} of 99 µm to 85% at a P_{80} of 46 µm. Comp B had gold extraction ranging from 83% at a P_{80} of 100 µm to 87% at a P_{80} of 53 µm. Comp C had gold extraction ranging from 86% at a P_{80} of 99 µm to 90% at a P_{80} of 53 µm.

Bulk sulphide flotation tests were also conducted using 10 kg charges to generate material for downstream cyanide leaching. The results were similar to the 2 kg charges. Comp A had a gold extraction of 82.1% and sulphide extraction of 96.7% at a P_{80} of 52 µm. Comp B had a gold extraction of 83.5% and sulphide extraction of 97% at a P_{80} of 50 µm. Comp C had a gold extraction of 89.3% and sulphide extraction of 97.5% at a P_{80} of 47µm.

Gravity tails from the gravity separation tests (G-1, G-2, and G-3) were also subjected to flotation tests using the same reagent regime as the whole ore flotation but with additional collector 3418A. However, the overall gold extraction (gravity combined with flotation of gravity tails) results did not improve when compared to the whole ore flotation results at the same grind size.

Coarse Bottle Roll Leach Testwork (Heap Leach Amenability)

The three composites were subjected to coarse bottle roll leach testwork at four different crush sizes, 19mm, 12.7 mm, 6.3 mm, and 1.7 mm, to study the ore's amenability to heap leaching. The leach conditions for the testwork were 50% solids pulp density, pH of 10.5 to 11, NaCN concentration of 0.5 g/L, 28 days retention time and 1-minute agitation every hour. Pregnant solution from each composite was submitted for gold analysis at 8 hours, 1, 2, 4, 6, 8, 14, 21 and 28 days to calculate the gold extraction percent. The results are shown in Table 13.14 and in Figure 13.9 to Figure 13.10.

Table 13.14 SGS 2014 Heap Leach Amenability

Feed	CN Test No.	Feed Size	NaCN Cons. kg/t	CaO Added kg/t	Extr'n % Au	Residue g/t Au	Calc'd Head Grade g/tAu	SFA Head Grade g/t Au	Direct Head g/t Au
Comp A	CN-1	19mm	0.26	0.77	34.7	1.02	1.56	1.22	1.81
	CN-2	12.7mm	0.24	0.69	33.6	1.25	1.89	1.76	
	CN-3	6.3mm	0.24	0.77	47.6	1.21	2.31	1.88	
	CN-4	1.7mm	0.26	0.79	62	0.72	1.88	1.87	
Comp B	CN-5	19mm	0.29	0.72	24	1.18	1.55	1.49	1.70
	CN-6	12.7mm	0.26	0.69	28.8	0.88	1.23	1.32	
	CN-7	6.3mm	0.25	0.73	42.5	1.11	1.93	2.05	
	CN-8	1.7mm	0.21	0.91	63.2	0.55	1.5	1.51	
Comp C	CN-9	19mm	0.42	0.71	22.5	1.54	1.99	1.22	1.01
	CN-10	12.7mm	0.51	0.76	32.5	0.7	1.03	1.12	
	CN-11	6.3mm	0.57	0.98	45.4	0.68	1.24	2.37	
	CN-12	1.7mm	0.54	1.01	71.6	0.29	1.04	1.09	

Figure 13.8 SGS 2014 Comp A Coarse Bottle Roll Gold Extraction vs. Time

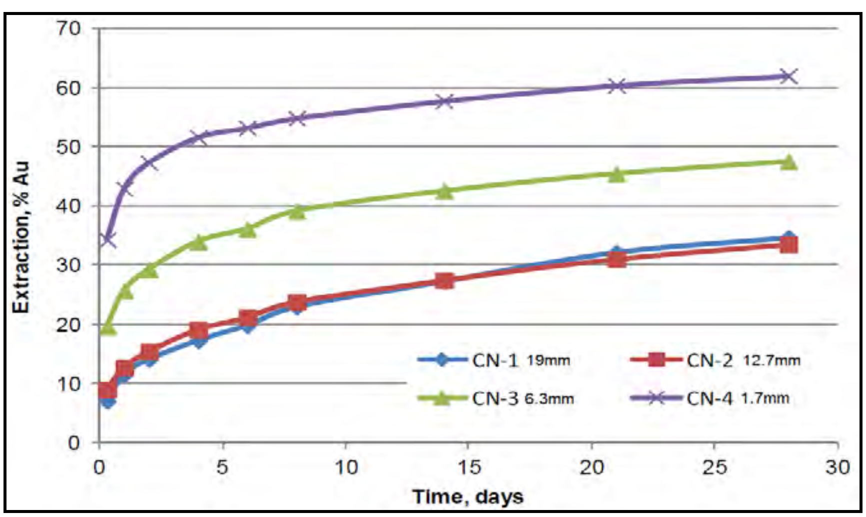


Figure 13.9 SGS 2014 Comp B Coarse Bottle Roll Gold Extraction vs. Time

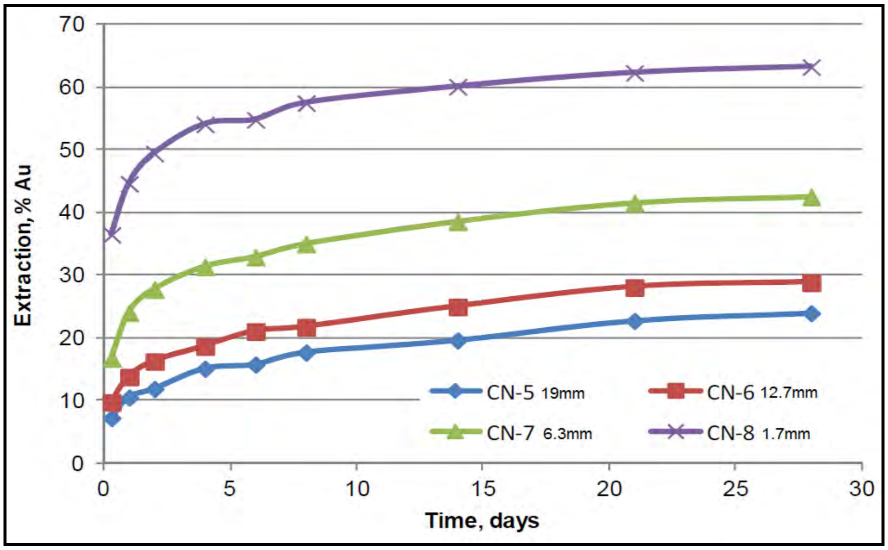
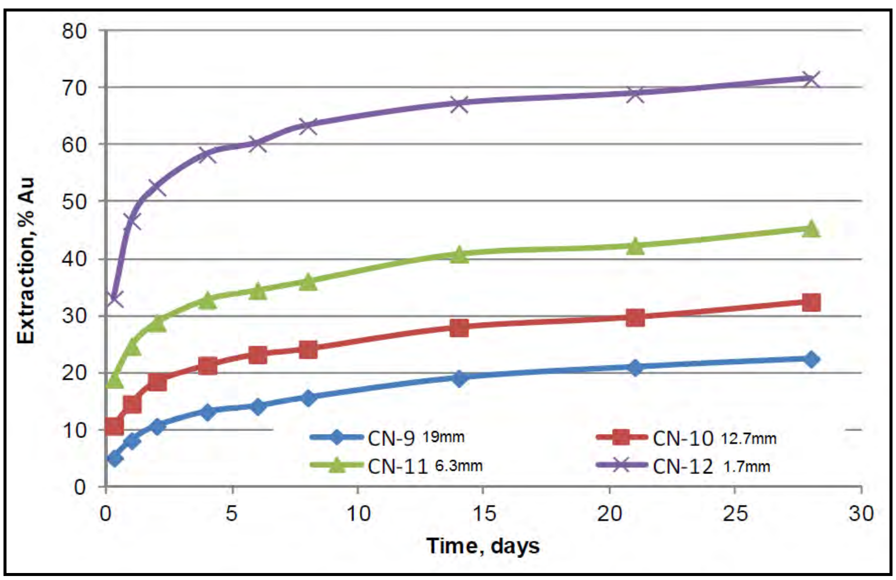


Figure 13.10 SGS 2014 Comp C Coarse Bottle Roll Gold Extraction vs. Time



The general trend from the coarse bottle roll leach results indicates that the finer the crush size, the higher the gold extraction. None of the composites reached a plateau in the leach curves during the 28 days duration. At the coarsest crush size, gold extractions after 28 days were 34.7%, 24.0%, and 22.5% for Comp A, B, and C, respectively. At the finest crush size, gold extractions were 62.0%, 63.2%, and 71.6% for Comp A, B, and C, respectively. The cyanide consumption was between 0.2 kg and 0.3 kg NaCN/t for Comp A and B, and slightly higher for Comp C between 0.4 kg to 0.6 kg NaCN/t. Lime consumption ranged from 0.7 kg to 1 kg CaO/t for all the composites.

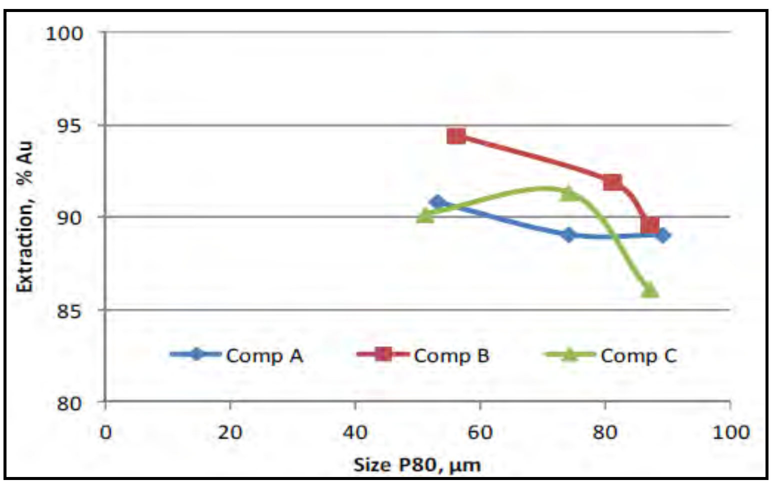
Whole Ore Cyanidation

Carbon-in-leach (CIL) bottle roll tests were conducted on whole ore at grind P_{80}'s 90, 75 and 53 µm for the three composites. The leach conditions were 40% solids, pH of 10.5 to 11, 0.5 g/L NaCN, 10g/L activated carbon addition, and 48-hours retention time. At the completion of the test, carbons were removed from the pulp by filtering and were then washed and submitted for gold analysis along with the final leach solution and residue samples. The results are shown in Table 13.15 and Figure 13.11.

Table 13.15 SGS 2014 Whole Ore Cyanidation Results

Feed	Test No.	Feed Size P80, µm	NaCN Cons. kg/t	CaO Added kg/t	Extr'n @ 48 h, %Au	Residue g/t Au	Calc'd Head Grade g/t Au	Direct Head Grade g/t Au
Comp A	CN-13	89	1.0	0.5	89.1	0.2	1.83	1.81
	CN-14	74	1.0	0.6	89.1	0.22	2.02	
	CN-15	53	1.0	0.6	90.9	0.17	1.86	
Comp B	CN-16	87	1.1	0.6	89.6	0.15	1.45	1.70
	CN-17	81	1.0	0.6	91.9	0.12	1.43	
	CN-18	56	1.1	0.6	94.4	0.08	1.44	
Comp C	CN-19	87	1.2	0.8	86.1	0.14	0.97	1.01
	CN-20	74	1.1	0.9	91.4	0.09	1.04	
	CN-21	51	1.4	1.0	90.2	0.12	1.17	

Figure 13.11 SGS 2014 Whole Ore Cyanidation Gold Extraction vs. Grind Size



The different grind size tested had a smaller impact on Comp A gold extractions than the Comp B and C's as seen in Figure 13.11. The gold extractions from coarsest to finest grind size ranged from 89.1% to 90.9% for Comp A, 89.6 % to 94.4% for Comp B, and 86.1% to 90.2% for Comp C. The gold extraction for the two finer grind sizes of Comp C were very similar.

Cyanide consumptions ranged from 1.0 kg to 1.4 kg NaCN/t and lime consumption ranged from 0.5 kg to 1.0 kg CaO/t.

Cyanidation of Flotation Concentrate

CIL tests were conducted on flotation concentrate from each composite to assess the impact of different sodium cyanide concentrations, aeration and regrinding. The leach conditions were as follows:

- 25% solids pulp density.
- pH of 10.5 to 11.
- NaCN concentrations of 1 g/L, 2 g/L or 5g/L.
- Activated carbon addition of 10 g/L.
- Aeration with air or oxygen.
- With or without addition of lead nitrate.
- Retention time of 72 hours.
- Regrinding to a P_{80} of 23 µm for Comp A, 16 µm for Comp B, and 19 µm for Comp C.

The results from this testwork are presented in Figure 13.12 to Figure 13.14.

Figure 13.12 SGS 2014 Comp A Flotation Concentrate CIL Overall Gold Recovery

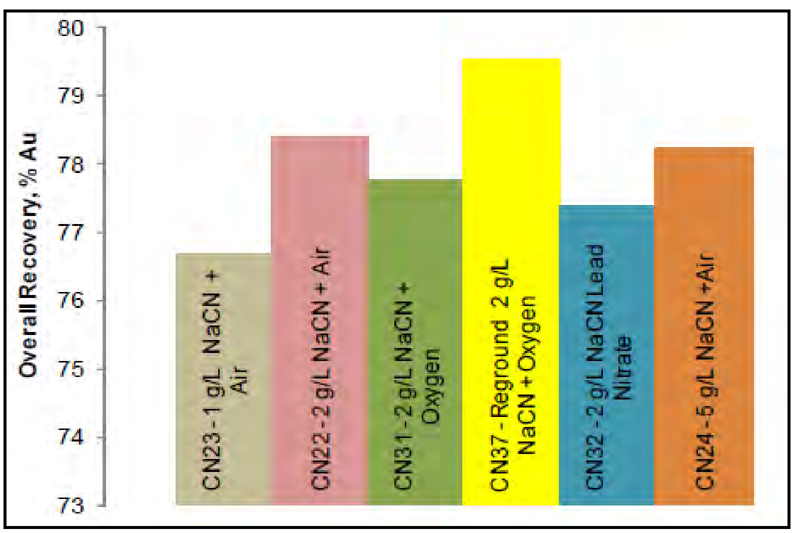


Figure 13.13 SGS 2014 Comp B Flotation Concentrate CIL Overall Gold Recovery

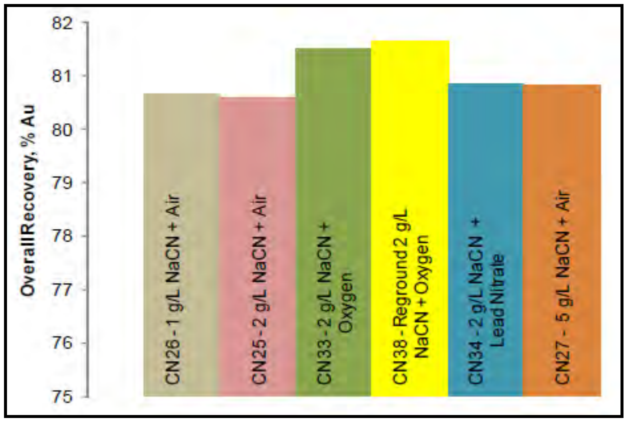
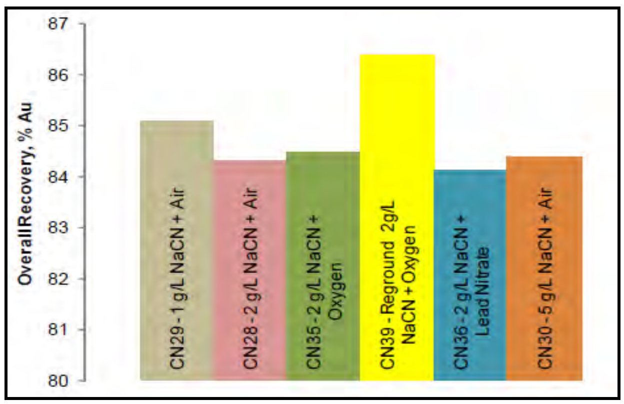


Figure 13.14 SGS 2014 Comp C Flotation Concentrate CIL Overall Gold Recovery



It was concluded by SGS that the optimum conditions for leaching flotation concentrate were having the sodium cyanide concentration at 2 g/L, sparging the pulp with oxygen and regrinding the pulp. Of all the parameters tested, regrinding the flotation concentrate provided the largest benefit for gold extraction.

Comparison of Different Process Options

The results for the three composites tested with different process options are presented in Table 13.16. Although the coarse bottle roll leach results for the 1.7 mm crush size provided better gold extraction, the results for the 12.7 mm crush size as provided in Table 13.16 is more realistic for heap leaching. Based on the comparison, the optimum metallurgical process was whole ore cyanidation, which yielded gold extraction exceeding 90% for the three composites.

Table 13.16 SGS 2014 Comparison of Selected Results for Different Process Options

Feed	Test Description	Test No.	Feed Size, P_{80}	NaCN Cons. kg/t	CaO Added kg/t	Overall Extr'n, % Au	Tailings Residue g/t Au	Calc'd Head Grade g/t Au	Direct Head Grade g/t Au
Comp A	Heap leach amenability	CN-2	12.7mm*	0.2	0.7	33.6	1.25	1.89	1.81
	Whole ore cyanidation	CN-15	53µm	1	0.6	90.9	0.17	1.86	
	Flotation	F7	53µm	---	---	84.9	0.31	1.99	
	Gravity + flotation	F10	53µm	---	---	81.2	0.36	1.83	
	Flotation concentrate CIL	CN-37	23µm	0.1	0.1	79.5	1.31	41.9	38.8
Comp B	Heap leach amenability	CN-6	12.7mm*	0.3	0.7	28.8	0.88	1.23	1.70
	Whole ore cyanidation	CN-18	56µm	0.6	0.6	94.4	0.08	1.44	
	Flotation	F8	53µm	---	---	87.1	0.21	1.54	
	Gravity + flotation	F11	53µm	---	---	80.5	0.28	1.38	
	Flotation concentrate CIL	CN-38	16µm	0.2	0.1	81.6	0.64	28.7	31.4
Comp C	Heap leach amenability	CN-10	12.7mm*	0.5	0.8	32.5	0.7	1.03	1.01
	Whole ore cyanidation	CN-21	51µm	1.4	1	90.2	0.12	1.17	
	Flotation	F9	46µm	---	---	90.3	0.11	0.97	
	Gravity + flotation	F12	53µm	---	---	87.2	0.15	1.09	
	Flotation concentrate CIL	CN-39	19µm	0.4	0.1	86.4	0.37	11.2	12.2

*100% passing

Environmental Testwork

The three composites were also subjected to modified ABA and Net Acid Generating (NAG) testing to determine the potential of the samples to generate acid drainage. The results are shown in Table 13.17 and Table 13.18.

Table 13.17 SGS 2014 Modified Acid-Base Accounting Results

Parameter	Unit	Comp A	Comp B	Comp C
Paste pH	units	9.27	9.56	8.65
Fizz Rate	---	3	3	3
Sample weight	g	2.02	2.04	2.01
HCl added	mL	53.8	84.8	256.8
HCl	Normality	0.1	0.1	0.1
NaOH	Normality	0.1	0.1	0.1
NaOH to pH=8.3	mL	21.61	32.52	64.68
Final pH	units	1.56	1.56	1.76
NP	t $CaCO_3$/1000 t	80	128	478
AP	t $CaCO_3$/1000 t	25.9	9.38	27.2
Net NP	t $CaCO_3$/1000 t	53.8	119	451
NP/AP	ratio	3.07	13.7	17.6
S	%	1.1	0.329	0.87
Acid Leachable SO_4 as S	%	0.27	< 0.01	< 0.01
Sulphide	%	0.83	0.30	0.87
C	%	1.03	1.68	6.27
CO_3	%	4.15	7.57	29.8

NP = Neutralization Potential
AP = Acid Generating Potential
PAG = Potential for Acid Generation

As seen in Table 13.17, all three composites had NP/AP ratio greater than 3 but less than 20 which classified them to be not potentially acid generating.

Table 13.18 SGS 2014 Net Acid Generation Results

Parameter	Unit	Comp A	Comp B	Comp C
Sample weight	g	1.5	1.5	1.5
Vol H_2O_2	mL	150	150	150
Final pH	units	10.89	11.20	11.23
NaOH	Normality	0.1	0.1	0.1
NaOH to pH=4.5	mL	0	0	0
NaOH to pH=7.0	units	0	0	0
NAG (pH 4.5)	kg H_2SO_4/tonne	0	0	0
NAG (pH 7.0)	kg H_2SO_4/tonne	0	0	0

The NAG results as seen in Table 13.18 further confirmed the conclusion that the three Malikoundi composites are not potentially acid generating with zero for all NAG values.

13.1.3 SGS 2015/2016 Testwork Program

In 2015, the Project entered into its prefeasibility study phase (PFS) and Mr. Jérôme Girard at IMG supervised additional metallurgical testwork with SGS Lakefield. A sample selection exercise with Mr. Guy Desharnais was also conducted at SGS Geostats during the time. The program began in 2015 with sample selection and grindability testwork, and ended with metallurgical testwork in 2016.

The metallurgical testing portion of the program included:

- Gravity separation testwork.
- Intensive cyanidation Testwork on Gravity Concentrate.
- Cyanidation on gravity tailings.
- Variability testwork.
- Environmental testwork.

The sections to follow provide a high-level description of the various aspects of the testwork program.

Sample Selection (2015)

The objective of the sample selection exercise was to provide a specific set of samples to meet the requirements for the Boto PFS. SGS Geostats analysed the Boto ore deposits and concluded that fresh rock from the Boto 2/Malikoundi pit and saprolite from the Boto 5 pit are the largest contributors to gold ounces (see Table 13.19).

Table 13.19 SGS 2015 Analysis on Gold Ozs Distribution for Boto Deposits

Material Type	Boto 2*	Boto 4	Boto 5	Boto 6	Total
Laterite	2%	1%	0%	0%	3%
Saprolite	2%	0%	**4%**	0%	7%
Transition	2%	0%	1%	0%	4%
Fresh Rock	**75%**	5%	1%	5%	86%
Total	81%	5%	7%	6%	100%

*Malikoundi and Boto 2 are denoted by only “Boto 2” in table.

The most abundant lithological types in the Boto 2/Malikoundi deposit were determined to be Pelite-rhythm, sandstone, cipolin and sandstone-rhythm. As for Boto 5, the most abundant were determined to be albitite, cipolin, and turbidite. Refer to Table 13.20 and Table 13.21 for analyses of the Geometallurgical database (from IMG) broken down by count, grade and lithology.

Table 13.20 SGS 2015 Analysis of Geometallurgical Database for Boto 2/Malikoundi

Lithology	Count	% Count	Mass (kg)	Length (m)	Rel% Au Ozs	Avg. Grade (g/t Au)
Pelite-rythm	1654	24%	5239.6	1659	28%	1.3
Pelite	1508	22%	4667.9	1514	18%	0.9
Sandstone	1251	18%	3942.9	1262	24%	1.4
Cipolin	580	8%	1828.9	582	5%	0.6
Sandst-rythm	330	5%	1057.5	330	8%	1.8
Sandstone-Lam	273	4%	885.5	273	2%	0.6
Diorite	237	3%	754.9	238	3%	1
Agglo-peli	210	3%	674	210	3%	1
Agglo	139	2%	440.5	139	2%	1.1
Pelite-carbo	128	2%	382.1	130	1%	0.3
Agglo-poly	111	2%	351.1	111	2%	1
Greywacke	74	1%	234.2	74	1%	0.6
Laterite	68	1%	131.1	71	0%	0
Andesite	60	1%	193.7	60	0%	0.2
Silt	49	1%	157.7	49	0%	0.3
Agglo-cipo	38	1%	121.1	38	0%	0.5
Agglo-carbo	36	1%	114	36	1%	1.7
Tuff	21	0%	68.7	21	0%	0.3
Fault-gouge	21	0%	64.2	21	0%	1
Tourmalinite	19	0%	61.7	19	1%	2.4
Mottled-zone	10	0%	21.5	10	0%	0
Agglo-sand	10	0%	32.5	10	0%	0.3
Rhyolite	8	0%	25.4	8	0%	1.9
Basalt	7	0%	23	7	0%	0.4
Alluvion	5	0%	9.9	6	0%	0
Quartz	4	0%	12.7	4	0%	1.7
Sandst-carbo	4	0%	13.2	4	0%	0.4
Breccia	2	0%	6.6	2	0%	1
Iron	2	0%	6.6	2	0%	10.9
NR	2	0%	2	2	0%	2.2
Total	**6861**	**100%**	**21524**	**6892**	**100%**	**1.08**

NR = Not Recognized

Table 13.21 SGS 2015 Analysis of Geometallurgical Database for Boto 5

Lithology	Count	% Count	Mass (kg)	Length (m)	Rel% Au Ozs	Avg. Grade (g/t Au)
Turbidite	10	6%	22.6	10	1%	0.5
Albitite	130	76%	274.9	116	92%	4.2
Laterite	3	2%	7.2	3	0%	0.6
Mottled-zone	2	1%	4.8	2	0%	0.3
Cipolin	17	10%	41.9	17	2%	0.6
Diorite	1	1%	2.6	1	0%	0
Tuff	7	4%	9.1	4.5	5%	6.7
Total	**170**	**100%**	**363**	**153.5**	**100%**	**3.5**

The ore vein structures at Boto were categorized by domain denoted as 119, 124_c, 121, 124b_c2, and 125 for Boto 2/Malikoundi, and 103, 104, 101, and 102 for the Boto 5. The primary domain for Boto 2/Malikoundi is 119 making up 77% of the gold ounces, while 103 is the primary domain for Boto 5, making up 59% of the gold ounces.

The main sampling criteria included selecting samples representative of the ore grade and lithological types, geographically well distributed with at least 60% from the first five years of production, and representative of some external and internal dilution.

Visual representation of where the selected samples were taken from within the Boto deposits are shown in Figure 13.15 to Figure 13.18 by superimposing the selected drill core samples over the pit shells.

Figure 13.15 SGS 2015 Selected Drill Core Samples for Boto 2/Malikoundi – Top View

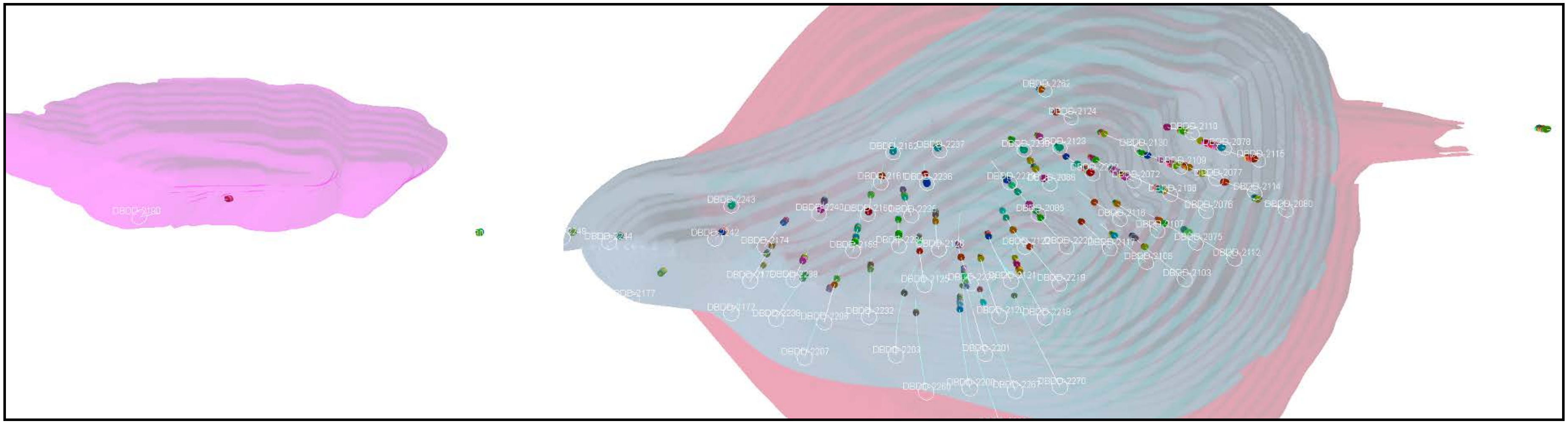


Figure 13.16 SGS 2015 Selected Drill Core Samples for Boto 2/Malikoundi – Side View

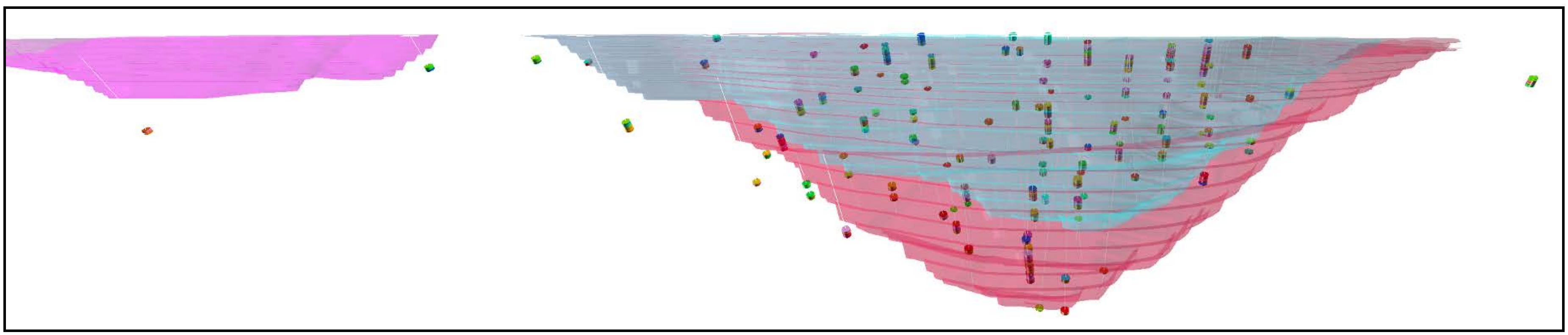


Figure 13.17 SGS 2015 Selected Drill Core Samples for Boto 5 – Top View

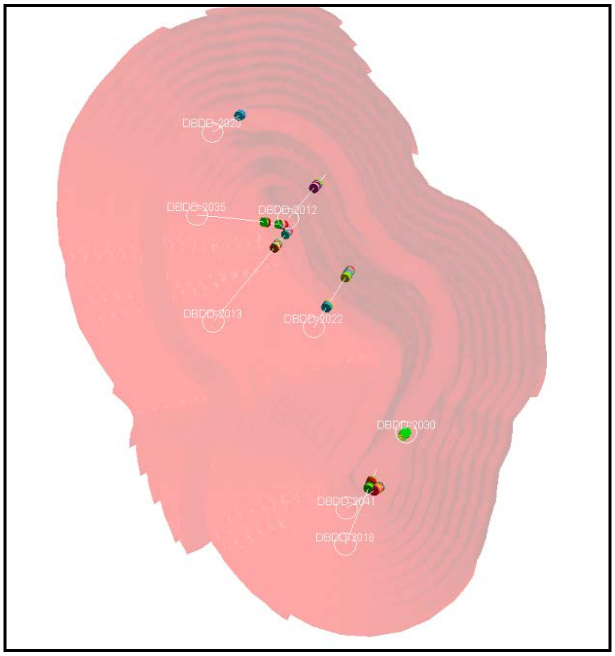
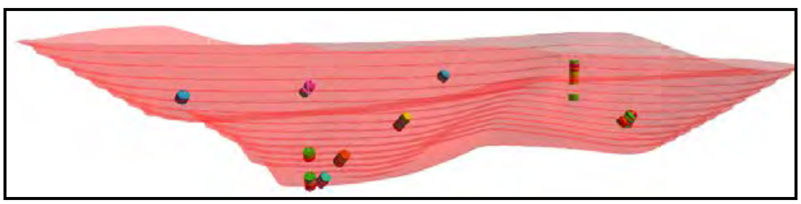


Figure 13.18 SGS 2015 Selected Drill Core Samples for Boto 5 – Side View



Grindability Testwork (2015)

Grindability testwork was conducted on three domain composites and 39 variability samples. The testwork consisted of tests on Bond ball mill grindability, Bond abrasion, JK drop-weight, SMC, and Bond low-energy impact. A summary of the grindability results are displayed in Table 13.22.

Table 13.22 SGS 2015 Summary of Grindability Results

Sample Name	Rock Type	Relative Density	JK Parameters A x b[1]	A x b[2]	t_a	SCSE	CWi (kWh/t)	BWi[3] (kWh/t)	BWi[4] (kWh/t)	AI (g)
GR-01	Fresh Rock	2.74	26.3	26.4	0.28	12.24	17.7	-	-	-
GR-02	Fresh Rock	2.74	30.5	31.0	0.28	11.33	11.4	-	-	-
GR-04	Fresh Rock	2.69	28.2	28.1	0.23	11.67	13.2	-	-	-
G01	Fresh Rock	2.70	-	36.9	0.35	10.25	-	18.7	-	0.636
G02	Fresh Rock	2.81	-	28.4	0.26	11.99	-	21.2	-	0.223
G03	Fresh Rock	-	-	-	-	-	-	19.3	-	0.566
G04	Fresh Rock	2.79	-	26.0	0.24	12.47	-	20.2	-	0.735
G05	Fresh Rock	2.74	-	33.0	0.31	10.90	-	16.3	-	0.597
G06	Fresh Rock	-	-	-	-	-	-	20.0	-	0.520
G07	Fresh Rock	2.54	-	41.4	0.42	9.52	-	20.9	-	0.739
G08	Fresh Rock	2.65	-	38.0	0.37	10.01	-	21.0	-	0.740
G09	Fresh Rock	-	-	-	-	-	-	21.5	-	0.823
G10	Fresh Rock	2.73	-	38.5	0.37	10.11	-	22.2	-	0.394
G11	Fresh Rock	2.72	-	38.8	0.37	10.05	-	21.3	-	0.456
G12	Fresh Rock	2.77	-	42.6	0.40	9.73	-	20.9	-	0.532
G13	Fresh Rock	-	-	-	-	-	-	16.1	-	0.259
G14	Fresh Rock	2.81	-	41.0	0.38	9.98	-	16.3	-	0.305
G15	Fresh Rock	2.86	-	51.7	0.47	9.08	-	13.8	-	0.273
G16	Fresh Rock	2.78	-	35.2	0.33	10.67	-	20.4	-	0.609
G17	Fresh Rock	-	-	-	-	-	-	19.4	-	0.476
G18	Fresh Rock	2.76	-	35.9	0.33	10.52	-	18.8	-	0.601
G19	Fresh Rock	2.77	-	28.4	0.26	11.86	-	23.8	-	0.293
G20	Fresh Rock	2.78	-	29.9	0.28	11.57	-	23.5	-	0.210
G21	Fresh Rock	-	-	-	-	-	-	20.3	-	0.359
G22	Fresh Rock	2.81	-	44.5	0.41	9.62	-	16.6	-	0.184
G23	Fresh Rock	-	-	-	-	-	-	17.4	-	0.252
G24	Fresh Rock	2.84	-	40.0	0.36	10.18	-	21.7	-	0.636
G25	Fresh Rock	-	-	-	-	-	-	19.7	-	0.543
G26	Fresh Rock	2.75	-	44.4	0.41	9.50	-	19.1	-	0.388
G27	Fresh Rock	2.75	-	32.7	0.31	10.99	-	18.2	-	0.448
G28	Saprolite	-	-	-	-	-	-	8.3	1.5	0.013
G29	Saprolite	-	-	-	-	-	-	14.2	7.8	0.075
G30	Saprolite	-	-	-	-	-	-	9.4	1.5	0.000
G31	Saprock	-	-	-	-	-	-	12.0	-	0.050
G32	Saprock	-	-	-	-	-	-	12.2	-	0.053
G33	Saprock	-	-	-	-	-	-	7.7	-	0.006
G34	Mottled-zone	-	-	-	-	-	-	13.9	-	-0.005
G35	Mottled-zone	-	-	-	-	-	-	10.8	-	-0.003
G36	Mottled-zone	-	-	-	-	-	-	9.2	-	0.000
G37	Fresh Rock	2.56	-	49.3	0.50	8.85	-	6.9	-	0.197
G38	Saprock	2.54	-	78.7	0.81	7.40	-	7.6	-	0.133
G39	Saprolite	-	-	-	-	-	-	6.0	2.1	0.005

[1] *A x b from DWT*
[2] *A x b from SMC*
[3] *Direct Measured Work Index*
[4] *Recalculated Work Index*

The three domain composites (all fresh rock type) were characterized as hard to very hard with respect to impact resistance and abrasion parameters (A x b and t_a), and as medium to very hard for the CWi's. The other fresh rock samples were characterized as moderately hard to hard with respect to the A x b values, as moderately hard to very hard with respect to the BWi's, and as medium to very abrasive with respect to the Ai's. The other rock types were all significantly softer and less abrasive than the fresh rock. The grindability statistics are summarized in Table 13.23.

Table 13.23 SGS 2015 Grindability Statistics by Rock Type

Statistics	Number of Samples		Relative Density	JK Parameters		Work Indices (kWh/t)			Ai (g)
	SMC	BWi / Ai		A x b[2]	SCSE	CWi	BWi[3]	BWi[4]	
Overall Minimum[1]	24	39	2.54	78.7	0.81	11.4	6.0	1.5	0.000
Overall Maximum[1]	24	39	2.86	26.0	12.47	17.7	23.8	7.8	0.823
Fresh Rock Average	23	28	2.74	37.1	10.57	14.1	19.1	-	0.464
Saprolite Average	0	4	-	-	-	-	9.5	3.2	0.023
Saprock Average	1	4	2.54	78.7	7.40	-	9.9	-	0.06
Mottled-zone Average	0	3	-	-	-	-	11.3	-	0.000
Total/Overall Average	**24**	**39**	**2.74**	**40.5**	**10.25**	**-**	**16.6**	**-**	**0.342**

[1] Minimum and maximum refer to softest and hardest results for the work and abrasion indices, but vice-versa for the A x b parameters.
[2] A x b from SMC
[3] Direct Measured Work Index
[4] Recalculated Work Index

Metallurgical Development Testwork (2016)

Three master composites were submitted for metallurgical development testwork. MC-1 contained fresh rock from Boto 2/Malikoundi, MC-2 contained fresh rock from both Boto 2/Malikoundi & Boto 5, and MC-3 contained only saprolite from Boto 5. Selected results for the head analysis are presented in Table 13.24.

Table 13.24 SGS 2016 Selected Head Analysis Results

Analysis	MC-1	MC-2	MC-3
Avg. Au, g/t	2.59	4.01	1.65
S %	0.93	1.58	1.27
S= %	0.77	1.52	1.2
Cu %	0.002	0.01	0.02
Cu(NaCN) %	<0.002	0.006	0.011
C(t) %	2.72	1.09	0.13
TOC leco %	0.07	0.43	0.13
Ag, g/t	<2	<2	<2
Al, g/t	58,200	69,400	144,000
Ba, g/t	171	148	66.9
Be, g/t	0.84	1.1	1.18
Ca, g/t	55,200	24,800	1,360
Co, g/t	24	35	65
Cr, g/t	92	126	182
Fe, g/t	55,200	42,900	30,100
K, g/t	6,210	7,880	1,730
Mg, g/t	33,400	20,300	3,550
Mn, g/t	729	314	65
Na, g/t	32,600	29,200	4,980
Ni, g/t	41	78	110
P, g/t	549	500	194
Sr, g/t	90.3	74.3	37.7
Ti, g/t	2,460	2,730	6,040
V, g/t	78	81	137
Y, g/t	10.5	12.4	18.5
Zn, g/t	14	72	108

The mineralogical analysis for the master composites are presented in Table 13.25.

Table 13.25 SGS 2016 Mineralogical Characteristics of the Gold Minerals

Comp	Au Grade g/t	Au Distribution by Association		Size Range µm	Average Size, µm	Au-Mineral Abundance	Minerals Associated with Exposed and Locked Au-Minerals
MC-1	2.55	Liberated	83.0	0.6 - 72.1	10.3	Gold (98%), Calaverite (1%), other (1%)	pyrite 69%, quartz 9%, pyrite/silicate 4%, Fe oxide/quartz 4%, Fe oxide 4%, dolomite/Fe oxide 3%, silicate/Fe oxide 2%, Fe oxide/silicate 1%, dolomite 1%, silicate/quatrz 1% silicate/dolomite 1% and other <1%.
		Exposed		0.6 - 16.3	2.9		
		Locked	17.0	0.6 - 15.5	2.6		
			100.0	**0.6 - 72.1**	**5.6**		
MC-2	3.78	Liberated	80.3	0.6 - 146	16.0	Gold (64%), Calaverite (35%), other (1%)	galena 42%, pyrite 28%, altaite 11%,silicate 10%, quartz 5%, Bi-Pb-Te 1%, Fe oxide 1%, silicate/ilmenite 1%, and other <1%.
		Exposed		0.6 - 108	7.1		
		Locked	19.7	0.6 - 12.6	2.0		
			100.0	**0.6 - 146**	**6.4**		
MC-3	1.64	Liberated	74.9	0.6 - 98.8	10.6	Gold (97%), Au-Te-Bi (2%), Calaverite (1%)	Fe oxide 42%, pyrite 30%, Bi-Pb-Te 16%, silicate 10%, pyrite/anhydrite 1% and other 1% .
		Exposed		0.6 - 59.5	4.7		
		Locked	25.1	0.6 - 14.0	2.3		
			100.0	**0.6 - 98.9**	**5.5**		

Native (liberated) gold (>75% Au, <25% Ag) was identified as being the main gold mineral in all three composites. The gold mineral abundance for MC-1, MC-2, and MC-3 were 98%, 64%, and 97%, respectively. Calaverite (AuTe) was found in MC-2 at ~35%, with only trace amount in MC-1 and MC-3.

Gravity separation testwork, intensive cyanidation of the gravity concentrate, and cyanidation of the gravity tailings were also conducted as part of the metallurgical development. A summary of the results is presented in Table 13.26.

Table 13.26 SGS 2016 Summary of Overall Results for Master Composites

Comp.	Description	Avg. Head Grade, g/t Au	Grind Size, P_{80}	Test No.	Gravity Concentrate			Conc. Cyanidation			Tailings Cyanidation			Comb. Au Extr'n %
					Mass Pull %	Assay, g/t Au	Au Extr'n %	Test No.	Au Extr'n %	O'all Au Rec'y %	Test No.	Au Extr'n %	% O'all Au Extr'n	
MC-1	Boto 2 Fresh Rock	2.59	69	G-45	0.18	221	16.4	CN-85	98.1	16.1	CN-88	88.8	74.5	90.6
MC-2	Boto 2 & 5 Fresh Rock	4.01	86	G-44	0.14	693	26.5	CN-84	98.9	26.2	CN-87	81.1	59.8	86.1
MC-3	Boto 5 Saprolite	1.65	49	G-46	0.19	235	27.2	CN-86	98.8	26.9	CN-89	64.4	47.1	74.0
											CN-20	82.3	60.2	87.1

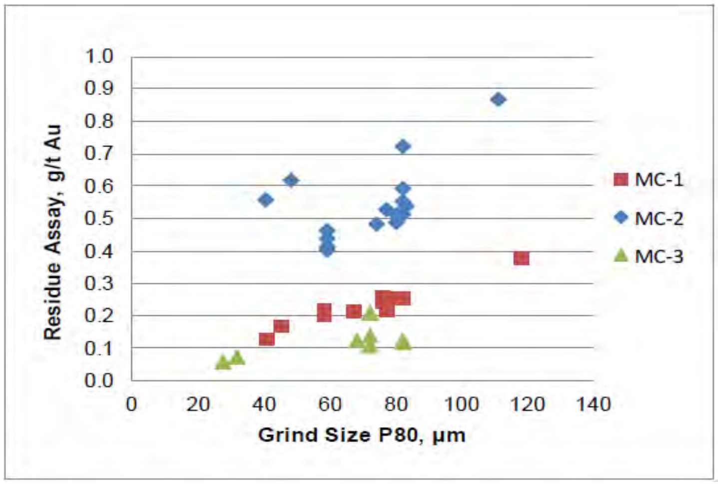
** Note: Boto 2/Malikoundi is denoted with only "Boto 2".*

For MC-3, the bulk cyanidation test (CN-89) had a viscosity issue which resulted in an overall gold extraction of only 74%. If this cyanidation result was replaced by the bottle roll test result (CN-20), also conducted on Boto 5 saprolite, the overall gold extraction for MC-3 would be 87%.

Gold extraction from gravity concentrate averaged at 23%, which at the time was believed to be sufficient to warrant inclusion of this step in the flowsheet.

The impact of finer grinding was also studied and a plot of the grind size versus the leach residue grade is shown in Figure 13.19. Although the leach conditions were not identical for the tests shown in this figure, a trend can be observed where the extraction of gold increases as the grind size decreases.

Figure 13.19 SGS 2016 Grind Size versus Leach Residue Assay



The following key observations were made for the SGS 2016 metallurgical development testwork:

- Decreasing the grind size from P_{80} of 118 µm to 41 µm for MC-1, P_{80} of 111 µm to 40 µm for MC-2, and P_{80} of 82 µm to 27 µm for MC-3 provided an increase in gold extraction of 9.7%, 7.7% and 3.6%, respectively.
- The addition of lead nitrate improved the initial leach kinetics of MC-1 but not the gold extraction after 24-hours. Lead nitrate addition to MC-2 showed inconclusive results.
- The addition of a pre-aeration stage resulted in a significant reduction in cyanide consumption.
- The use of oxygen during MC-1 and MC-2 leach tests at the time did not show any obvious benefits.
- MC-2 showed slower leach kinetics than MC-1, possibly due to its higher Calaverite (AuTe) content. Higher pH or lead nitrate addition may potentially be beneficial to this composite.
- Increasing cyanide concentration did not reveal noticeable improvement in leach kinetics or gold extraction for MC-1 or MC-3. Increasing cyanide concentration to 0.75 g/L NaCN did show some benefit for MC-2.
- Leach tests conducted with and without carbon showed all three composites to be non preg-robbing.

Variability Testwork (2016)

40 samples were submitted for gold extraction variability testwork as part of the 2016 SGS metallurgical testwork. A summary of the results is presented in Table 13.27.

Table 13.27 SGS 2016 Summary Results of Variability Testwork

Sample	Test No.	Grind P_{80}, µm	Reagent Con., kg/t NaCN	Reagent Con., kg/t CaO	% Au Extraction / Recovery 6-h CN	24-h CN	48-h CN	Grav Sep	Grav + CN	Residue g/t Au	Calc. Head g/t Au
R1	CN-65	76	0.08	2.79	87	88	92.4	27.9	94.5	0.22	2.91
R2	CN-78	96	0.07	2.19	62	70	74.3	9.80	76.8	2.11	8.22
R3	CN-47	80	0.06	1.96	83	85	87.1	23.1	90.0	0.21	1.58
R4	CN-59	86	0.10	2.88	79	86	84.1	19.0	87.1	0.25	1.57
R5	CN-66	93	0.01	1.84	79	82	81.5	11.3	83.5	0.15	0.81
R6	CN-71	74	0.05	2.89	57	65	70.9	23.0	77.6	1.4	4.79
R7	CN-67	82	0.06	2.24	76	81	83.8	13.2	85.9	0.54	3.30
R8	CN-75	77	0.14	2.28	74	84	84.9	37.7	90.6	0.14	0.92
R9	CN-46	80	0.24	1.86	66	77	82.5	26.2	87.1	0.11	0.60
R10	CN-48	76	0.33	2.83	81	84	89.6	14.1	91.0	1.09	10.4
R11	CN-49	75	0.05	1.95	84	82	85.7	23.0	89.0	0.21	1.47
R12	CN-62	63	0.03	2.05	83	87	87.6	48.8	93.7	0.09	0.69
R13	CN-77	42	0.06	2.84	91	94	93.7	29.3	95.5	0.19	2.92
R14	CN-69	89	0.04	1.79	81	83	83.5	26.6	87.9	0.17	1.00
R15	CN-54	72	0.05	2.06	83	84	85.1	26.7	89.1	0.12	0.81
R16	CN-64	73	0.08	2.78	83	84	86.3	25.2	89.7	0.32	2.33
R17	CN-60	72	0.02	2.11	85	88	91.3	18.4	92.9	0.25	2.87
R18	CN-52	97	0.03	2.07	80	84	84.3	15.4	86.7	0.19	1.21
R19	CN-68	64	0.06	1.98	81	84	85.5	14.0	87.5	0.29	2.00
R20	CN-74	79	0.08	2.17	73	79	79.8	10.6	81.9	0.16	0.77
R21	CN-70	68	0.06	2.29	80	85	85.8	19.3	88.5	0.09	0.60
R22	CN-63	75	0.06	2.38	76	85	87.1	8.7	88.2	0.36	2.74
R23	CN-57	73	0.04	2.7	87	88	90.5	15.8	92.0	0.25	2.63
R24	CN-76	93	0.06	2.12	72	79	81.6	11.6	83.7	0.26	1.41
R25	CN-50	73	0.06	2.20	80	85	86.3	13.7	88.2	0.19	1.35
R26	CN-51	81	0.05	1.84	84	86	87.9	20.0	90.3	0.09	0.70
R27	CN-73	73	0.05	2.42	81	84	84.9	18.1	87.6	0.14	0.93
R28	CN-56	85	0.04	4.96	77	92	93.1	8.1	93.7	0.27	3.94
R29	CN-61	36	0.03	5.22	76	95	96.3	10.6	96.7	0.06	1.50
R30	CN-44	50	0.01	4.25	66	77	94.2	17.8	95.3	0.04	0.70
R31	CN-55	79	0.06	6.55	57	88	89.4	18.9	91.4	0.85	7.96
R32	CN-45	77	0.00	6.03	62	86	89.2	9.7	90.2	0.19	1.76
R33	CN-72	80	0.00	7.61	76	87	89.9	27.1	92.6	0.18	1.73
R34	CN-53	80	0.04	2.14	89	93	94.2	65.5	98.0	0.06	1.03
R35	CN-79	69	0.73	5.51	82	83	84.4	24.2	88.2	0.09	0.55
R36	CN-80	20	0.05	2.25	66	76	85.8	23.7	89.2	0.18	1.23
R37	CN-81	86	0.52	4.01	80	86	84.5	12.8	86.5	0.20	1.29
R38	CN-58	56	0.16	2.89	72	96	95.7	24.7	96.7	0.03	0.69
R39	CN-82	77	0.31	5.86	78	74	90.1	33.9	93.5	0.17	1.72
R40	CN-83	78	0.38	3.41	87	91	92.7	20.9	94.3	0.09	1.17
Overall Average		74	0.11	3.06	77	84	86.9	21.2	89.6	0.30	2.17
Maximum		97	0.73	7.61	91	96	96.3	65.5	98.0	2.11	10.40
Minimum		20	0.00	1.79	57	65	70.9	8.1	76.8	0.03	0.55
Standard Deviation		15	0.15	1.50	8	6	5.3	11.1	4.7	0.40	2.18

Gap Analysis on 2015/2016 Testwork Program

In 2017, a review of the 2005/2016 metallurgical testwork was completed by Lycopodium to identify some potential gaps prior to commencing the metallurgical testwork program for the Boto feasibility study (FS) phase.

The recommendations or gaps identified include but are not limited to the following:

- Silver and Tellurium detection limits were too high and Mercury content was not assessed.
- Multiple variables were changed in the same test – this should be avoided in future testwork.
- Impact of pre-aeration or pre-oxygenation on gold extractions or leach kinetics was not fully confirmed due to lack of a base case for the tested grind size.
- True specific gravity should be determined by pycnometer. SMC tests provided relative density.
- Drill intercepts that will be used to form future master composites should be assayed, and results reported first prior to forming composites to ensure proper material assignment.
- Whole ore leach tests should be conducted to study effect of excluding gravity separation step.
- Lime demand test and rheology testwork to precede leach testwork such that realistic pulp pH and pulp density ranges can be selected.
- Kinetic sampling to be conducted at shorter time intervals (e.g. 2, 4, 6, 12, 24, 36 and 48-hours).
- Effect of lead nitrate should also be assessed on saprolitic material to ensure no negative effects.
- The 40 variability samples should be tested for Tellurium (Te) to potentially establish location of Calaverite mineralization in the geological model.
- Grind calibration should be conducted to determine grind times for each targeted grind size.
- A leach test with site water should be conducted to ensure no negative effects.

Lycopodium also consulted Orway Mineral Consultants (OMC) to conduct a review of the 2015 comminution data and the following recommendations were provided for Boto 2/Malikoundi:

- Conduct 12 more CWi tests with priority given to pelite and agglomerate lithologies.
- Conduct four more BWi tests on Boto 2/Malikoundi material, two on saprolite and two on saprock.
- Conduct SMC tests on 5 new agglomerate and 3 new cipolin samples (optional only if budget is not a constraint).

13.2 Review of Most Recent Metallurgical Tests

13.2.1 SGS 2017/2018 Testwork Program

In 2017, the Project entered into its FS phase and Ms. Ryda Peung at Lycopodium supervised the metallurgical testwork at SGS Lakefield with involvement from Mr. Ricardo Esteban at IAMGOLD. A sample selection exercise with Mr. Matthew Halliday was also conducted at SGS Geostats at the start of the program.

The key objectives of this testwork included confirming the requirement of a gravity circuit, confirming the optimum leach conditions such as grind size, cyanide concentration, pulp density, addition rate for lead nitrate, and oxygen addition during leaching. The program also included CIP modelling, and tests for solids-liquid separation, pH neutralization, oxygen uptake, preg-robbing, and cyanide destruction.

Sample Selection (2017)

For the Boto FS phase, SGS Geostats organised the drillhole data provided by IMG in such a way as to select samples respecting distance from previous metallurgical sampling and matching the grade distribution of the deposit. The main objective was to select samples representative of the different grades, lithology, and spatial locations across the entire deposit, but also within the ultimate pit shell. Samples within the mineralized envelopes or may potentially contain Calaverite were also given strong preferences. SGS Geostats provided a list of the selected samples respecting these parameters to Lycopodium. The samples used in forming the master composites and the variability composites were then selected from this list by Lycopodium with agreement from IMG.

Three master composites were formed for use in the metallurgical development testwork:

- MC-1: 50% saprolite, 50% saprock, 100% of the ore from Boto 5 deposit.
- MC-2: ~90% fresh rock, ~5% saprolite, ~5% saprock, 96% of the ore from Boto 2/Malikoundi deposit and 4% from Boto 5 deposit.
- SAP: 100% saprolite, ~85% of the ore from Boto 2/Malikoundi deposit and ~15% from Boto 5 deposit.

The majority of the soft ore (10%) fed to the plant come from Boto 5 deposit, therefore MC-1 was formed to better understanding the metallurgical behaviour of this material. MC-2 was formed to represent the expected LOM blend and ore grade. The SAP (saprolite) sample was formed without grade consideration, and will only be used in solids-liquid testwork. The samples used in forming the master composites shown in Figure 13.20 and Figure 13.21 for Boto 2/Malikoundi pit, and in Figure 13.22 and Figure 13.23 for the Boto 5 pit.

Since Boto 2/Malikoundi makes up close to 95% of the overall life-of-mine (LOM) tonnage, the variability samples selected were heavily weighted for the Boto 2/Malikoundi pit. Boto 5 material had limited amount of mass, hence, the majority of it was used in forming the master composite, MC-1, and only a small amount remained for the variability testwork. The samples used in forming the variability composites for the recovery variablity testwork are shown in Figure 13.24 and Figure 13.25.

Figure 13.20 SGS 2017 – Boto 2/Malikoundi Samples Used in Master Composites – Top View

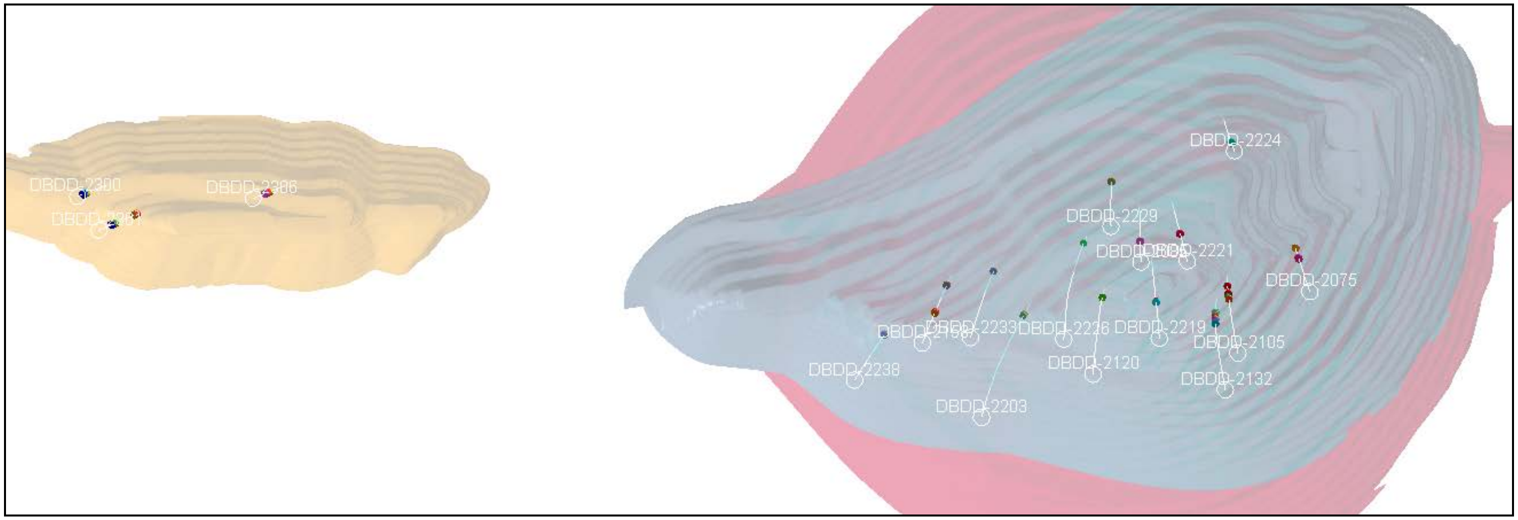


Figure 13.21 SGS 2017 – Boto 2/Malikoundi Samples Used in Master Composites – Side View

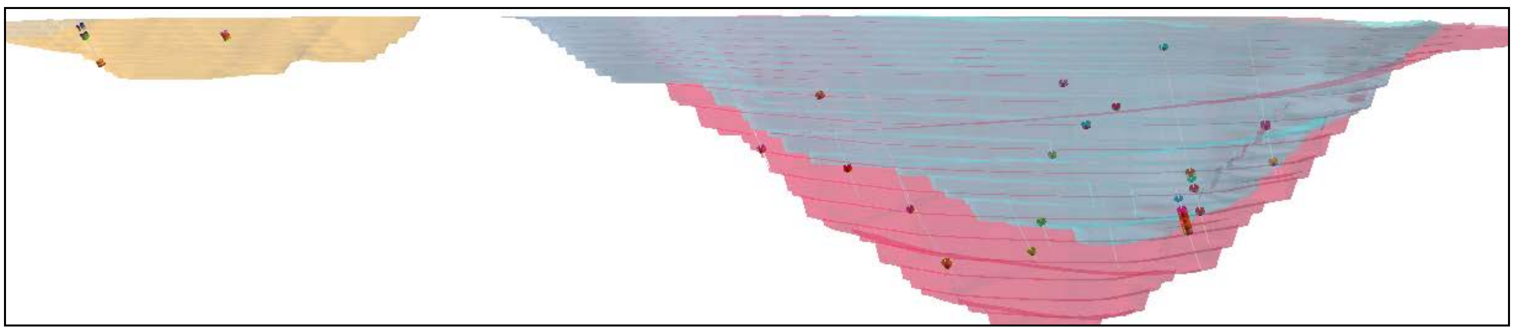


Figure 13.22 SGS 2017 – Boto 5 Samples Used in Master Composites – Top View

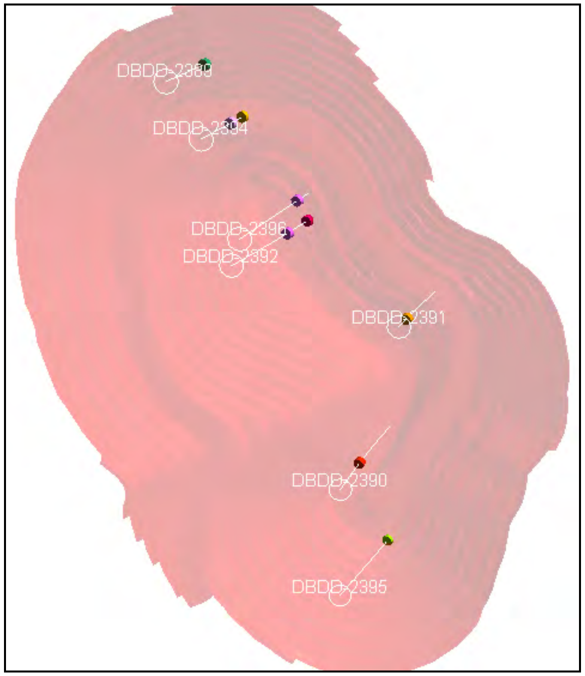


Figure 13.23 SGS 2017 – Boto 5 Samples Used in Master Composites – Side View

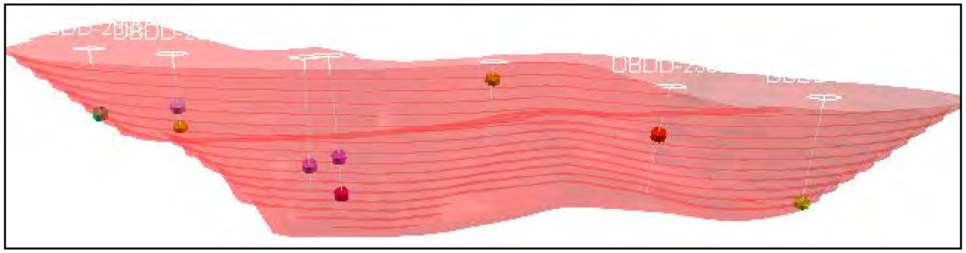


Figure 13.24 SGS 2017 – Boto 2/Malikoundi Samples Used in Variability Composites – Top View

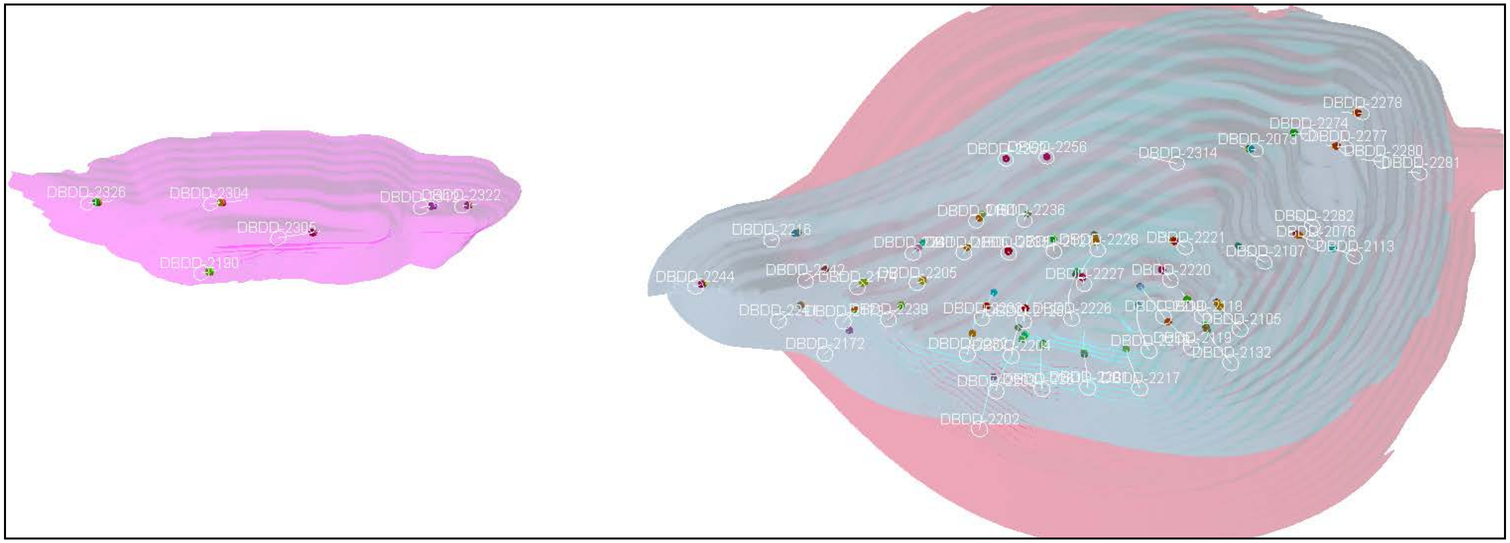
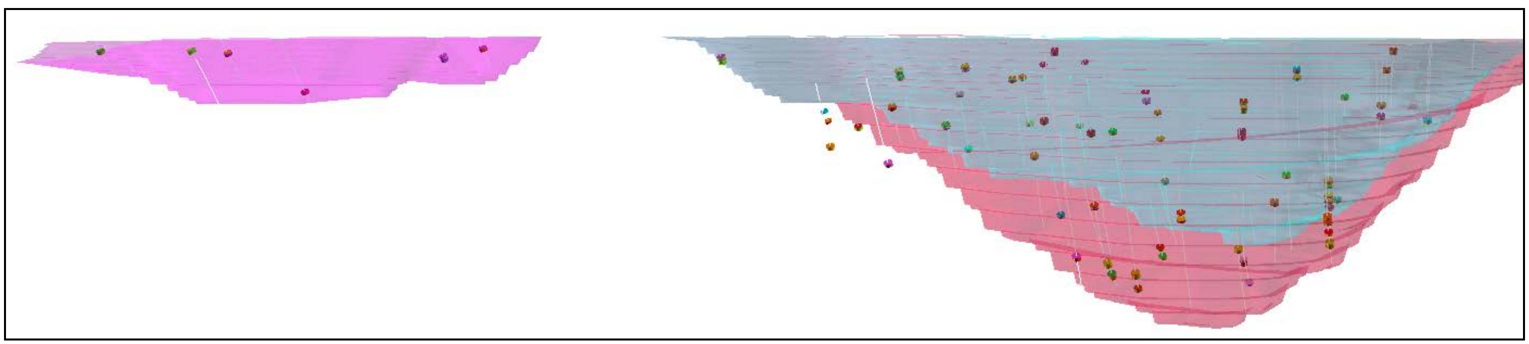


Figure 13.25 SGS 2017 – Boto2/Malikoundi Samples Used in Variability Composites – Side View



Grindability Testwork (2017/2018)

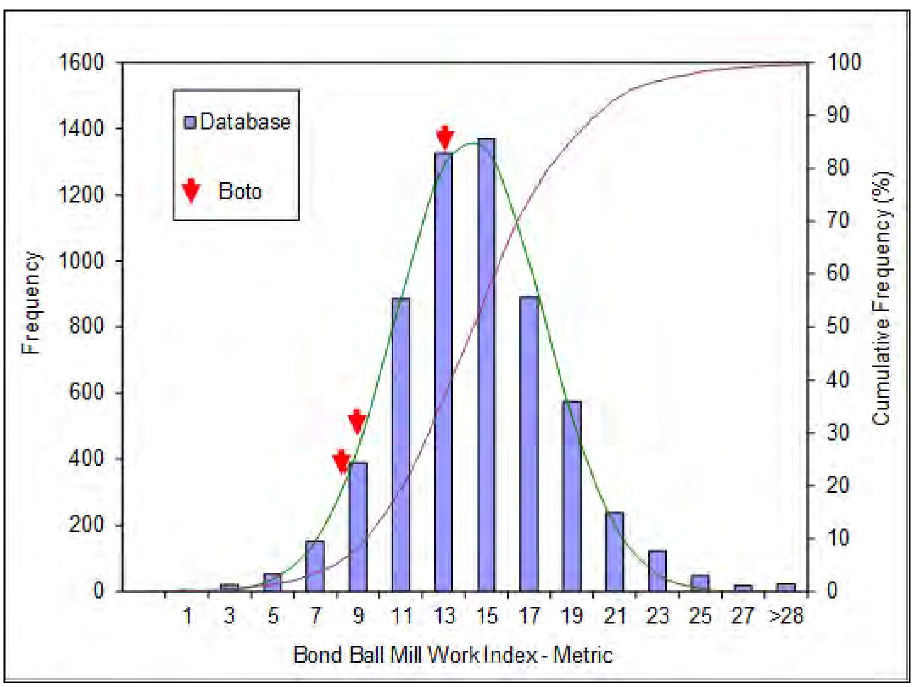
Following the recommendations from OMC's review of the previous comminution data, two saprolite and two saprock samples from Boto 2/Malikoundi were selected for additional BWi tests. Unfortunately, one of the saprolite samples was too fine and disintegrated before testwork can be done. The results for the three new BWi tests are presented in Table 13.28 and Figure 13.26.

Table 13.28 SGS 2017/2018 Bond Ball Mill Grindability Testwork Summary

Sample Name	Closing Screen		F_{80} µm	P_{80} µm	Grams per Rev.	Work Index (kWh/t)		Hardness Percentile vs. SGS Database	% Undersize in Feed	Bulk Density, kg/m³
	Mesh	µm				Direct	Overall, including fines			
Saprock 1	170	90	2,247	70	1.43	13.1		36	13.2	1,774
Saprock 2	170	90	2,065	67	2.25	9.0		6	16.2	1,643
Saprolite 2	170	90	1,816	71	1.35	* 14.4	* 8.3	6	16.2	1,643

** The modified BWi procedure, involving the removal of excess U/S (naturally passing the closing screen) was applied to the Saprolite 2 material. The purpose of this procedure is to generate an accurate power requirement for the material in the feed that requires comminution (the "Direct" BWI value) and eliminating the material that requires no power as it is naturally finer than the closing screen size. The "Overall Including Fines" BWi value is the proportionally recombined (based on mass split) BWi's for the coarse and fines fractions.*

Figure 13.26 SGS 2017/2018 BWi Results Compared to BWi Database



Additional CWi tests were not conducted due to limited material availability that meet the testwork rock size requirement.

Head Analysis and Mineralogy Analysis on Master Composites

The specific gravities, determined by pycnometer, were 2.79, 2.81 and 2.83 for MC-1, MC-2, and SAP, respectively.

Selected head analysis results for the master composites are shown in Table 13.29.

Table 13.29 SGS 2017/2018 Selected Head Analysis Results

Analysis		MC1	MC2	SAP
Au (dup. Fire Assays)	g/t	2.87	1.90	0.63
Au (SFA)	g/t	3.45	1.68	1.31
Ag	g/t	1.4	<0.5	<0.5
S	%	3.35	1.59	<0.01
$S^{=}$	%	3.17	1.37	<0.05
Fe	%	3.99	5.39	6.55
WO_3	%	0.023	0.01	<0.01
Cu	%	0.041	0.011	0.005
As	%	0.002	<0.001	<0.001
Cu (CN Sol)	%	0.029	0.004	<0.002
C(t)	%	0.04	2.22	0.02
C(g)	%	N/D	<0.05	N/D
TOC (Leco)	%	N/D	<0.05	N/D
CO_3	%	N/D	10.4	N/D
Hg	g/t	<0.3	<0.3	<0.3
Te	g/t	51	11	<4
Rb	g/t	<0.002	<0.002	0.003
Al	g/t	141,000	59,500	118,000
Ba	g/t	62.8	158	177
Be	g/t	1.28	0.88	1.18
Bi	g/t	49	<20	<20
Ca	g/t	782	52,300	458
Co	g/t	116	32	16
Cr	g/t	40	108	81
K	g/t	1,150	5,130	6,750
Mg	g/t	4,190	28,100	3,710
Mn	g/t	87	598	303
Na	g/t	932	28,600	688
Ni	g/t	219	48	25
P	g/t	<70	468	215
Sr	g/t	49.4	74	34.4
Ti	g/t	5,200	2,550	5,640
V	g/t	78	83	163
Y	g/t	19.3	13.3	15.6
Zn	g/t	33	22	29

One of the main objectives for the 2017/2018 testwork was to further study the distribution of Tellurides in the ore body and also to study the gold deportment and the gold mineral types. The mineralogy analysis on MC-1 and MC-2 is summarized in Table 13.30, Table 13.31, and in Figure 13.27.

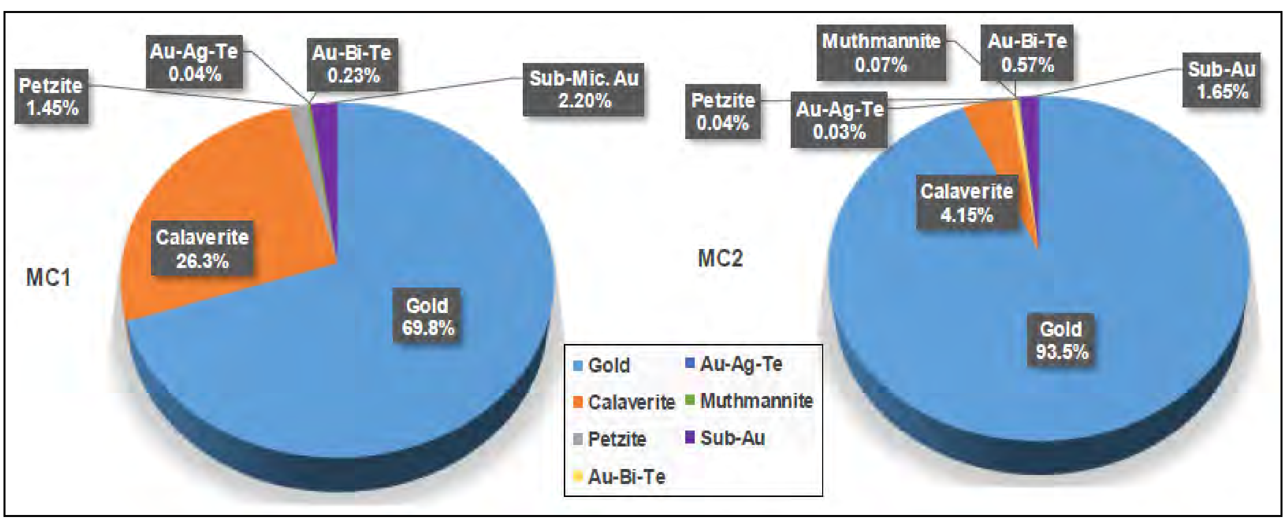
Table 13.30 SGS 2017/2018 Characteristic of Microscopic Gold in MC-1 and MC-2

Sample ID	Association	Gold Distribution (%)	Size Range (µm)	Average Size (µm)	Gold Carriers and abundance	Minerals Associated with Exposed and Locked Au-Minerals
MC1	Liberated	36.9	0.6 - 51.8	10.6	Gold (53%), calaverite (42%), petzite (4%), and Au-Bi-Te (1%)	silicates 43%, iron oxide 35%, NiTe 14%, pyrite 4%, BiTe/silicates 2%, others 2%
	Exposed	40.9	0.6 - 283	13.0		
	Locked	20.0	0.6 - 32.0	2.7		
		97.8	**0.6 - 283**	**7.6**		
MC2	Liberated	57.7	0.6 - 101	10.6	Gold (90%), calaverite (8%), and Au-Bi-Te (2%)	pyrite 58%, silicates 20%, PbTe/iron oxide 9%, PbTe/dolomite 5%, iron oxide/calcite 3%, dolomite 2, pyrite/NiTe 1%, and other minerals 2%.
	Exposed	16.3	0.6 - 20.5	3.8		
	Locked	24.3	0.6 - 22.4	2.7		
		98.3	**0.6 - 101**	**5.1**		

Table 13.31 SGS 2017/2018 Gold Deportment by Gold Mineral Type and Sub-Au for MC-1 and MC-2

Sample ID	Gold Mineral	Gold Mineral Abundance (%)	Au% of Each Gold Minerals	Overall Gold Distribution by Gold Type (%)
MC1	Gold	53.5	89.3	69.8
	Calaverite	42.1	42.7	26.3
	Petzite	3.67	27.0	1.45
	Au-Bi-Te	0.62	25.1	0.23
	Au-Ag-Te	0.07	43.5	0.04
	Sub-Mic. Au	-	-	2.20
	Sum	***100***	**-**	**100**
MC2	Gold	89.8	95.5	93.5
	Calaverite	8.36	45.5	4.15
	Petzite	0.13	35.2	0.04
	Au-Bi-Te	1.49	39.5	0.57
	Au-Ag-Te	0.06	30.2	0.03
	Muthmannite	0.17	43.8	0.07
	Sub-Au	-	-	1.65
	Sum	***100***	**-**	**100**

Figure 13.27 SGS 2017/2018 Gold Distribution by Gold Mineral Type and Sub-Au for MC-1 and MC-2



Te in MC-1 and MC-2 is associated with Au-Bi-Te, Au-Ag-Te, Muthmannite, Petzite and Calaverite. Based on the number of grains observed, MC-1 contains 26.3% of the gold in Calaverite and MC-2 contains 4.15% of the gold in Calaverite.

Te Assay Analysis on Old Pulp Samples

Te assays were analysed on pulps from the 2016 variability testwork to identify possible areas of Calaverite in the geological model. SGS was only able to provide Te analysis at a detection limit of 4 g/t Au. Results higher than the detection limit were linked back to their corresponding drillholes as shown in Table 13.32.

Table 13.32 Tellurium Assays on 2016 Variability Testwork Pulps

Sample	Te, g/t	Pit	Weather	Drillhole - From/To
R01A	< 4	Boto 2/Mali	Fresh Rock	
R02A	9	Boto 2/Mali	Fresh Rock	DBDD-2200 - 302/317
R03A to R05A	< 4	Boto 2/Mali	Fresh Rock	
R06A	23	Boto 2/Mali	Fresh Rock	DBDD-2072 - 49/51, 55/57, 72/73, 98/99
R07A	4	Boto 2/Mali	Fresh Rock	DBDD-2075 - 121/122, 142/147 DBDD-2110 - 102/106
R08A to R27A	< 4	Boto 2/Mali	Fresh Rock	
R28	8	Boto 2/Mali	Saprolite	DBDD-2078 - 8/9, 16/26 DBDD-2162 - 44/47 DBDD-2240 - 27/31 DBDD-2262 - 25/27
R29	< 4	Boto 2/Mali	Saprolite	
R30A	< 4	Boto 2/Mali	Saprolite	
R31	36	Boto 2/Mali	Saprolite	DBDD-2109 - 16/18, 30/43
R32	< 4	Boto 2/Mali	Saprock	
R33	< 4	Boto 2/Mali	Saprock	
R34A	4	Boto 5	Fresh Rock	DBDD-2035 - 145/158
R35A	< 4	Boto 5	Fresh Rock	
R36	15	Boto 5	Saprolite	DBDD-2030 - 10/29
R37	47	Boto 5	Saprolite	DBDD-2030 - 5/10, DBDD-2022 - 36/43 DBDD-2012 - 74/78
R38	10	Boto 5	Saprolite	DBDD-2022 - 98/116
R39A	11	Boto 5	Saprock	DBDD-2018 - 76/78, 83/84, 85/89 DBDD-2012 - 81/81.5, 82/83, 83.5/85 DBDD-2041 - 71/72
R40A	< 4	Boto 5	Saprolite	

Gravity Separation Testwork

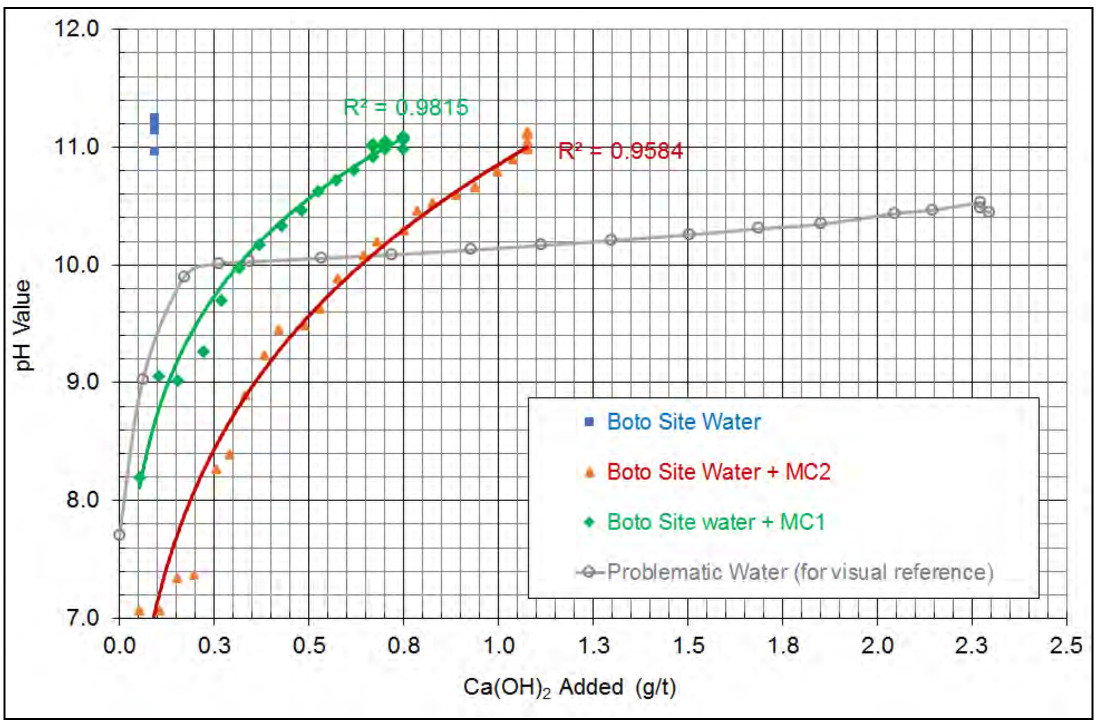
Gravity separation testwork was conducted on MC-1 and MC-2 using a Knelson concentrator coupled with a Mozley laboratory separator to generate material for grind calibration and for downstream testwork such as pH neutralization, rheology and cyanidation tests. Initially, one test was conducted on MC-1 and two were conducted on MC-2. The results showed gravity recoverable gold at 19.2% for MC-1 (Test No. G2) and 18.8% and 4.9% for MC-2 (Tests No. G1 and G3, respectively).

Additional gravity separation tests were later conducted to generate material for the leach optimization testwork. The results showed gravity recoverable gold at 25.7% for MC-1 (Test No. G4), and 15.2% and 19.0% for MC-2 (Tests No. G3R and G5, respectively).

pH Neutralization Testwork

pH neutralization tests are performed by adding Milk of lime to the MC-1 and MC-2 slurries. Boto site water was used in grinding both of the composites and in preparing the Milk of lime during this testwork. The results from the pH neutralization testwork are shown in Figure 13.28.

Figure 13.28 SGS 2017/2018 pH Neutralization Testwork



The Boto site water, and MC-1 and MC-2 slurries reacted well to the lime addition as seen in Figure 13.28. The figure also shows the results for a problematic site water in the grey line as a benchmark. Excess lime consumption is not expected during the testwork and the downstream tests should be able to maintain pH 10.5 to 11.0 for the pulp.

Rheology Testwork

Rheology tests were conducted on two gravity tailings samples, G1 and G2. The G1 sample represents MC-2 gravity tailings, while the G2 sample represents MC-1's. The main objective of this testwork is to determine the maximum solids density that each slurry can be operated safely at. Figure 13.29 and Figure 13.30 show plots of solids density versus yield stress for both G2 and G1 samples, respectively.

Figure 13.29 SGS 2017/2018 Solids Density vs. Yield Stress for MC-1 Gravity Tailings (G2)

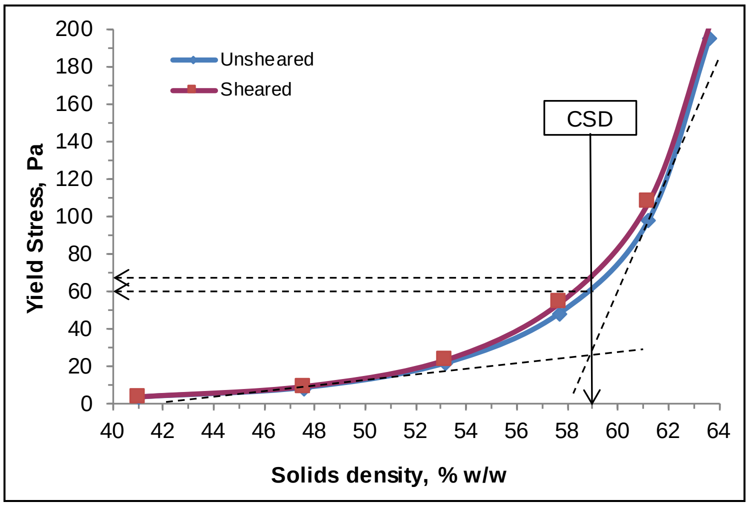
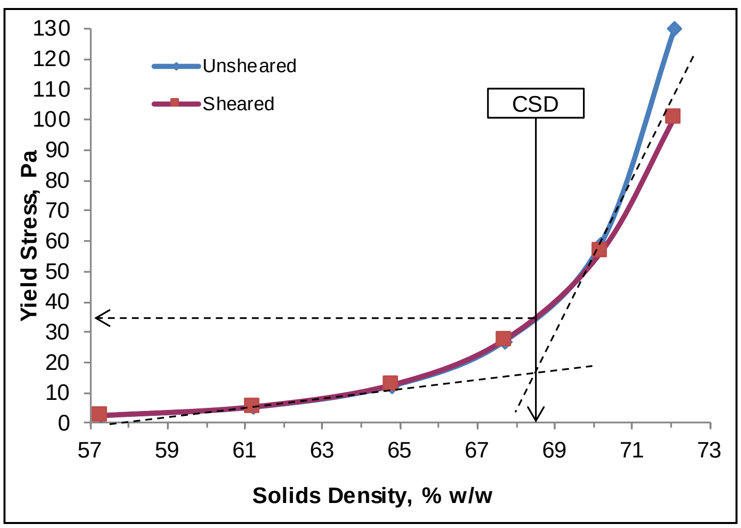


Figure 13.30 SGS 2017/2018 Solids Density vs. Yield Stress for MC-2 Gravity Tailings (G1)

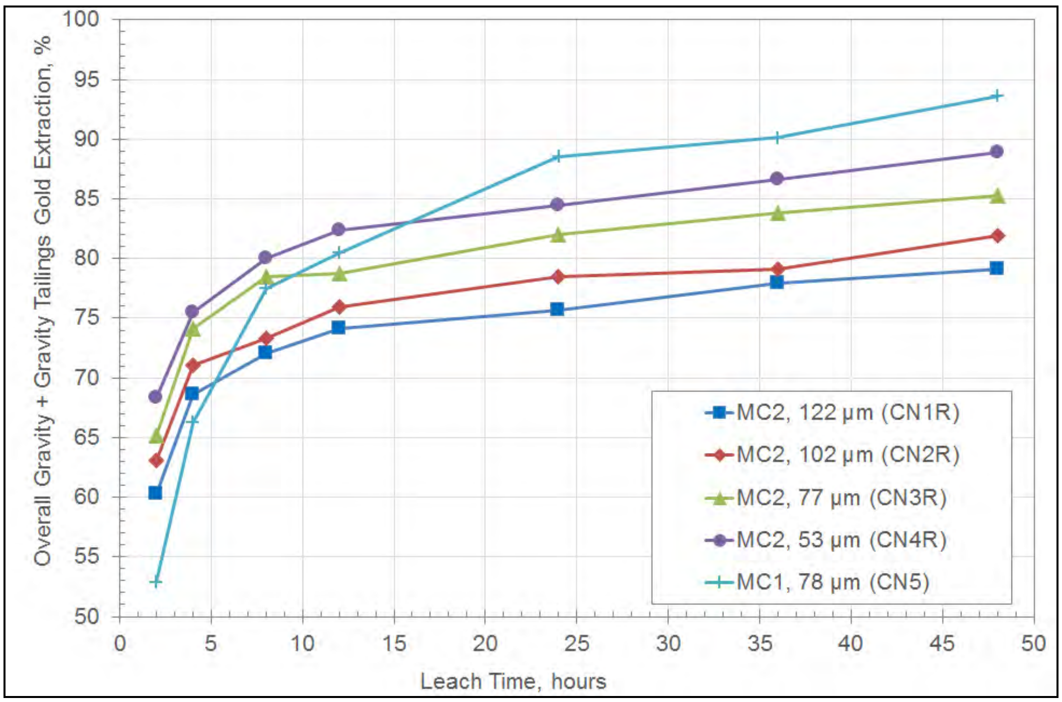


Based on the rheology results, the safe operating density where yield stress is under 10 Pa is ~50% solids for MC-1 and ~64% solids for MC-2.

Grind Optimization Testwork

The grind optimization testwork series included four tests on G3 (MC-2) at targeted grind P_{80}'s of 125 µm, 106µm, 75 µm and 53 µm. Only one test at a targeted grind P_{80} of 75 µm for G1 (MC-1) was conducted as it is anticipated that the different targeted grind sizes for saprolitic material has insignificant impact on gold extraction. Figure 13.31 describes the results from the grind optimization testwork.

Figure 13.31 SGS 2017/2018 Grind Optimization Testwork Results



Note: CN1R to CN4R are repeated tests of the CN1 to CN4 tests due to contamination issues at the laboratory.

As expected, an improvement in leach kinetics and gold extraction is observed in general at decreasing grind sizes for MC-2. A trade-off study was conducted by Lycopodium and the conclusion recommended using a targeted grind size of 75 µm. More details can be found in Section 13.3.2, Grind Size Selection.

Leach Optimization Testwork

The leach optimization testwork series were all conducted at the targeted grind size P_{80} of 75 µm. The testwork included varying lead nitrate concentration, cyanide concentration, pulp density, incorporating pre-aeration/ pre-oxygenation, and adding oxygen during leaching to increase the dissolved oxygen level.

The results for all the leach optimization testwork conducted on gravity tailings are shown in Table 13.33.

Table 13.33 SGS 2017/2018 Summary of Leach Optimization Results

Test	Feed = Tails from Test	Feed Size, µm	Pre-aer. / Pre-ox. Lead Nitrate	Air or O_2 Pre.	Air or O_2 Leach	Time , h	DO_2, mg/L (Avg)	Pulp Density (% Solids)	NaCN, g/L, (start/maint.)	Reagents (kg/t of CN Feed) Added NaCN	Added CaO	Consumed NaCN	Consumed CaO	Au Extraction / Recovery (%) 2 h	4 h	8 h	12 h	24 h	36 h	48 h	Grav.	Grav + CN	1 O'all Norm Rec.,%	Leach Res. Au, g/t	Head Grade, Au, g/t CN (calc)	Grav + CN (calc)	Direct
Composite MC-1																											
CN9		77	250	Air	--	6	6.8	38.2	0.5/0.35	2.07	3.62	1.84	3.14	71	84	88	88	91	90	**91.7**		**93.4**	**94.3**	0.20	2.35		
CN10	G2		75	Air	--	6	8.3	40.6	0.5/0.35	2.62	2.95	2.47	2.75	68	78	84	69	90	88	**90.3**	**19.9**	**92.3**	**93.6**	0.22	2.28		
CN5		78	0	Air	--	6	5.5	40.2	0.5/0.35	1.31	1.85	0.88	1.83	42	58	72	76	86	88	**92.1**		**93.7**	**94.3**	0.20	2.54		
CN17			0	Air	--	0	5.5	36.9	0.5/0.35	1.29	1.85	0.76	1.68	44	63	79	87	91	92	**92.0**		**94.0**	**94.8**	0.18	2.25		
CN18			0	O_2	--	4	17.8	36.9	0.5/0.35	1.21	2.45	0.73	2.17	73	83	92	95	98	94	**92.0**		**94.0**	**95.1**	0.17	2.09	3.06	3.45
CN19			0	O_2	--	8	15	36.2	0.5/0.35	1.26	2.03	0.75	1.87	65	73	82	89	91	93	**91.2**		**93.7**	**94.3**	0.19	2.48		
CN26	G4	82	200	Mix	Mix	4	14.1	35.8	0.5/0.35	1.59	1.17	0.82	1.05	73	82	90	93	90	**92.2**	--	**25.7**	**94.2**	**94.9**	0.18	2.28		
CN27			75	Mix	Mix	4	13.8	36.9	0.5/0.35	1.37	0.93	0.89	0.76	75	81	84	86	91	--	--		**93.0**	**93.8**	0.22	2.27		
CN30			200	Mix	Mix	4	15.2	34.4	0.5/0.4	1.57	1.38	0.94	1.31	75	88	92	95	94	**90.8**	--		**93.2**	**94.0**	0.21	2.27		
CN31			200	Mix	Mix	4	14.1	43.4	0.5/0.4	1.29	1.76	0.82	1.71	77	85	90	91	91	**91.7**	--		**93.3**	**94.8**	0.18	2.18		
Composite MC-2																											
CN6			250	Air	--	6	6.9	54.5	0.5/0.35	2.95	2.06	2.89	2.06	74	77	79	80	81	81	**81.9**		**84.6**	**84.8**	0.26	1.41		
CN7	G3R	76	125	Air	--	6	7.5	56.4	0.5/0.35	1.68	1.94	1.61	1.77	74	78	82	79	83	82	**80.8**	**15.2**	**83.7**	**83.9**	0.27	1.41	1.74	
CN8			75	Air	--	6	8.1	52.3	0.5/0.35	1.96	2.10	1.94	1.90	71	77	76	80	81	79	**78.7**		**81.9**	**82.1**	0.30	1.41		
CN3R		77	0	Air	--	6	5.1	51.3	0.5/0.35	1.08	1.89	0.85	1.86	59	70	75	75	79	81	**82.7**		**83.5**	**85.1**	0.25	1.44		
CN11			0	Air	--	0	5.9	47.9	0.5/0.35	0.86	1.64	0.51	1.64	35	58	70	78	82	85	**85.5**		**88.2**	**87.7**	0.21	1.42		
CN12			0	O_2	--	4	19.8	47.8	0.5/0.35	0.65	1.65	0.34	1.57	76	78	81	79	85	85	**87.5**		**89.9**	**89.6**	0.18	1.41		
CN13			75	O_2	--	4	19.6	47.0	0.5/0.35	0.60	1.85	0.30	1.79	81	84	85	86	88	89	**87.2**		**89.6**	**89.5**	0.18	1.37		
CN15			0	Air	--	4	6.0	48.7	0.5/0.35	1.06	2.45	0.91	2.41	51	64	72	73	82	85	**84.2**		**87.2**	**86.3**	0.23	1.46		
CN16			0	Air	--	8	6.7	50.1	0.5/0.35	1.09	3.47	0.97	3.41	58	69	76	79	82	80	**82.4**		**83.5**	**85.1**	0.25	1.42		1.68
CN20			200	Mix	Mix	4	13.9	46.4	0.5/0.35	0.59	1.59	0.15	1.45	75	82	83	85	88	**86.6**	--		**89.1**	**88.7**	0.19	1.42		
CN21	G5	75	200	Mix	Mix	4	14.0	46.5	0.35/0.25	0.45	1.46	0.20	1.39	73	82	83	85	82	**85.8**	--	**19.0**	**88.5**	**88.1**	0.20	1.41	1.73	
CN22			75	Mix	Mix	4	12.8	47.5	0.5/0.35	0.59	1.19	0.19	1.17	72	79	81	81	**84.6**	--	--		**87.6**	**86.9**	0.22	1.43		
CN23			200	Air	Air	4	6.1	47.3	0.5/0.35	0.76	1.53	0.47	1.51	74	76	79	80	81	**83.4**	--		**84.6**	**86.6**	0.23	1.36		
CN24			0	Mix	Air	4	5.9	44.4	0.5/0.35	1.00	1.76	0.64	1.71	64	71	77	78	83	**83.0**	--		**86.2**	**85.4**	0.25	1.44		
CN25			200	Mix	Air	4	5.8	48.0	0.5/0.35	0.95	1.70	0.71	1.70	68	75	78	79	81	**84.1**	--		**87.2**	**86.3**	0.23	1.45		
CN28			200	Mix	Mix	4	15.3	45.1	0.5/0.4	0.71	2.08	0.30	2.06	75	80	84	85	88	**87.8**	--		**90.1**	**89.3**	0.18	1.47		
CN29			200	Mix	Mix	4	14.9	52.7	0.5/0.4	0.59	2.22	0.33	2.20	73	75	78	83	84	**86.8**	--		**89.3**	**89.0**	0.19	1.40		

(1) Overall normalized recovery % is calculated by using the direct head grade and the leach residue grade.
(2) All other extraction/recovery values are calculated based on pregnant solution samples.

Varying the lead nitrate addition was the first set of leach optimization tests. Figure 13.32 and Figure 13.33 show plots of the gold extraction at different leach time for Tests No. CN6 to CN10 in comparison to the base cases, CN5 and CN3R, for MC-1 and MC-2, respectively. Based on the results, the addition of lead nitrate improved the leach kinetics but did not improve the gold extraction significantly. It was also expected that cyanide consumption would decrease, however, this effect was not observed in the results. It was noted that this set of tests encountered evaporation problems possibly due to the use of smaller sample size (i.e., 500 g) in an attempt to conserve sample mass. Later tests conducted all used standard sample mass size of 1,000 g.

Figure 13.32 SGS 2017/2018 Effect of Lead Nitrate on MC-1 Gravity Tailings Cyanidation

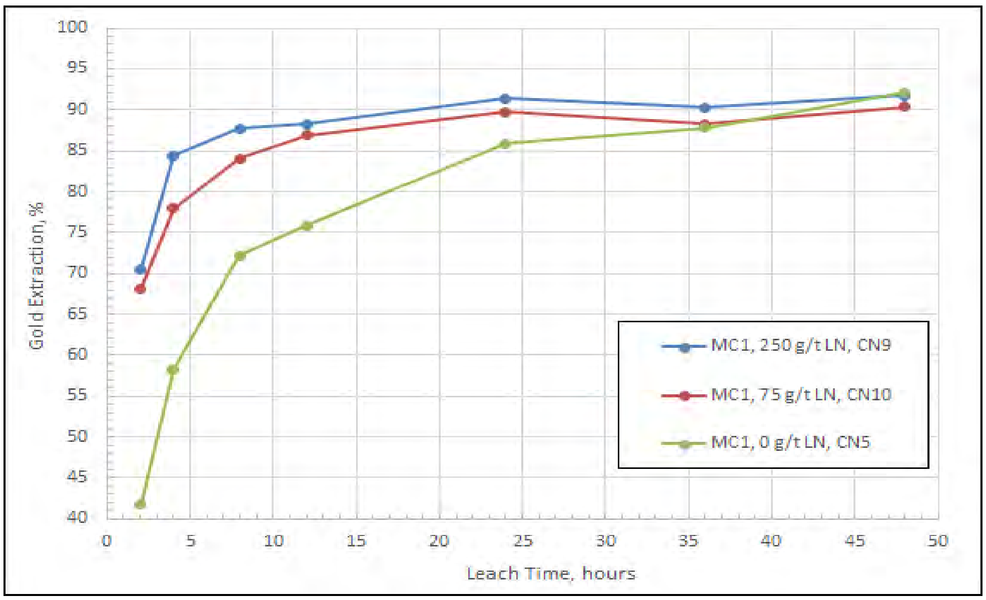
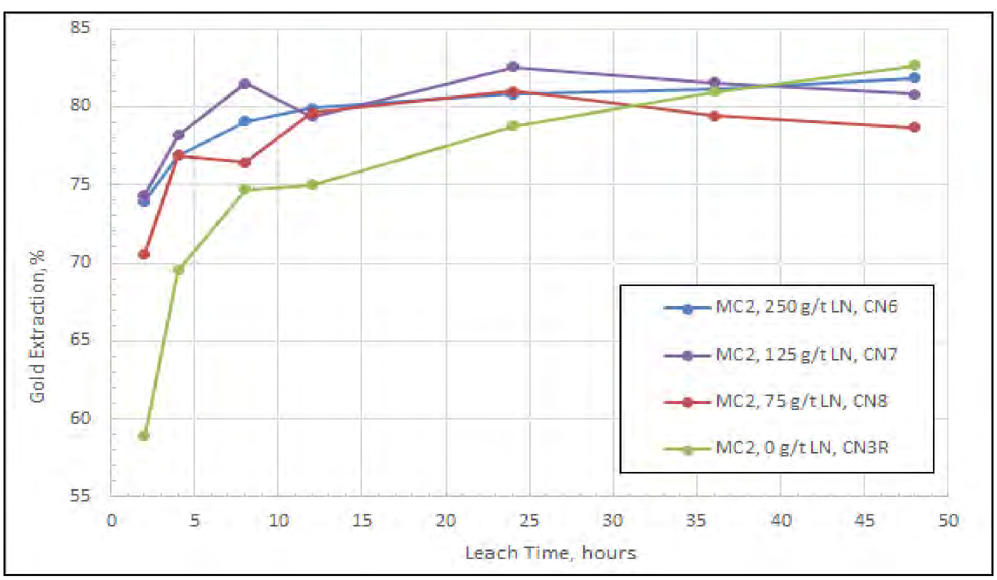


Figure 13.33 SGS 2017/2018 Effect of Lead Nitrate on MC-2 Gravity Tailings Cyanidation



Additional leach optimization tests were conducted using standard sample mass size of 1,000 g to repeat a few of the lead nitrate tests and also to study the effect of pre-aeration, pre-oxygenation, and addition of oxygen during leaching. The results for Tests No. CN11 to CN27 are plotted in Figure 13.34 and Figure 13.35 with comparison to the new base cases, CN17 and CN11, for MC-1 and MC-2, respectively.

Figure 13.34 SGS 2017/2018 Effect of Lead Nitrate, Pre-aer., Pre-oxy. and O_2 Addition on MC-1

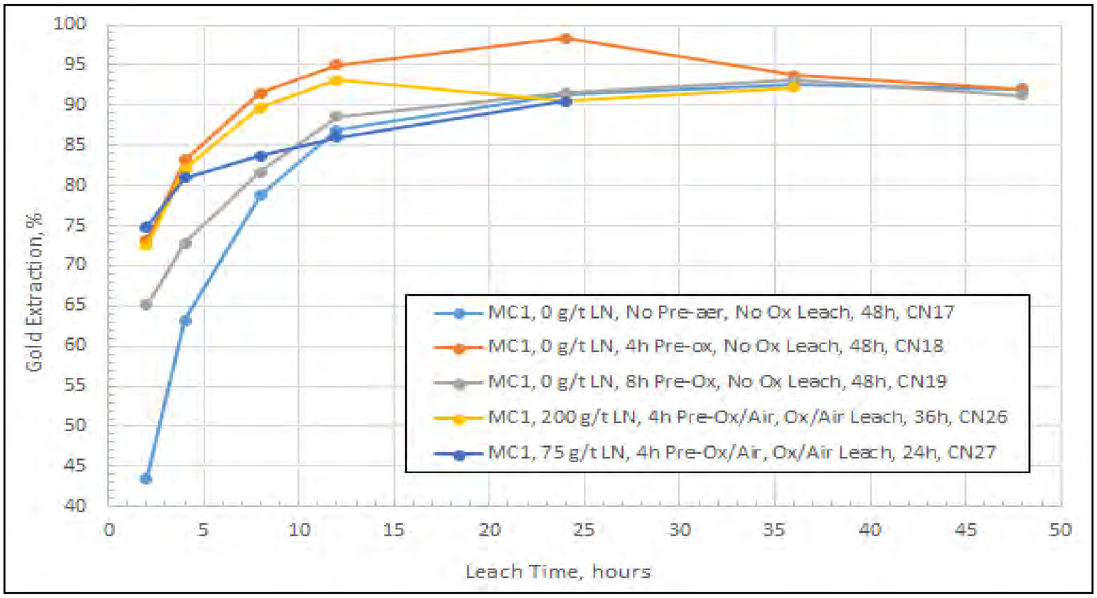
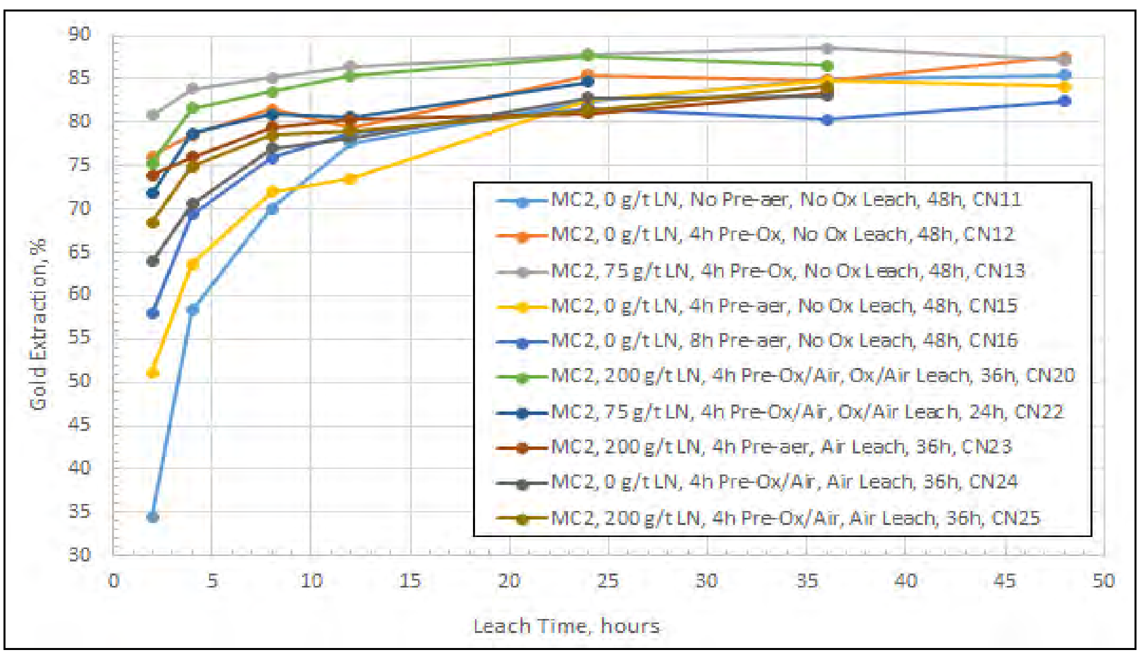


Figure 13.35 SGS 2017/2018 Effect of Lead Nitrate, Pre-aer., Pre-oxy. and O_2 Addition on MC-2

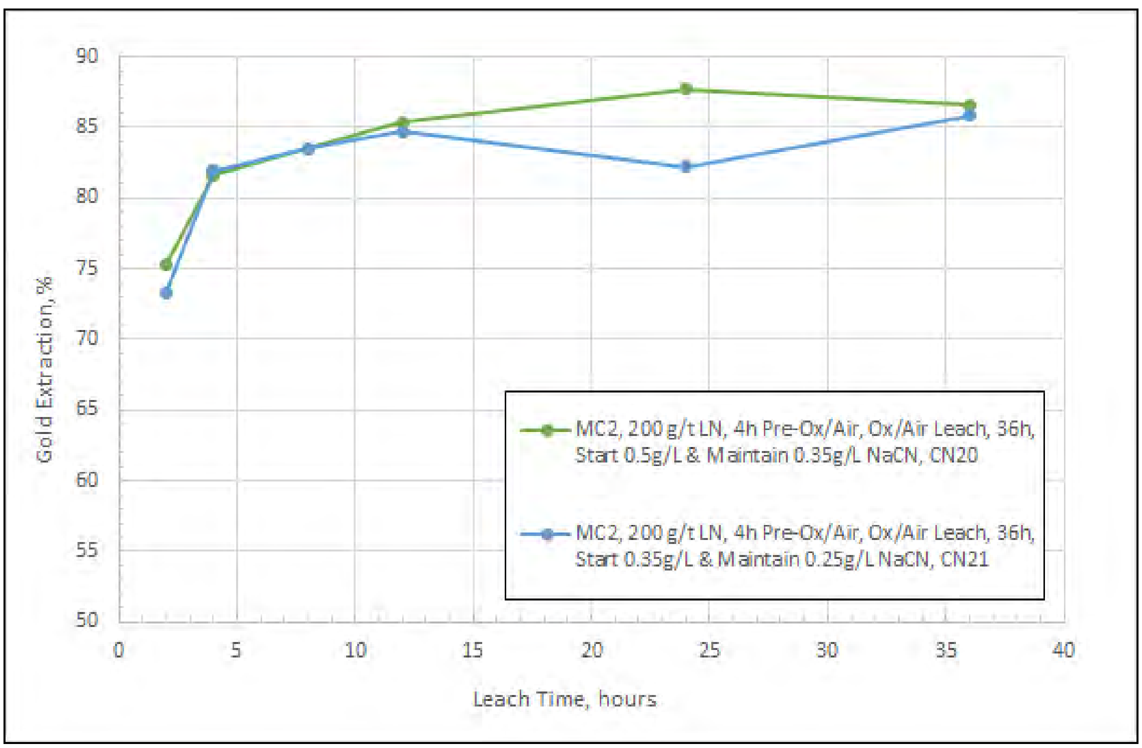


MC-1 gold extractions are consistently high, ranging from 90 % to 92%. The addition of lead nitrate, and addition of oxygen in the pre-treatment step and during leaching provided insignificant improvement to the gold extraction for MC-1. However, improvement is seen in the leach kinetics during the initial leach period. There were also no improvement in the cyanide consumption.

MC-2 overall gold extractions improved by an estimated 1.8% with the addition of lead nitrate and oxygen. The leach kinetics were also significantly more rapid indicating that the leach time can essentially be decreased from 48 hours to 36 hours, and possibly even as low as 24-hours. Lead nitrate addition of 200 g/t Au provided better leach performance than the test with lead nitrate added at 75 g/t Au (i.e., CN20 versus CN22). Adding oxygen mixed with air instead of only air during leaching also provided better leach performance as seen in CN20 vs. CN25. Lastly, cyanide consumption was seen to be reduced by approximately 50% or more with the addition of lead nitrate and oxygen.

Cyanide concentrate was varied for one of the MC-2 leach tests. Figure 13.36 shows the comparison between CN20 and CN21 where one had 0.5 g/L NaCN at the start and maintained at 0.35 g/L NaCN throughout the test, and the other had 0.35 g/L NaCN at the start and maintained at 0.25 g/L NaCN throughout the test. The leach performance was very similar between the two tests with CN20 only slightly better. However, it was also noted that some tests showed signs of negative effect when the cyanide concentration dropped below 0.35 g/L by accident, hence, it was agreed that instructions for the lab technicians should be changed to dropping cyanide concentration to 0.4 g/L prior to topping up to avoid going below the 0.35 g/L level again.

Figure 13.36 SGS 2017/2018 Effect of Lowering Cyanide Concentration on MC-2



A set of leach conditions was agreed upon prior to starting the pulp density optimization leach tests (CN28 to CN31) and those conditions were as follows:

- Pulp pH at 10.5 to 11 and temperature at 35 °C maintained.
- Pre-treatment with air and oxygen mixture for 4 hours, with lead nitrate addition at 200 g/t.
- Air/oxygen mixture addition during leaching to maintain a dissolved oxygen level at ~15 mg/L.
- Maintain 0.5 g/L NaCN at start of test and drop to 0.4 g/L before topping up.

The effect on gold extraction from varying pulp densities are shown in Figure 13.37 and Figure 13.38.

Figure 13.37 SGS 2017/2018 Leach Tests at Varying Pulp Densities for MC-1

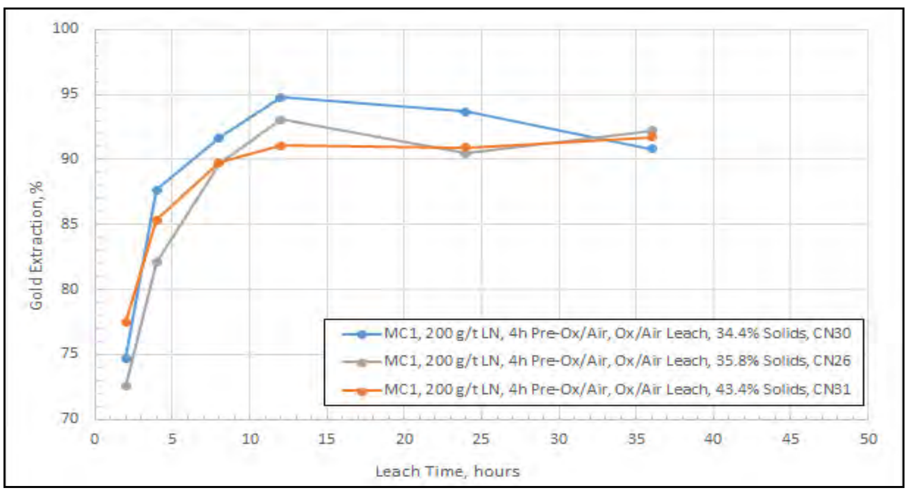
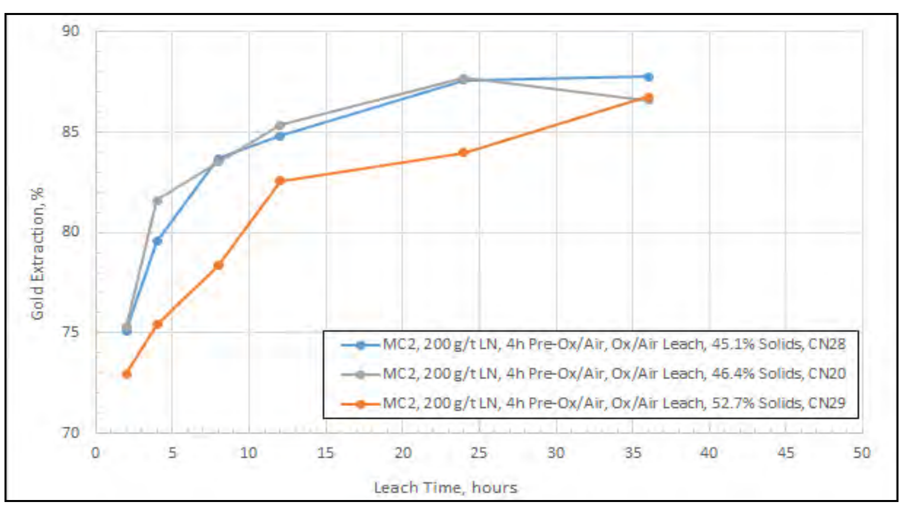


Figure 13.38 SGS 2017/2018 Leach Tests at Varying Pulp Densities for MC-2



As seen in Figure 13.37 and Figure 13.38, both MC-1 and MC-2 gravity tailings showed slower leach kinetics with increasing pulp densities, however, the overall gold extractions were not impacted as significantly. It was later suggested by SGS that conducting the pulp density leach tests on whole ore could yield different results from the tests conducted on gravity tailings. Consequently, three additional pulp density leach tests were later conducted and the results are presented as part of the whole ore cyanidation testwork section.

The pulp density selected for use in subsequent tests is 40% solids for MC-1 and 50% solids for MC-2.

Mineralogy Analysis on Gravity Tailings and Leach Residues

The mineralogy analysis on MC-1 and MC-2 gravity tailings and their leach residues is summarized in Table 13.34 and Table 13.35. The gold in the residue is mainly locked with Calaverite.

Table 13.34 SGS 2017/2018 Characteristics of Microscopic Gold for Gravity Tailings & Leach Residues

Sample ID	Au Grade (g/t)	Association	# of Gold Grains	Size Range (µm)	Average Size (µm)	Gold Minerals Type by Association (%)			Overall Gold Mineral Abundfance (%)
						Gold	Clv	Ptz, etc.	
MC2 (G1 Tail)	1.33	Liberated	32	0.6 - 19.6	7.6	7.20	8.31	-	gold (85.8%), calaverite (12.5%), petzite and Au-Cu-Zn-Ag (1.7%).
		Exposed	43	0.6 - 36.4	4.6	63.3	0.55	-	
		Locked	127	0.6 - 11.7	2.4	15.3	3.61	1.70	
			202	**0.6 - 36.4**	**3.6**	**85.8**	**12.5**	**1.70**	**100**
MC1 (G2 Tail)	2.37	Liberated	150	0.6 - 58.5	7.7	63.3	3.43	2.97	gold (80.9%), petzite (10.4%) and calaverite (8.76%).
		Exposed	47	0.6 - 16.9	4.2	2.79	1.15	4.29	
		Locked	121	0.6 - 21.3	2.6	14.8	4.18	3.09	
			318	**0.6 - 58.5**	**5.2**	**80.9**	**8.76**	**10.4**	**100**
CN3R (Comp MC2) CN Res	0.25	Liberated	9	1.6 - 13.7	7.2	0.0	33.4	-	calaverite (75.7%) and gold 24.3%.
		Exposed	3	1.7 - 5.2	3.1	0.8	3.17	-	
		Locked	93	0.6 - 8.4	1.7	23.5	39.2	-	
			105	**0.6 - 13.7**	**2.2**	**24.3**	**75.7**	**-**	**100**
CN5 (Comp MC1) CN Res	0.20	Liberated	0	0.0 - 0.0	0.0	-	-	-	calaverite (80.1%) and gold 19.9%.
		Exposed	0	0.0 - 0.0	0.0	-	-	-	
		Locked	101	0.3 - 8.6	2.0	19.9	80.1	-	
			101	**0.3 - 8.6**	**2.0**	**19.9**	**80.1**	**-**	**100**

Table 13.35 SGS 2017/2018 Overall Gold Deportment for Gravity Tailings & Leach Residues

Sample ID	Gold Dist. by Association			Gold Dist. by Gold Mineral Type			Minerals Associated with Exposed and Locked Au-Minerals
	Association Type	Distribution %	Grade g/t	Mineral Type	Distribution %	Grade g/t	
MC2 (G1 Tail)	Liberated	25.3	0.34	Gold	93.2	1.24	silicates 70.3%, pyrite 17.4%, calcites 3.74%, Pb-Te/rutile 3.63%, silicates/iron oxide 1.01%, quartz 0.97%, other minerals 0.78%, and pyrite/other minerals 2.17%.
	Exposed	25.7	0.34	Calaverite	6.28	0.08	
	Locked	49.0	0.65	Petzite, etc	0.52	0.01	
		100	**1.33**		**100**	**1.33**	
MC1 (G2 Tail)	Liberated	58.8	1.39	Gold	91.3	2.16	pyrite 81.4%, silicates 12.8%, pyrite/silicates 1.45%, quartz 1.42%, petzite 1.03%, gold 0.98%, calcites 0.36%, Bi-Te 0.34%, and other minerals 0.29%.
	Exposed	12.2	0.29	Calaverite	5.8	0.14	
	Locked	29.0	0.69	Petzite	3.0	0.07	
		100	**2.37**		**100**	**2.37**	
CN3R (Comp MC2) CN Res	Liberated	12.3	0.03	Gold	39.8	0.10	pyrite 49.3%, silicates 33.6%, dolomite 11.8%, quartz 5.24%, iron oxide 0.02%, and pyrite/chalcopyrite 0.02%.
	Exposed	0.89	0.00	Calaverite	60.2	0.15	
	Locked	86.8	0.22	-	-	-	
		100	**0.25**		**100**	**0.25**	
CN5 (Comp MC1) CN Res	Liberated	0.00	-	Gold	32.4	0.06	silicates 48.1%, pyrite 40.6%, iron oxide 7.07%, Bi-Te/pyrite 2.45%, quartz/ zircon 0.75%, Ni-Te/pyrite 0.45%, rutile/silicates 0.35%, and other minerals 0.17%.
	Exposed	0.00	-	Calaverite	67.6	0.14	
	Locked	100	0.20	-	-	-	
		100	**0.20**		**100**	**0.20**	

Oxygen Uptake Testwork

Oxygen uptake tests were conducted on MC-1 and MC-2 gravity tailings to estimate the rate at which oxygen is consumed in the slurry. Refer to Figure 13.39 and Figure 13.40 for details.

Figure 13.39 SGS 2017/2018 Oxygen Uptake Results for MC-1

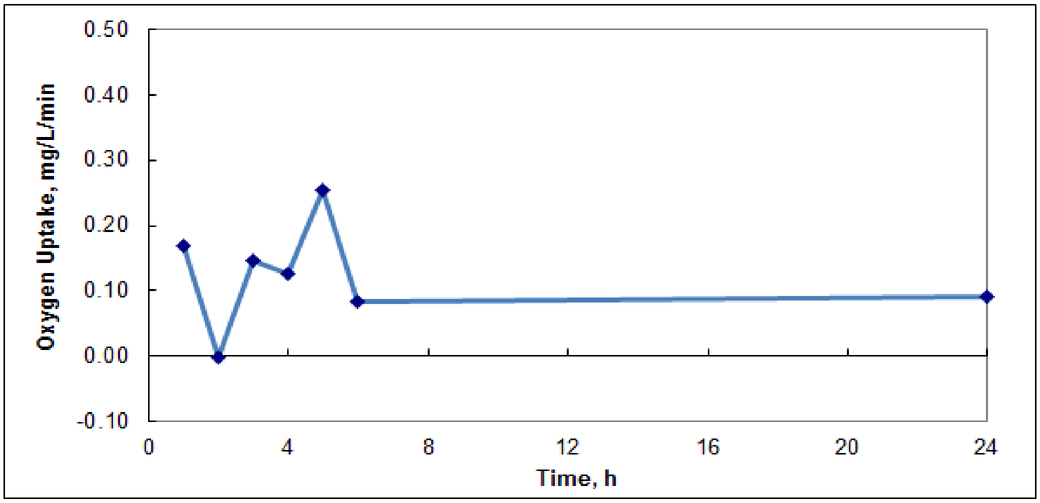
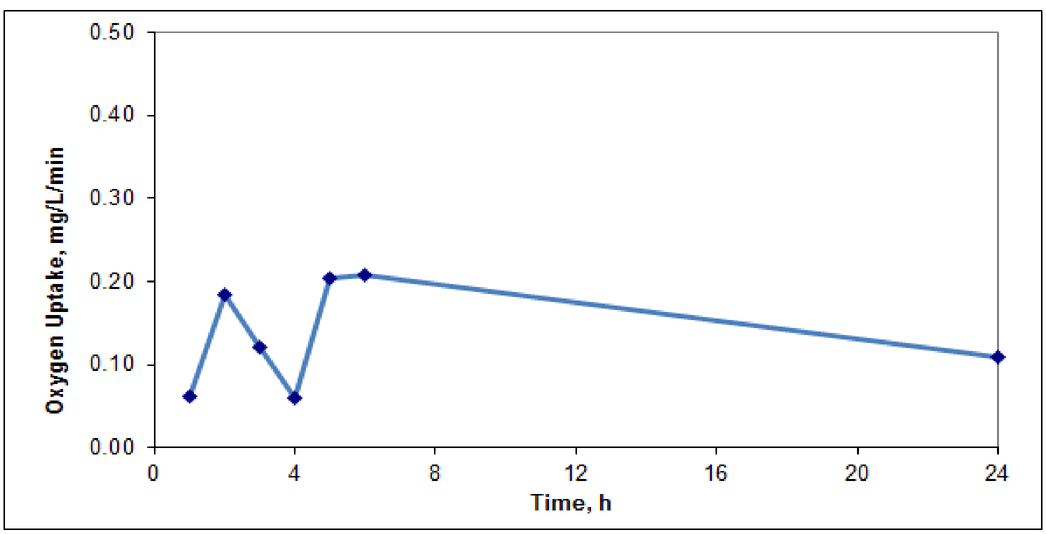


Figure 13.40 SGS 2017/2018 Oxygen Uptake Results for MC-2



The industry rule of thumb for oxygen uptake rate is 0.04 mg/L/min for saprolitic ore and 0.10 mg/L/min for primary ore (fresh rock). A low oxygen uptake rate indicates that conventional air sparging will be suitable while a high rate indicates that oxygen sparging will be required.

Based on the results, oxygen is consumed on average at a rate of 0.123 mg/L for MC-1 and 0.135 mg/L for MC-2, which is considered to be on the higher end of the range, hence, sparging with oxygen is recommended.

Preg-robbing Assessment

Results from the preg-robbing assessment are shown in Table 13.36.

Table 13.36 SGS 2017/2018 Preg-robbing Assessment Results

Test	Feed = Tail from Test	Feed Size, µm	Carbon Concentration (g/L)	Au Extraction/Recovery %			Leach Res. g/t Au	Head Grade, g/t Au		
				48 h	Grav.	Grav + CN		CN (calc)	Grav + CN	Direct
Composite MC1										
PR2a	G2	76	0	**87.7**	**19.9**	**90.2**	0.31	2.49	2.95	3.45
PR2b			20	**88.3**		**90.6**	0.27	2.27		
Composite MC2										
PR1a	G3R	77	0	**79.8**	**15.2**	**82.8**	0.30	1.48	1.74	1.68
PR1b			20	**81.8**		**84.6**	0.27	1.46		

There is no evidence of preg-robbing activity in both MC-1 and MC-2 gravity tailings as the results of the tests with carbon added are very similar to the ones conducted with no carbon.

Extended Gravity Recoverable Gold (E-GRG) and Gravity Circuit Modelling by FLSmidth

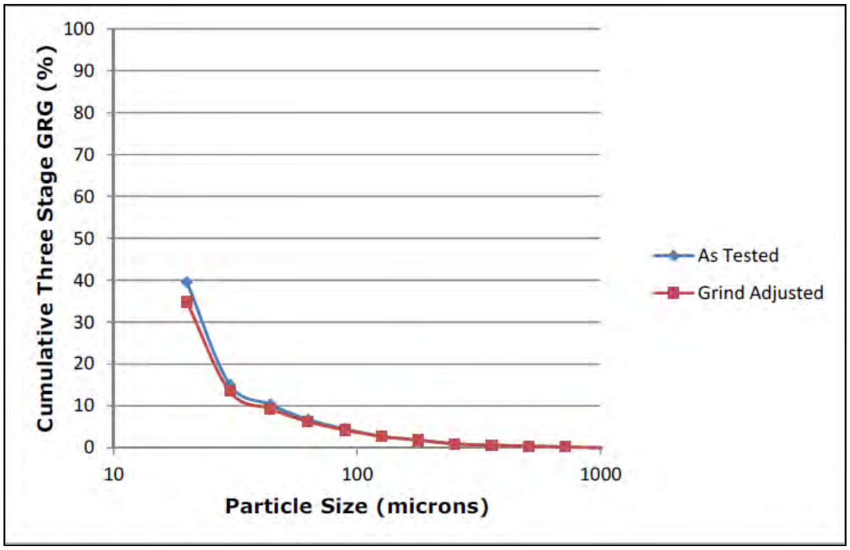
E-GRG test was conducted on MC-2 composite to determine the gravity-recoverable gold (GRG) number. The results from the E-GRG test is presented in Table 13.37 and the GRG number was determined to be 39.6.

Table 13.37 SGS 2017/2018 E-GRG Test Results for MC-2

Grind Size	Product	Mass		Assay (g/t Au)	Units Au	Dist'n %
		grams	%			
P_{80} = 626 µm	Stage 1 Conc	83.0	0.83	26.9	2,236	**11.6**
603 µm	Sampled Tails	286.1	2.87	1.59	455	**2.36**
P_{80} = 195 µm	Stage 2 Conc	87.7	0.88	32.2	2,826	**14.7**
165 µm	Sampled Tails	262.3	2.63	1.49	391	**2.03**
P_{80} = 75 µm	Stage 3 Conc	65.4	0.66	38.9	2,543	**13.2**
47 µm	Final Tails	9,171	92.1	1.18	10,778	**56.1**
	Totals (Head)	9,955	100.0	1.93	19,228	**100.0**
	Knelson Conc	236.1	2.37	32.2	7,605	**39.6**

FLSmidth was consulted by SGS to conduct a gravity circuit modelling exercise. The three stage E-GRG test ground the sample to 47 µm, however, the plant is expected to grind the ore to 75 µm, thus there will be less GRG liberated at this coarser grind. FLSmidth used the “as-tested” data and adjusted them to predict the “grind adjusted” three stage cumulative curves in Figure 13.41.

Figure 13.41 SGS/FLSmidth Cumulative Three Stage GRG as a Function of Particle Size



Based on FLSmidth's assessment, the GRG number is moderate but the majority of the GRG is classified as "very fine" on the AMIRA gold grain size classification scale.

A gravity circuit model was generated based on the E-GRG results and plant inputs, and the results are presented in Table 13.38.

Table 13.38 FLSmidth Gravity Circuit Modelling Results

Knelson Model	Feed Rate to Gravity (mtph)	Concentrating Cycle Time (minutes)	Circulating Load Treated (%)	Gravity Recovery (% Au)	Concentrate Data (kg/day)	Concentrate Data (g/tonne)
KC-QS40	200	30	14	5.7	2640	312
KC-QS48	300	30	21	7.3	3120	336
2 x KC-QS48	900	30	43	10.7	6240	246

According to the assessment done by FLSmidth, the amenability for gravity recovery is low and it will be difficult to recover the very fine GRG at full scale. Based on the current E-GRG results, FLSmidth concluded that the benefits to having a gravity circuit for the Boto project will be limited and thus it is not recommended.

Whole Ore Cyanidation Testwork

In order to confirm that a gravity circuit has limited benefits to the overall gold extraction, whole ore cyanidation tests were conducted on both MC-1 and MC-2. The results for the whole ore leach tests are presented in Table 13.39. One gravity tailings leach test for each composite is also included in the table for comparison purposes.

Table 13.39 SGS 2017/2018 Whole Ore Cyanidation Test Results

Test	Purpose	Feed = Tail from Test	Feed Size, µm	Pulp Density, % Solids (w/w)		DO_2, mg/L (Avg)	NaCN, g/L, (start/maint.)	Reagents (kg/t of CN Feed)				Au Extraction / Recovery (%)								[1] % Overall Normalized Au Recovery Based on			Residue, Au, g/t					Head Grade, Au, g/t		
								Added		Consumed														Cut						
				Target	Actual			NaCN	CaO	NaCN	CaO	2 h	4 h	8 h	12 h	24 h	36 h	Grav	2 O'all	Average Head	Direct Head	Diff.	Avg.	a	b	c	d	CN (calc)	[2] O'all	Direct
Composite MC1																														
CN26	Gravity Tail CN	G4	82	40	36	14.1	0.5/0.35	1.59	1.17	0.82	1.05	73	82	90	93	90	**92.2**	**25.7**	**94.2**	94.4	94.9	0.5	0.18	0.24	0.16	0.15	0.16	2.28	3.06	3.45
CN32	Whole Ore CN	W/O	67	40	39	12.9	0.5/0.4	1.33	2.20	0.70	2.15	66	77	80	86	94	**95.5**	--	**95.5**	95.4	95.8	0.4	0.15	0.14	0.15	--	--	3.23		
Composite MC2																														
CN20	Gravity Tail CN	G5	75	50	47	13.9	0.5/0.35	0.59	1.59	0.15	1.45	75	82	83	85	88	**86.6**	**19.0**	**89.1**	89.1	88.7	0.4	0.19	0.20	0.18	--	--	1.42	1.73	
CN33	Whole Ore CN	W/O	74	50	50	12.6	0.5/0.4	0.69	1.76	0.29	1.74	78	81	86	85	85	**88.9**	--	**88.9**	88.8	88.4	0.4	0.20	0.18	0.21	--	--	1.76		
CN48	With Site Water	W/O	~75	50	50	11.0	0.5/0.4	0.57	1.95	0.20	1.95	80	83	84	86	87	**89.1**	--	**89.1**	87.4	86.9	0.5	0.22	0.22	0.22	--	--	2.02		
CN72	At Finer Grind	W/O	56	50	51	11.4	0.5/0.4	0.60	1.57	0.24	1.54	84	85	87	88	89	**90.8**	--	**90.8**	91.1	90.8	0.3	0.16	0.16	0.15	--	--	1.68		
CN95	With Stirred Reactor	W/O	77	50	49	18.2	0.5/0.4	0.56	2.19	0.16	2.16	82	83	84	85	86	**87.7**	--	**87.7**	88.2	87.8	0.4	0.21	0.19	0.22	--	--	1.67		1.68
CN114	Pulp Density	W/O	~75	45	45	13.0	0.5/0.4	0.91	1.50	0.48	1.45	82	84	87	87	92	**87.7**	--	**87.7**	87.6	87.2	0.4	0.22	0.21	0.22	--	--	1.74		
CN115	Pulp Density	W/O	~75	50	50	12.2	0.5/0.4	0.58	1.49	0.25	1.46	79	82	74	78	94	**87.6**	--	**87.6**	87.4	86.9	0.5	0.22	0.22	0.22	--	--	1.77		
CN116	Pulp Density	W/O	~75	55	54	11.6	0.5/0.4	0.52	1.53	0.24	1.50	68	68	59	78	91	**86.8**	--	**86.8**	88.2	87.8	0.4	0.21	0.21	0.20	--	--	1.55		

1 The normalized recoveries are calculated by comparing the actual assayed residue grades from each test to the calculated average or direct ore head grade.

2 Overall refers to "gravity separation + tailing cyanide leach" for Tests CN20 and CN26 and "Leach Only" for other tests in the table.

The following observations can be made from the whole ore leach tests:

- MC-1 overall gold extraction at 36-hours was similar for whole ore leaching compared to the leach results for gravity tailings (CN26 versus CN32, Figure 13.42).
- MC-2 overall gold extraction at 36 hours was similar for whole ore leaching compared to the leach results for gravity tailings (CN20 versus CN33, CN48, & CN95, Figure 13.43).
- The results for whole ore leaching using site water and using stirred reactor were similar to the results for whole ore leach with tap water.
- The results for whole ore leaching at the finer grind (P_{80} 51µm) showed 2% to 3% improvement in the gold extraction (Average of CN33, CN48, & CN95 vs CN72, Figure 13.44).
- The results for whole ore leaching at varying pulp densities for MC-2 showed that leach kinetics during the initial leach period is less rapid as the pulp density increases. The overall gold extraction at 36 hours for MC-2 was not impacted significantly at the different pulp densities tested. Refer to CN114 to CN116 in Figure 13.45.

Figure 13.42 SGS 2017/2018 Whole Ore vs. Gravity Tailings Cyanidation for MC-1

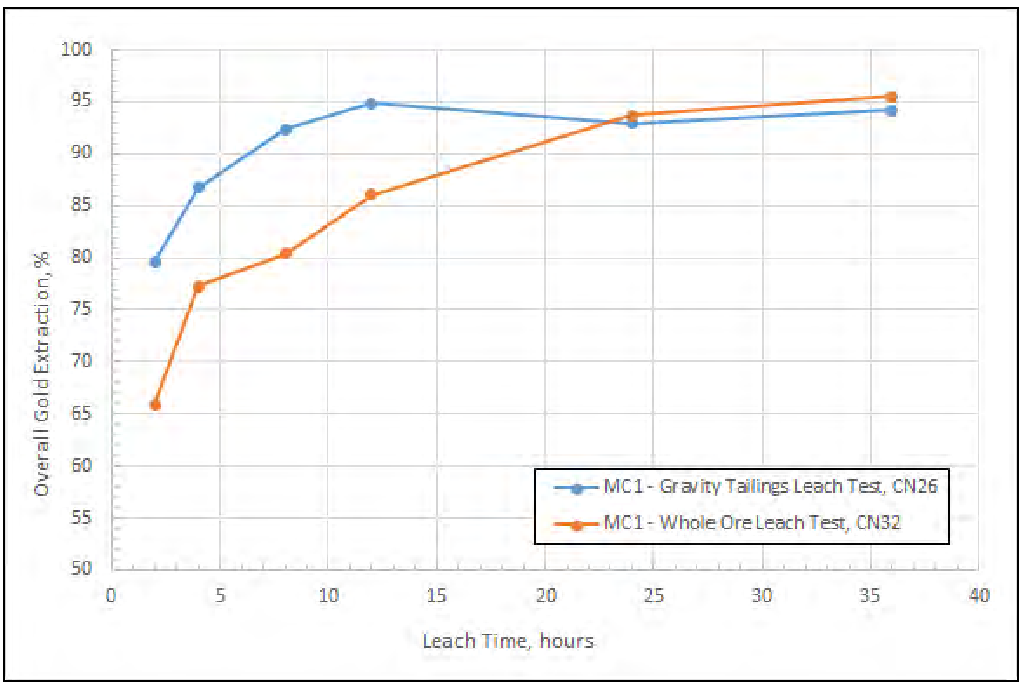


Figure 13.43 SGS 2017/2018 Whole Ore vs. Gravity Tailings Cyanidation for MC-2

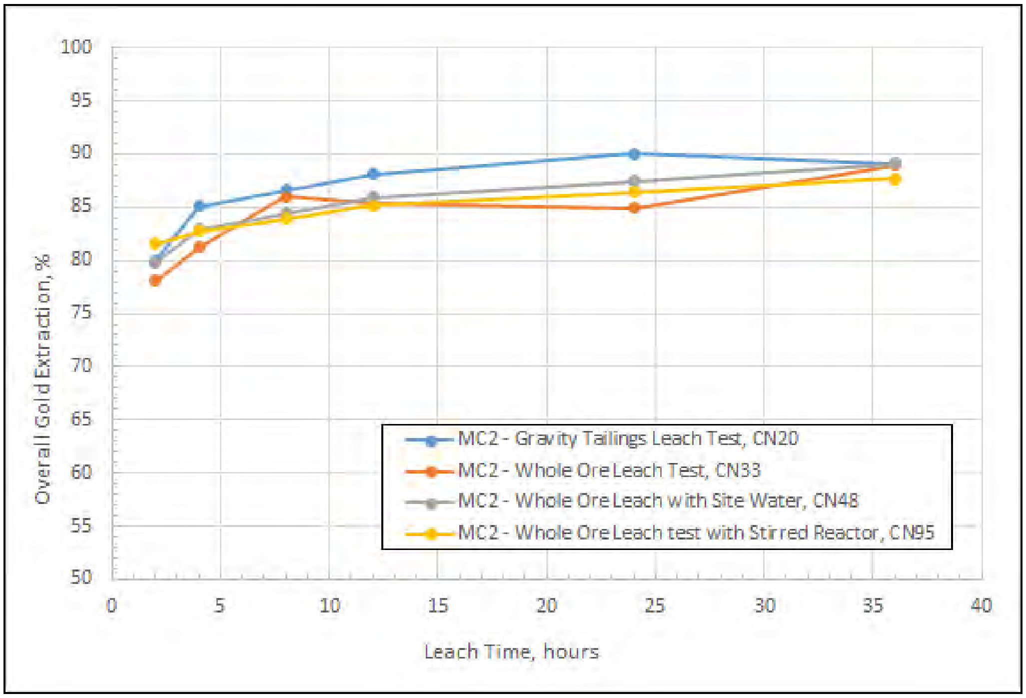


Figure 13.44 SGS 2017/2018 Effect of Whole Ore at Finer Grind for MC-2

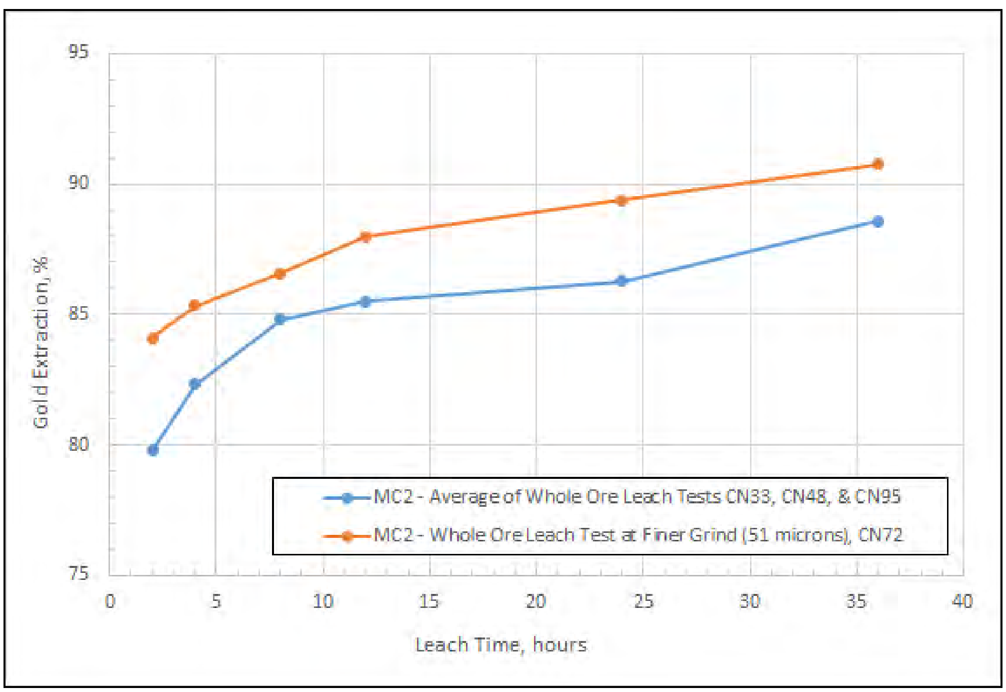
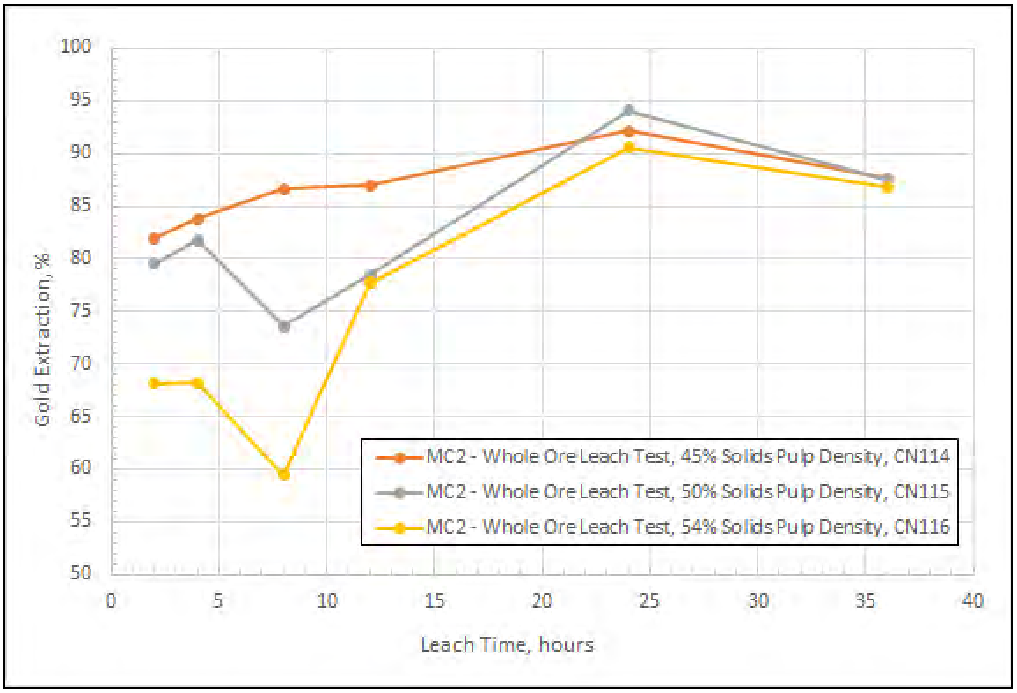


Figure 13.45 SGS 2017/2018 Effect of Pulp Densities on Whole Ore Leaching for MC-2



Carbon-in-Pulp (CIP) Modelling

In order to perform CIP modelling, results from leach kinetics testwork, carbon absorption isotherm testwork, and carbon absorption kinetics testwork are required. Figure 13.46 to Figure 13.48 illustrate the results from these tests.

Figure 13.46 SGS 2017/2018 Leach Kinetics for CIP Modelling

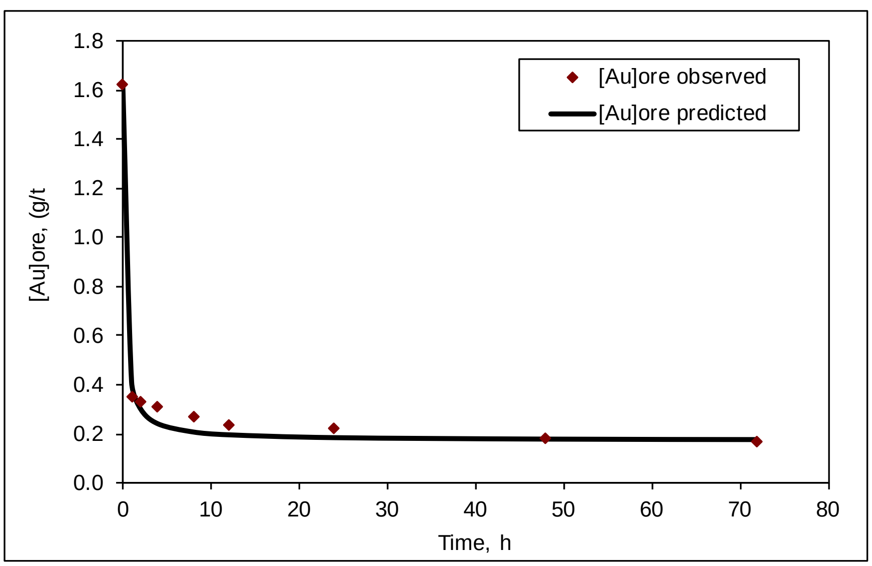


Figure 13.47 SGS 2017/2018 Carbon Absorption Kinetics for CIP Modelling

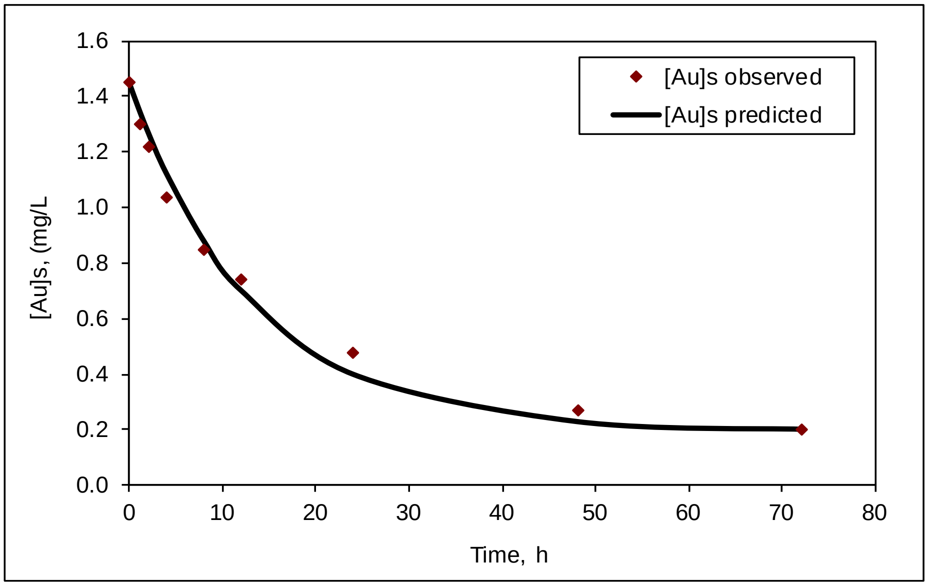
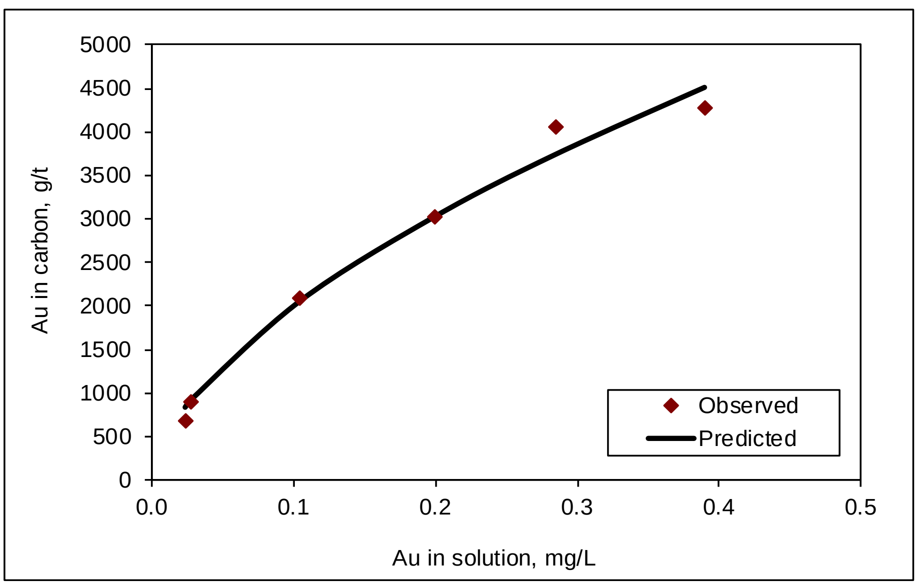


Figure 13.48 SGS 2017/2018 Carbon Absorption Isotherms for CIP Modelling



The CIP modelling results are shown in Table 13.40. Based on the modelling results, the currently selected daily carbon advancement rate of 5 tpd will be adequate for the Boto project. The various scenarios demonstrated that the carbon advancement rate can be lowered to as low as 2.5 tpd if operated more aggressively by allowing carbon to load a higher amount of gold.

Table 13.40 SGS 2017/2018 CIP Modelling Results for MC-2

Inputs \ Scenarios	1	2	3	4	5	6	7	8	9	10
Slurry feed rate (m^3/h)	462	462	462	462	462	462	462	462	462	462
Solids (t/h)	341	341	341	341	341	341	341	341	341	341
Solution (m^3/h)	341	341	341	341	341	341	341	341	341	341
Consider Leach after Carbon addition	N	N	N	N	N	N	N	N	N	N
Gold on stripped carbon, g/t	120	120	120	120	120	120	120	**80**	**50**	**120**
Adsorption tank(s) size, m^3	100	100	100	100	**50**	**100**	100	100	100	100
Carbon frequency advance (% in 24 hours)	100%	**90%**	**80%**	**70%**	100%	100%	100%	100%	100%	100%
Leaching										
Au leached before Carbon addition	89.2%	89.2%	89.2%	89.2%	89.2%	89.2%	89.2%	89.2%	89.2%	89.2%
Leach time before Carbon addition (h)	34	34	34	34	34	34	34	34	34	34
Leach only total tankage (m^3)	15720	15720	15720	15720	15720	15720	15720	15720	15720	15720
Number of Leaching tanks	5	5	5	5	5	5	5	5	5	5
Volume of Leaching tanks (m^3)	3144	3144	3144	3144	3144	3144	3144	3144	3144	3144
CIP/CIL										
Leach Kinetic Constant (ks)	3.508	3.508	3.508	3.508	3.508	3.508	3.508	3.508	3.508	3.508
Model output kinetic constant (k)	0.010	0.010	0.010	0.010	0.010	0.010	0.010	0.010	0.010	**0.008**
Model output equilibrium constant (K)	29361	29361	29361	29361	29361	29361	29361	29361	29361	**23489**
Product of equilibrium and kinetic constants (kK)	303	303	303	303	303	303	303	303	303	**194**
Number of stages	6	6	6	6	6	6	6	6	6	6
Total CIP/CIL volume (m^3)	600	600	600	600	300	600	600	600	600	600
Slurry residence time in each adsorption tank (h)	0.2	0.2	0.2	0.2	**0.1**	0.2	0.2	0.2	0.2	0.2
Gold grade in residue (g/t)	0.182	0.182	0.182	0.182	0.182	0.182	0.182	0.182	0.182	0.182
Gold in final barren solution (mg/L)	0.004	0.004	0.004	0.004	0.009	0.005	0.006	0.003	0.002	0.006
Gold in loaded carbon (g/t)	2565	2837	3177	3613	4997	3176	4191	2528	2499	2562
Carbon residence time/stage (h)	24	27	30	34	24	24	24	24	24	24
Carbon Concentration (g/L pulp)	50	50	50	50	50	40	30	50	50	50
Equivalent transferred carbon unit flowrate (kg/h)	208	188	167	146	104	167	125	208	208	208
Daily carbon transfer / batch elution capacity (kg/day)	5000	**4500**	**4000**	**3500**	**2500**	**4000**	**3000**	**5000**	5000	5000
Carbon Inventory per stage (kg)	5000	5000	5000	5000	2500	4000	3000	5000	5000	5000
Carbon inventory all stages (tons)	30	30	30	30	15	**24**	**18**	**30**	30	30
Gold Lock-Up on Carbon (kg)	19.5	21.3	23.7	26.8	21.4	19.7	20.6	18.3	17.4	21.3
CIP/CIL Gold recovery per day (g/day)	12227	12227	12227	12226	12192	12224	12212	12238	12247	12210
Overall Gold Leaching Efficiency	89.2%	89.2%	89.2%	89.2%	89.2%	89.2%	89.2%	89.2%	89.2%	89.2%
Overall Gold Adsorption Efficiency	99.7%	99.7%	99.7%	99.7%	99.4%	99.7%	99.6%	99.8%	99.9%	99.6%
Overall Gold Recovery	88.9%	88.9%	88.9%	88.9%	88.7%	88.9%	88.8%	89.0%	89.1%	88.8%
Au in loaded carbon / Au in feed	1527	1689	1891	2151	2974	1890	2494	1505	1488	1525
Upgrading ratio	1712	1894	2120	2411	3335	2120	2797	1687	1668	1710
Circuit filling time - slurry (days)	1.5	1.5	1.5	1.5	1.4	1.5	1.5	1.5	1.5	1.5
Ramp-up time (days) *	1.6	1.7	1.9	2.2	1.8	1.6	1.7	1.5	1.4	1.7

**Ramp-up time (days) = Gold lock-up (kg)/ Gold Produced (kg/day)*
***Bold Values** indicate key values that changed between scenarios*

Cyanide Detoxification Testwork

The current plant design does not include a cyanide detoxification circuit, however, cyanide detoxification tests were conducted to provide results for designing this circuit in case this option is considered in the future.

Cyanide detoxification testwork was conducted for MC-2 using the SO_2/Air method, and the results are presented in Table 13.41.

Table 13.41 SGS 2017/2018 Cyanide Detoxification Test Results for MC-2

Feed / Test	Time, minutes		Temp °C	Product Analysis (Solution Phase), mg/L						Reagent Addition								
						CN_{WAD}				g/g CN_{WAD}			g/L Feed Pulp			kg/t Solids		
	Dur-ation	Ret.		pH	CN_T	(3) Ana.	Picric Acid	Cu	Fe	(1) SO_2 Equiv	Lime	(2) Cu	(1) SO_2 Equiv	Lime	(2) Cu	(1) SO_2 Equiv	Lime	(2) Cu
Feed (CN101)		--		*10.7*	*112*	*108*	--	*26.1*	*5.14*	--	--	--	--	--	--	--	--	--
Batch Test																		
CND 1	180	180	35.0	8.5	--	--	0.20	--	--	5.38	5.03	0.09	0.44	0.42	0.007	0.58	0.54	0.01
Continuous Tests																		
CND 1-1	150	61	35.1	8.5	--	--	**11.3**	--	--	**4.71**	0.36	0.09	0.37	0.03	0.007	0.51	0.04	0.01
CND 1-2	180	60	35.0	8.5	2.60	<0.1	0.98	2.90	0.80	**5.93**	1.40	0.09	**0.47**	0.11	0.007	0.64	0.15	0.01
CND 1-3	180	62	35.1	8.6	4.40	<0.1	0.68	4.70	1.50	**5.96**	1.80	**0.00**	0.47	0.15	0.000	0.64	0.19	0.01

(1) SO_2 added using sodium metabisulphite ($Na_2S_2O_5$)
(2) Cu added using $CuSO_4 \bullet 5H_2O$
(3) Samples were submitted for distillation method analyses when picric acid method indicates <1 m/L CN_{WAD}.

The 4.7 g SO_2 / g CN_{WAD}, added in CND 1-1 is the standard starting dosage after initial batch treatment to the approximate detoxification target. Running at that concentration resulted in a gradual increase in CN_{WAD} content from 0.02 mg/L (CND 1 Batch Test) to >11 mg/L (CND 1-1).

A moderate increase in SO_2 dosage to ~5.9 g/g CN_{WAD} was required to achieve a CN_{WAD} level of <1 mg/L (by picric acid method) in test CND 1-2.

Eliminating Cu (often required to catalyze the reaction) and running at a similar SO_2 dosage yielded a similar CN_{WAD} level of <1 mg/L (by picric acid method) in test CND 1-3.

Variability Testwork (2018)

A total 76 samples were submitted for whole ore cyanidation testwork. Table 13.42 shows the type of lithology that the samples were associated with, along with the sample count, and ranges of gold head grades.

Table 13.42 SGS 2017/2018 Variability Samples Head Grade Range by Lithology Type

Lithology Association	Count	Head Grade, Au g/t		
		Min	Max	Average
Pelite	23	0.21	20.64	2.33
Sandstone	23	0.17	10.90	1.99
Cipolin	11	0.30	4.81	1.73
Agglomerates	10	0.16	1.81	0.44
Diorite	6	0.09	0.32	0.20
Albitite	1	0.26	0.26	0.26
Basalt	1	0.40	0.40	0.40
Unknown	1	0.08	0.08	0.08
Overall	**76**	**0.08**	**20.64**	**1.64**

The variability composites selected in the FS phase also included lower grade samples in order to study the metallurgical behaviour of lower grade ore. A summary of the results is shown in Table 13.43.

Table 13.43 SGS 2017/2018 Summary of Variability Testwork

Composite	Test No.	Feed Size, µm	DO_2, mg/L (Avg)	Targeted Pulp Density % Solids	Reagents (kg/t of CN Feed)				% Au Extraction			Leach Residue g/t Au	Head Grade			Ore Weathering Type
					Added		Consumed		24 h	36 h			Au		Te	
					NaCN	CaO	NaCN	CaO	Calc'd Head	Calc'd Head	Direct Head		CN (calc)	Direct	Direct	
Boto2_007	CN034b	73	12.8	40	0.80	2.43	0.06	2.29	80	92.3	91.0	<0.02	0.26	0.22	<4	Saprolite
Boto2_010	CN080	71	12.2	50	0.53	1.30	0.14	1.25	85	88.0	90.6	<0.02	0.17	0.21	<4	Fresh Rock
Boto2_019	CN035	77	13.8	50	0.50	1.42	0.08	1.38	77	78.6	77.5	0.08	0.37	0.36	<4	Fresh Rock
Boto2_023	CN096	79	14.4	50	0.50	1.42	0.10	1.36	84	81.8	85.7	0.05	0.25	0.31	<4	Fresh Rock
Boto2_024	CN081	75	12.6	50	0.58	1.36	0.15	1.31	73	77.1	72.6	0.05	0.22	0.18	<4	Fresh Rock
Boto2_026	CN097	102	14.4	40	0.75	2.34	0.05	2.30	91	85.6	83.8	0.04	0.24	0.22	<4	Mix of saprock & fresh rock
Boto2_026	CN097R	71	14.8	40	0.86	3.26	0.31	3.19	87	93.0	91.3	0.02	0.29	0.23	<4	Mix of saprock & fresh rock
Boto2_027	CN036	76	13.1	50	0.51	0.99	0.05	0.97	88	88.6	85.5	0.03	0.22	0.17	<4	Fresh Rock
Boto2_028	CN047	73	14.5	50	0.55	1.31	0.12	1.27	86	86.9	87.9	0.02	0.15	0.17	<4	Fresh Rock
Boto2_035	CN037	76	13.8	50	0.53	1.32	0.06	1.27	83	77.9	78.0	0.04	0.18	0.18	<4	Fresh Rock
Boto2_037	CN038	76	14.4	50	0.52	1.78	0.08	1.74	91	91.6	92.8	0.29	3.39	3.98	6	Fresh Rock
Boto2_039	CN049	74	13.5	40	0.75	1.62	0.10	1.61	89	91.6	87.5	0.04	0.48	0.32	<4	Saprolite
Boto2_049	CN043	74	15.3	50	0.51	1.19	0.11	1.15	79	81.0	78.1	0.05	0.26	0.23	<4	Fresh Rock
Boto2_051	CN064	72	13.0	50	0.51	1.52	0.05	1.49	89	89.8	85.1	0.04	0.34	0.24	<4	Fresh Rock
Boto2_055	CN082	76	13.3	50	0.56	1.52	0.17	1.46	78	77.7	76.6	0.06	0.25	0.23	<4	Fresh Rock
Boto2_059	CN039	73	13.8	50	0.50	1.65	0.09	1.61	94	95.9	96.1	0.81	19.7	20.6	11	Fresh Rock
Boto2_060	CN040	74	13.4	50	0.57	1.86	0.12	1.83	87	88.5	88.6	0.02	0.17	0.18	<4	Fresh Rock
Boto2_061	CN041	71	14.1	50	0.53	1.62	0.12	1.59	81	79.8	80.2	0.16	0.79	0.81	<4	Fresh Rock
Boto2_062	CN094	74	14.1	50	0.50	1.56	0.09	1.52	92	92.2	92.2	0.30	3.84	3.83	<4	Fresh Rock
Boto2_065	CN046	69	15.0	50	0.50	1.17	0.07	1.13	95	94.8	95.6	0.04	0.67	0.79	<4	Fresh Rock
Boto2_066	CN079	77	14.2	50	0.51	1.18	0.06	1.12	75	92.5	96.4	0.03	0.33	0.70	<4	Fresh Rock
Boto2_071	CN044	74	14.3	50	0.60	1.66	0.19	1.62	81	84.6	85.6	0.06	0.39	0.42	<4	Fresh Rock
Boto2_072	CN045	73	12.8	50	0.51	1.22	0.09	1.20	90	86.9	87.5	0.04	0.30	0.32	<4	Fresh Rock
Boto2_075	CN050	75	13.4	50	0.61	1.35	0.18	1.35	91	92.0	90.0	0.05	0.63	0.50	<4	Fresh Rock
Boto2_077	CN051	76	12.7	50	0.54	1.70	0.13	1.69	94	86.0	90.7	<0.02	0.14	0.21	<4	Fresh Rock
Boto2_080	CN052	74	12.4	50	0.50	1.24	0.11	1.24	84	87.1	86.5	0.04	0.31	0.30	<4	Fresh Rock
Boto2_082	CN053	80	11.4	50	0.51	1.34	0.12	1.33	97	92.3	92.2	<0.02	0.26	0.26	<4	Fresh Rock
Boto2_083	CN054	72	11.2	50	0.71	1.53	0.31	1.53	84	83.2	87.3	0.04	0.24	0.31	<4	Fresh Rock

Composite	Test No.	Feed Size, µm	DO_2, mg/L (Avg)	Targeted Pulp Density % Solids	Reagents (kg/t of CN Feed) Added NaCN	Added CaO	Consumed NaCN	Consumed CaO	% Au Extraction 24 h Calc'd Head	36 h Calc'd Head	36 h Direct Head	Leach Residue g/t Au	Head Grade Au CN (calc)	Au Direct	Te Direct	Ore Weathering Type
Boto2_084	CN042	75	13.3	50	0.52	1.37	0.08	1.34	93	86.3	87.3	<0.02	0.15	0.16	<4	Fresh Rock
Boto2_089	CN055	74	10.4	50	0.71	1.49	0.33	1.49	85	81.5	80.7	0.06	0.30	0.29	<4	Fresh Rock
Boto2_090	CN056	78	11.1	50	0.59	1.41	0.24	1.40	71	77.0	77.8	0.15	0.63	0.65	<4	Fresh Rock
Boto2_100	CN057	73	10.4	50	0.75	1.90	0.16	1.90	95	95.4	96.0	0.31	6.60	7.60	<4	Fresh Rock
Boto2_103	CN058	75	10.7	40	0.76	2.29	0.11	2.29	87	87.4	86.8	0.64	5.05	4.81	<4	Saprolite
Boto2_106	CN108	78	13.5	50	0.50	1.73	0.12	1.66	78	78.9	78.8	0.14	0.66	0.66	<4	Fresh Rock
Boto2_107	CN073	74	13.6	50	0.82	1.45	0.51	1.41	84	86.0	82.1	0.06	0.43	0.34	<4	Fresh Rock
Boto2_109	CN065	80	13.0	50	0.51	1.72	0.14	1.71	90	91.1	92.0	0.07	0.73	0.81	<4	Fresh Rock
Boto2_110	CN074	84	13.2	50	0.50	1.33	0.10	1.31	85	90.2	88.2	0.11	1.12	0.94	<4	Fresh Rock
Boto2_111	CN075	74	13.1	50	0.53	1.26	0.16	1.25	86	85.2	83.4	0.18	1.18	1.06	<4	Fresh Rock
Boto2_112	CN066	68	12.5	40	0.75	4.26	0.06	4.24	95	95.0	95.4	0.15	3.02	3.24	<4	Saprolite and some saprock
Boto2_114	CN067	74	10.8	50	0.50	1.51	0.06	1.50	81	84.5	85.8	0.17	1.10	1.20	<4	Fresh Rock
Boto2_115	CN109	73	13.5	50	0.50	1.34	0.05	1.29	90	89.3	87.2	0.11	0.98	0.82	<4	Fresh Rock
Boto2_117	CN071	72	13.6	50	0.51	1.42	0.11	1.41	72	82.5	79.0	1.86	10.6	8.85	22	Fresh Rock
Boto2_118	CN060	81	10.1	50	0.50	1.30	0.12	1.30	80	82.0	81.2	0.28	1.55	1.49	<4	Fresh Rock
Boto2_119	CN076	74	10.5	50	0.59	1.26	0.20	1.21	76	78.1	80.0	0.04	0.18	0.20	<4	Fresh Rock
Boto2_120	CN061	75	8.6	50	0.76	2.29	0.04	2.28	91	91.7	90.9	0.11	1.27	1.15	<4	Saprock
Boto2_121	CN062	81	8.8	50	0.50	1.27	0.02	1.27	75	75.6	74.2	0.12	0.47	0.45	<4	Fresh Rock
Boto2_122	CN063	74	10.2	50	0.51	0.74	0.12	0.73	79	79.9	78.9	0.02	0.10	0.10	<4	Fresh Rock
Boto2_123	CN077	71	15.5	40	0.87	3.12	0.28	2.99	88	91.8	92.4	0.06	0.67	0.73	<4	Saprolite
Boto2_126	CN078	79	13.9	50	0.51	1.27	0.12	1.24	86	90.3	89.9	0.07	0.72	0.69	<4	Fresh Rock
Boto2_128	CN083	77	12.7	50	0.68	1.79	0.21	1.75	62	64.4	65.7	2.06	5.79	6.00	8	Fresh Rock
Boto2_129	CN084	75	12.5	50	0.72	1.88	0.37	1.88	86	87.1	86.9	1.39	10.7	10.6	<4	Fresh Rock
Boto2_130	CN085	76	12.2	50	0.73	1.73	0.24	1.69	87	87.9	87.7	0.24	1.95	1.92	<4	Fresh Rock
Boto2_132	CN086	75	13.8	50	0.62	1.17	0.14	1.13	93	93.1	90.7	0.07	0.94	0.70	<4	Fresh Rock
Boto2_133	CN110	49	13.7	50	0.50	1.17	0.03	1.04	89	88.8	86.2	0.13	1.16	0.94	<3	Fresh Rock
Boto2_133	CN087	74	14.0	50	0.53	1.28	0.16	1.25	84	83.6	81.7	0.13	0.79	0.71	<4	Fresh Rock
Boto2_135	CN110R	78	14.0	50	0.50	1.54	0.11	1.46	89	89.9	88.9	0.12	1.18	1.09	<3	Fresh Rock
Boto2_134	CN088	76	13.6	50	0.53	1.35	0.16	1.30	82	83.1	79.0	0.18	1.03	0.84	<4	Fresh Rock
Boto2_136	CN068	74	14.5	50	0.50	1.21	0.05	1.19	90	92.7	90.4	0.09	1.23	0.94	<4	Fresh Rock
Boto2_138	CN089	75	14.1	50	0.50	1.29	0.08	1.20	81	81.3	80.0	0.03	0.16	0.15	<4	Fresh Rock
Boto2_139	CN090	82	13.6	50	0.50	1.27	0.08	1.20	90	89.9	90.5	0.07	0.70	0.74	<4	Fresh Rock

Composite	Test No.	Feed Size, µm	DO_2, mg/L (Avg)	Targeted Pulp Density % Solids	Reagents (kg/t of CN Feed)				% Au Extraction			Leach Residue g/t Au	Head Grade			Ore Weathering Type
					Added		Consumed		24 h	36 h			Au		Te	
					NaCN	CaO	NaCN	CaO	Calc'd Head	Calc'd Head	Direct Head		CN (calc)	Direct	Direct	
Boto2_140	CN091	76	14.3	50	0.52	1.62	0.14	1.57	93	93.4	93.8	0.12	1.81	1.92	<4	Fresh Rock
Boto2_141	CN092	72	12.7	50	0.50	1.85	0.07	1.76	90	91.9	90.3	0.07	0.86	0.72	<4	Fresh Rock
Boto2_142	CN093	71	13.6	50	0.50	1.18	0.08	1.15	91	90.7	89.6	0.04	0.43	0.39	<4	Fresh Rock
Boto2_143	CN098	75	13.5	50	0.58	1.67	0.15	1.63	91	93.8	93.0	0.65	10.4	9.27	<4	Fresh Rock
Boto2_144	CN069	72	11.5	50	0.55	1.51	0.11	1.50	69	78.1	73.0	0.19	0.87	0.71	<4	Fresh Rock
Boto2_145	CN070	76	13.1	50	0.54	1.91	0.10	1.81	77	78.8	76.7	0.12	0.57	0.52	<4	Fresh Rock
Boto2_146	CN099	75	13.3	50	0.50	1.55	0.13	1.53	93	93.7	93.2	0.23	3.60	3.33	<4	Fresh Rock
Boto2_149	CN100	75	14.3	50	0.50	1.36	0.08	1.32	82	82.6	78.8	0.14	0.81	0.66	<4	Fresh Rock
Boto2_151	CN102	77	12.8	50	0.50	1.50	0.10	1.43	92	88.0	85.7	0.03	0.25	0.21	<4	Fresh Rock
Boto2_152	CN103	78	14.0	50	0.50	1.72	0.14	1.70	82	83.8	80.6	0.07	0.43	0.36	<4	Fresh Rock
Boto2_153	CN104	77	15.7	40	0.75	1.63	0.04	1.59	97	97.6	97.0	0.02	0.83	0.66	<4	Saprolite
Boto2_154	CN105	80	12.0	50	0.50	1.41	0.06	1.35	61	56.5	58.5	0.17	0.39	0.41	<4	Fresh Rock
Boto2_155	CN106	78	12.9	40	0.75	3.79	0.12	3.75	94	95.0	94.1	0.07	1.39	1.18	<4	Saprolite
Boto2_156	CN111	78	7.7	40	0.75	4.02	0.06	3.76	81	89.3	88.5	0.03	0.28	0.26	<4	Saprolite
Boto2_157	CN112 (Avg)	73	11.6	40	0.76	1.97	0.09	1.84	90	95.5	95.4	0.05	0.99	0.97	<4	Saprolite
Boto2_157	CN112	94	9.6	40	0.75	1.67	0.06	1.58	89	93.0	93.0	0.07	1.00	1.01	<4	Saprolite
Boto2_157	CN112R	52	13.6	40	0.76	2.26	0.12	2.10	90	97.9	97.9	0.02	0.97	0.94	<4	Saprolite
Boto2_159	CN107	74	13.9	50	0.69	3.05	0.54	3.05	95	94.5	93.8	0.22	3.98	3.55	<4	Fresh Rock
Boto5_007	CN059	75	10.2	40	0.75	2.93	0.09	2.93	83	87.8	92.3	0.02	0.16	0.26	<4	Saprock
Boto5_012	CN113	73	8.4	40	0.89	1.78	0.34	1.53	55	75.6	60.0	0.02	0.08	0.05	12	Saprolite
Average Value =		75	12.8		0.59	1.70	0.13	1.65	85.0	86.7	85.9	0.17	1.60	1.56		
Minimum Value =		49	7.7		0.50	0.74	0.02	0.73	55.5	56.5	58.5	0.02	0.08	0.05	<4	
Maximum Value =		102	15.7		0.89	4.26	0.54	4.24	97.2	97.9	97.9	2.06	19.7	20.6	22.0	
Standard Dev. =		6	1.7		0.12	0.65	0.10	0.64	8.2	7.2	7.9	0.35	3.03	3.03	6.18	

Note: Composite ID denoted as "Boto 2_XXX" represents Malikoundi composites.

Table 13.44 was generated for the purposes of understanding the variation in leach performance based on the type of lithology that the 76 samples are associated with.

Table 13.44 Summary of Variability Testwork Gold Extraction by Lithology Type

Lithology Association	Count	(1) % Au Extraction		
		Min	Max	Average
Pelite	23	58.0	96.4	88.8
Sandstone	23	64.0	97.8	83.1
Cipolin	11	62.6	94.2	88.9
Agglomerates	10	72.6	90.7	83.9
Diorite	6	77.3	91.0	85.4
Albitite	1	92.3	92.3	92.3
Basalt	1	90.1	90.1	90.1
Unknown	1	76.3	76.3	76.3
Overall	**76**	**58.0**	**97.8**	**86.1**

(1) Based on direct head grade and leach residue.

13.2.2 2018 Solids-Liquid Settling Testwork

Solids-liquid testwork were conducted at both Outotec and Pocock testing facilities. The purpose of this testwork is to investigate the most effective type of flocculant, flocculant dosing rates, and the solids flux or loading rates for thickener sizing.

Outotec Testwork

The samples submitted to Outotec were SAP and MC-2 with targeted grind P_{80} of 75µm. The flocculant screening results for SAP and MC-2 are presented in Figure 13.49 and Figure 13.50, respectively.

Figure 13.49 Outotec 2018 Flocculant Screening Results for SAP Composite

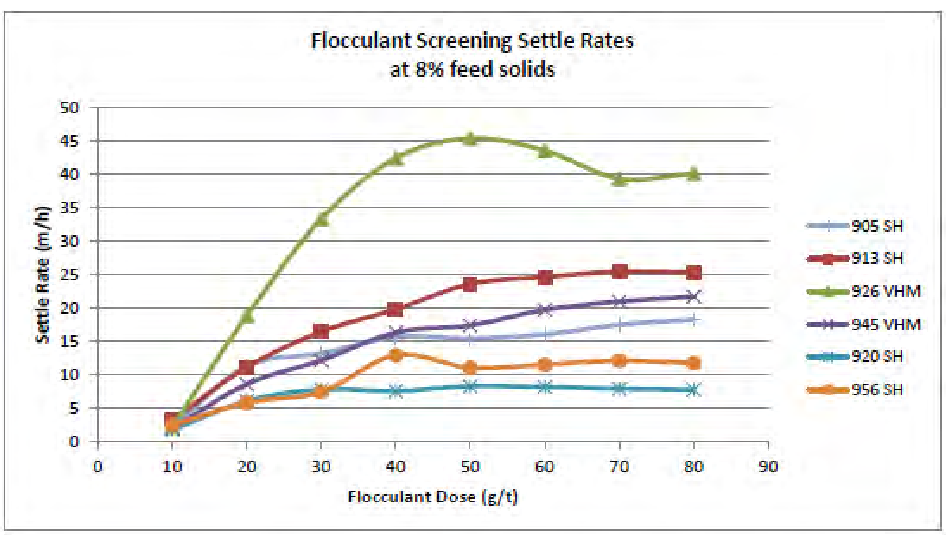
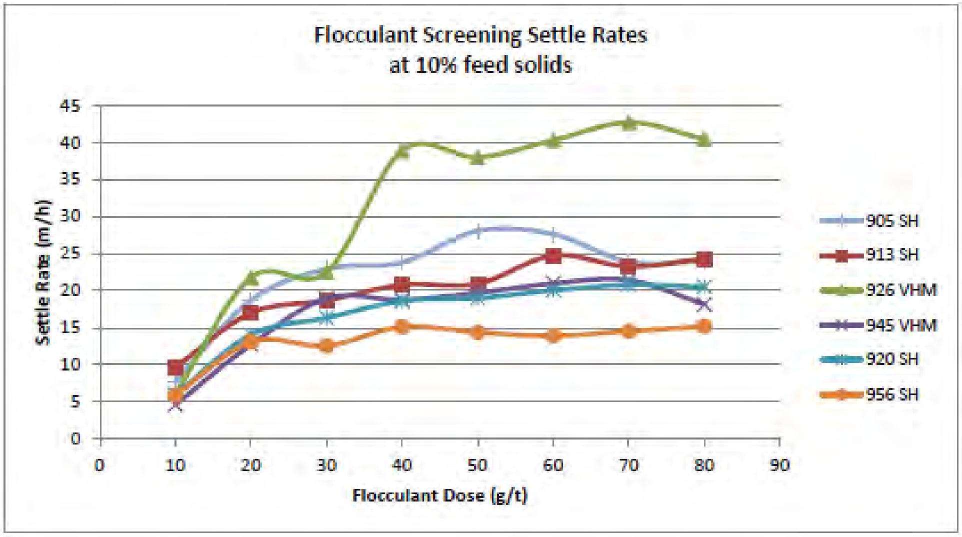


Figure 13.50 Outotec 2018 Flocculant Screening Results for MC-2 Composite



The flocculant with the best overall performance for both composites was SNF 926 VHM as it provided the clearest overflow at the lowest dosages.

The dynamic thickening test results are presented in Table 13.45 and Table 13.46.

Table 13.45 Outotec 2018 Dynamic Thickening Test Results for SAP Composite

Run No.	Feed Flux (tph/m²)	Feed Liquor RR (m/h)	Flocculant Type	Flocculant Dose (g/t)	Underflow Meas. Solids (% (w/w))	Underflow YS (Pa)	Overflow Solids (mg/L)
1	0.60	4.42	926 VHM	15	44.9	22	<200
2	0.50	3.68		15	46.0	31	<200
3	0.30	2.21		15	47.8	40	<200
4	0.20	1.47		15	48.9	43	<200
5	0.10	0.74		15	50.3	49	<200
6	0.20	1.47		10	48.3	38	<200

Table 13.46 Outotec 2018 Dynamic Thickening Testwork Results for MC-2 Composite

Run No.	Feed Flux (tph/m²)	Feed Liquor RR (m/h)	Flocculant Type	Flocculant Dose (g/t)	Underflow Meas. Solids (% (w/w))	Underflow YS (Pa)	Overflow Solids (mg/L)
1	0.80	3.20	926 VHM	15	62.3	41	<200
2	0.60	2.40		15	63.5	43	<200
3	0.40	1.61		15	64.8	41	<200
4	1.00	4.03		15	62.5	39	<200
5	1.20	4.83		15	61.9	38	<200
6	1.40	5.64		15	61.2	35	<200
7	1.00	4.03		10	62.0	31	<200

Based on the dynamic thickening results, Outotec recommended sizes of thickeners for both pre-leach and post-leach in the case that a tailings thickening option is required in the future. The thickener sizes in Table 13.47 and Table 13.48 are based on MC-2 composite with only limited feed from the SAP composite.

Table 13.47 Outotec 2018 Recommended Pre-leach Thickener Size and Flocculant Dosage

Pre-Leach 17m thickener					
Feed: MC2 Blend					
Feed t/h dry solids	Flux t/m2/h	Underflow % solids	Overflow ppm	Flocculant Dose g/t	Yield Stress Pa
310	1.40	61 - 63	<200	15	35
Feed: 100% Sap					
Feed t/h dry solids	Flux t/m2/h	Underflow % solids	Overflow ppm	Flocculant Dose g/t	Yield Stress Pa
68	0.30	48 - 50	<200	15	40
136	0.60	45 - 47	<200	15	22

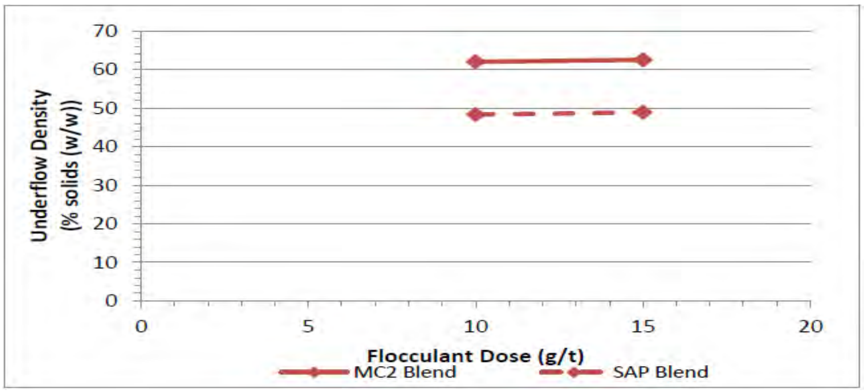
Table 13.48 Outotec 2018 Recommended Pre-leach Thickener Size and Flocculant Dosage

Post-Leach 26m thickener					
Feed: MC2 Blend					
Feed t/h dry solids	Flux t/m2/h	Underflow % solids	Overflow ppm	Flocculant Dose g/t	Yield Stress Pa
310	0.60	63 - 65	<200	15	43
Feed: 100% Sap					
Feed t/h dry solids	Flux t/m2/h	Underflow % solids	Overflow ppm	Flocculant Dose g/t	Yield Stress Pa
68	0.13	49 - 51	<200	15	49
136	0.27	48 - 50	<200	15	40

During the FS phase, the plant through increased from 2.5 Mtpa to 2.75 Mtpa, hence the mill feed rate became 341 t/h of dry solids instead of the 310 t/h indicated in the Outotec report. Based on this throughput increase, the recommended thickener diameter for the pre-leach thickener and tailings thickener would be 18 m and 27 m, respectively. The pre-leach thickener diameter is currently designed at 23 m to prevent a possible bottleneck from occurring at this unit operation. The tailings thickener is currently not included in the design.

Note that a lower flocculant dosage of 10 g/t Au can also be used as the tests conducted at this lower dosage showed similar results to ones conducted at 15 g/t Au. Refer to Figure 13.51 for details.

Figure 13.51 Outotec 2018 – Effect of Flocculant Dosage on Underflow Density



Pocock Testwork

The samples submitted to Pocock were MC-1 (slightly modified) and MC-2 with targeted grind P_{80} of 53 µm. MC-1 was slightly modified by adding a small portion of new saprolite material to make up sufficient mass for the testwork requirement. The flocculant screening results are presented in Table 13.49. Based on Pocock's observation, many types of flocculants responded well, however, SNF AN 905 SH was shown to produce a slightly more robust floccule structure than the other types tested.

Table 13.49 Pocock 2018 Flocculant Screening Results for MC-1 and MC-2

Material	Tested pH	Temp (°C)	Initial Solids Concentration of Slurry Tested	(1)Flocculant Concentration (g/l)	Flocculant Selected
MC-1	10.7	20	15%	0.1	(2)SNF AN 905 SH
MC-2	10.7	20	15%	0.1	(2)SNF AN 905 SH

(1) Flocculant solution concentration prior to contact with the pulp.
(2) SNF AN 905 SH is a medium to high molecular weight, 5% charge density, anionic polyacrylamide. Equivalent products may also serve.

Pocock assessed the operating parameter ranges for both conventional and high rate thickeners on the composites. The recommendation for either of the options are presented in Table 13.50 and Table 13.51.

Table 13.50 Pocock 2018 Recommended Conventional Thickener Operating Parameter Ranges

Material Tested	Flocculant			Minimum Unit Area at Specified Feed Solids Concentration and Underflow Density(2) (m²/MTPD)					Maximum Underflow Solids Conc.(%)
	Type	Dose (g/MT)	Conc.(1) (g/l)	5% Feed Solids(3)	10% Feed Solids(3)	15% Feed Solids(3)	20% Feed Solids(3)	25% Feed Solids(3)	
MC 1	SNF 905	30	0.1	0.558	0.791	1.068(4)	---	---	48%
MC 2	SNF 905	20	0.1	---	---	0.224	0.286	0.337	62%

(1) Flocculant concentration used for testing. Actual flocculant concentration should be maintained between 0.1 to 0.2 g/l prior to contact with the pulp.
(2) Unit Area includes a 1.25 scale-up factor. The range of unit areas provided corresponds to the range of feed solids concentration & u/f densities shown.
(3) Thickener feed solids concentration range by weight.
(4) Specified condition is not recommended due to the poor settling performance of the material at the given feed solids concentration.

Table 13.51 Pocock 2018 Recommended High Rate Thickener Operating Parameter Ranges

Material Tested	Tested Feed Solids(1) (%)	Flocculant			Design Basis Net Feed Loading (m^3/m^2hr)(5)	Predicted Overflow TSS Conc. Range (mg/l)(6)	Predicted Underflow Density(7)
		Type(2)	Dose(3) (g/MT)	Conc. (4) (g/l)			
MC 1	5.8	SNF AN 905 SH	35 - 40	0.1 – 0.2	2.22	150 – 250	48%
MC 2	20.2	SNF AN 905 SH	25 - 30	0.1 – 0.2	4.16	150 - 250	62%

(1) Feed solids concentration range required for thickener operation (wt. %) at maximum design Net Feed Loading Rate. Note: Maintaining feed solids concentration in the ranges shown is critical to thickener performance and operation at design rates shown.
(2) Flocculants from other manufacturers with similar specifications would also serve.
(3) Recommended flocculant dose in grams per metric ton (g/MT).
(4) Recommended flocculant concentration prior to contact with the pulp.
(5) Recommended net feed loading rate in cubic meters of feed slurry per hour per square meter of thickener area. This can be used to calculate the required thickener area based on the volumetric feed rate at the design solids concentration. Since hydraulic design bases are specified independent of solids tonnage, an operable feed solids concentration range is required to properly specify a thickener designed using hydraulic feed loading rate. Recommended design net feed loading rates are provided without scale-up or safety factors.
(6) Overflow suspended solids conc. in milligrams per liter as measured using a 0.45m septum.
(7) Maximum underflow solids concentration recommended based on viscosity considerations and experience.

13.3 Results Interpretation

The sections to follow describes the main testwork results that contributed towards the development of the process flow diagrams and process design criteria.

13.3.1 Comminution Parameters

OMC compiled and analysed all comminution testwork data based on the lithology weighted average values per weathering type. The weighted average bond grindability and JK parameters are presented in Table 13.52.

Table 13.52 Bond Grindability and JK Parameters for Comminution Circuit Design

Material Type	Statistic	BWi (kWh/t)		CWi (kWh/t)		Ai (g)		Axb	
		50th %	85th %	50th %	85th %	50th %	85th %	50th %	85th %
Boto 2/Malikoundi									
Saprolite	Wt. Ave	10.5	10.8	n/a	n/a	0.033	0.033	*100.0*	*100.0*
Saprock	Wt. Ave	11.2	11.2	n/a	n/a	0.042	0.043	*78.7*	*78.7*
Fresh Rock	Wt. Ave	19.9	20.6	13.2	16.4	0.474	0.542	36.0	31.3

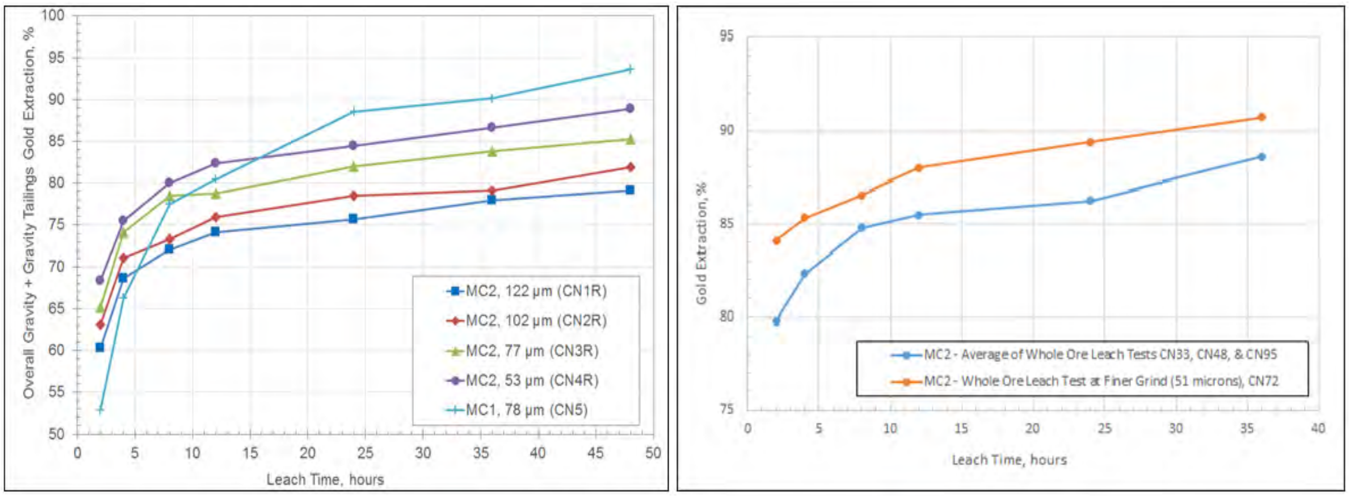
Note: Italicized values are assumed by OMC.

The comminution circuit design is based on the Boto 2/Malikoundi primary ore (fresh rock) parameters where BWi, CWi and Axb design values must meet the 85th percentile requirement, and the Ai design value must meet the 50th percentile requirement.

13.3.2 Grind Size Selection

Cyanidation tests were conducted on both gravity tailings and whole ore samples at the finer grind size (targeted P_{80} 53µm) as seen in Figure 13.52.

Figure 13.52 Review of Cyanidation Tests at Varying Grind P_{80}



The results for the finer grind, ~P_{80} 53µm, yielded 2% to 3% higher gold extraction than the results for grind P_{80} 75 µm. A grind size trade-off study was conducted by Lycopodium (Memorandum No. 5084-MEM-006) to assess the benefit of incorporating a regrind milling circuit into the plant design. The study showed a negative NPV and no payback on the IRR for the addition of a regrind circuit.

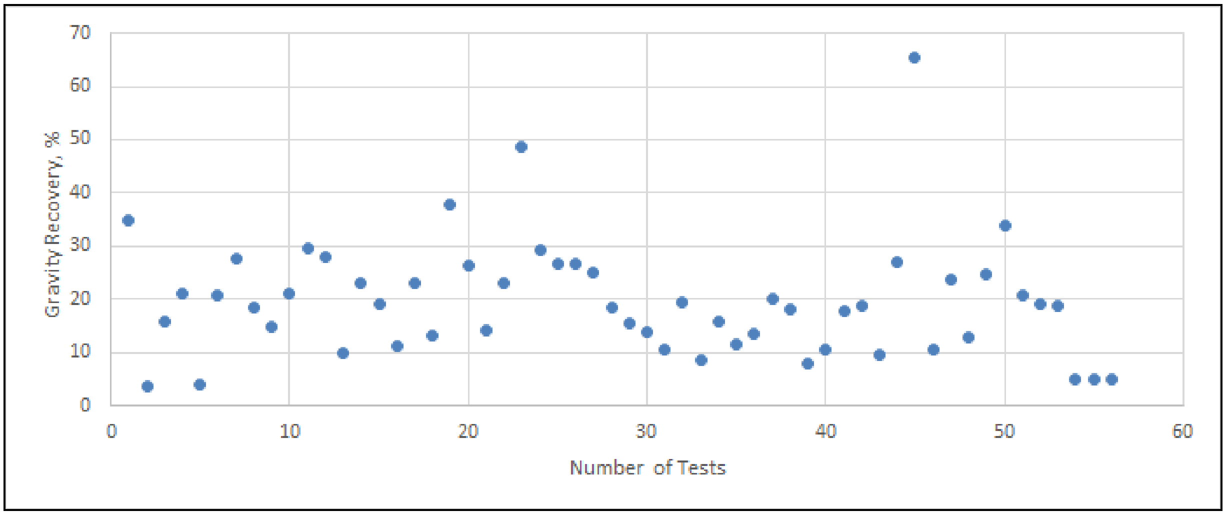
The financial analysis was based on a set of inputs which included a base case gold price of $1,200/oz. A sensitivity analysis showed that the NPV and IRR becomes positive with a payback at the end of a 10-year mine life if gold price becomes $1,250/oz.

It was concluded that a regrind milling circuit is not justifiable at the base case gold price and is not recommended for inclusion in the design at this stage. However, the plant layout should include provision for future installation.

13.3.3 Gravity Circuit

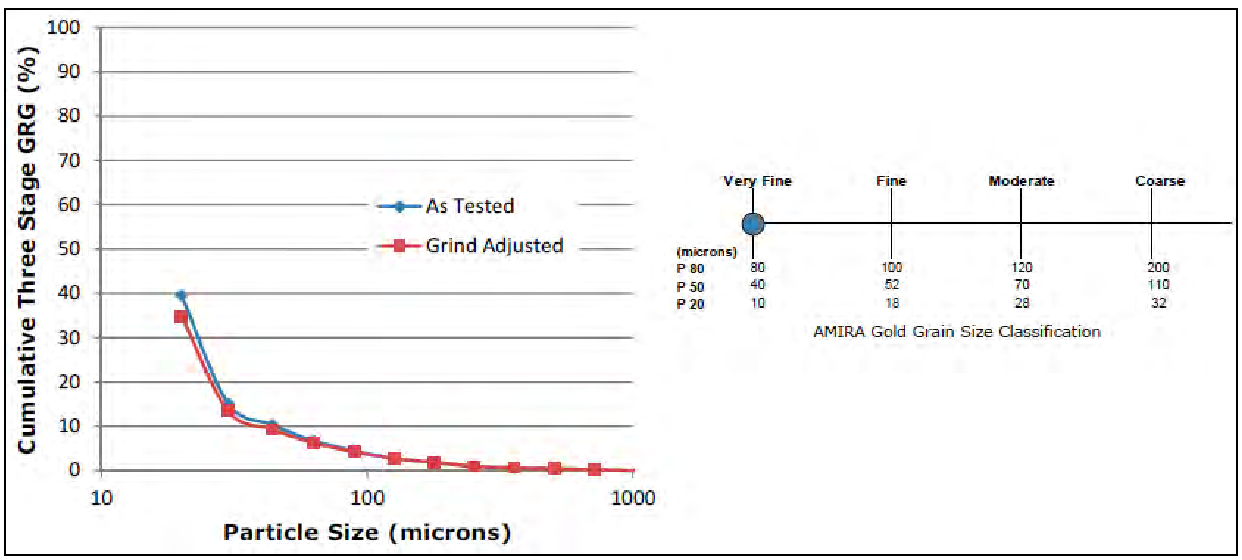
A review of all the previous gravity separation results showed that out of 56 tests (from 2013 to 2018), only three yielded a GRG close to 35%, one close to 50%, and one Boto 5 sample over 65%. The overall GRG average is at approximately 20%. This is considered to be on the low end of the GRG range and inclusion of a gravity circuit cannot be justified. Refer to **Error! Not a valid bookmark self-reference.** for details.

Figure 13.53 Gravity Separation Results from 2013 to 2018



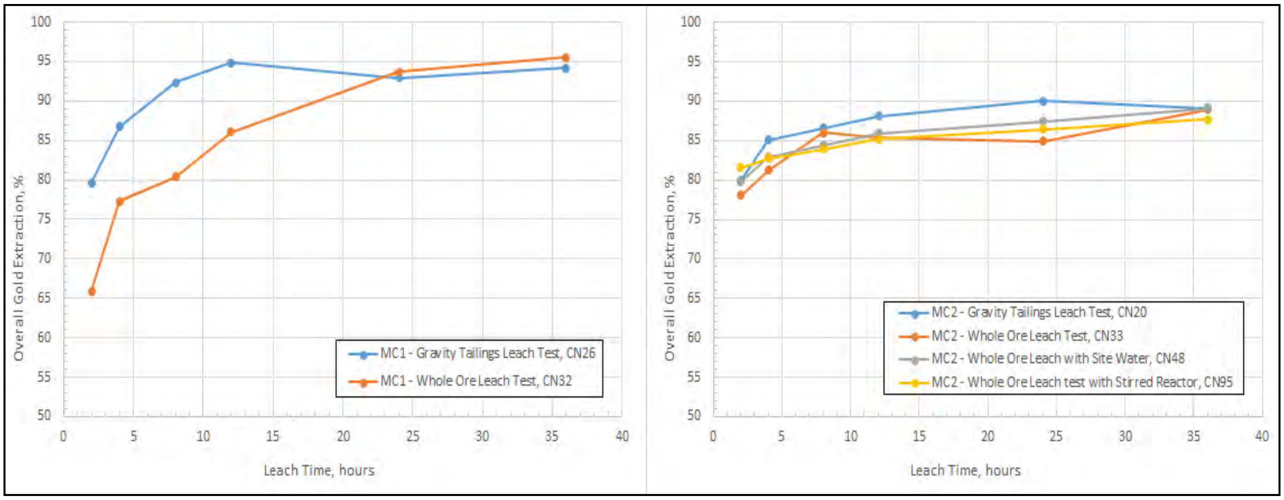
The E-GRG test results combined with the results from whole ore leaching during the SGS 2017/2018 program also did not provide justification for including a gravity circuit in the plant design.

Figure 13.54 Review of SGS 2017/2018 E-GRG Test Results on MC-2



The E-GRG test results as shown in Figure 13.54 indicated that the majority of the GRG amount in MC-2 is classified as very fine. Although the GRG number is considered to be moderate at 39.6, recovery with gravity at the full scale would be difficult due to this fine nature and therefore a gravity circuit is not recommended.

Figure 13.55 Review of Whole Ore versus Gravity Tailings Leach Tests



The whole ore leaching results for MC-1 and MC-2, as presented in Figure 13.55, produced similar results to the leach tests on gravity tailings. This further validates the recommendation to omit a gravity circuit from design.

13.3.4 Pre-treatment and Leach Conditions

The latest metallurgical program conducted extensive leach optimization testwork (see Table 13.33 in Section 13.2.1) to determine the ideal pre-treatment and leach conditions for the Project. The results showed that the use of oxygen in pre-treatment and during leaching, along with lead nitrate addition will yield at least 1.8% higher gold extraction for MC-2 than if these additions were not considered. Furthermore, the leach kinetics will be more rapid, and the cyanide consumption will also be reduced significantly (see Table 13.53). For MC-1 (Boto 5 Master Composite), the lead nitrate and oxygen addition provided some leach kinetic benefits, but did not improve gold extraction.

Table 13.53 Comparison of Leach Tests with and without Oxygen and Lead Nitrate Addition

Composite Name	Tests in Comparison	Δ in Gold Extraction %	Δ in Leach Kinetics	Δ in Cyanide Consumption
MC2	CN11 (no Pb & no O_2) Vs. CN13 (with Pb & O_2)	CN11 - 87.7% Au CN13 - 89.5% Au Increase by 1.8% Au	CN11 - Leach completion @36hr CN13 - Leach completion @24hr Decrease leach time by 8hr	CN11 - 0.51 kg/t CN13 - 0.30 kg/t 41% Reduction in Consumption
MC2	CN15 (no Pb & no O_2) Vs. CN13 (with Pb & O_2)	CN15 - 86.3% Au CN13 - 89.5% Au Increase by 3.2% Au	CN15 - Leach completion @36hr CN13 - Leach completion @24hr Decrease leach time by 8hr	CN15 - 0.91 kg/t CN13 - 0.30 kg/t 67% Reduction in Consumption
MC2	CN16 (no Pb & no O_2) Vs. CN13 (with Pb & O_2)	CN16 - 85.1% Au CN13 - 89.5% Au Increase by 4.4% Au	CN16 - Leach completion @24hr CN13 - Leach completion @24hr Insignificant Impact	CN16 - 0.97 kg/t CN13 - 0.30 kg/t 69% Reduction in Consumption
MC2	CN15 (no Pb & no O_2) Vs. CN20 (with Pb & O_2)	CN15 - 86.3% Au CN20 - 88.7% Au Increase by 2.4% Au	CN15 - Leach completion @36hr CN20 - Leach completion @24hr Decrease leach time by 8hr	CN15 - 0.91 kg/t CN20 - 0.15 kg/t 76% Reduction in Consumption

A trade-off study was conducted by Lycopodium (Memorandum No. 5084-MEM-005) to justify the inclusion of the oxygen plant and the lead nitrate system. The outcomes from this study suggested that inclusion of these systems could yield a positive NPV of $15.3M and an IRR of 157%, providing a payback period within the first year.

Other leach conditions recommended based on the results of the leach optimization tests include:

- Pulp pH at 10.5 to 11 to be maintained with the addition of lime.
- Pre-treatment with oxygen, and leaching with oxygen spargers to maintain a dissolved oxygen level of ~15 mg/L or higher.
- Lead nitrate addition of 200 g/t ore.
- Pre-treatment time of 4-hours minimum, and leach time of approximately 36-hours.
- Cyanide concentration of 0.5 g NaCN/L to be maintained in first leach tank.

13.3.5 Gold Recovery Models

Gold Extraction Models

A relationship between the leach residue and head grades was developed from the SGS 2017/2018 variability testwork results. Figure 13.56 and Figure 13.57 illustrate this relationship for the Boto 2/Malikoundi fresh rock and saprolite/saprock, respectively. The current cut-off-grade for Boto 2/Malikoundi fresh rock is 0.63 g/t Au, therefore the leach results for samples with head grades below 0.5 g/t Au were not used as those materials will not likely get mined. Similarly, the current cut-off-grades for the Boto 2/Malikoundi saprolite and saprock are 0.46 g/t Au and 0.5 g/t Au, respectively, therefore leach results for samples with head grades below 0.25 g.t Au were not used as those materials will also not likely get mined.

Figure 13.56 Gold Extraction Model for Boto 2/Malikoundi Fresh Rock

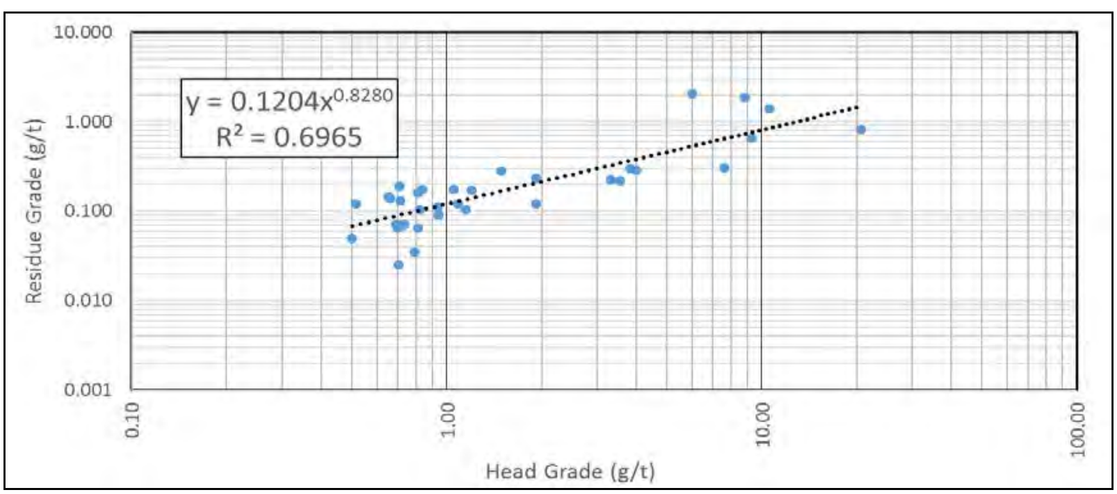
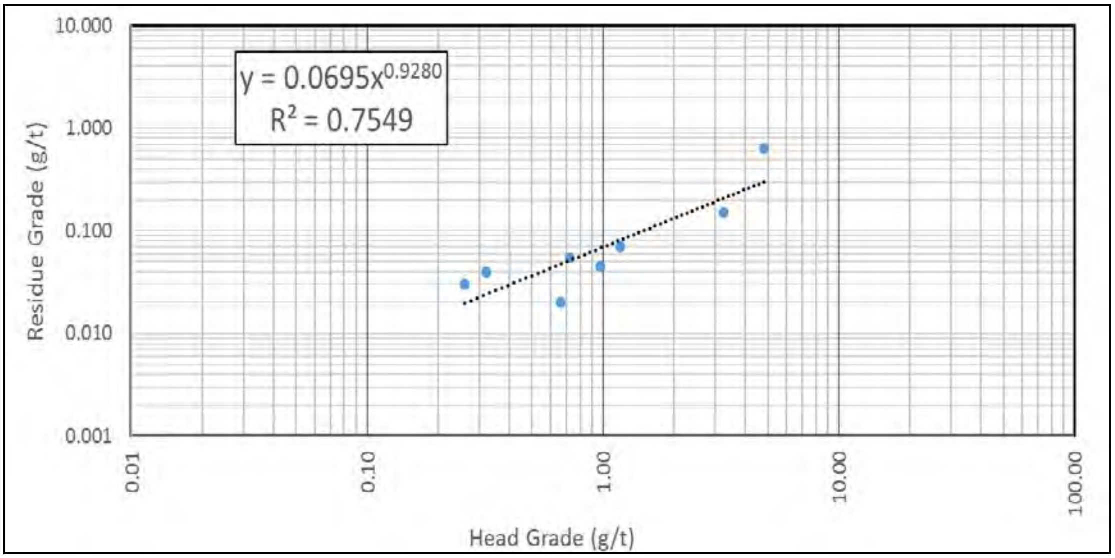


Figure 13.57 Gold Extraction Model for Boto 2/Malikoundi Saprolite/Saprock

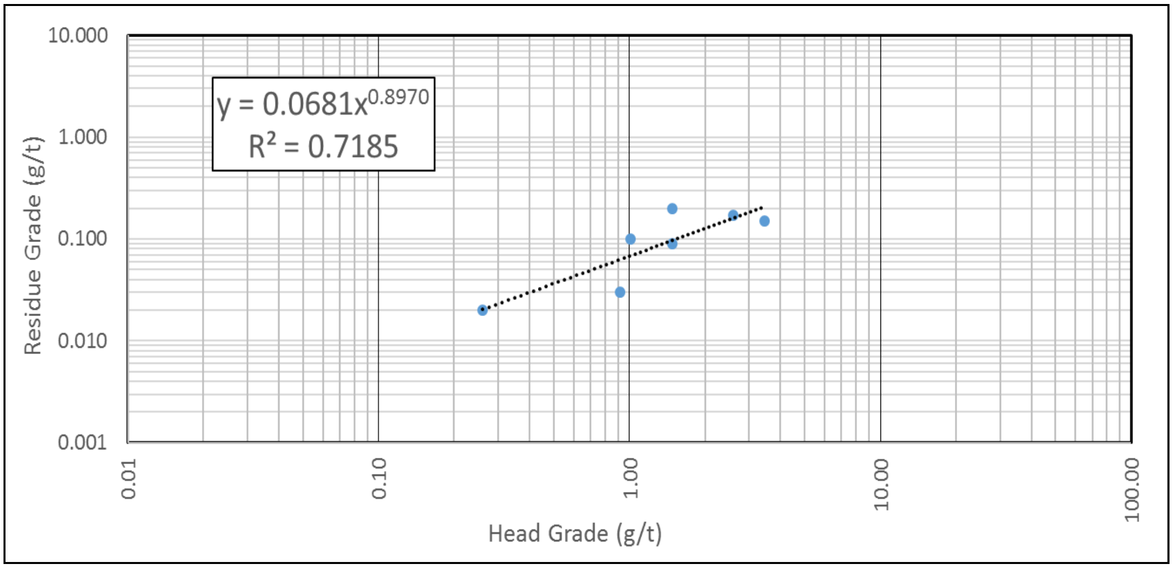


Since MC-1 (Boto 5 Master Composite) showed little gold extraction benefit from lead nitrate and oxygen addition, a combination of old and new test results as seen in Table 13.54 were used to plot the Boto 5 saprolite/saprock gold extraction model in Figure 13.58. Note that the current cut-off grades for Boto 5 saprolite and saprock are 0.48 g/t Au and 0.49 g/t Au, respectively, therefore leach results for samples with head grades below 0.25 g/t Au were not used as those materials will not likely get mined.

Table 13.54 Boto 5 Saprolite/Saprock Leach Data for Extraction Model

Year	Test No.	Head Grade (g/t Au)	Leach Residue (g/t Au)	Extraction %
2013	CN-16/17/18 (avg)	1.01	0.10	90.1
2016	CN-82	2.60	0.17	93.5
2016	CN-81	1.48	0.20	86.5
2016	CN-58	0.92	0.03	96.7
2016	CN-83	1.48	0.09	93.9
2017/2018	Boto5_007	0.26	0.02	92.3
2017/2018	CN-33 (MC1)	3.45	0.15	95.7

Figure 13.58 Gold Extraction Model for Boto 5 Saprolite/Saprock



The lowest leach residue grade assayed in the variability testwork was 0.02 g/t Au, hence, Lycopodium considered this to be the lowest leach residue grade that can realistically be achieved during cyanidation. The extraction equations shown in Figure 13.56, Figure 13.57 and Figure 13.58 should then be bounded by a lower limit of head grade so that each equation does not give a residue grade less than 0.02 g/t Au.

The following calculations were used to determine the lower limit ("x" is head grade and "y" is residue):

Boto 2/Malikoundi Fresh Rock → $y = 0.1204\ x^{0.828}$ → $0.02 = 0.1204\ x^{0.828}$ → $x = 0.12$ g/t Au.

Boto 2/Malikoundi Saprolite/Saprock → $y = 0.0695\ x^{0.928}$ → $0.02 = 0.0695\ x^{0.928}$ → $x = 0.27$ g/t Au.

Boto 5 Saprolite/Saprock → $\mathbf{y} = 0.0681\ \mathbf{x}^{0.897}$ → $0.02 = 0.0681\ \mathbf{x}^{0.897}$ → $\mathbf{x} = 0.26$ g/t Au.

The lower limit for head grade is calculated to be 0.12 g/t Au and 0.27 g/t Au for Boto 2/Malikoundi fresh rock and saprolite/saprock, respectively. The lower limit for Boto 5 saprolite/saprock is calculated to be 0.26 g/t Au.

Gold Losses Models

In order to account for gold losses, models for estimating the gold losses by solution were generated based on CIP inputs provided by KEMIX. Refer to Figure 13.59, Figure 13.60, and Figure 13.61 for details.

Figure 13.59 Gold Losses Model for Boto 2/Malikoundi Fresh Rock

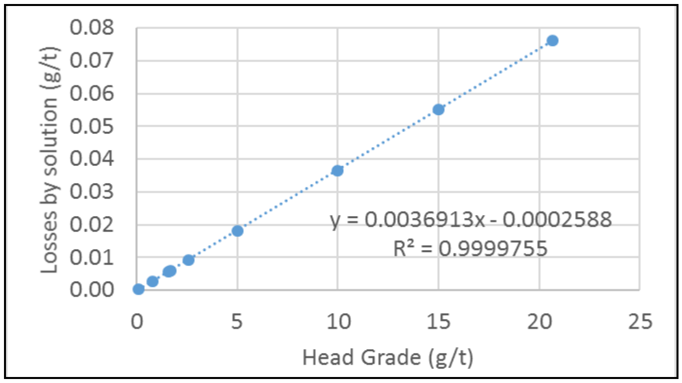


Figure 13.60 Gold Losses Model for Boto 2/Malikoundi Saprolite/Saprock

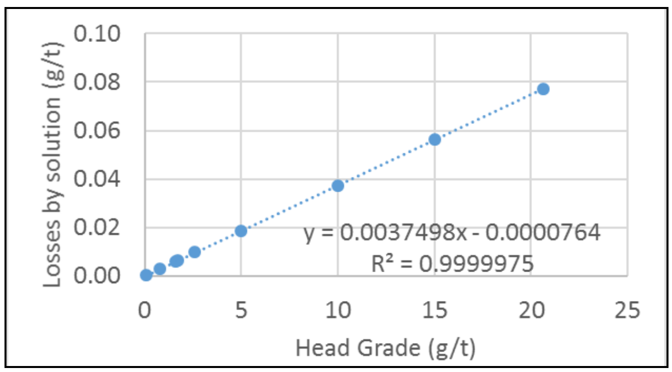
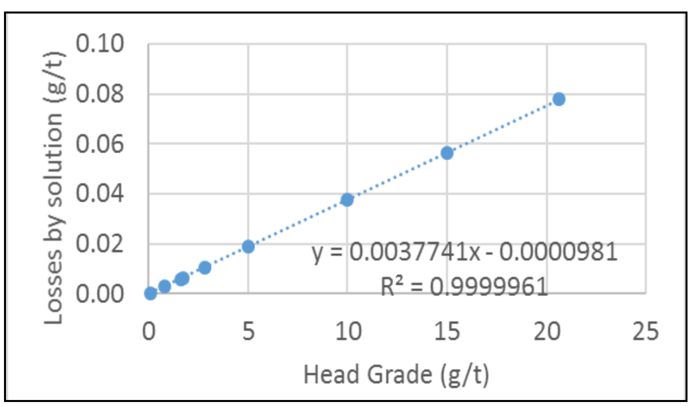


Figure 13.61 Gold Losses Model for Boto 5 Saprolite/Saprock



Gold Recovery Models

Combining the gold extraction and gold losses models, a gold recovery model can then be generated for the Boto 2/Malikoundi fresh rock and saprolite/saprock, and Boto 5 saprolite/saprock. Refer to Figure 13.62, Figure 13.63 and Figure 13.64 for details.

Figure 13.62 Gold Extraction and Recovery Models for Boto 2/Malikoundi Fresh Rock

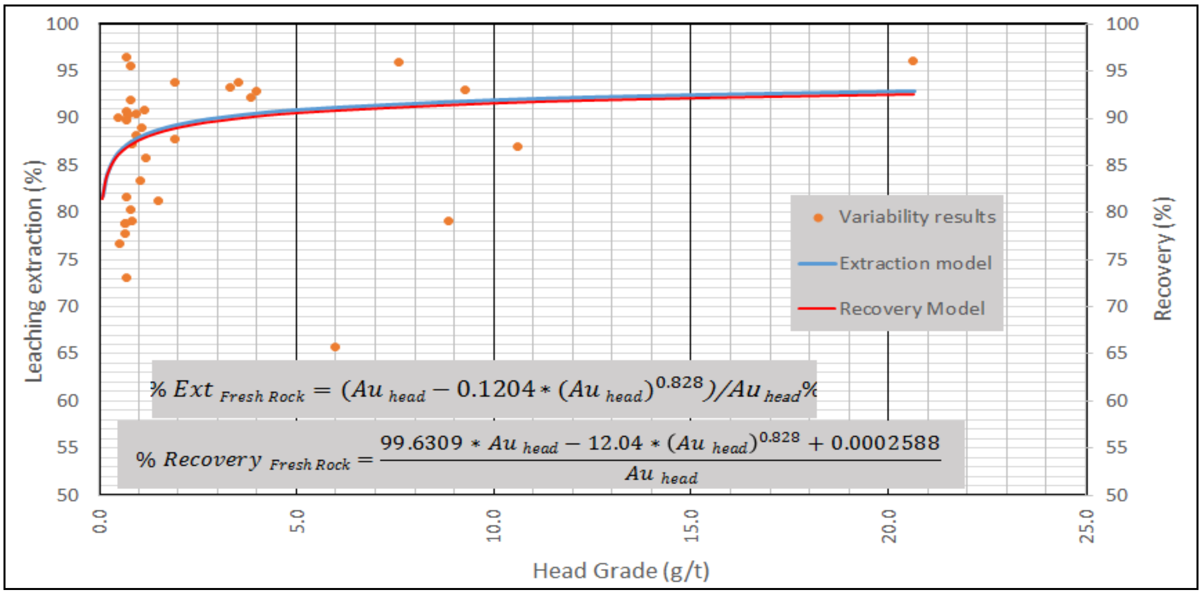


Figure 13.63 Gold Extraction and Recovery Models for Boto 2/Malikoundi Saprolite/Saprock

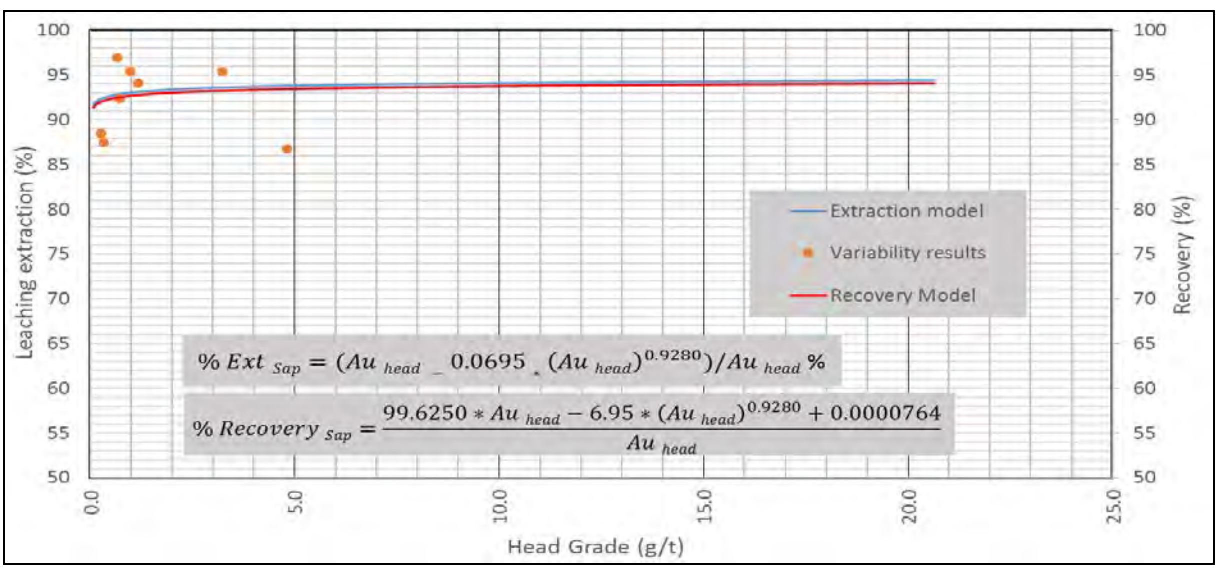
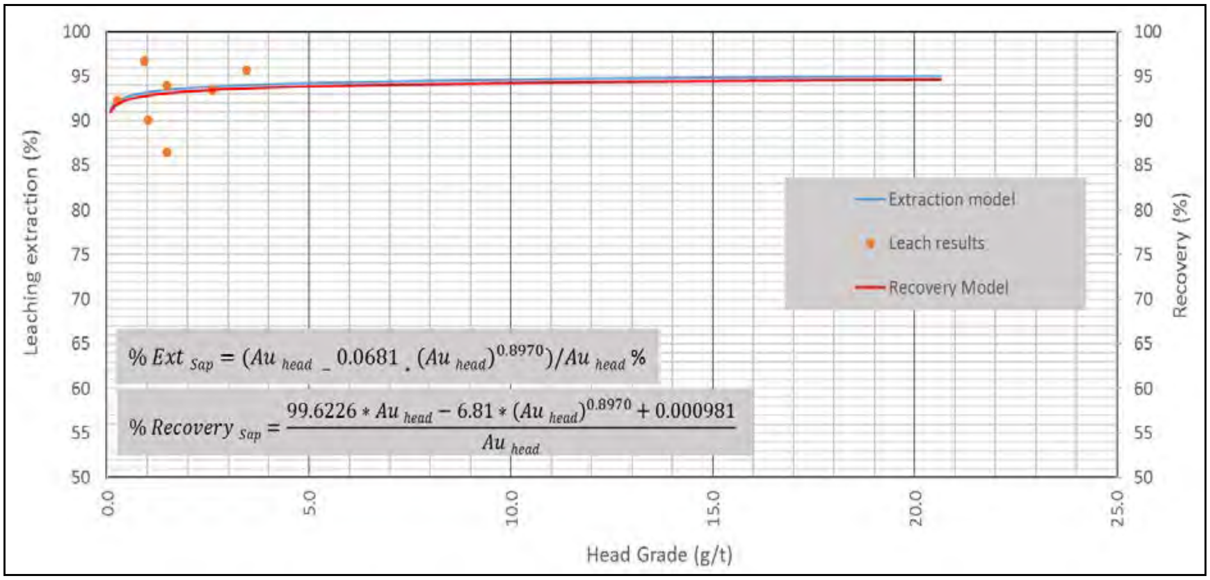


Figure 13.64 Gold Extraction and Recovery Models for Boto 5 Saprolite/Saprock



In summary, the three recovery equations are as follows after combining the extraction equations with the gold loss equations:

Boto 2/Malikoundi

Fresh Rock Rec % = $(99.6309Au_{head} - 12.04Au_{head}^{0.828} + 0.0002588)/Au_{head}$ where $Au_{head} \geq 0.12$ **(Eq. 1)**

Sap/saprock Rec % = $(99.6250Au_{head} - 6.95Au_{head}^{0.928} + 0.0000764)/Au_{head}$ where $Au_{head} \geq 0.27$ **(Eq. 2)**

Boto 5

Sap/saprock Rec % = $(99.6226Au_{head} - 6.81Au_{head}^{0.897} + 0.000981)/Au_{head}$ where $Au_{head} \geq 0.26$ **(Eq. 3)**

13.3.6 Summary of Metallurgical Criteria

A summary of the metallurgical inputs in the process design criteria is presented in Table 13.55.

Table 13.55 Summary of Metallurgical Criteria

Criteria	Units	Design	Notes / Source
Plant Throughput	tpa	2,750,000	Mine plan
Ore Type	- - -	Fresh Rock Saprolite Saprock	Mine plan
Ore Blend Per Mine Plan - Fresh Rock - Saprolite/Saprock	% %	90 10	Lycopodium
Ore Blend for Comminution Design - Fresh rock - Saprolite/Saprock	 % %	 100 0	Lycopodium
Head Grade - Gold (Design) - Silver	Au g/t Ag g/t	1.71 negligible	LOM average head grade Testwork
Ore Specific Density Ore Bulk Density Crushing Work Index (CWi, average) Bond Ball Mill Work Index (BWi, average) Bond Abrasion Index (Ai)	t/m^3 t/m^3 kWh/t kWh/t 	2.81 1.55 17.7 20.6 0.476	Testwork Lycopodium Lycopodium Lycopodium Lycopodium
Targeted Grind Size P_{80} Leach & CIP Circuit Residence Time Targeted Pulp Slurry Density	µm hrs % solids	75 36 50%	Lycopodium Testwork Testwork
Pre-leach Thickener Solids Loading	$t/m^2{\cdot}h$	1.40	Testwork
Pre-oxygenation / Leach Aeration - Oxygen Uptake Rate - Targeted Dissolved Oxygen Level	 mg/L/min ppm	 0.135 >15	 Testwork Lycopodium
Lead Nitrate Addition	kg/t	0.20	Testwork
Sodium Cyanide Consumption - Consumption per Blended Ore - NaCN Loss to Tails	kg NaCN/t kg NaCN/t kg NaCN/t	0.27 0.13 0.14	Calculated Testwork Calculation
Lime Consumption (100% purity) - Fresh rock average - Sap/saprock average - Per Design Ore Blend	 kg/t kg/t kg/t	 1.51 2.81 1.64	 Testwork Testwork Calculation
Daily Loaded Carbon Advance Rate Maximum Loaded Carbon Grade Targeted Loaded Carbon Grade	t/d g /t Au g g/t Au	5.0 Up to 5,000 2,643	Lycopodium Lycopodium Lycopodium

13.4 Conclusions and Recommendations

13.4.1 Conclusions

The following conclusions can be made from the metallurgical testwork with regards to a leach/CIP process:

- Fresh rock, saprolite, and saprock at Boto are readily amenable to whole ore cyanidation.
- The optimum grind size was determined to be P_{80} of 75 µm.
- Gold recovery is predicted to be 88.7% for Boto 2/Malikoundi fresh rock, 92.9% for Boto2/Malikoundi saprolite/saprock, and 93.2% for Boto 5 saprolite/saprock, at the design head grade of 1.71 g/t Au.
- The ore at Boto is not expected to have preg-robbing properties.
- A pre-oxygenation step with oxygen sparging during leach, combined with lead nitrate addition is critical in achieving the maximum possible recovery.
- Leach extraction rates are essentially completed by 24-hours to 36-hours.
- Cyanide consumption rates are expected to be low, averaging about 0.13 kg NaCN/t ore. When accounting for cyanide residue in CIP tailings, an addition rate of 0.27 kg NaCN/t ore is expected.
- Lime consumption rates are expected to be moderate, average at 1.64 kg CaO/t ore at the design ore blend. When accounting for 85% purity of the supplied lime, and an addition rate of 1.93 kg CaO/t ore is expected.

13.4.2 Recommendations

Material meeting minimum 3-inch rock size from future drilling activities should be set aside for CWi tests since the additional CWi tests planned for in the FS phase were not conducted due to material availability.

During plant operations, the following items are recommended:

- Natural cyanide attenuation (free and WAD) be monitored in the tailings storage facility.
- Site water quality (raw and process) be monitored during the initial wet and dry seasons to document the seasonal impact of water quality.
- Gold adsorption rate and equilibrium loading on carbon be monitored as the plant head grade varies during the life of the operation to ensure that carbon movement and management is optimized.
- Slurry percent solids in the leaching stage be monitored during start-up and operation as this parameter could reduce gold extraction if allowed to increase to over 50% solids.

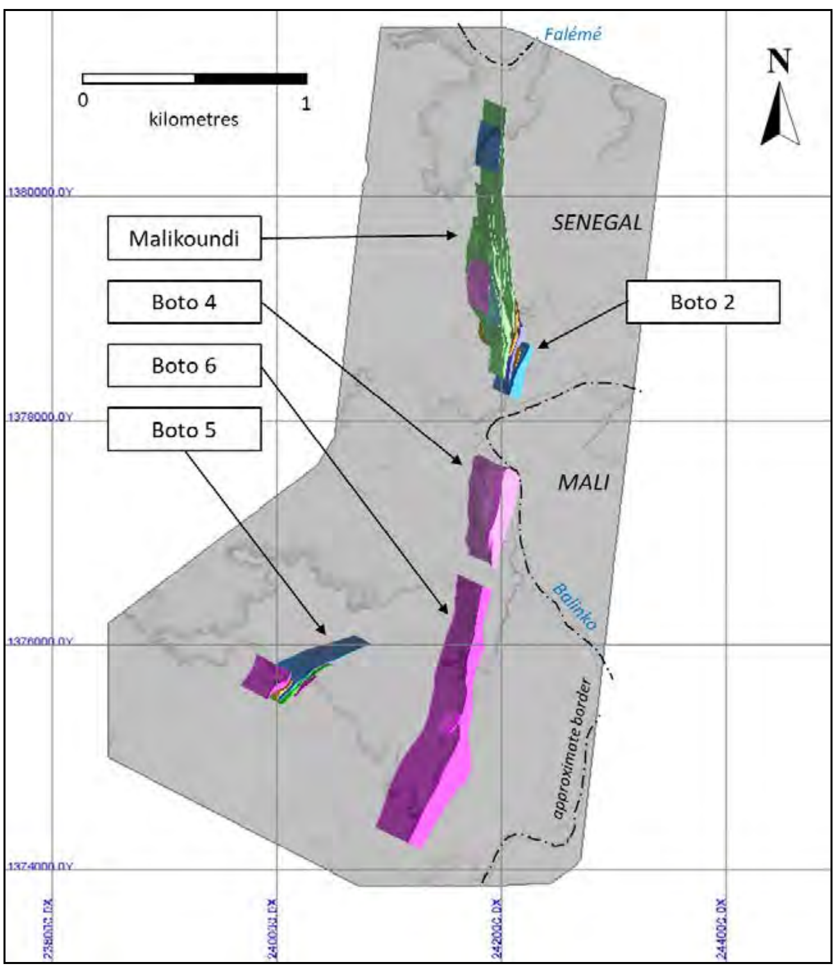
14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

This section discloses the mineral resources for the Project, prepared and disclosed in accordance with the CIM Standards and Definitions for Mineral Resources and Mineral Reserves (2014). The QP responsible for these resource estimates is Mr. Paul Daigle, P.Geo., Associate Senior Geologist for AGP. The effective date of this mineral resource is May 8, 2018.

The resource estimate has been prepared using interpreted mineralized veins (domains) for four deposits that comprise the Project; these include, in order of priority: Malikoundi/Boto 2, Boto 5, Boto 6, and Boto 4 (Figure 14.1). The resource estimates were conducted using Geovia GEMS™ 6.8.1 resource estimation software. The blocks models were estimated using inverse distance cubed.

Figure 14.1 Plan View of the Project



The mineral resources are reported at variable cut-off grades based on alteration zones that vary between 0.37 g/t Au and 0.51 g/t Au. Gold grades were capped prior to compositing, with capping levels varying between 1.71 g/t Au and 42.02 g/t Au depending on mineralized zone and sub-domain. Several mineralized zones did not require capping.

The mineral resources are reported within optimized constraining shells. The optimized constraining shells were developed for each deposit by AGP using Hexagon Mining MineSight 3D and incorporates metal recovery, geotechnical parameters, and assumed costs for each alteration zone. The mineral resources are classified as Indicated Resources or Inferred Resources in accordance with the CIM Definitions of Mineral Resources and Mineral Reserves (2014).

Table 14.1 presents a summary of the Mineral Resources for the Project.

Table 14.1 Summary of Mineral Resources for the Project; Effective Date 8 May, 2018

Classification	Tonnes (,000t)	Grade (g/t Au)	Contained Metal (,000 oz Au)
Indicated	48,045	1.61	2,487
Inferred	2,483	1.80	144

***Notes**:*

- *Mineral resources are reported within optimized constraining shells using MineSight 3D software.*
- *Summation errors may occur due to rounding;*
- *Cut-off grades vary between 0.37 g/t Au and 0.51 g/t Au, depending on the deposit and the alteration type of material;*
- *Mineral resources were estimated based on a gold price of $US 1,500/ oz;*
- *Capping of grades varied between 1.71 g/t Au and 42.02 g/t Au on raw assays by mineralized zone or sub-domain;*
- *The density varies between 1.70 g/cm^3 and 2.76 g/cm^3 depending on alteration zone.*

The mineral resources for the Boto project include the Malikoundi/Boto 2, Boto 5, and Boto 6 deposits. The Boto 4 deposit is currently not classified as mineral resources because the deposit is situated within a 500 m exclusion zone of the Balinko River (the border of Senegal and Mali) and underneath the village of Guémédji. Should the 500 m zone be reduced or lifted the Boto 4 deposit will be re-evaluated.

14.1.1 Database

The database was provided by IAMGOLD on November 23, 2017 and updated on April 15, 2018 and is comprised of diamond drill holes (DDH) and reverse circulation (RC) drill holes completed up to March 2018. All drill holes were positioned on a local grid and the final collar coordinates were surveyed using a DGPS system using the UTM WGS 84 datum. The current database consists of 784 drill holes (410 DDH and 375 RC) totalling approximately 126,429 m, where 565 of the drill holes are used in the current resource estimates, totalling 102,442 m. A summary of the drill holes is listed in Table 14.2 below.

Table 14.2 Summary of Drill Holes for the Project up to March 2018

Year	Type	Drill Holes		Number of Drill Holes for Resource	
		Number	Metres	Number	Metres
2000	DDH	8	1,117	6	853
2000	RC	2	177	2	177
2001	DDH	13	2,057	11	1,921
2001	RC	23	2,080	23	2,080
2002	RC	24	1,593	5	387
2003	RC	52	3,292	43	2,980
2007	DDH	11	2,639	11	2,639
2007	RC	107	10,687	73	7,289
2008	DDH	18	3,721	18	3,721
2009	DDH	17	3,880	17	3,880
2009	RC	73	7,618	39	4,276
2011	DDH	1	284	1	284
2012	DDH	50	13,322	50	13,322
2013	DDH	52	13,130	52	13,130
2014	DDH	60	16,223	52	13,473
2015	DDH	58	14,856	46	13,098
2016	DDH	38	6,139	38	6,139
2017	DDH	69	11,853	69	11,853
2018	DDH	14	2,452	2	290
2018	RC	94	9,309	7	652
TOTAL		**784**	**126,429**	**565**	102,444

AGP received the database as a GEMS project for all deposits of the Project. The database was made up of several tables that included, but are not limited to, collar, survey, assay, lithology, alteration, and structural mineralization (including oriented core information and principal textures). The GEMS validation tool was used to verify the database for the collar, survey, assay, and lithology and alteration tables; there were no errors found.

As described in Section 12 of this report, AGP reviewed approximately 12% of the assay database distributed over the four deposits and made a comparison with the results found in the assay certificates issued by the laboratory. No errors found. The author is of the opinion that the database is adequate for the purposes of mineral resource estimation for the Project.

14.2 MALIKOUNDI/BOTO 2

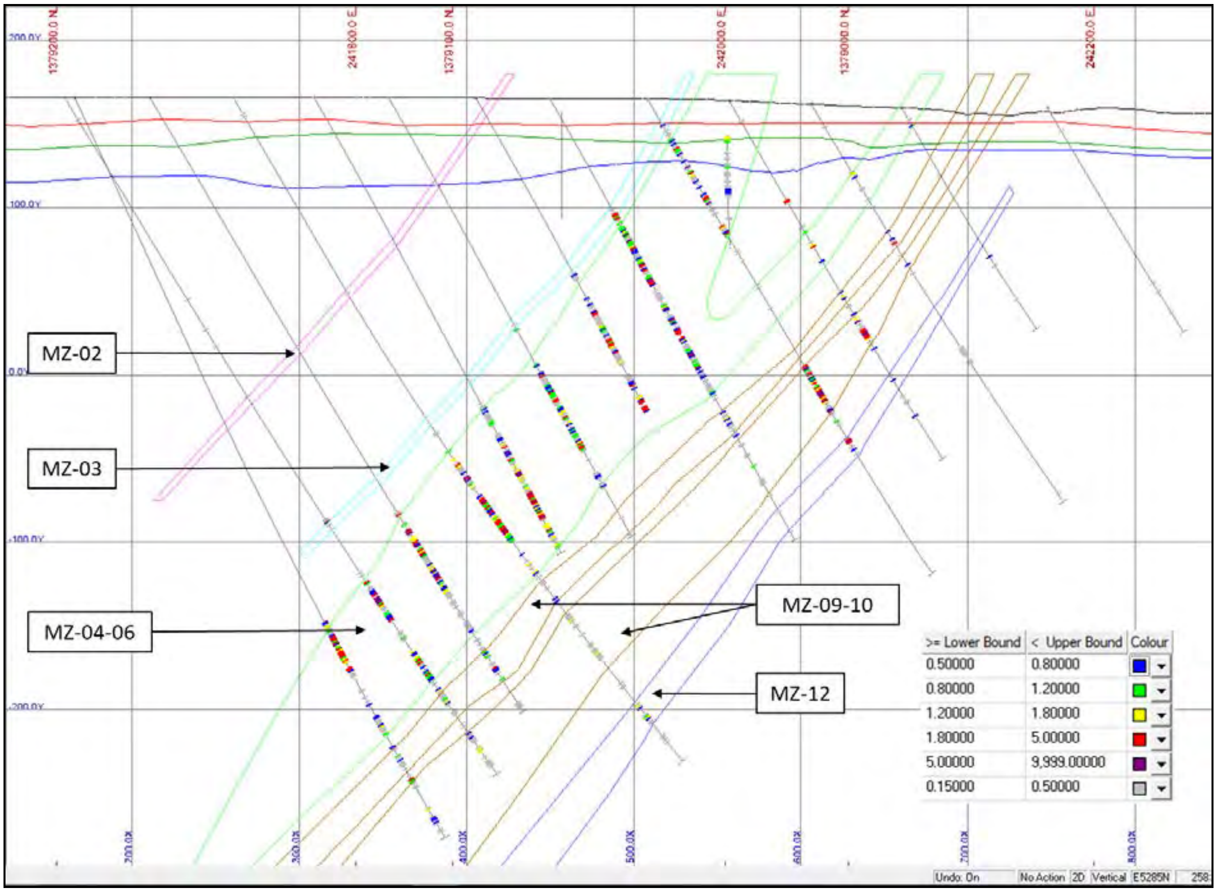
14.2.1 Geological Interpretation

In order to carry out mineral resource estimation at the Malikoundi/Boto 2 deposit, mineralized zones were interpreted by IAMGOLD's on-site geologists. The interpreted mineralized wireframes were completed using conventional polylines on vertical sections defined along the 50 m to 100 m spaced drill fences and matched section by section for better continuity. The rings were snapped to the assay 'From's' and 'To's' and subsequently, were connected by tie lines to create a 3-dimensional (3D) solid wireframe.

In January 2018 and April 2018, the eight solid wireframes that represent the mineralized zones for the Malikoundi/Boto 2 deposit were updated by IAMGOLD geologists based on most recent drill information. A minimum true thickness of 3 m to 4 m was used in modelling the various mineralized zones. The interpreted mineralized envelopes were used to capture a minimum nominal grade of 0.15 g/t Au, supported by geology, strong alteration, and a larger concentration of sulphide minerals, mainly pyrite. True thickness of these mineralized envelopes varies between < 2 m and 119 m, with an average of 20 m. Several short intervals exist where drilling was stopped within a mineralized zone. The statistical evaluation of the gold assays, capping and compositing, was carried out based on these interpreted mineralized zones.

The geological models used for the current mineral resources for Malikoundi/Boto 2 are shown in Figure 14.2.

Figure 14.2 Cross-section E5285N at Malikoundi/Boto 2; Showing Gold Grades Greater than or Equal to 0.15 g/t Au; looking northeast 025°Az



Note: Grid is 100 m by 100 m (AGP, May 2018)

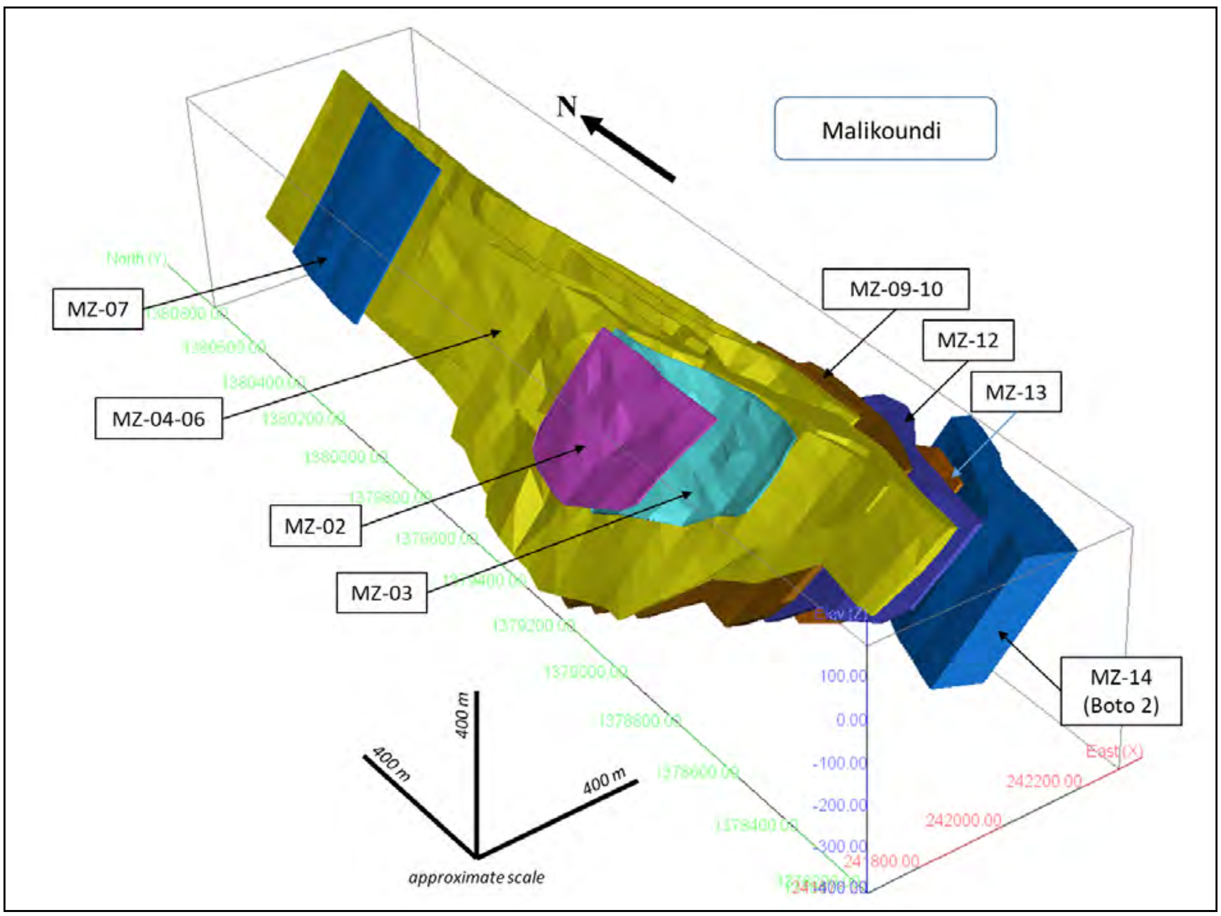
The eight 3D wireframes were considered by AGP as a good representation of the mineralized structures present and were accepted by AGP. No triangulation errors existed in any of the wireframes reviewed. In order to undertake the estimation of the Malikoundi/Boto 2 deposit, two of the larger wireframes (MZ-04-06 and MZ-09-10) were divided into sub-domains to avoid the influence of the neighbouring parallel zones.

Table 14.3 shows the mineralized zones and their corresponding rock codes and sub-domain codes. Figure 14.3 illustrates the eight interpreted mineralized wireframes. Figure 14.4 identifies the sub-domains for MZ-04-06 and MZ-09-10.

Table 14.3 List of Mineralized Zones and Rock Codes and Sub-domain Codes for Malikoundi/Boto 2

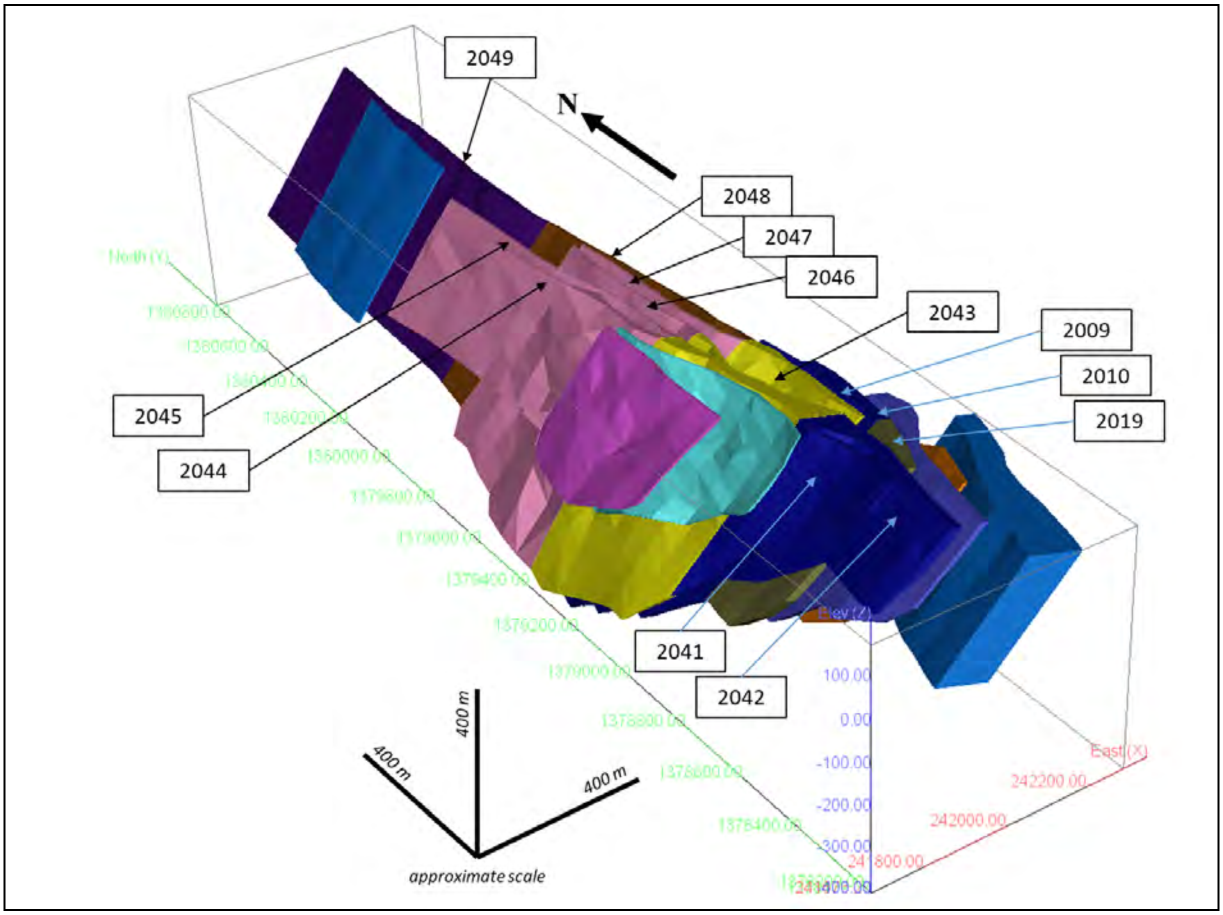
Mineralized Zone (Wireframe)	Volume (,000 m^3)	Rock Code	Sub-domain Code
MZ-02	598	202	202
MZ-03	850	203	203
MZ-04-06	33,265	204	2041, 2042, 2043, 2044, 2045, 2046, 2047, 2048, 2049
MZ-07	739	207	207
MZ-09-10	14,149	209	2009, 2010, 2019
MZ-12	6,980	212	212
MZ-13	318	213	213
MZ-14	441	214	214

Figure 14.3 Interpreted Mineralized Zones for Malikoundi/Boto 2; perspective view looking Northeast)



(AGP, May 2018)

Figure 14.4 Interpreted Mineralized Zones for Malikoundi/Boto 2; perspective view looking northeast; showing Sub-domains for MZ-04-06 and MZ-09-10



(AGP, May 2018)

Alteration Surfaces

The alteration surfaces were modelled by IAMGOLD geologists and were updated in April 2018 based on the most recent drill information. The alteration surfaces distinguish the zones of tropical weathering for laterite, saprolite, transition (or saprock), and fresh rock. The alteration zones for laterite, saprolite, and transition do represent a large part of the Malikoundi/Boto 2 deposit. The laterite zone is generally barren of gold mineralization and is excluded from mineral resources. Each block is coded into the block model based on a 50/50 rule.

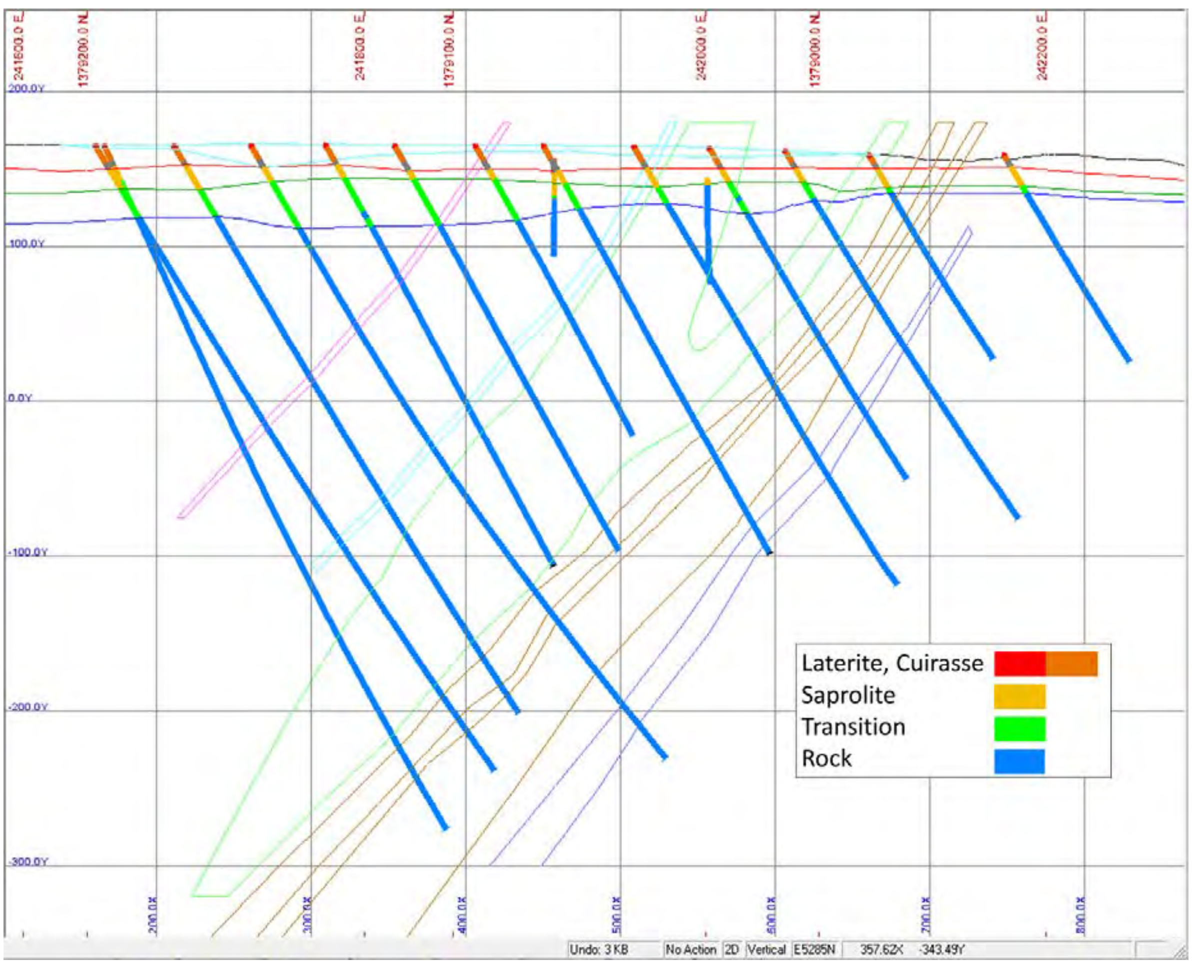
In order to capture some of the cuirass layer (also known as Ferrocrete or hardpan layer of laterite) AGP modelled the cuirass based on the descriptions in the drill hole database. This model was used to capture a possible volume of cuirass that may be used as construction material within the laterite zone. The laterite and cuirass zones are not considered for mineral resources.

Table 14.4 presents the alteration codes, or Weather model, used in the Malikoundi/Boto 2 block model. Figure 14.5 presents a cross-section of the alteration profile for the Malikoundi/Boto 2 deposit.

Table 14.4 List of Alteration Zones and Codes for Malikoundi/Boto 2

Alteration Zone	Code
Cuirass (Ferrocrete)	30
Laterite	40
Saprolite	50
Transition (or Saprock)	60
Rock	70

Figure 14.5 Cross-section E5285N of the Alteration (Weather) Model for the Malikoundi/Boto 2 Deposit; looking 025°Az northeast



Note: Grid is 100 m x 100 m (AGP May 2018)

14.2.2 Statistical Analysis

Raw Assays

The database for the Malikoundi/Boto 2 deposit consists of a total of 53,496 assays values, where a total of 16,161 assays values intercept the interpreted mineralized zones. The majority of the samples were collected on 1 m sample intervals. The descriptive statistics of the raw assays, by mineralized zones and sub-domains are shown in Table 14.5 below. Descriptive statistics for all raw assay values and sample lengths are shown in Table 14.6. Figure 14.6 shows the box plots for the raw assay values for Malikoundi/Boto 2.

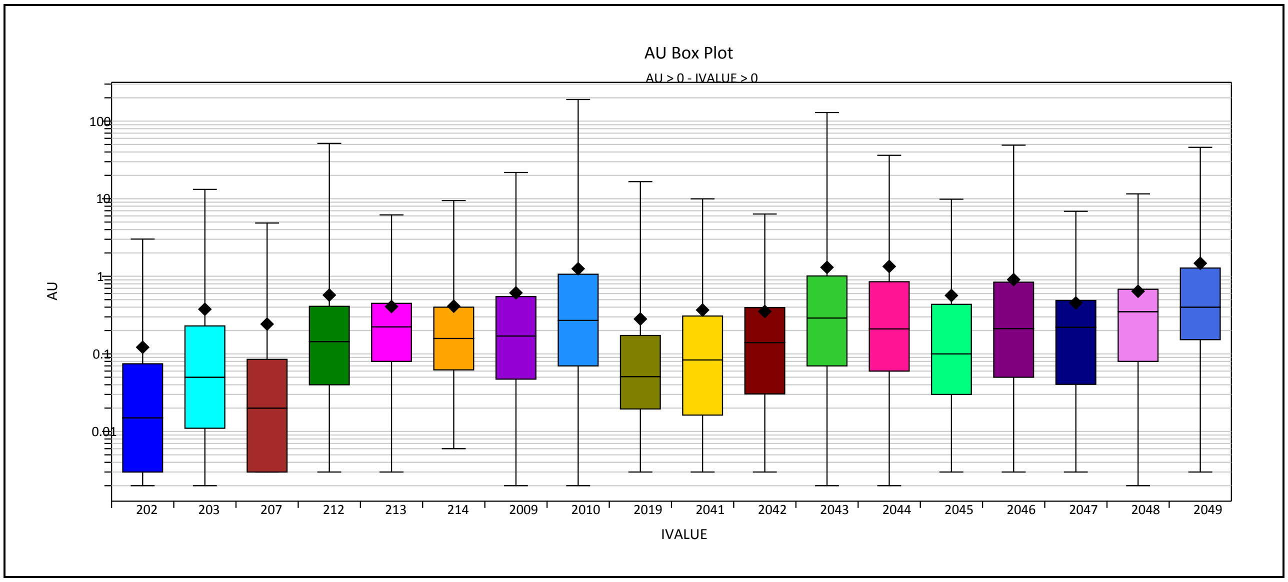
Table 14.5 Descriptive Statistics for Malikoundi/Boto 2 Deposit by Mineralized Zones and Sub-Domains (no zeroes)

Zone/Sub-domain	202	203	2041	2042	2043	2044	2045	2046	2047
	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au
Count	250	435	125	358	5910	707	543	1174	375
Min	0.002	0.002	0.003	0.003	0.002	0.002	0.003	0.003	0.003
Max	3.02	13.18	9.96	6.35	129.13	36.26	9.85	49.00	6.88
Mean	0.12	0.36	0.37	0.35	1.30	1.31	0.47	0.89	0.44
Std Dev	0.37	1.22	1.00	0.67	4.25	3.56	1.19	2.63	0.77
CV	3.07	3.25	2.72	1.90	3.27	2.72	2.56	2.95	1.74
Zone/Sub-domain	**2048**	**2049**	**207**	**2009**	**2010**	**2019**	**212**	**213**	**214**
	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au
Count	286	561	90	1133	1995	334	1098	165	633
Min	0.002	0.003	0.003	0.002	0.002	0.003	0.003	0.003	0.006
Max	11.55	45.95	4.86	21.77	190.00	16.60	51.64	6.19	9.47
Mean	0.66	1.47	0.24	0.61	1.25	0.28	0.57	0.41	0.41
Std Dev	1.19	3.81	0.66	1.38	5.06	1.08	2.05	0.67	0.80
CV	1.81	2.59	2.74	2.26	4.04	3.82	3.58	1.64	1.95

Table 14.6 Descriptive Statistics for Malikoundi/Boto 2 Deposit

	g/t Au	Length (m)
Count	16172	16172
Minimum	0.002	0.5
Maximum	190.00	5.00
Average	0.97	1.05
Standard Deviation	3.48	0.26
CV	3.58	0.25

Figure 14.6 Box plot of Raw Assays for Malikoundi/Boto 2 by Mineralized Zone and Sub-domain



Capping Analysis

To reduce the influence of outliers on the average grade, and the co-efficient of variation (CV) of the sample populations, a capping analysis in the form of decile analyses, disintegration analyses, histogram, and cumulative plots were used to assess the sample populations of gold grade outliers within each of the mineralized zones and sub-domains for Malikoundi/Boto 2.

Table 14.7 shows the capping levels and number of values affected for each of the mineralized zones and sub-domains.

Table 14.7 List of Capping Levels by Mineralized Zone/Sub-domain

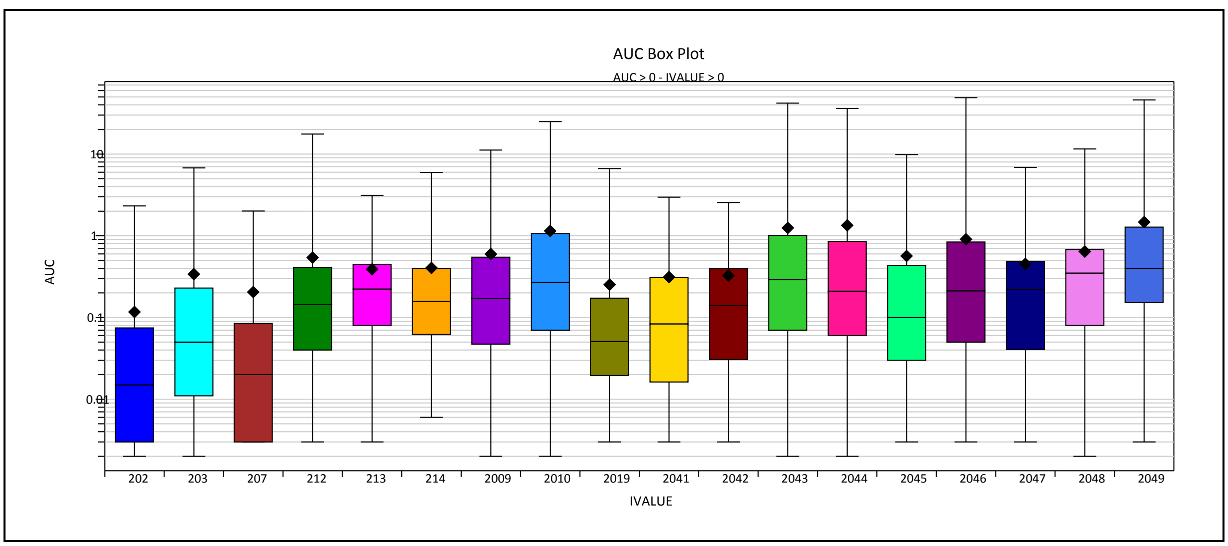
Mineralized Zone	Zone/Sub-domain (Code)	Capping Grade (g/t Au)	Nº of Values Affected	% of Total Number of Values Affected
MZ02	202	2.32	2	0.8 %
MZ03	203	6.77	3	0.7 %
MZ04-06	2041	2.97	1	0.8 %
	2042	2.55	3	0.8 %
	2043	42.02	10	0.2 %
	2044	24.82	4	0.6 %
	2045	6.20	5	0.9 %
	2046	20.37	4	0.3 %
	2047	no capping	0	-
	2048	no capping	0	-
	2049	24.78	3	0.5 %
MZ07	207	2.01	2	2.2 %
MZ09-10	2009	11.20	2	0.2 %
	2010	25.00	4	0.2 %
	2019	6.64	1	0.3 %
MZ12	212	17.58	1	0.1 %
MZ13	213	3.13	1	0.6 %
MZ14	214	5.97	2	0.3 %

Table 14.8 presents the summary of descriptive statistics for capped gold grades by mineralized zone and sub-domain. Figure 14.7 presents the box plot of capped gold grades by mineralized zone and sub-domain.

Table 14.8 Descriptive Statistics for Capped Gold Grades by Mineralized Zone and Sub-domain

Zone/Sub-domain	202	203	2041	2042	2043	2044	2045	2046	2047
	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au
Count	250	435	125	358	5910	707	543	1174	375
Min	0.002	0.002	0.003	0.003	0.002	0.002	0.003	0.003	0.003
Max	2.32	6.77	2.97	2.55	42.02	24.82	6.20	20.37	6.88
Mean	0.12	0.36	0.31	0.33	1.24	1.26	0.44	0.85	0.45
Std Dev	0.34	0.89	0.56	0.50	3.24	3.09	1.00	2.02	0.78
CV	2.91	2.65	1.79	1.53	2.61	2.47	2.30	2.38	1.71
Zone/ Sub-domain	**2048**	**2049**	**207**	**2009**	**2010**	**2019**	**212**	**213**	**214**
	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au	g/t Au
Count	286	561	90	1133	1995	334	1098	165	633
Min	0.002	0.003	0.003	0.002	0.002	0.003	0.003	0.003	0.006
Max	9.11	24.78	2.01	11.20	25.00	6.64	17.58	3.13	5.97
Mean	0.64	1.40	0.21	0.60	1.15	0.25	0.54	0.39	0.40
Std Dev	1.17	2.66	0.46	1.21	2.06	0.69	1.45	0.54	0.74
CV	1.82	2.25	2.24	2.02	2.24	2.75	2.67	1.38	1.82

Figure 14.7 Box Plots for Capped Gold Grades for Malikoundi/Boto 2 by Mineralized Zone and Sub-Domain



Composites

After capping the raw assay values, the raw assay values were composited to 2 m intervals within the interpreted mineralized zone wireframe. The composites were adjusted along the drill hole to avoid remnants at the hanging wall or foot wall of the mineralized zone.

A nominal composite length of 2 m was selected which was at or above the 75th percentile of the raw assay sampling length. Composites were generated downward from the collar of the hole within the mineralized wireframes. Composites length were adjusted to avoid un-representative remnants at the wireframe boundaries where the minimum composition length is 1 m. Capping was applied to the raw assays prior to compositing. Table 14.9 shows the descriptive statistics of the capped 2 m composite values by mineralized zone and sub-domain. Descriptive statistics for all raw assays and composite lengths are shown in Table 14.9 and Table 14.10. Figure 14.8 presents the box plot of the capped 2 m composite values by mineralized zone and sub-domain.

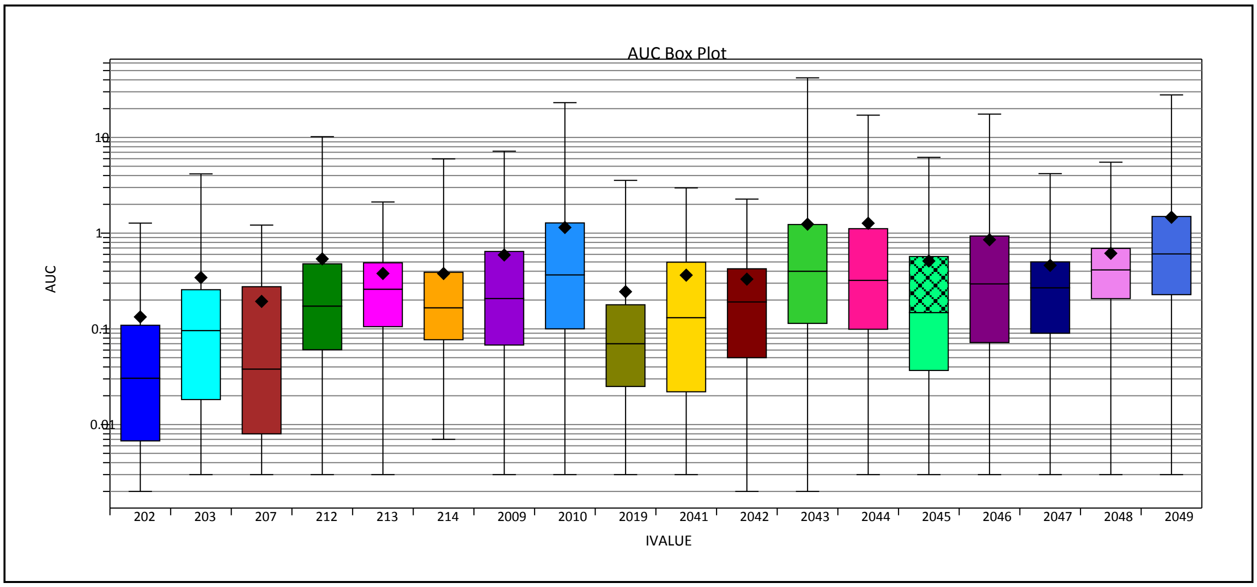
Table 14.9 Descriptive Statistics of the Capped 2 m Capped Composites by Mineralized Zones and Sub-domains

Zone/ Sub-domain	**202**	**203**	**2041**	**2042**	**2043**	**2044**	**2045**	**2046**	**2047**
	g/t Au	**g/t Au**	**g/t Au**	**g/t Au**	**g/t Au**	**g/t Au**	**g/t Au**	**g/t Au**	**g/t Au**
Count	127	221	66	189	3002	351	242	588	185
Min	0.002	0.003	0.003	0.002	0.002	0.003	0.003	0.003	0.003
Max	1.27	4.17	2.97	2.27	41.98	17.09	6.20	17.53	4.19
Mean	0.13	0.34	0.36	0.33	1.24	1.27	0.47	0.85	0.46
Std Dev	0.26	0.69	0.59	0.39	2.62	2.58	0.89	1.66	0.65
CV	1.98	2.00	1.61	1.17	2.11	2.03	1.91	1.96	1.40
Zone/ Sub-domain	**2048**	**2049**	**207**	**2009**	**2010**	**2019**	**212**	**213**	**214**
	g/t Au	**g/t Au**	**g/t Au**	**g/t Au**	**g/t Au**	**g/t Au**	**g/t Au**	**g/t Au**	**g/t Au**
Count	144	302	49	567	996	178	566	91	539
Min	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.007
Max	5.51	21.89	1.21	7.18	23.10	3.56	10.19	2.12	5.97
Mean	0.61	1.39	0.19	0.59	1.15	0.24	0.54	0.38	0.38
Std Dev	0.78	2.30	0.30	0.98	2.12	0.53	1.15	0.42	0.61
CV	1.28	1.65	1.56	1.66	1.85	2.18	2.14	1.09	1.62

Table 14.10 Descriptive Statistics for all Composite Values and Lengths for Malikoundi/Boto 2 Deposit

	g/t Au	Length (m)
Count	8403	8403
Minimum	0.002	1.00
Maximum	41.98	3.00
Average	0.91	2.00
Standard Deviation	2.01	0.12
CV	2.21	0.06

Figure 14.8 Box Plots for the Capped 2 m Composites for Malikoundi/Boto 2 by Mineralized Zones and Sub-domains



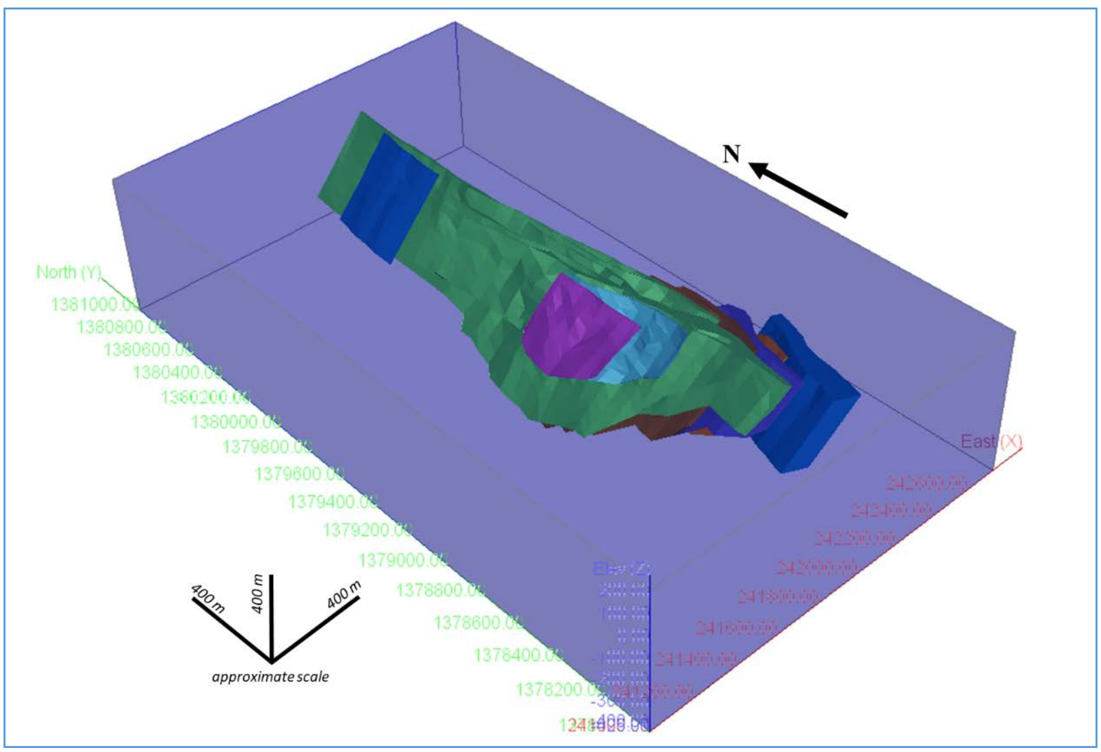
14.2.3 Block Model

The block model for the Malikoundi/Boto 2 deposit was set up with a block matrix of 5 m long by 10 m long by 5 m high. The block model is not rotated, with the elongated blocks aligned to the preferred mainly north-south strike of the deposit. The block matrix was selected as appropriate for the 50 m x 50 m drill pattern and the block height was selected in consideration of a medium sized open pit operation. Table 14.11 summarizes the block model parameters and Figure 14.9 presents the block model over the interpreted mineralized zones for the Malikoundi/Boto 2 Deposit.

Table 14.11 Block Model Parameters for the Malikoundi/Boto 2 Deposit

	Parameters
Easting	241000 mE
Northing	1378000 mN
Maximum Elevation	225 m
Rotation Angle	No rotation°
Block Size (X, Y, Z in metres)	5 x 10 x 5
Number of blocks in the X direction	350
Number of blocks in the Y direction	301
Number of blocks in the Z direction	130

Figure 14.9 Block Model for the Malikoundi/Boto 2 Deposit; perspective view looking Northeast



(AGP May 2018)

Grade Interpolation

The block model was interpolated in two passes. The gold grades were estimated using the 2 m composites using the inverse distance cubed (ID^3) interpolation (anisotropic) method. This method was selected due as there were no reasonable variograms obtained for the individual mineralized zones. The block model was also interpolated using ID^2 and NN interpolation methods for validation purposes. Table 14.12 shows estimation parameters for each pass used to estimate gold grades.

Table 14.12 Estimation Parameters for the Malikoundi/Boto 2 Block Model

	Min Nº composites	Max Nº composites	Max Nº composites per Drill Hole	Min Nº of Drill Holes
Pass 1	4	12	3	2
Pass 2	3	12	3	1

Search Ellipses

The search ellipses used for the Malikoundi/Boto 2 block model interpolation used similar ranges as in the previous resource estimates in consideration of the regular 50 m x 50 m drill pattern and their intersections of the mineralized zones. Search ellipses were oriented in alignment with the interpreted mineralized zones and sub-domains.

In the mineralized zone MZ03, there are three shifts in orientation of the zone, from north to south. To better estimate this zone, three separate search ellipses were used on selected blocks in the north, centre, and south portions of this zone. In the mineralized zone MZ09-10, there were two changes in orientation of the zone. To better estimate these zones, two search ellipses were used on selected blocks in the north portion of MZ09 and south portion of MZ10.

Table 14.13 lists the search ellipse parameters used to estimate the Malikoundi/Boto 2 Deposit.

Table 14.13 Search Ellipse Parameters for Pass 1 and Pass 2 for Malikoundi/Boto 2

Profile Name	Search Anisotropy	Azimuth (°)	Dip (°)	Azimuth (°)	X Range (m)	Y Range (m)	Z Range (m)	Search Type
Pass 1								
M1_02	Az,Dip,Az	270	-50	1	75.0	75.0	20.0	Ellipsoidal
M1_03N	Az,Dip,Az	330	-50	10	75.0	75.0	20.0	Ellipsoidal
M1_03M	Az,Dip,Az	320	-50	5	75.0	75.0	20.0	Ellipsoidal
M1_03S	Az,Dip,Az	270	-60	0	75.0	75.0	20.0	Ellipsoidal
M1_09N	Az,Dip,Az	275	-55	0	75.0	75.0	20.0	Ellipsoidal
M1_09S	Az,Dip,Az	260	-50	1	75.0	75.0	20.0	Ellipsoidal
M1_10N	Az,Dip,Az	290	-60	0	75.0	75.0	20.0	Ellipsoidal
M1_10S	Az,Dip,Az	260	-50	1	75.0	75.0	20.0	Ellipsoidal
M1_12_42	Az,Dip,Az	260	-50	1	75.0	75.0	20.0	Ellipsoidal
M1_13_14	Az,Dip,Az	330	-47	35	75.0	75.0	20.0	Ellipsoidal
M1_19	Az,Dip,Az	290	-47	20	75.0	75.0	20.0	Ellipsoidal
M1_41	Az,Dip,Az	275	-50	1	75.0	75.0	20.0	Ellipsoidal
M1_43	Az,Dip,Az	275	-60	0	75.0	75.0	20.0	Ellipsoidal
M1_44_45	Az,Dip,Az	288	-60	0	75.0	75.0	20.0	Ellipsoidal
M1464748	Az,Dip,Az	280	-58	0	75.0	75.0	20.0	Ellipsoidal
M1_MKN	Az,Dip,Az	260	-60	0	75.0	75.0	20.0	Ellipsoidal
Pass 2								
M2_02	Az,Dip,Az	270	-50	1	100.0	100.0	20.0	Ellipsoidal
M2_03N	Az,Dip,Az	330	-50	10	100.0	100.0	20.0	Ellipsoidal
M2_03M	Az,Dip,Az	320	-50	5	100.0	100.0	20.0	Ellipsoidal
M2_03S	Az,Dip,Az	270	-60	0	100.0	100.0	20.0	Ellipsoidal
M2_09N	Az,Dip,Az	275	-55	0	100.0	100.0	20.0	Ellipsoidal
M2_09S	Az,Dip,Az	260	-50	1	100.0	100.0	20.0	Ellipsoidal
M2_10N	Az,Dip,Az	290	-60	0	100.0	100.0	20.0	Ellipsoidal
M2_10S	Az,Dip,Az	260	-50	1	100.0	100.0	20.0	Ellipsoidal
M2_12_42	Az,Dip,Az	260	-50	1	100.0	100.0	20.0	Ellipsoidal
M2_13_14	Az,Dip,Az	330	-47	35	100.0	100.0	20.0	Ellipsoidal
M2_19	Az,Dip,Az	290	-47	20	100.0	100.0	20.0	Ellipsoidal
M2_41	Az,Dip,Az	275	-50	1	100.0	100.0	20.0	Ellipsoidal
M1_43	Az,Dip,Az	275	-60	0	100.0	100.0	20.0	Ellipsoidal
M2_44_45	Az,Dip,Az	288	-60	0	100.0	100.0	20.0	Ellipsoidal
M2464748	Az,Dip,Az	280	-58	0	100.0	100.0	20.0	Ellipsoidal
M1_MKN	Az,Dip,Az	260	-60	0	100.0	100.0	20.0	Ellipsoidal

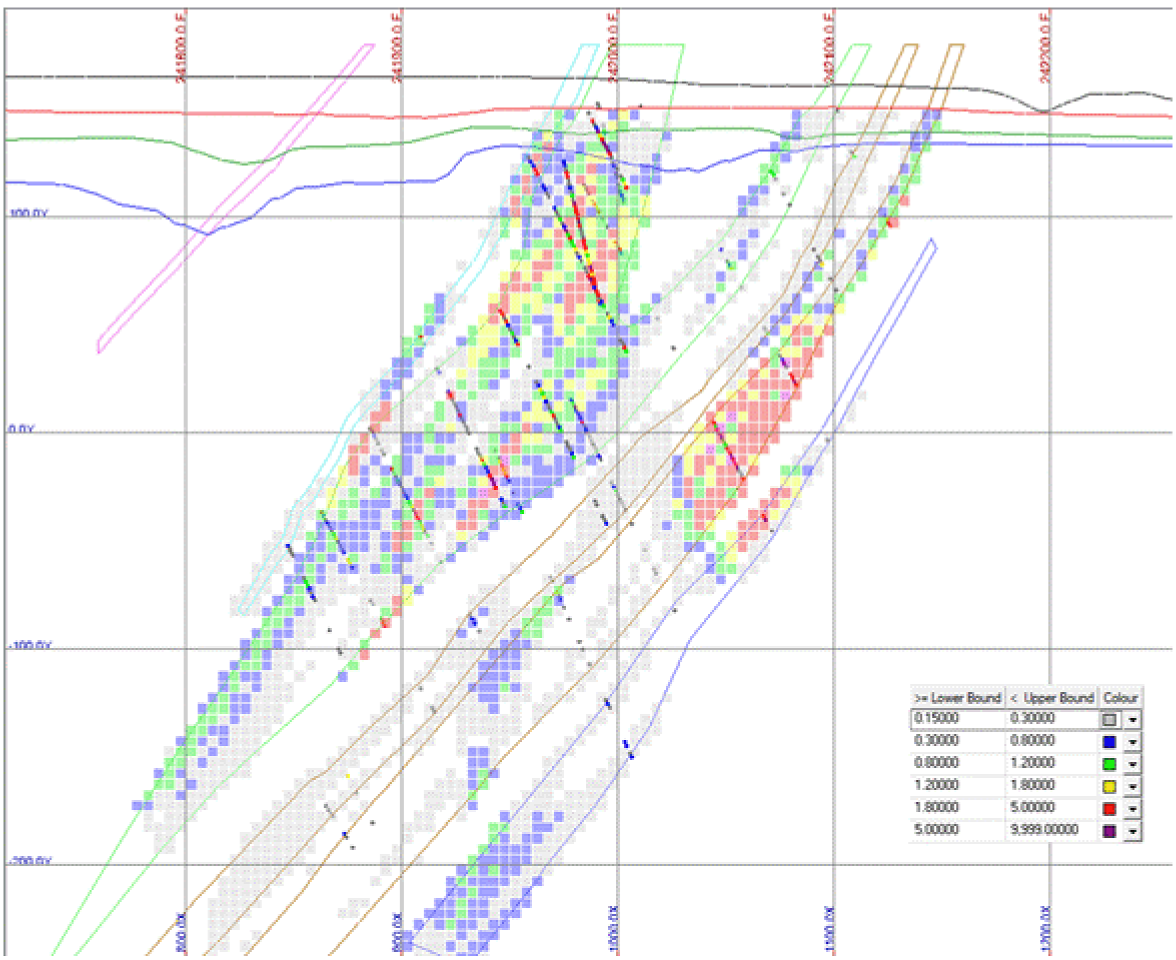
14.2.4 Validation

The Malikoundi/Boto 2 block model and resource estimate was validated by the methods described below. AGP is satisfied that the block model gold grades reflect the gold grades from the drill core samples.

Visual Validation

The block model was validated by visually inspecting the block model results on section to compare with the drill hole composite data. The grades of the blocks by section agreed well with the composite data used in the interpolation. Figure 14.10 presents a selected cross-section (1379005N) that shows the gold grades in blocks with the gold grades from the 2 m composites.

Figure 14.10 Cross-section 1379005N Comparing Block Grades with the 2 m Composites for the Malikoundi/Boto 2 Deposit; looking north



(AGP May 2018)

Statistics

AGP reviewed the statistics for the mineralized zones and sub-domains for Malikoundi/Boto 2 and found no global bias between the different interpolation methods and the 2 m composites. Table 14.14 shows the average gold grades (no zeroes) of the mineralized zone for the Malikoundi/Boto 2 Deposit.

Table 14.14 Average Gold Grades (no zeroes) for the Malikoundi/Boto 2 Deposit

Malikoundi/Boto2	202	203	204	207	209	212	213	214
Data	**Average (g/t Au)**	**Average (g/t Au)**	**Average (g/t Au)**	**Average (g/t Au)**	**Average (g/t Au)**	**Average (g/t Au)**	**Average (g/t Au)**	**Average (g/t Au)**
ID3	0.11	0.33	0.94	0.50	0.75	0.50	0.37	0.38
ID2	0.11	0.33	0.94	0.49	0.76	0.50	0.37	0.38
NN	0.12	0.33	0.90	0.46	0.72	0.46	0.40	0.38
2 m Composites	0.13	0.34	1.08	0.54	0.87	0.54	0.38	0.38

Swath Plots

AGP interpolated gold grades using the ID^3, ID^2, and NN interpolation methods and compared the grades of blocks to the 2 m composites along a particular swath of the block model. The comparison shows no apparent local bias. In the charts, the composite line is generally above the interpolated grade and the peaks and valleys are well represented in the block model which also display normal smoothing of the composite data. Figure 14.11, Figure 14.12, and Figure 14.13 show the swath plots for the selected mineralized zone, MZ04-06, in the Malikoundi/Boto 2 Deposit by Easting, Northing, and Elevation respectively.

Figure 14.11 Swath plots for Gold Grades by Easting for MZ04-06 in the Malikoundi/Boto 2 Deposit

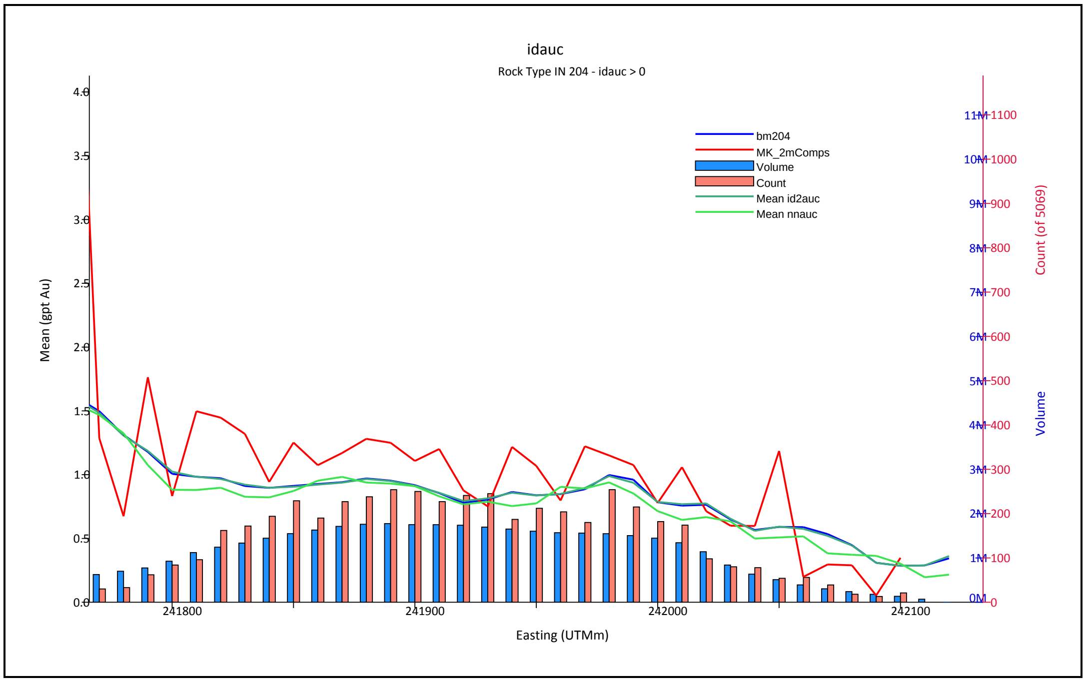


Figure 14.12 Swath plots for Gold Grades by Northing for MZ04-06 in the Malikoundi/Boto 2 Deposit

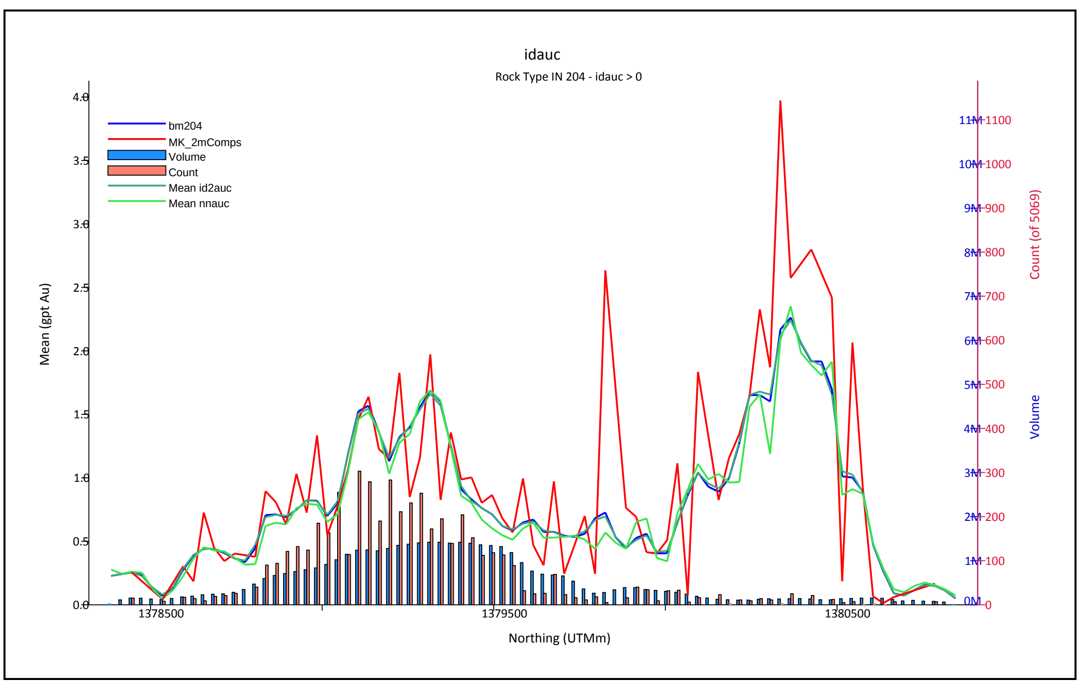
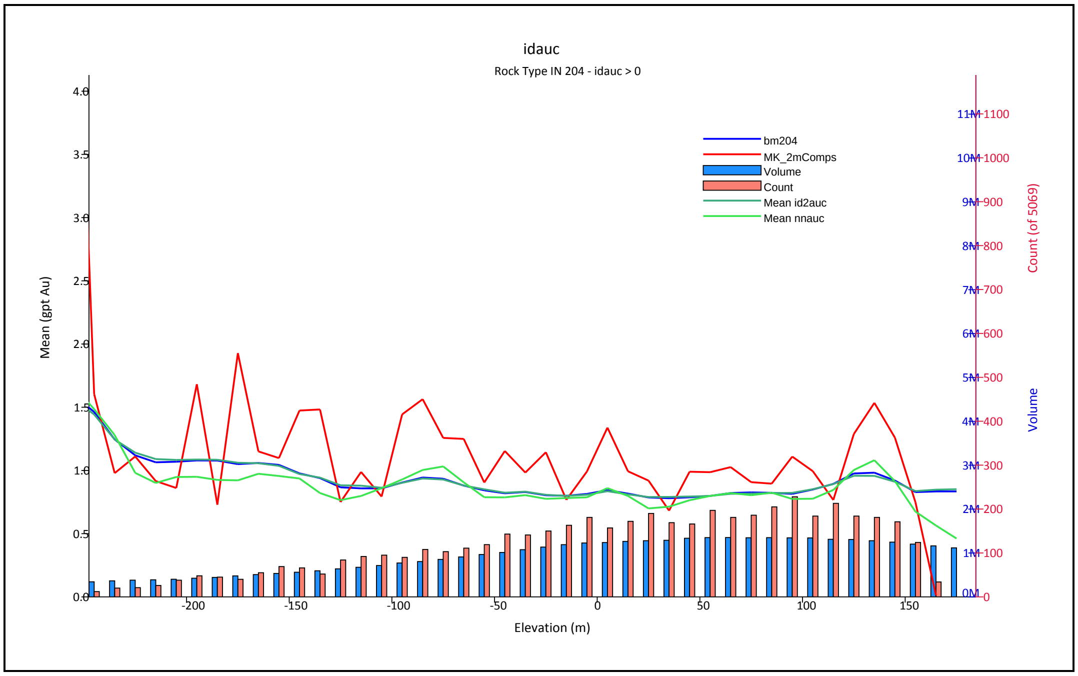


Figure 14.13 Swath plots for Gold Grades by Elevation for MZ04-06 in the Malikoundi/Boto 2 Deposit



14.3 BOTO 5

14.3.1 Geological Interpretation

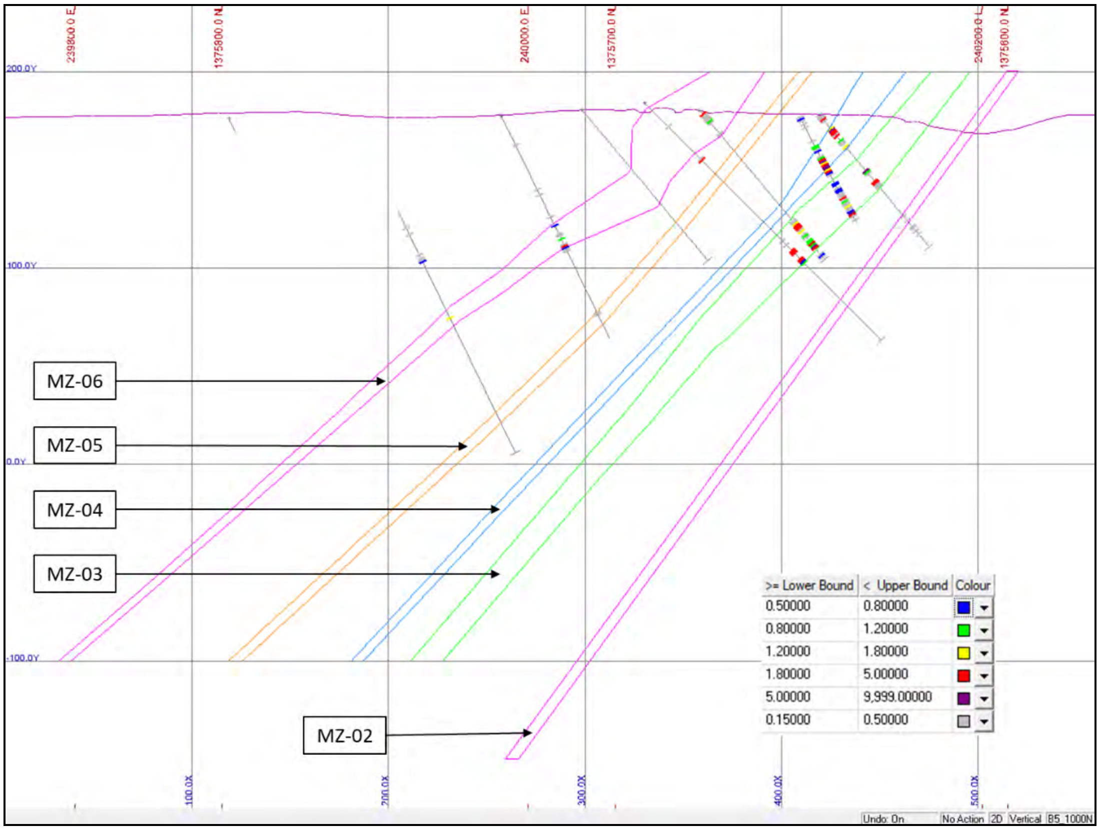
In order to carry out mineral resource estimation at the Boto 5 deposit, five mineralized zones were interpreted by IAMGOLD's on-site geologists. The interpreted mineralized wireframes were completed using 3D rings on vertical sections along the 50 m to 100 m spaced drill fences and matched for better continuity. The rings were snapped to the assay 'From's' and 'To's' and subsequently, connected by tie lines to create a 3D solid wireframe.

In August 2017, the five solid wireframes that represent the mineralized zones were updated by IAMGOLD geologists based on the most recent drill information. A minimum true thickness of 3 m to 4 m was used in modelling the various mineralized zones. The interpreted mineralized envelopes were used to capture a minimum nominal grade of 0.15 g/t Au supported by geology, strong alteration, and a larger concentration of sulphide minerals, mainly pyrite. True thickness of these mineralized envelopes varies between 2 m and 50 m, with an average of 11 m. The statistical evaluation of the gold assays, capping, and compositing was carried out based on these interpreted mineralized zones.

The five 3D wireframes were considered by AGP as a good representation of the mineralized structures present and were accepted by AGP. No triangulation errors existed in any of the wireframes reviewed.

Figure 14.14 shows a cross section (B5_1000N) of the five mineralized zones. Table 14.15 lists the five mineralized zones and their corresponding rock codes.

Figure 14.14 Cross-section B5_1000N at Boto 5; showing gold grades greater than or equal to 0.15 g/t Au; looking northeast 028.5°Az



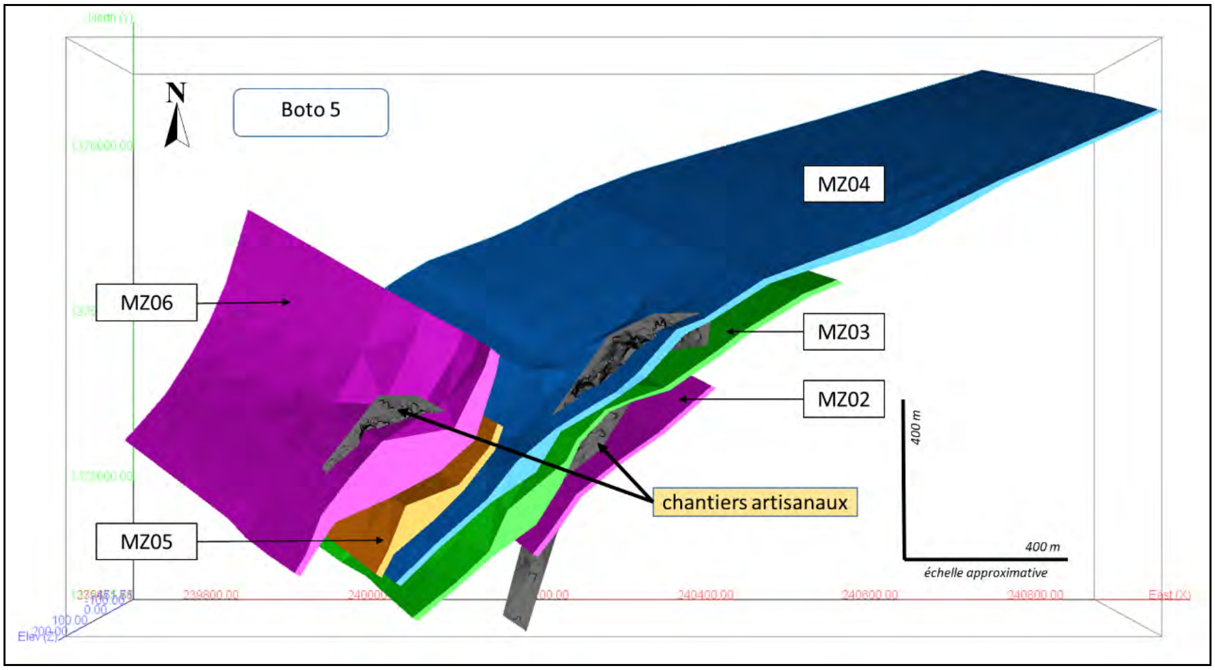
Note: grid is 100 m by 100 m (AGP May 2018)

Table 14.15 List of Mineralized Zones and Rock Codes for Boto 5

Deposit	Volume (m^3)	Mineralized Zone (Wireframe)	Rock Code
Boto 5	597	MZ02	502
	2,204	MZ03	503
	2,097	MZ04	504
	588	MZ05	505
	1,453	MZ06	506

It was observed that a portion of the surface of the Boto 5 deposit and surroundings have been affected by ongoing small scale artisanal mining. In order to account for these workings, two 3D wireframes were constructed using the topographic Lidar surface, which captured the majority of these surface workings. With this information, two portions of the topographic surface were clipped out (west and east) and copied 5 m below surface. Two wireframes were built between these two sets of surfaces to represent material removed or condemned by the artisanal workings. Figure 14.15 shows the five mineralized zones and the two areas of artisanal workings.

Figure 14.15 Plan View of the Boto 5 Deposit; showing the five mineralized zones and artisanal workings



(AGP May 2018)

Alteration Surfaces

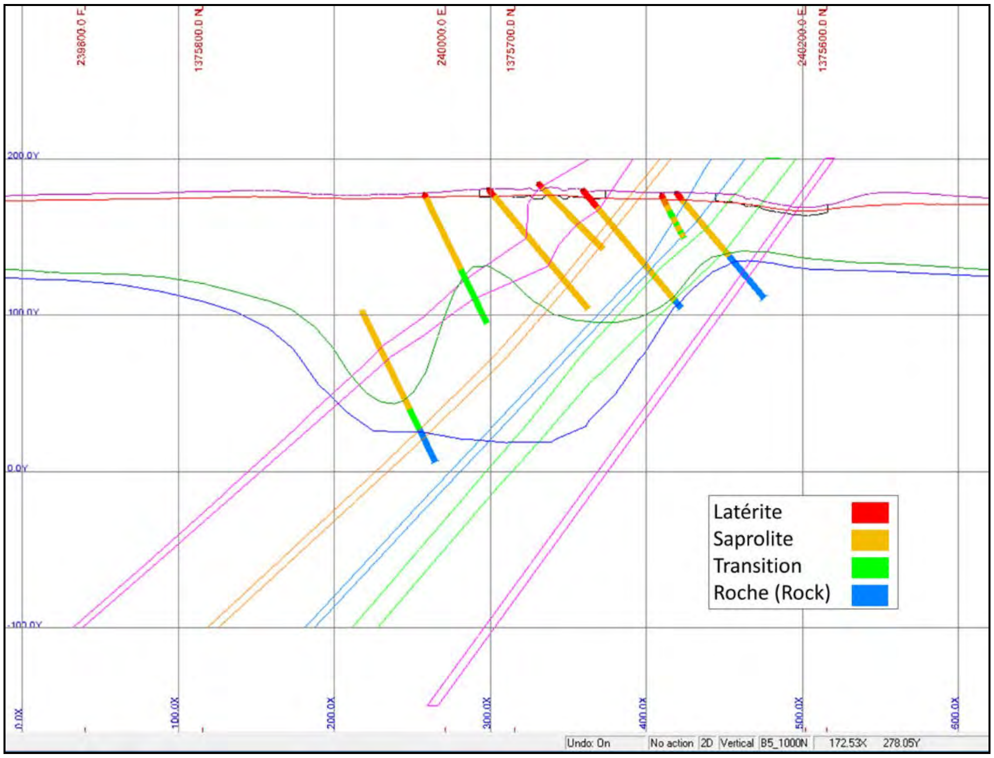
The alteration surfaces were modelled by IAMGOLD geologists and were updated based on the most recent drill information. The alteration surfaces distinguish the zones of tropical weathering for laterite, saprolite, transition (or saprock), and fresh rock for purposes of assigning density and or the optimization of the constraining shell. The laterite zone is generally barren of gold mineralization and is excluded from mineral resources. Each block is coded into the block model based on a 50/50 rule.

Table 14.16 lists the alteration codes for the resources model. Figure 14.16 presents a cross-section, B5_1000N, of the alteration zones for Boto 5.

Table 14.16 List of Alteration Zones and Codes for Boto 5

Alteration Zone	Rock Code
Laterite	40
Saprolite	50
Transition (or Saprock)	60
Rock	70

Figure 14.16 Cross-section B5_1000N of the Alteration (Weather) Model for the Boto 5; looking 028.5°Az northeast



Note: the grid is 100 m by 100 m (AGP May 2018)

14.3.2 Statistical Analysis

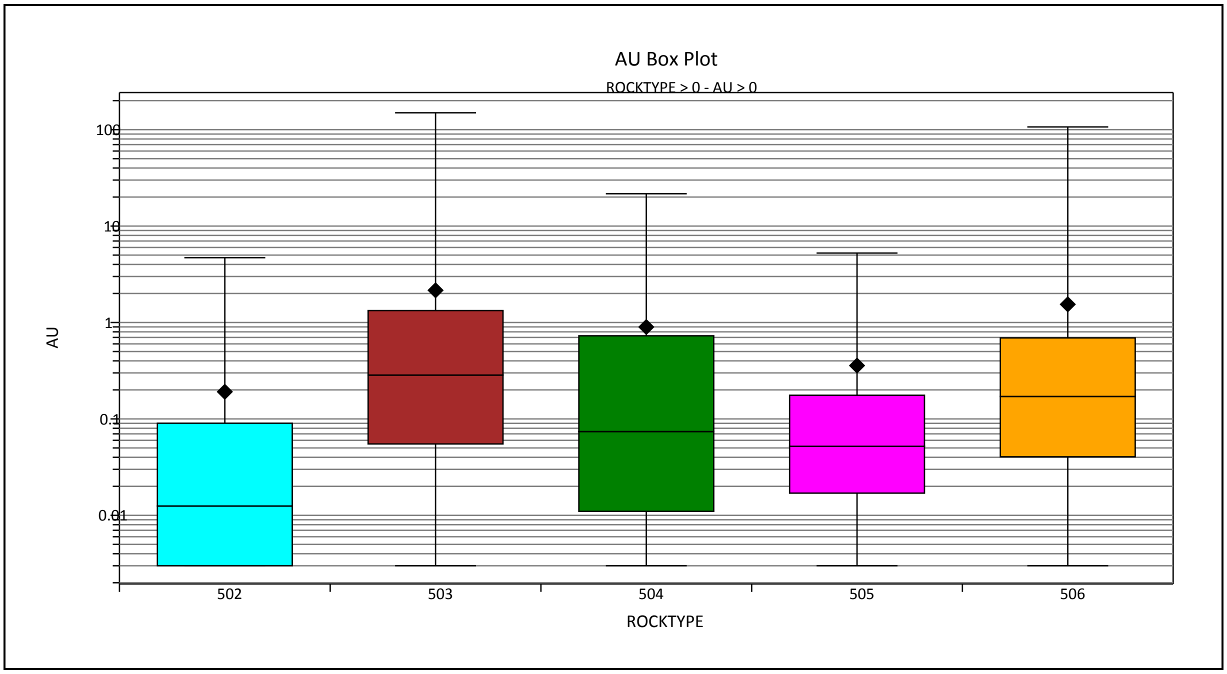
Raw Assays

A total of 1,487 assay values were used for the resource estimate for Boto 5. The descriptive statistics for the raw assays are presented in Table 14.17. Figure 14.17 shows the box plots for the raw assays, by mineralized zone.

Table 14.17 Descriptive Statistics for Boto 5 Deposit by Mineralized Zones (no zeroes)

Zone	502 g/t Au	503 g/t Au	504 g/t Au	505 g/t Au	506 g/t Au	Length (m)
Count	77	592	270	166	382	1487
Min	0.003	0.003	0.003	0.003	0.003	0.5
Max	4.70	149.59	21.64	5.25	106.69	2.00
Mean	0.19	2.16	0.90	0.36	1.54	1.02
Std Dev	0.61	8.08	2.36	0.92	6.61	0.17
CV	3.21	3.74	2.62	2.58	4.28	0.17

Figure 14.17 Box Plots for Raw Assay Values for Boto 5 by Mineralized Zone



Capping Analysis

To reduce the influence of outliers on the average grade and the coefficient of variation (CV) of the sample populations, a capping analysis in the form of decile analyzes, disintegration analyzes, histogram, and cumulative probability plots were used to assess the sample populations of gold grade outliers within each of the mineralized zones for Boto 5. Table 14.18 shows the capping levels and number of values affected for each mineralized zone. Table 14.19 shows descriptive statistics for the capped assay values by mineralized zone.

Table 14.18 List of Capping Levels by Mineralized Zone

Deposit	Rock Code	Capping Level (g/t Au)	N° of Values affected	% of Total Number of Values affected
Boto 5	502	1.71	2	2.6 %
	503	32.33	3	0.5 %
	504	10.73	2	0.7 %
	505	3.315	5	3.0 %
	506	26.013	2	0.5 %

Table 14.19 Descriptive Statistics for Capped Values for Boto 5 by Mineralized Zone

	502 g/t Au	503 g/t Au	504 g/t Au	505 g/t Au	506 g/t Au
Count	77	592	270	166	382
Minimum	0.003	0.003	0.003	0.003	0.003
Maximum	1.71	32.33	10.73	3.315	26.013
Mean	0.15	1.86	0.84	0.35	1.32
Std Dev	0.35	4.50	1.95	0.90	3.94
CV	2.37	2.42	2.33	2.55	2.99

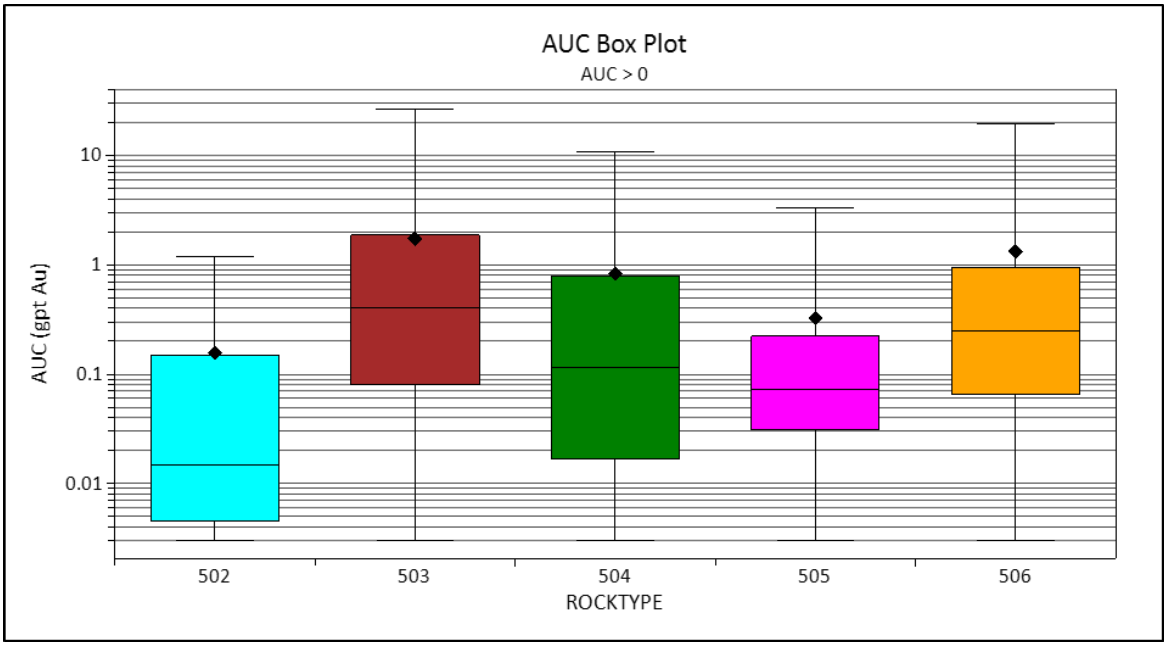
Composites

A nominal composite length of 2 m was selected which was at or above the 75th percentile of the raw assay sampling length. Composites were generated downward from the collar of the hole within the mineralized wireframes. Composites length were adjusted to avoid un-representative remnants at the wireframe boundaries where the minimum composition length is 1 m. Capping was applied to the raw assays prior to compositing. Table 14.20 shows the descriptive statistics of the capped 2 m composite values by mineralized zone. Figure 14.18 presents the box plot of the capped 2 m composite values by mineralized zone.

Table 14.20 Descriptive Statistics of the Capped 2 m Composites by Mineralized Zone

	502 g/t Au	503 g/t Au	504 g/t Au	505 g/t Au	506 g/t Au	Length (m)
Count	34	290	156	88	215	783
Minimum	0.003	0.003	0.003	0.003	0.000	1.00
Maximum	1.20	26.47	10.73	3.32	19.44	3.00
Mean	0.16	1.74	0.78	0.30	1.17	1.98
Std Dev	0.29	3.36	1.67	0.67	2.85	0.17
CV	1.80	1.93	2.14	2.22	2.43	0.09

Figure 14.18 Box Plots for Capped 2 m Composite Values for Boto 5 by Mineralized Zone



14.3.3 Block Model

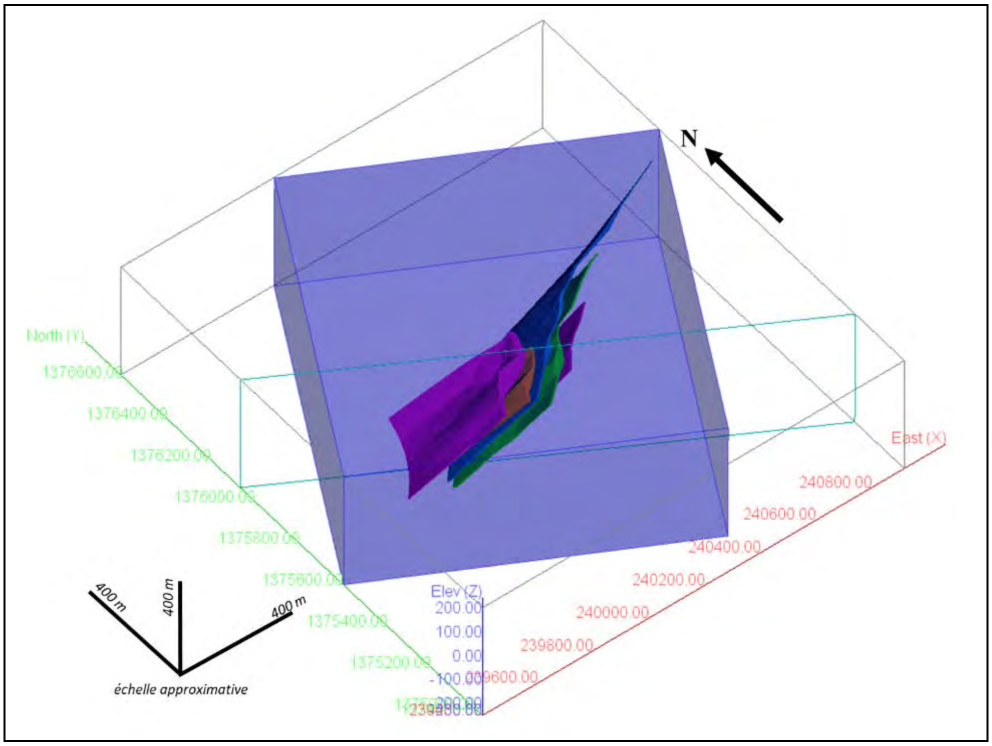
The block model for the Boto 5 deposit was set up with a block matrix of 5 m long by 5 m long by 5 m high. The block model is rotated -28.5° (clockwise), with the elongated blocks aligned to the preferred mainly north-south strike of the deposit. The block matrix was selected as appropriate for the 50 m x 50 m drill pattern and the block height was selected in consideration of a medium sized open pit operation. Table 14.21 summarizes the block model parameters and Figure 14.19 presents the block model over the interpreted mineralized zones for the Boto 5 deposit.

Table 14.21 Block Model Parameters for the Boto 5 Deposit

	Parameters
Easting	239 400 E
Northing	1 375 600 N
Maximum Elevation	225 m
Rotation Angle*	-28.5°
Block Size (X, Y, Z in metres)	5 x 5 x 5
Number of blocks in the X direction	220
Number of blocks in the Y direction	230
Number of blocks in the Z direction	90

**GEMS convention: negative is clockwise*

Figure 14.19 Block Model for the Boto 5 Deposit; perspective view looking northeast



(AGP May 2018)

Grade Interpolation

The block model for Boto 5 was interpolated in two passes. The gold grades were estimated using the 2 m composites, using the ID^3 interpolation method (anisotropic). The block model was also interpolated using ID^2 and NN interpolation methods for validation purposes. Table 14.22 shows estimation parameters for each pass used to estimate gold grades.

Table 14.22 Estimation Parameters for the Boto 5 Block Model

	Min N° composites	Max N° composites	Max N° composites per Drill Hole	Min N° of Drill Holes
Pass 1	4	12	3	2
Pass 2	3	12	3	1

Search Ellipses

The search ellipses used for the Malikoundi/Boto 2 block model interpolation used similar ranges as in the previous resource estimates in consideration of the regular 50 m x 50 m drill pattern and their intersections of the mineralized zones. Search ellipses were oriented in alignment with the interpreted mineralized zones. MZ04 in the northeast has a different alignment than in the southwest and separate interpolation was made for selected blocks in this zone.

Table 14.23 lists the search ellipse parameters used to estimate the Boto 5 deposit.

Table 14.23 Search Ellipse Parameters for Pass 1 and Pass 2 for Boto 5

Profile Name	Search Anisotropy	Z (°)	X (°)	Z (°)	X Range (m)	Y Range (m)	Z Range (m)	Search Type
Pass 1								
B502_P1	ZXZ	72	-52	0	60.0	60.0	10.0	Ellipsoidal
B503_P1	ZXZ	66	-52	0	60.0	60.0	10.0	Ellipsoidal
B504_P1	ZXZ	70	-45.7	0	60.0	60.0	10.0	Ellipsoidal
B505_P1	ZXZ	70	-45.7	0	60.0	60.0	10.0	Ellipsoidal
B5NE_P1	ZXZ	50	-60	0	60.0	60.0	10.0	Ellipsoidal
B506_P1	ZXZ	70	-45.7	0	60.0	60.0	10.0	Ellipsoidal
Pass 2								
B502_P1	ZXZ	72	-52	0	120.0	120.0	20.0	Ellipsoidal
B503_P1	ZXZ	66	-52	0	120.0	120.0	20.0	Ellipsoidal
B504_P1	ZXZ	65	-45.7	0	120.0	120.0	20.0	Ellipsoidal
B505_P1	ZXZ	70	-45.7	0	120.0	120.0	20.0	Ellipsoidal
B5NE_P1	ZXZ	50	-60	0	120.0	120.0	20.0	Ellipsoidal
B506_P1	ZXZ	70	-45.7	0	120.0	120.0	20.0	Ellipsoidal

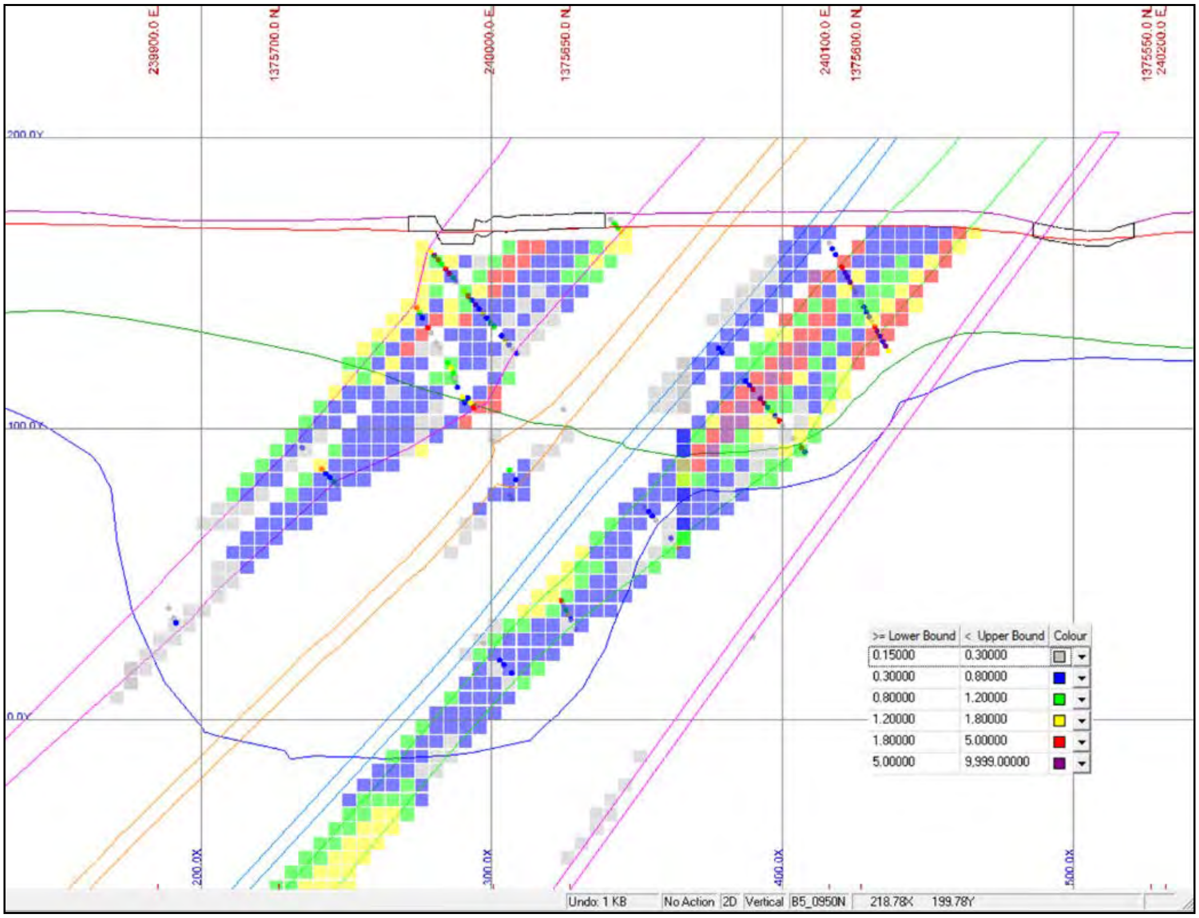
14.3.4 Validation

The Boto 5 block model and resource estimate was validated by the methods described below. AGP is satisfied that the block model gold grades reflect the gold grades from the drill core samples.

Visual Validation

The block model was validated by visually inspecting the block model results on section, to compare with the drill hole composite data. The grades of the blocks by section agreed well with the composite data used in the interpolation. Figure 14.20 presents a selected cross-section (B5_0950E) that shows the gold grades in blocks with the gold grades from the 2 m composites.

Figure 14.20 Cross-section B5_0950E Comparing Block Grades with the 2m Composites for the Boto 5 Deposit; looking 028.5°Az northeast



Note: Grid is 100 m (AGP May 2018)

Statistics

AGP reviewed the statistics for the mineralized zones and sub-domains for Boto 5 and generally found no bias between the different interpolation methods and the 2 m composites. It is noted that the de-clustered mean (from NN), for the MZ03 and MZ06, is closer to interpolated grades than the overall mean of the 2 m composites, in part, due to the lower sample density in these mineralized zones. Table 14.24 shows the average gold grades (no zeroes) of the mineralized zone for the Boto 5 deposit.

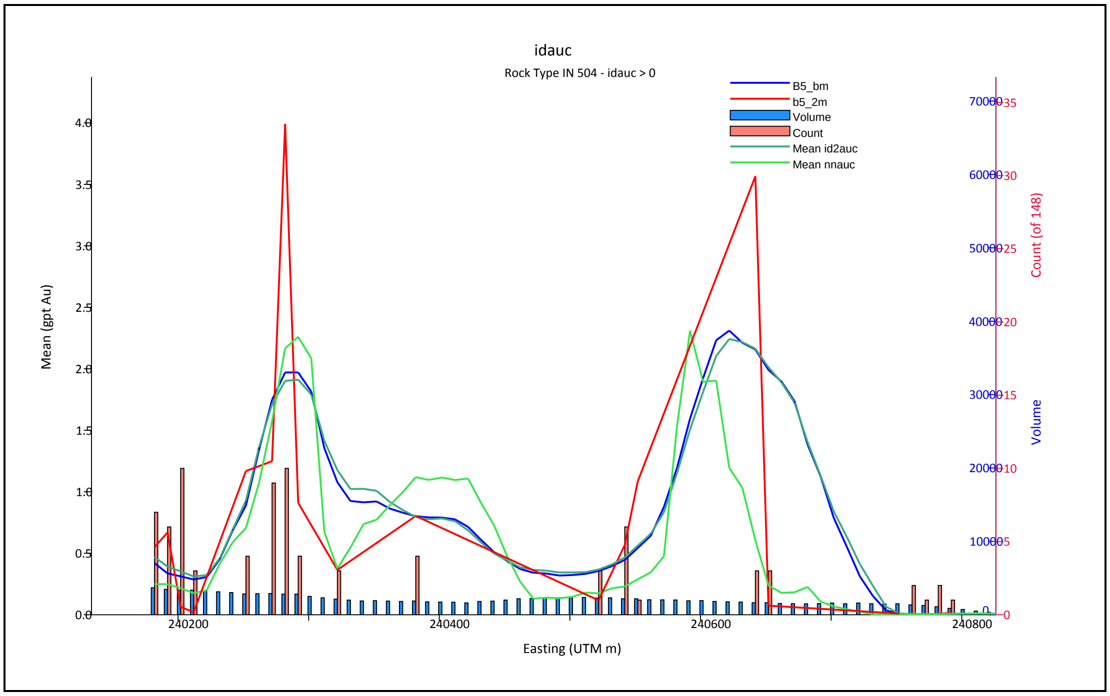
Table 14.24 Average Gold Grades (no zeroes) for the Boto 5 Deposit

Boto 5	502	503	504	505	506
Data	Average (g/t Au)	Average (g/t Au)	Average (g/t Au)	Average (g/t Au)	Average (g/t Au)
ID3	0.17	1.02	0.69	0.16	0.66
ID2	0.18	1.05	0.71	0.16	0.69
NN	0.17	0.92	0.50	0.16	0.57
2m Composites	0.16	1.74	0.83	0.33	1.32

Swath Plots

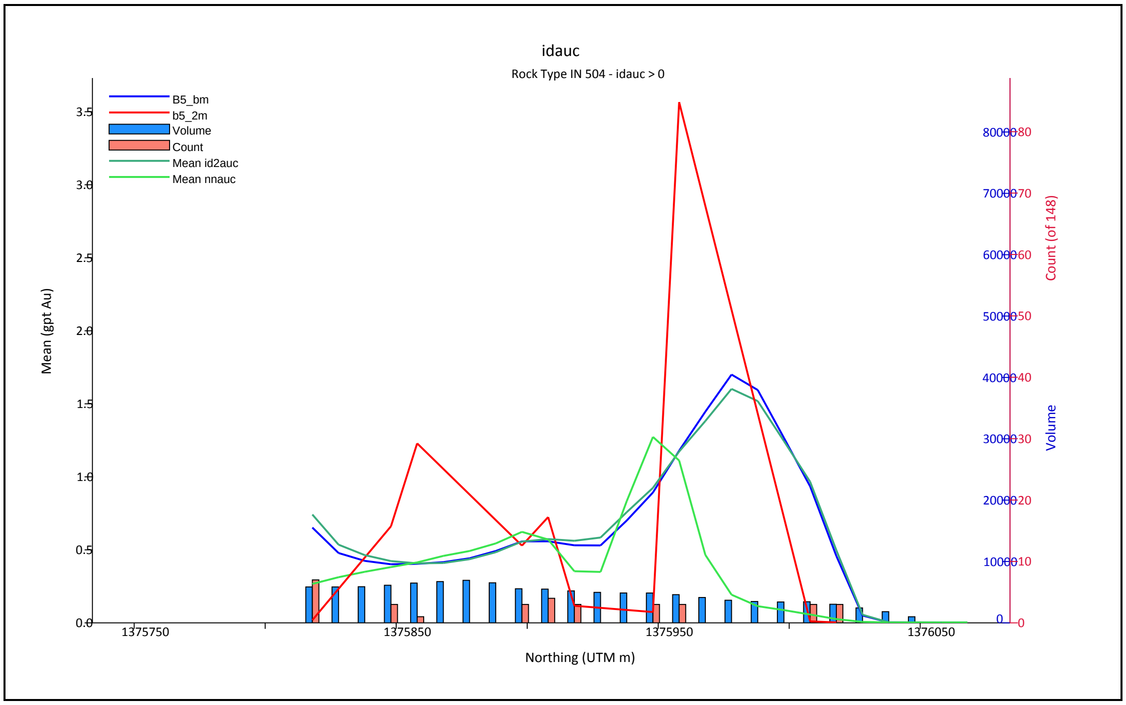
AGP interpolated gold grades using the ID^3, ID^2, and NN interpolation methods and compared the grades of blocks to the 2 m composites along a particular swath of the block model. The comparison shows no apparent bias. It has been noted that the NN plot is more skewed than the ID interpolation plots due to limited sample support. Figure 14.21, Figure 14.22, and Figure 14.23 show the swath plots for the selected mineralized zone, MZ04, in the Boto 5 deposit by Easting, Northing, and Elevation respectively.

Figure 14.21 Swath plots for Gold Grades by Easting for MZ04 in the Boto 5 Deposit



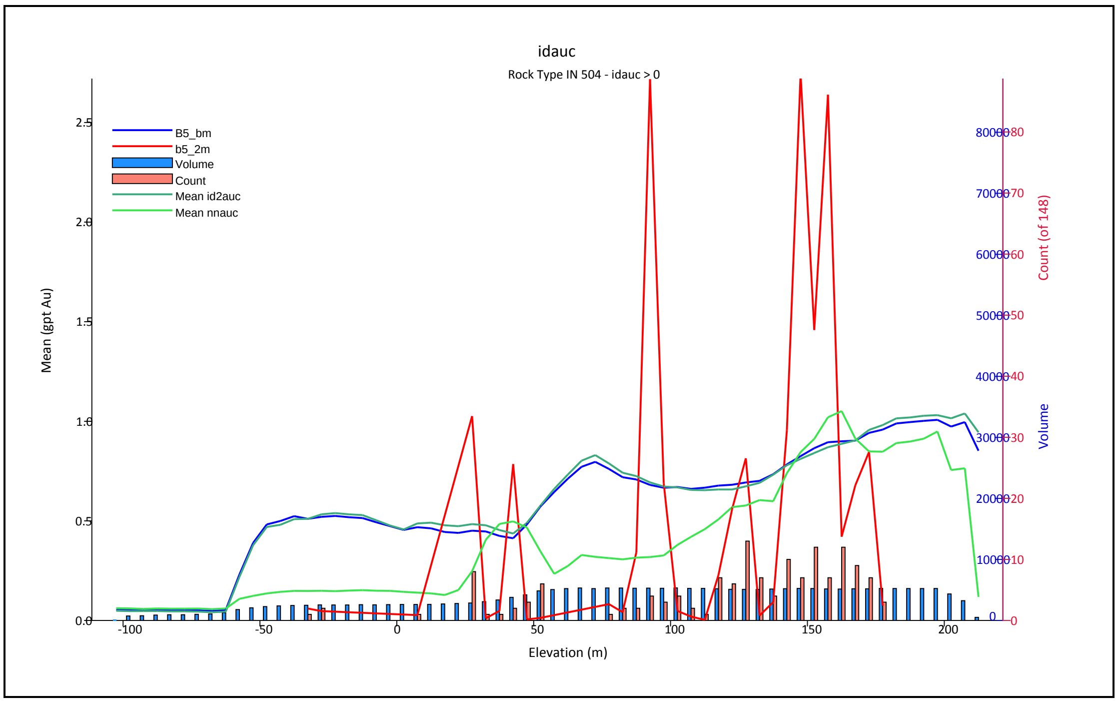
Note: B5_bm = ID3 capped gold grades

Figure 14.22 Swath plots for Gold Grades by Easting for MZ04 in the Boto 5 Deposit



Note: B5_bm = ID3 capped gold grades

Figure 14.23 Swath plots for Gold Grades by Easting for MZ04 in the Boto 5 Deposit



Note: B5_bm = ID3 capped gold grades

14.4 BOTO 6

14.4.1 Geological Interpretation

In order to carry out mineral resource estimation at the Boto 6 deposit, two mineralized zones were interpreted by IAMGOLD's on-site geologists. The interpreted mineralized wireframes were completed using 3D rings on vertical sections along the 50 m to 100 m spaced drill fences and matched for better continuity. The rings were snapped to the assay 'From's' and 'To's' and subsequently connected by tie lines to create a 3D solid wireframe.

In 2017, the interpreted mineralized zones were updated by IAMGOLD geologists based on most recent drill information. A minimum thickness of 3 m to 4 m was used in modelling the various mineralized zones. The interpreted mineralized envelopes were used to capture a minimum nominal grade of 0.15 g/t Au supported by geology, strong alteration, and a larger concentration of sulphide minerals, mainly pyrite. The statistical evaluation of the gold assays, capping, and compositing was carried out based on these interpreted mineralized zones.

The two 3D wireframes were considered by AGP as a good representation of the mineralized structures present and were accepted by AGP. No triangulation errors existed in any of the wireframes reviewed.

Figure 14.24 presents the two mineralized zones for the Boto 6 deposit. Table 14.25 lists the mineralized zones and rock codes.

Figure 14.24 Interpreted Mineralized Zones for Boto 6; perspective view looking northeast

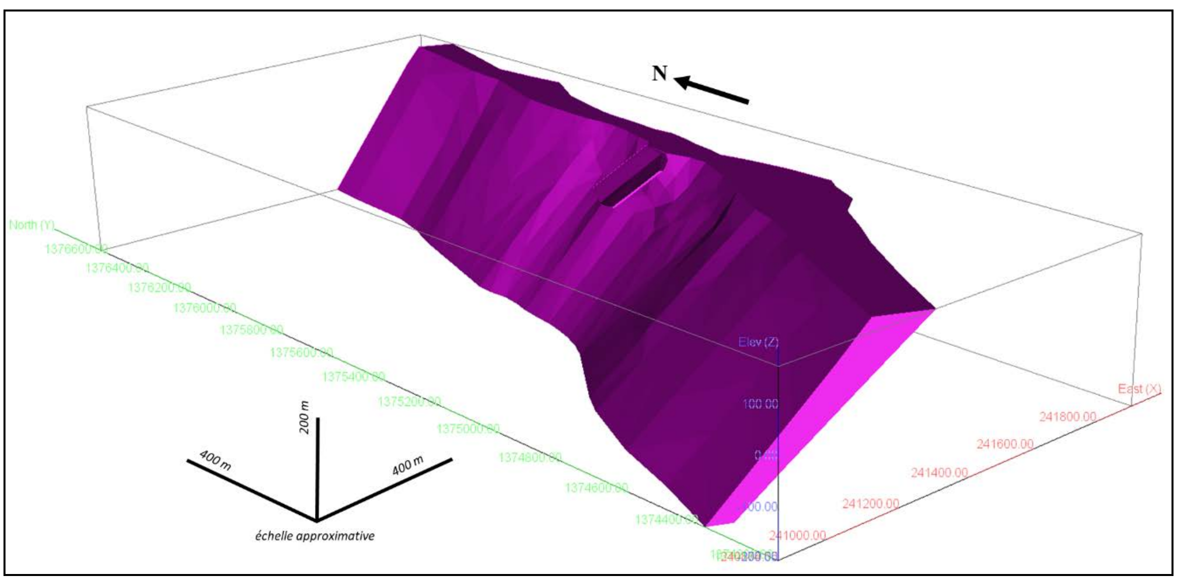


Table 14.25 List of Mineralized Zones and Rock Codes for Boto 6

Deposit	Volume (m^3)	Mineralized Zone	Rock Codes
Boto 6	186	MZ-W	601
	128,993	MZ-Main	602

Alteration Surfaces

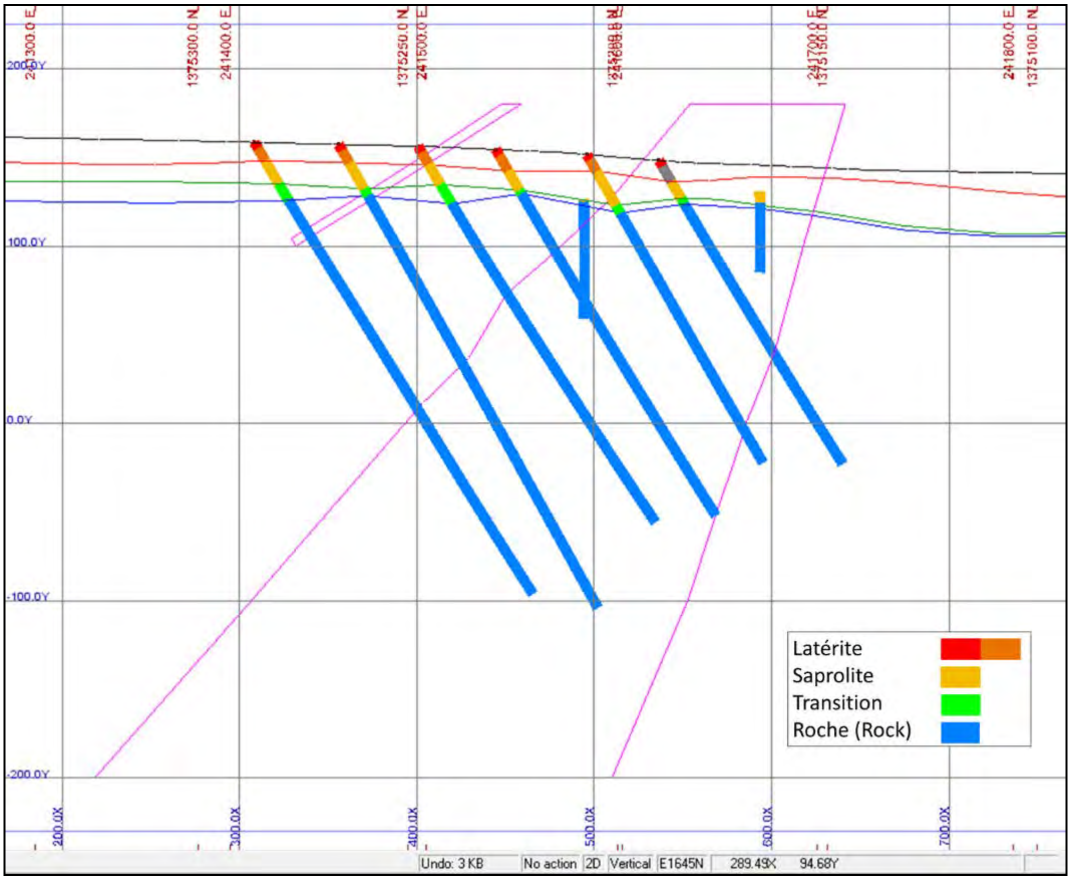
The alteration surfaces were modelled by IAMGOLD geologists and were updated based on the most recent drill information. The alteration surfaces distinguish the zones of tropical weathering for laterite, saprolite, transition (or saprock), and fresh rock for purposes of assigning density, and/or the optimization of the constraining shell. The laterite zone is generally barren of gold mineralization and is excluded from mineral resources. Each block is coded into the block model based on a 50/50 rule.

Table 14.26 lists the alteration codes for the resources model. Figure 14.25 presents a cross-section, E1645N, of the alteration zones for Boto 6.

Table 14.26 List of Alteration Zones and Codes for Boto 6

Alteration Zone	Code
Laterite	40
Saprolite	50
Transition (or Saprock)	60
Rock	70

Figure 14.25 Cross-section E1645N of the Alteration (Weather) Model for the for Boto 6; looking 025°Az northeast



Note: grid is 100 m by 100 m

14.4.2 Statistical Analysis

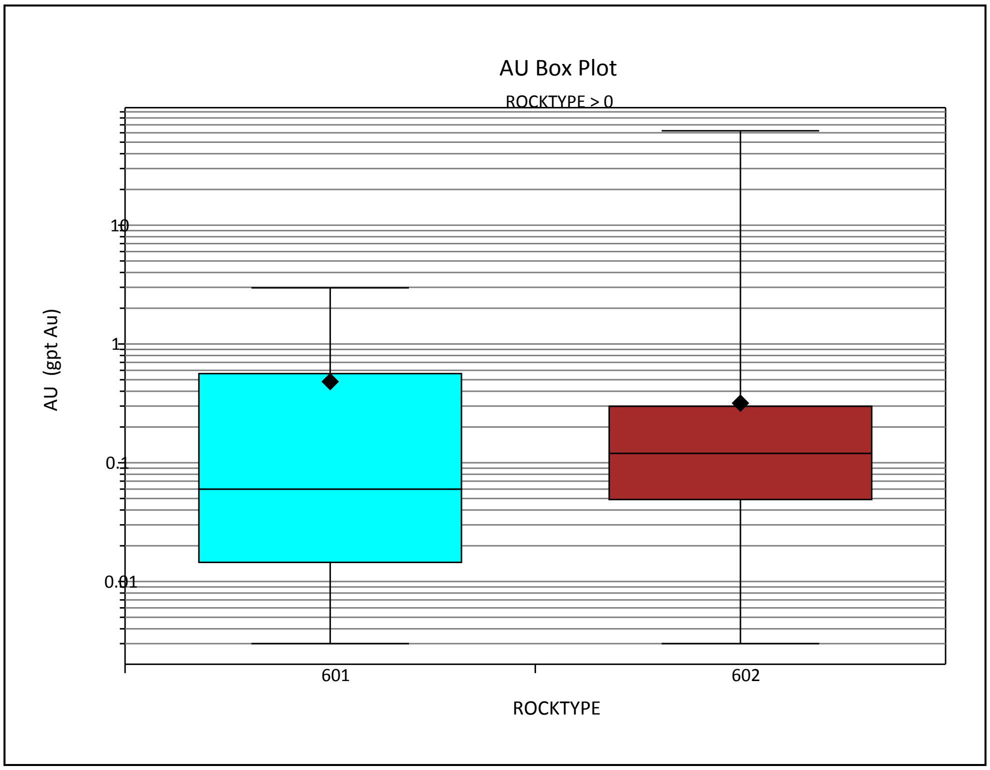
Raw Assays

A total of 9,710 assays values were used for the resource estimate for Boto 6. The descriptive statistics for the raw assays are presented in Table 14.27. Figure 14.26 shows the box plots for the raw assays by mineralized zone.

Table 14.27 Descriptive Statistics for Boto 6 Deposit by Mineralized Zones (no zeroes)

Zone	601 Au (g/t)	602 Au (g/t)	Length (m)
Count	46	9,664	9,710
Min	0.003	0.003	1.00
Max	2.97	62.38	3.00
Mean	0.48	0.32	1.13
Std Dev	0.78	1.16	0.34
CV	1.62	3.66	0.30

Figure 14.26 Box Plots for Raw Assays Values for Boto 6 by Mineralized Zone



Capping Analysis

To reduce the influence outliers on the average grade and the co-efficient of variation of the sample populations, a capping analysis in the form of decile analyses, disintegration analyses, histogram, and cumulative probability plots was used to assess the sample populations of gold grade outliers within each of the mineralized zones and sub-domains for Boto 6. Table 14.4.4 shows the capping levels and number of values affected for each mineralized zone. Table 14.28 shows descriptive statistics for the capped assay values by mineralized zone.

Table 14.28 List of Capping Levels by Mineralized Zone

Deposit	Rock Code	Capping Level (g/t Au)	Nº of Values affected	% of Total Number of Values affected
Boto 6	601	no capping	0	-
	602	10.00	7	0.1%

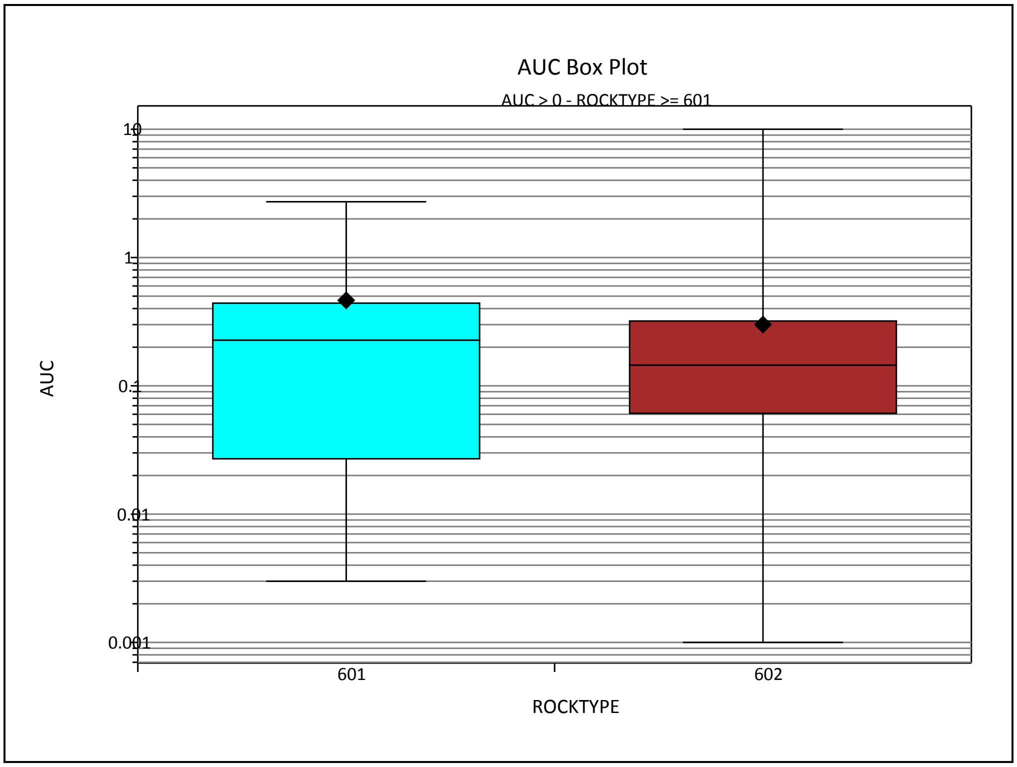
Composites

A nominal composite length of 2 m was selected which was at or above the 75th percentile of the raw assay sampling length. Composites were generated downward from the collar of the hole within the mineralized wireframes. Composites length were adjusted to avoid un-representative remnants at the wireframe boundaries where the minimum composition length is 1 m. Capping was applied to the raw assays prior to compositing. Table 14.29 shows the descriptive statistics of the capped 2 m composite values. Figure 14.27 presents the box plot of the capped 2 m composite values.

Table 14.29 Descriptive Statistics for the 2 m Capped Composites – Boto 6

	601	602	Length
	g/t Au	g/t Au	(m)
Count	24	5564	5588
Minimum	0.003	0.001	0.03
Maximum	2.72	10.00	2.50
Mean	0.46	0.30	2.00
Std. Deviation	0.67	1.54	0.05
CV	1.44	1.81	0.02

Figure 14.27 Box Plots for Capped 2 m Composite Values for Boto 6 by Mineralized Zone



14.4.3 Block Model

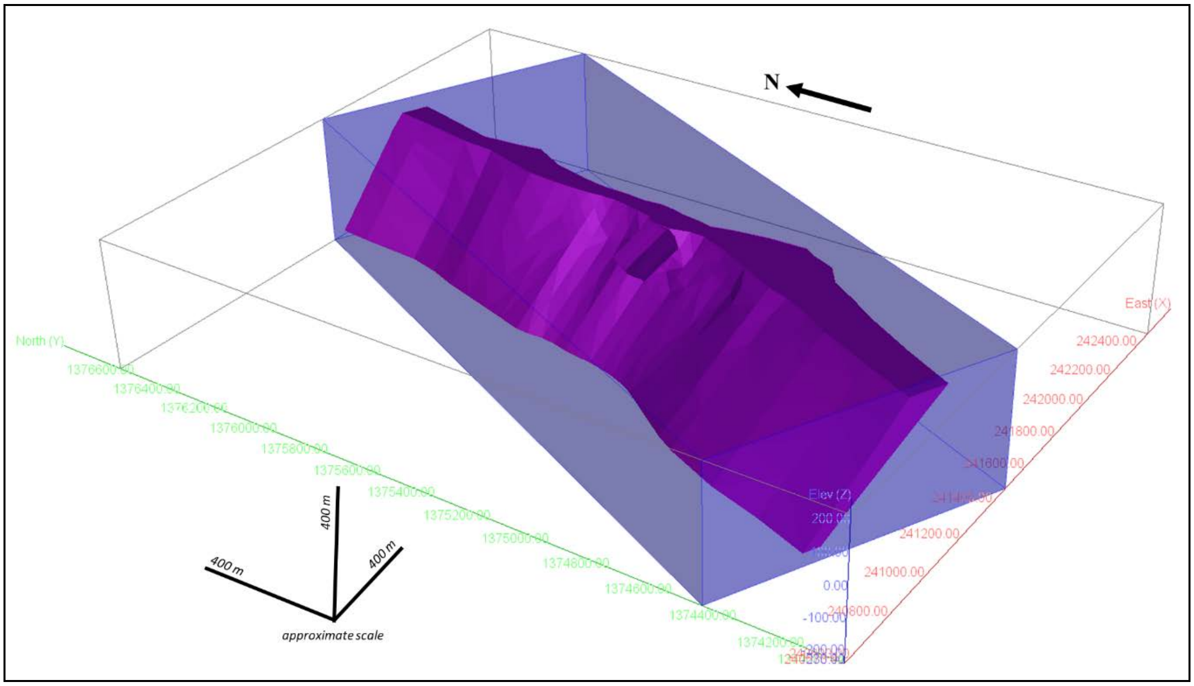
The block model for the Boto 6 deposit was set up with a block matrix of 5 m long by 5 m long by 5 m high. The block model is rotated -28.5° clockwise, with the elongated blocks aligned to the preferred mainly north-south strike of the deposit. The block matrix was selected as appropriate for the 50 m x 50 m drill pattern and the block height was selected in consideration of a medium sized open pit operation. Table 14.30 summarizes the block model parameters and Figure 14.28 presents the block model over the interpreted mineralized zones for the Boto 6 deposit.

Table 14.30 Block Model Parameters for the Boto 6 Deposit

	Parameters
Easting	240 573.423 E
Northing	1 374 505.655 N
Maximum Elevation	225 m
Rotation Angle*	25°
Block Size (X, Y, Z in metres)	5 x 5 x 5
Number of blocks in the X direction	200
Number of blocks in the Y direction	475
Number of blocks in the Z direction	91

**GEMS convention: negative is clockwise*

Figure 14.28 Block Model for the Boto 6 Deposit; perspective view looking northeast



Grade Interpolation

The block model for Boto 6 was interpolated in two passes. The gold grades were estimated using the 2 m composites using the ID^3 interpolation method (anisotropic). The block model was also interpolated using ID^2 and NN interpolation methods for validation purposes. Table 14.31 shows estimation parameters for each pass used to estimate gold grades.

Table 14.31 Estimation Parameters for the Boto 6 Block Model

	Min Nº composites	Max Nº composites	Max Nº composites per Drill Hole	Min Nº of Drill Holes
Pass 1	4	12	3	2
Pass 2	3	12	3	1

Search Ellipses

The search ellipses used for the Boto 6 block model interpolation used similar ranges as in the previous resource estimates in consideration of the regular 50 m x 50 m drill pattern and their intersections of the mineralized zones. Search ellipses were oriented in alignment with the interpreted mineralized zones. Within the mineralized zone, MZ-Main, the north and south portions used different search ellipse orientations in the interpolation passes for the selected blocks in this zone.

Table 14.32 lists the search ellipse parameters used to estimate the Boto 6 deposit.

Table 14.32 Search Ellipse Parameters for Pass 1 and Pass 2 for Boto 6

Profile Name	Search Anisotropy	Azimuth (°)	Dip (°)	Azimuth (°)	X Range (m)	Y Range (m)	Z Range (m)	Search Type
Pass 1								
B601_P1	Az,Dip,Az	295	-35	28	60.0	60.0	20.0	Ellipsoidal
B602N_P1	Az,Dip,Az	105	-55	0	60.0	60.0	20.0	Ellipsoidal
B602S_P1	Az,Dip,Az	320	-45	38	60.0	60.0	20.0	Ellipsoidal
Pass 2								
B601_P2	Az,Dip,Az	295	-35	28	120.0	120.0	20.0	Ellipsoidal
B602N_P2	Az,Dip,Az	105	-55	0	120.0	120.0	20.0	Ellipsoidal
B602S_P2	Az,Dip,Az	320	-45	38	120.0	120.0	20.0	Ellipsoidal

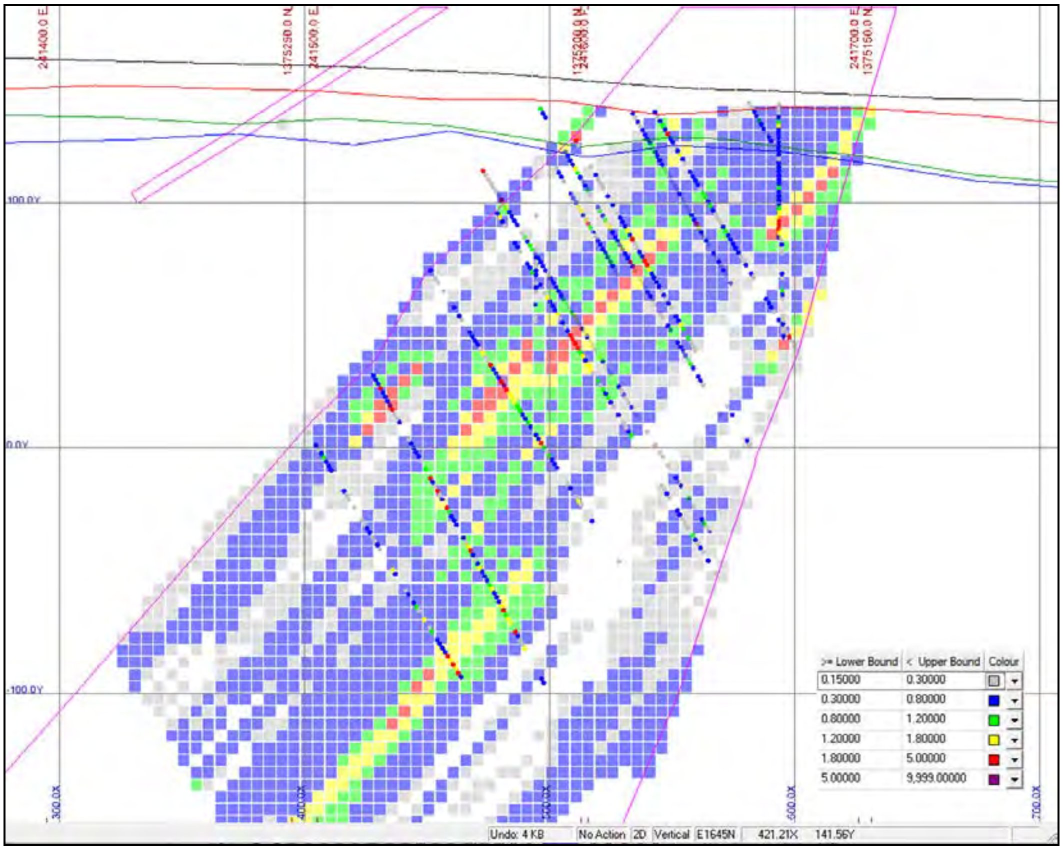
14.4.4 Validation

The Boto 6 block model and resource estimate was validated by the methods described below. AGP is satisfied that the block model gold grades reflect the gold grades from the drill core samples.

Visual Validation

The block model was validated by visually inspecting the block model results on section to compare with the drill hole composite data. The grades of the blocks by section agreed well with the composite data used in the interpolation. Figure 14.29 presents a selected cross-section (E1645N) that shows the gold grades in blocks with the gold grades from the 2 m composites.

Figure 14.29 Cross-section E1645N Comparing Block Grades with the 2m Composites for the Boto 6 Deposit; looking 025°Az northeast



Note: Grid is 100 m by 100 m

Statistics

AGP reviewed the statistics for the mineralized zones and sub-domains for Boto 6 and no bias was noted. Table 14.33 shows the average gold grades (no zeroes) of the mineralized zone for the Boto 6 deposit.

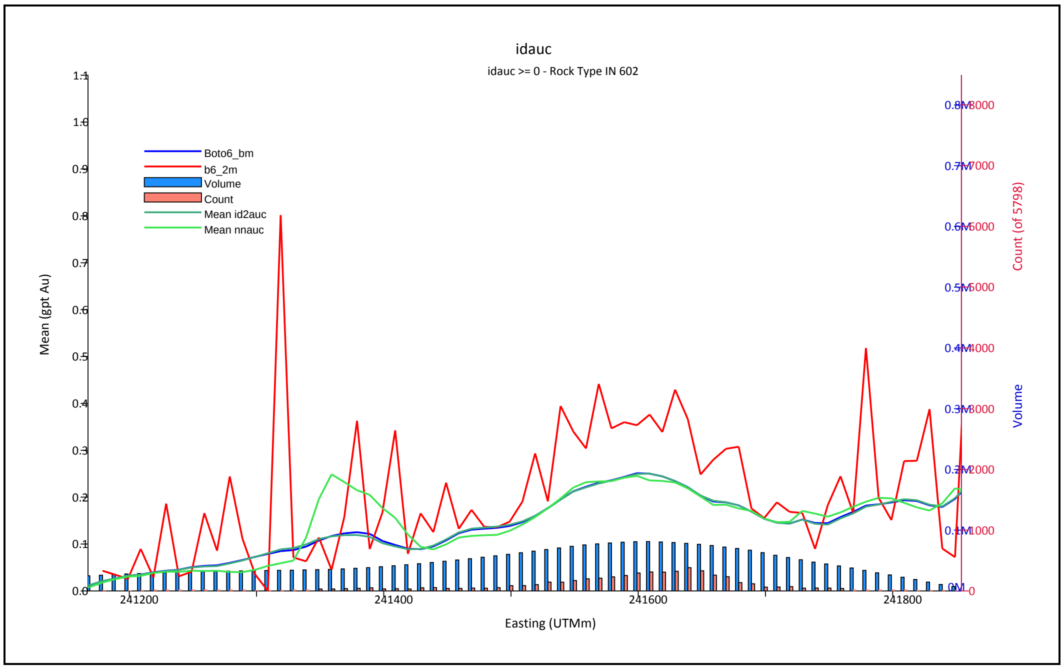
Table 14.33 Average Gold Grades (no zeroes) for the Boto 6 Deposit

Boto 6	601	602
Data	Average (g/t Au)	Average (g/t Au)
ID3	0.50	0.24
ID2	0.51	0.24
NN	0.46	0.26
2m Composites	0.46	0.30

Swath Plots

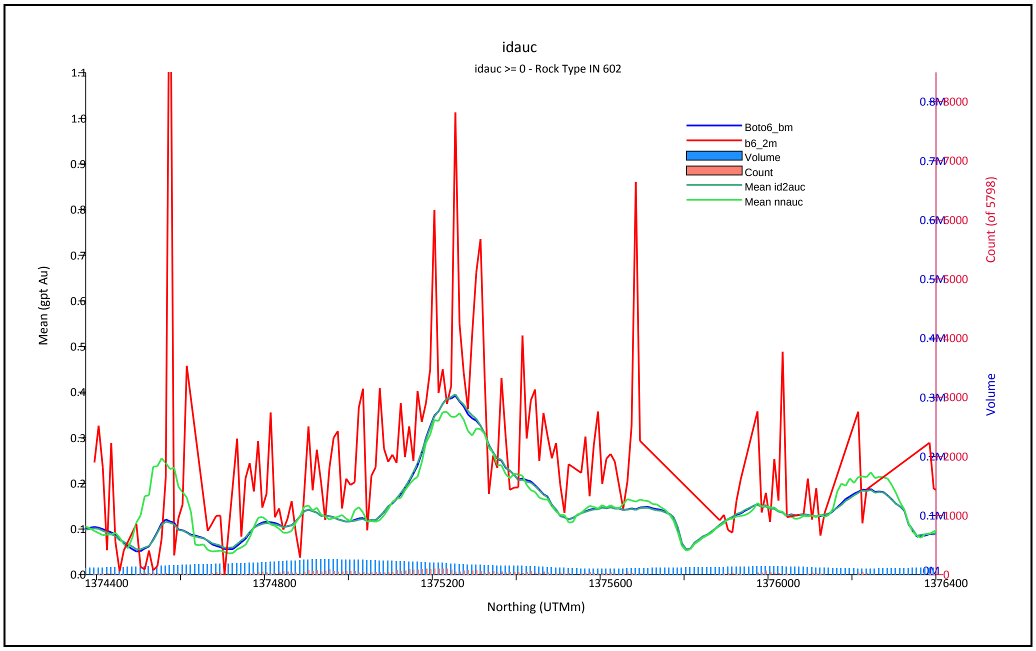
AGP interpolated gold grades using the ID^3, ID^2, and NN interpolation methods and compared the grades of blocks to the 2 m composites along a particular swath of the block model. The comparison shows no apparent bias. It is noted that the NN plot is more skewed than the ID interpolation plots in the south end of Boto 6 due to limited sample support in this area. Figure 14.30, Figure 14.31, and Figure 14.32 show the swath plots for the selected mineralized zone, MZ-Main, in the Boto 6 deposit by Easting, Northing and Elevation respectively.

Figure 14.30 Swath plots for Gold Grades by Easting for MZ-Main in the Boto 6 Deposit



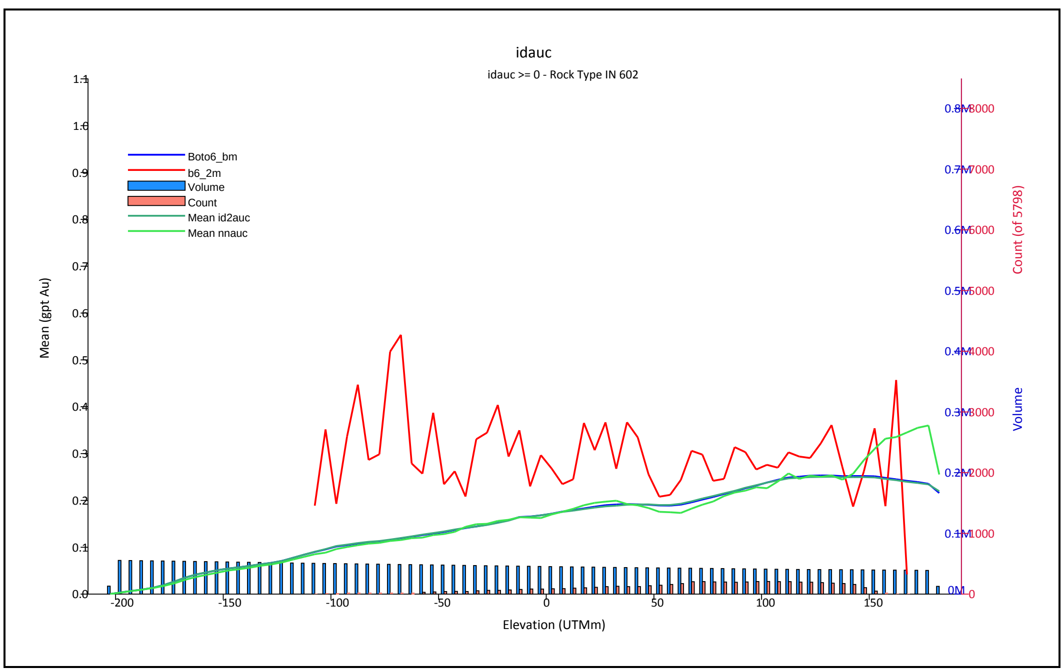
Note: Boto6_bm = ID3 capped gold grades

Figure 14.31 Swath plots for Gold Grades by Northing for MZ-Main in the Boto 6 Deposit



Note: Boto6_bm = ID3 capped gold grades

Figure 14.32 Swath plots for Gold Grades by Elevation for MZ-Main in the Boto 6 Deposit



14.5 BOTO 4

14.5.1 Geological Interpretation

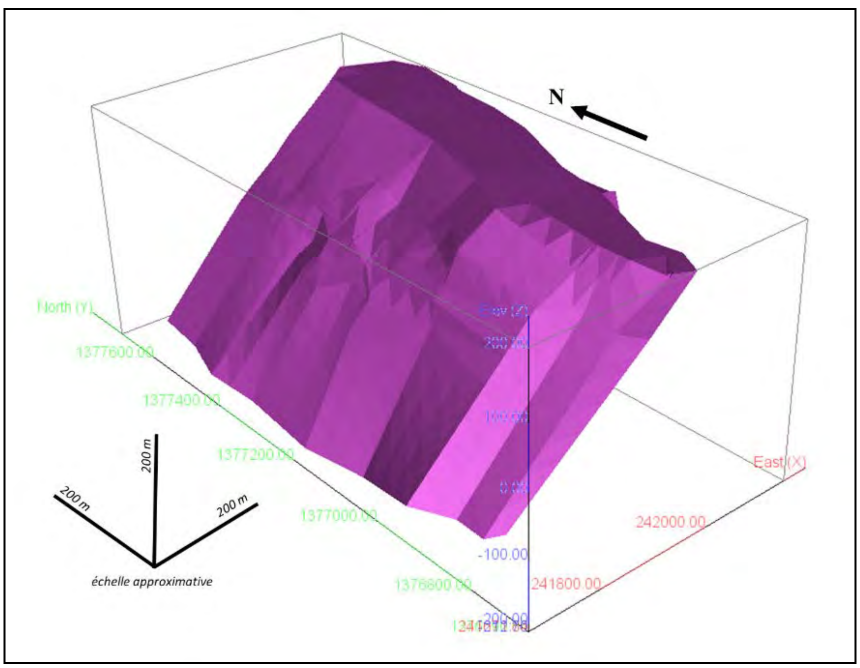
The Boto 4 deposit consists of a single interpreted mineralized zone. The 3D wireframe used to represent the Boto 4 deposit was interpreted for the June 2015 Resource Estimate and has not been subject to further drilling since. The 3D wireframe was considered by AGP as a good representation of the mineralized structure present and was accepted by AGP. No triangulation errors existed in any of the wireframes reviewed. Table 14.34 lists the Rock Code used for the wireframe. Figure 14.33 presents the interpreted mineralized wireframe for the Boto 4 deposit.

Table 14.34 List of Mineralized Zones and Rock Codes for Boto 4

Deposit	Volume (m^3)	Mineralized Zone	Rock Code
Boto 4	53,531	MZ-W	401

+

Figure 14.33 Interpreted Mineralized Wireframe for the Boto 4 Deposit; perspective view, looking northeast



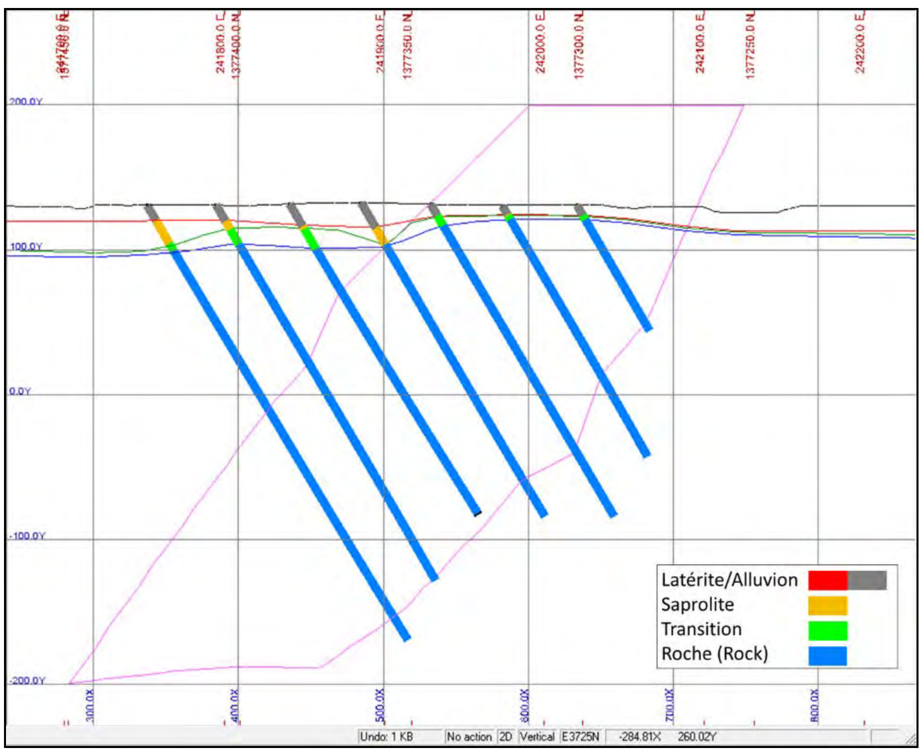
Alteration Surfaces

The alteration surfaces were modelled by IAMGOLD geologists to distinguish between the laterite, saprolite, transition (or saprock), and fresh rock. These surfaces were used to analyse and assign density to the block model and, in the optimization of the constraining shell. The laterite is generally barren are not considered as mineral resources. Table 14.35 shows the Alteration (Weather) Codes for the Boto 4 deposit. Figure 14.34 presents a cross-section as an example of the alteration profile for the Boto 4 deposit.

Table 14.35 List of Alteration Zones and Codes for Boto 4

Alteration Zone	Alteration Code (Weather Code)
Laterite	40
Saprolite	50
Transition (or Saprock)	60
Rock	70

Figure 14.34 Cross-section (E3725E) of the Alteration (Weather) Model for the Boto 4 Deposit; looking 025°Az northeast



Note: grid is 100 m by 100 m

14.5.2 Statistical Analysis

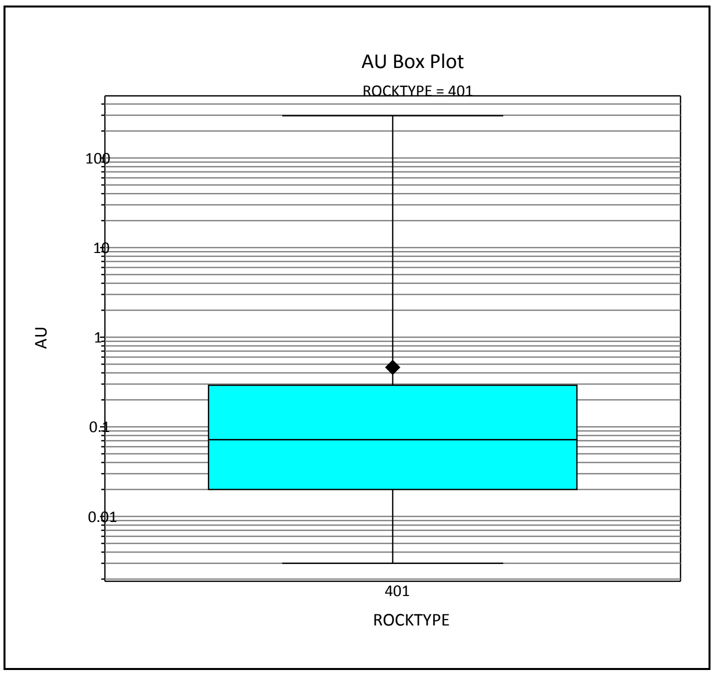
Raw Assays

A total of 7,112 assay values were used for the resource estimation for the Boto 4 deposit. The descriptive statistics for gold grades and sample lengths are shown in Table 14.36. Figure 14.35 shows the box plot for the raw assays values for Boto 4.

Table 14.36 Descriptive Statistics for the Boto 4 Deposit

	401	
	Au (g/t)	**Length (m)**
Count	7112	7112
Minimum	0.003	0.40
Maximum	296.00	3.00
Mean	0.46	1.08
Std. Deviation	3.83	0.28
CV	8.30	0.26

Figure 14.35 Box Plots for Raw Assays Values for Boto 4



Capping

A capping analysis in the form of decile analysis, degradation/disintegration analysis, histogram, and log-probability plots were used to assess the sample population of gold grades for the mineralized zone at Boto 4. Table 14.37 shows the capping level used for the Boto 4 deposit.

Table 14.37 Capping Level for the Boto 4 Deposit

Deposit	Mineralized Zone (Rock Code)	Capping Value (g/t Au)	Number of Values Affected	% of Total Number of Values Affected
Boto 4	401	16.00	8	0.1 %

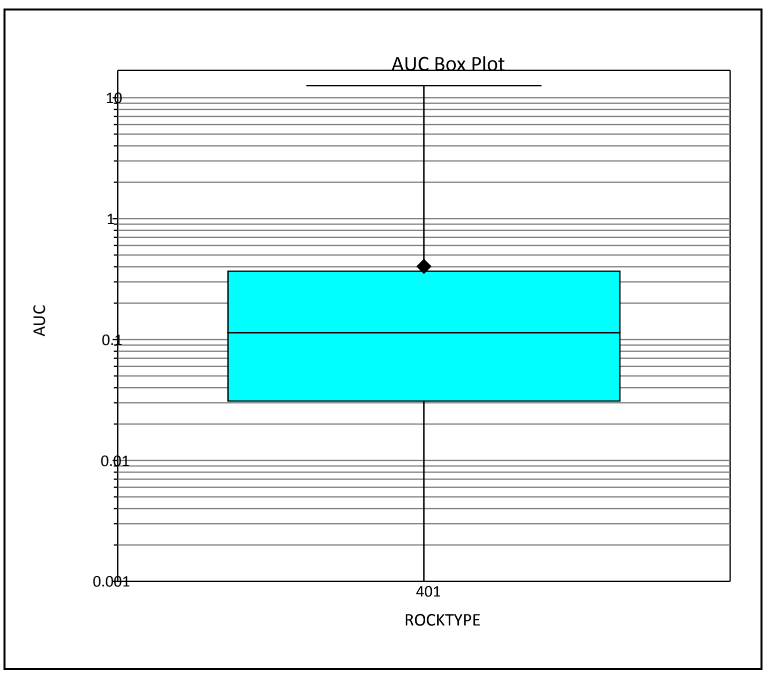
Composites

A nominal composite length of 2 m was selected which was at or above the 75th percentile of the raw assay sampling length. Composites were generated downward from the collar of the hole within the mineralized wireframes. Composites length were adjusted to avoid un-representative remnants at the wireframe boundaries where the minimum composition length is 1 m. Capping was applied to the raw assays prior to compositing. Table 14.38 shows the descriptive statistics of the capped composite values. Figure 14.36 shows the box plot for the capped composite values for Boto 4.

Table 14.38 Descriptive Statistics for Capped 2 m Composite Values for the Boto 4 Deposit

	401	
	Au (g/t)	**Length (m)**
Count	3823	3823
Minimum	0.001	0.99
Maximum	12.59	2.02
Mean	0.40	2.00
Std. Deviation	0.89	0.05
Variance	0.80	0.00
CV	2.22	0.02

Figure 14.36 Box Plots for Capped 2 m Composite Values for Boto 4



14.5.3 Block Model

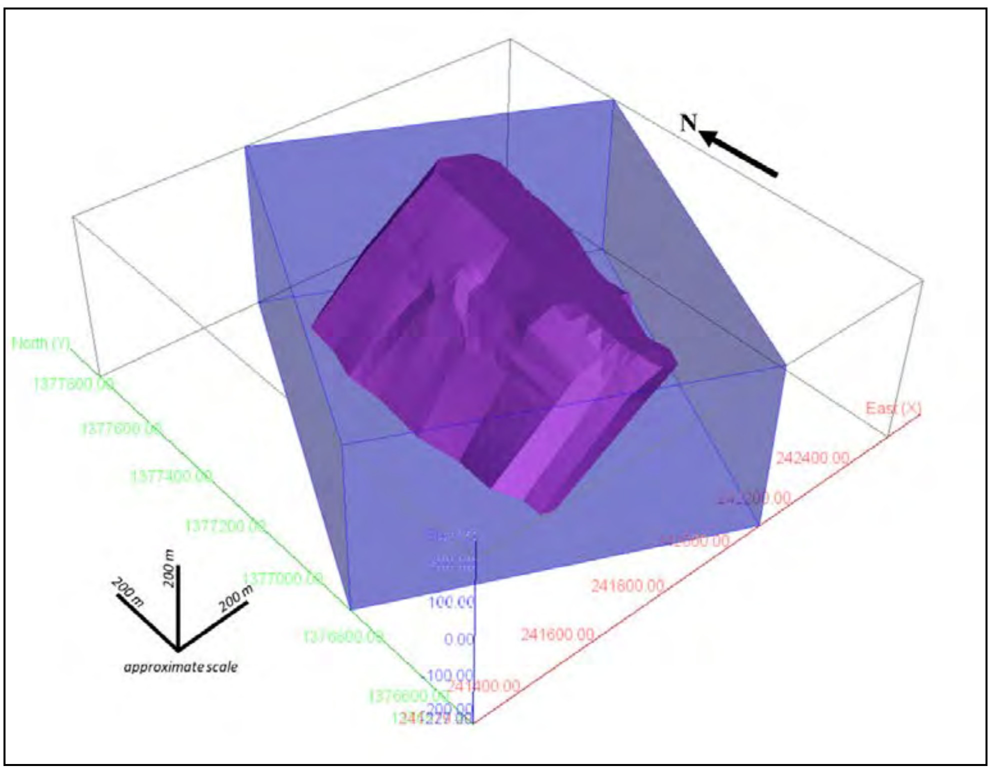
For the current resource estimation of Boto 4, a block matrix of 5 m length x 5 m width x 4 m height was used. The block model is rotated 25° clockwise in the orientation of the drill fences. The summary of the block model parameters is shown in Table 14.39. Figure 14.37 shows the block model over the interpreted mineralized zone at Boto 4.

Table 14.39 Block Model Parameters for the Boto 4 Deposit

	Parameters
Easting	241 279.287 E
Northing	1 376 909.779 N
Maximum Elevation	225 m
Rotation *	-25°
Block Size (X, Y, Z in metres)	5 x 5 x 4
Number of Blocks, X Direction	180
Number of Blocks, Y Direction	210
Number of Blocks, Z Direction	113

** GEMS Convention: negative = clockwise*

Figure 14.37 Boto 4 Block Model; perspective view looking northeast



Grade Interpolation

The block model was interpolated in two passes. The gold grades were estimated using the 2 m composites using ID^3 interpolation method (true). The block model was also interpolated using ID^2 and NN interpolation methods for validation purposes. Table 14.40 shows estimation parameters for each pass used to estimate gold grades.

Table 14.40 Estimation Parameters for the Boto 4 Block Model

	Min Nº composites	Max Nº composites	Max Nº composites per Drill Hole	Min Nº of Drill Holes
Pass 1	4	12	3	2
Pass 2	3	12	3	1

Search Ellipses

The search ellipses used for the Boto 4 block model interpolation used the same ranges as the Boto 4 deposit in consideration of the regular 50 m x 50 m drill pattern and their intersections of the mineralized zones. Search ellipses were aligned to the interpreted mineralized zone. Table 14.41 presents the search ellipse parameters for the Boto 4 deposit.

Table 14.41 Search Ellipse Parameters for Pass 1 and Pass 2 for the Boto 4 Deposit

Profile Name	Search Anisotropy	Z (°)	X (°)	Z (°)	X Range (m)	Y Range (m)	Z Range (m)	Search Type
Pass 1								
B4_P1	ZXZ	100	-55	0	60.0	60.0	15.0	Ellipsoidal
Pass 2								
B4_P2	ZXZ	100	-55	0	120.0	120.0	20.0	Ellipsoidal

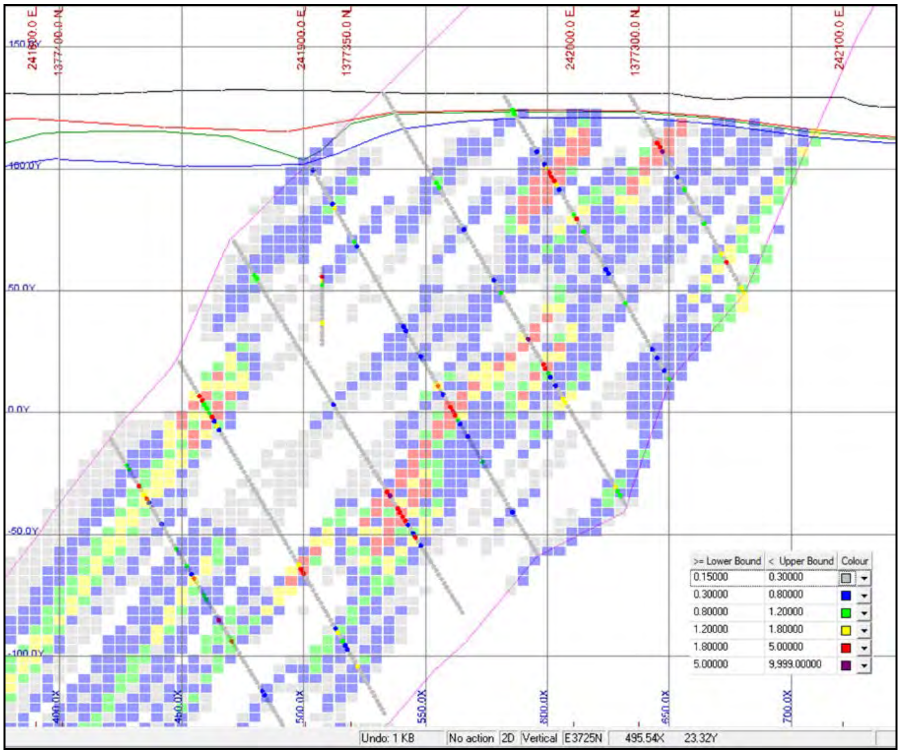
14.5.4 Validation

The Boto 4 block model and resource estimate was validated by the methods described below. AGP is satisfied the block model gold grades reflect the gold grades from the drill core samples.

Visual Validation

The block model was validated by visually inspecting the block model results on section to compare with the drill hole composite data. The grades of the blocks by section agreed well with the composite data used in the interpolation. Figure 14.38 presents a selected cross-section (E3725N) that shows the gold grades in blocks with the gold grades from the 2 m composites.

Figure 14.38 Cross-section E3725N Comparing Block Grades with the 2m Composites for the Boto 4 Deposit; looking 025° Az northeast



Note: grid is 100 m by 100 m

Statistics

AGP reviewed the statistics for the mineralized zone for Boto 4 and found no bias between the different interpolation methods and the 2 m composites. Table 14.42 shows the average gold grades (no zeroes) of the mineralized zone for the Boto 4 deposit.

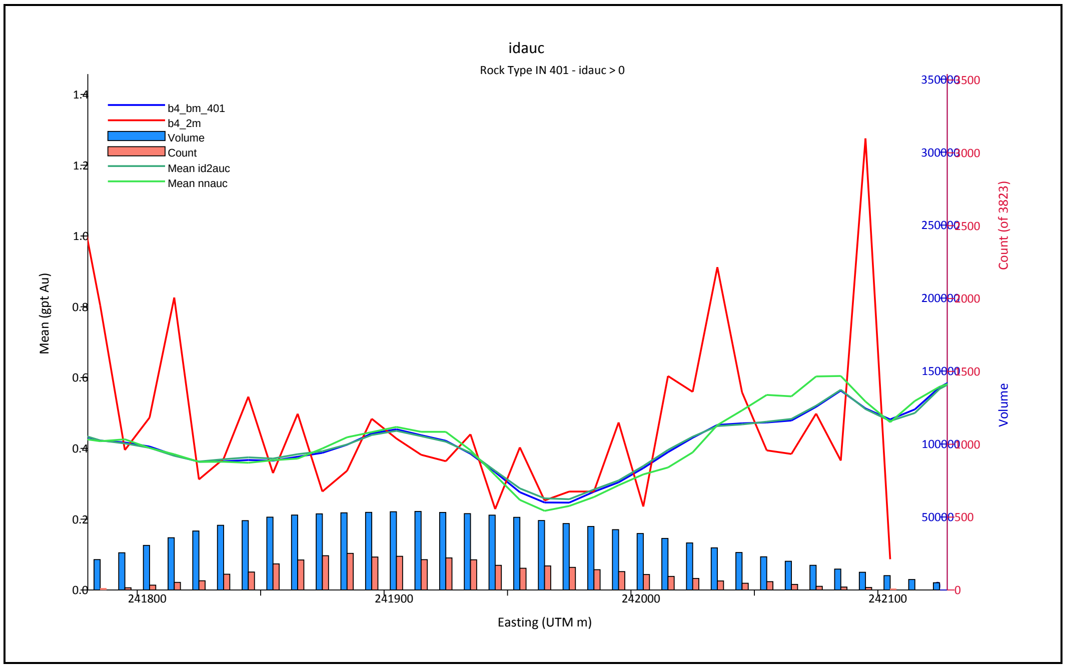
Table 14.42 Average Gold Grades (no zeroes) for the Boto 4 Deposit

	Boto 4
Method	**Average (g/t Au)**
ID3	0.39
ID2	0.39
NN	0.39
2m Composites	0.40

Swath Plots

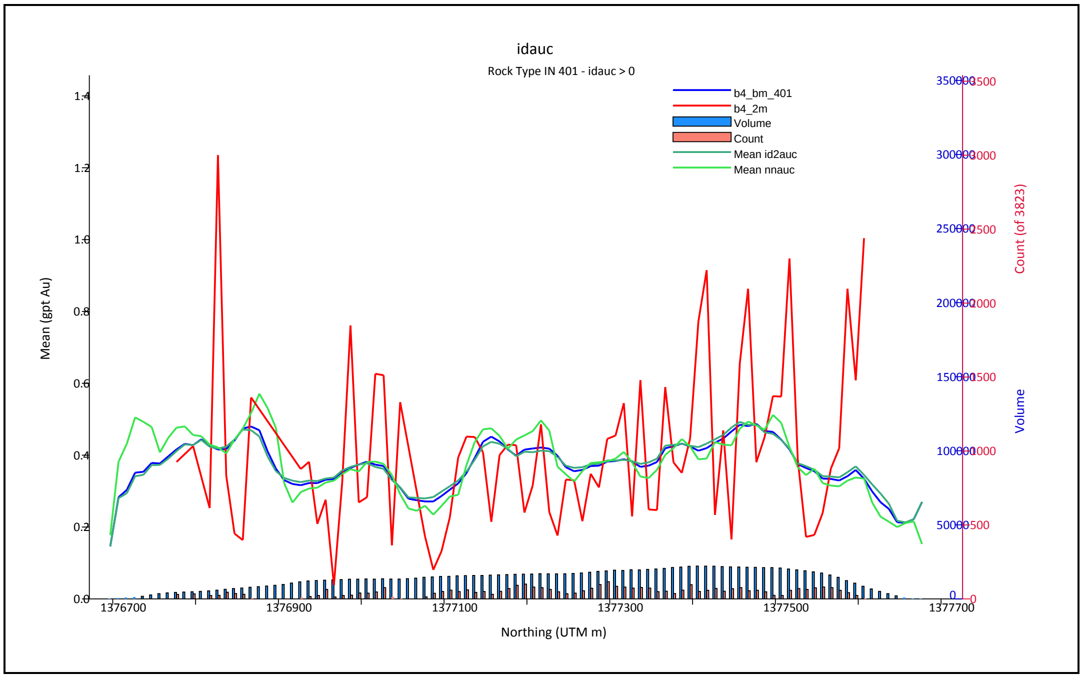
AGP interpolated gold grades using the ID^3, ID^2, and NN interpolation methods and compared the grades of blocks to the 2 m composites along a particular swath of the block model. The comparison shows no apparent bias. Figure 14.39, Figure 14.40, and Figure 14.41 show the swath plots for the interpreted mineralized zone by Easting, Northing, and Elevation respectively.

Figure 14.39 Swath plots for Gold Grades by Easting for Boto 4



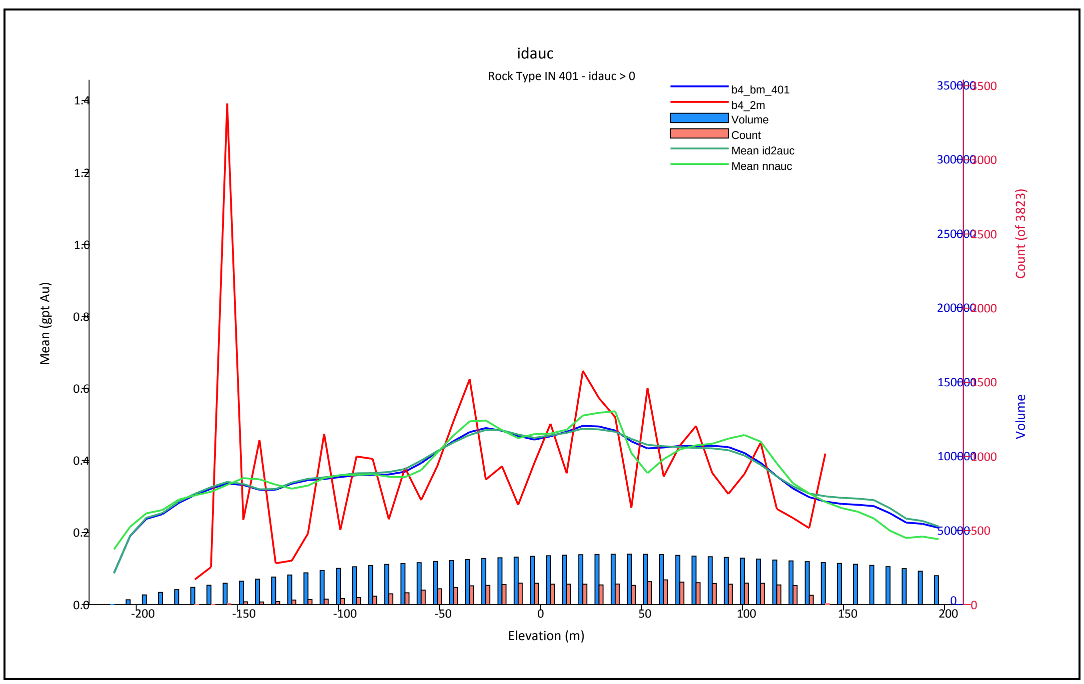
Note: B4-bm = ID3 capped gold grades

Figure 14.40 Swath plots for Gold Grades by Northing for Boto 4



Note: b4_bm_401 = ID3 capped gold grades

Figure 14.41 Swath plots for Gold Grades by Elevation for Boto 4



Note: b4_bm_401 = ID3 capped gold grades

14.6 MINERAL RESOURCES

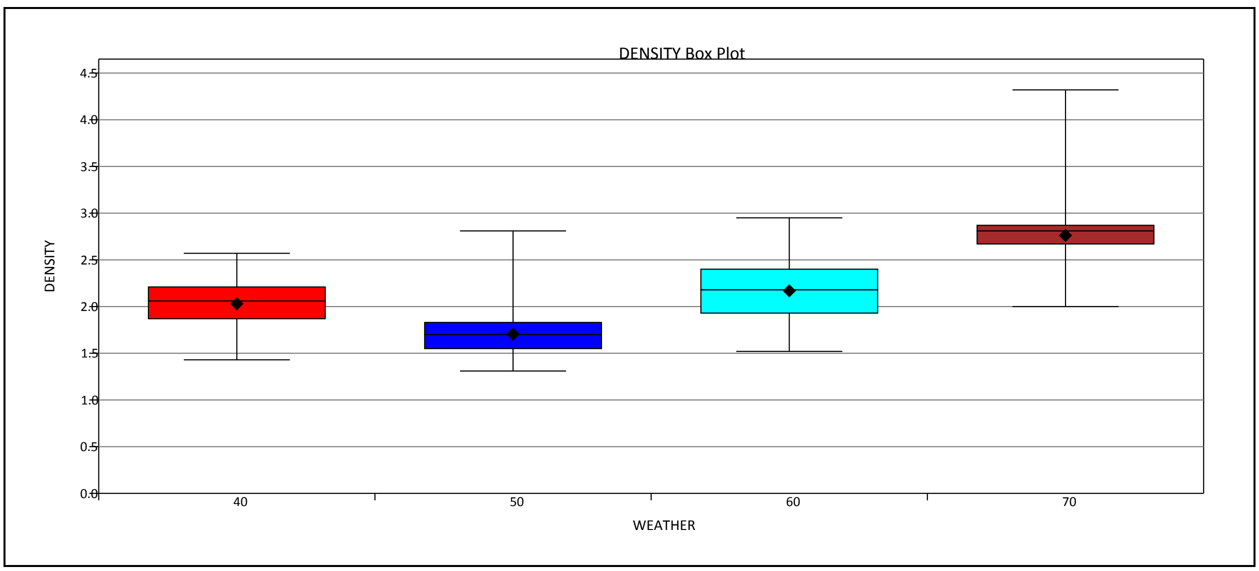
14.6.1 Density

Approximately 3,400 measurements were collected for specific gravity from the Project between the Malikoundi/Boto 2, Boto 6, and Boto 4 deposits. The majority of measurements were collected from the Malikoundi deposit. A total of 3,282 measurements were used to assign the density values to the block models by alteration zone (Weather Code). Density values were attributed to the block models based on alteration zone (Weather Code) using the 50/50 rule. Table 14.43 presents the descriptive statistics for density values by alteration zone. Figure 14.42 presents the box plot for the density values by alteration zone.

Table 14.43 Descriptive Statistics for Density Values by Alteration Zone

Alteration Zone	Laterite [40]	Saprolite [50]	Transition [60]	Rock [70]
Count	279	366	343	2294
Minimum	1.43	1.31	1.52	2.01
Maximum	2.57	2.81	2.95	4.32
Mean	2.03	1.70	2.17	2.76
Std Dev	0.23	0.22	0.33	0.18
CV	0.11	0.13	0.15	0.07

Figure 14.42 Box Plots for Density Values by Alteration Zone



14.6.2 Mineral Resource Classification

Mineral Resource Classification

Mineral resources were classified in accordance with definitions provided by CIM (2014) Standards and Definitions. The mineral resources for the Malikoundi/Boto 2, Boto 5 and Boto 6 deposits were classified as Indicated and Inferred mineral resources and captured within optimized constraining shells.

Indicated Resources are classified where estimated blocks are situated within the 50 m by 50 m drill hole grid, interpolated with a minimum of two drill holes, and nominally within 25 m for the last drill hole. Inferred Resources are classified as blocks estimated with a minimum of two drill holes, with a nominal distance to the closest composite of less than 70 m.

The Malikoundi/Boto 2 deposit and Boto 6 deposits are situated in proximity to the Falémé and Balinko Rivers, the border of Senegal with Mali. With respect to prospects of eventual economic extraction, a 500 m offset, or exclusion zone, was applied from these rivers edges' as a protected zone. There are no mineral resources declared within the 500 m offset zone.

14.6.3 Mineral Resources – Malikoundi/Boto 2

Optimized Constraining Shell and Cut-off Grades

In order to demonstrate 'reasonable prospects for eventual economic extraction' an optimized constraining shell was used to report mineral resources for the Malikoundi/Boto 2 deposit. The constraining shell was developed using Hexagon Mining MineSight® 3D. Table 14.44 lists the parameters used for the optimized constraining shell by alteration zone.

Table 14.44 Parameters for the Optimized Constraining Shell for the Malikoundi/Boto 2 Deposit

		Malikoundi/Boto 2 Parameters			
Description	**Units**	**Laterite [40]**	**Saprolite [50]**	**Transition [60]**	**Rock [70]**
Gold Price	$/oz	n/a	1,500	1,500	1,500
Mining Cost (Ore)	$/ t milled	n/a	1.41	1.75	1.92
Mining Cost (Waste)	$/ t milled	1.34	1.34	1.67	1.84
Incremental Mining Cost (Ore)	$/ t milled	n/a	0.022	0.022	0.022
Incremental Mining Cost (Waste)	$/ t milled	0.028	0.028	0.028	0.028
Mining Throughput	Mtpa	2.75	2.75	2.75	2.75
Processing Cost	$/ t milled	n/a	10.09	10.73	14.35
Metallurgical Recovery	%		94.8	92.1	89.8
G&A	$/ t milled	4.35	4.35	4.35	4.35
Slope Angle	°	33	33	55-60	55-75
Cut-off Grade	g/t Au	n/a	0.37	0.40	0.51

Note: n/a = not applicable

The cut-off grades established for the Malikoundi/Boto 2 deposit by alteration zone are:

- Saprolite 0.37 g/t Au.
- Transition 0.40 g/t Au.
- Rock 0.51 g/t Au.

Mineral Resources

The mineral resources are reported inclusive of mineral reserves. The mineral resources for the Malikoundi/Boto 2 deposit are: An Indicated Resource of 41.9 Mt at 1.66 g/t Au; and an Inferred Resource of 2.0 Mt at 2.00 g/t Au. Table 14.45 presents the mineral resources for the Malikoundi/Boto 2 deposit.

Table 14.45 Mineral Resources for the Malikoundi/Boto 2 Deposit; reported within the optimized constraining shell

Classification	Tonnes ('000 t)	Grade (g/t Au)	Contained Gold ('000 oz Au)
Indicated	41,915	1.66	2,240
Inferred	1.974	2.00	127

Notes:

1. The mineral resources are reported within an optimized constraining shell using a gold price of US$1,500/oz.
2. Summation errors may occur due to rounding.
3. Mineral Resources are reported inclusive of Mineral Reserves.
4. Mineral Resources are classified in accordance with the CIM (2014) Standards and Definitions of mineral resources.
5. Cut-off grades used to report mineral resources vary from 0.37 g/t Au and 0.51 g/t Au depending on alteration zone.
6. Capping of grade outliers varies between 2.01 g/t Au and 42.02 g/t Au depending on interpreted mineralized zone and sub-domain.
7. The density varies between 1.70 g/cm^3 and 2.76 g/cm^3 depending on alteration zone.

AGP is not aware of any information not already discussed in this report, which would affect their interpretation or conclusions regarding the subject property. AGP is required to inform the public that the quantity and grade of reported Inferred resources in this estimation must be regarded as conceptual in nature and are based on limited geological evidence and sampling. The geological evidence is sufficient to imply, but not verify, geological grade or quality of continuity. For these reasons, an Inferred resource has a lower level of confidence than an Indicated resource. It is reasonably expected that most of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. The rounding of values, as required by the reporting guidelines, may result in apparent differences between tonnes, grade, and metal content.

It is also noted that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.6.4 Mineral Resources – Boto 5

Optimized Constraining Shell and Cut-off Grades

In order to demonstrate 'reasonable prospects for eventual economic extraction' an optimized constraining shell was used to report mineral resources for the Boto 5 deposit. The constraining shell was developed using Hexagon Mining MineSight® 3D. Table 14.46 lists the parameters used for the optimized constraining shell by alteration zone.

Table 14.46 Parameters for the Optimized Constraining Shell for the Boto 5 Deposit

		Boto 5 Parameters			
Description	**Units**	**Laterite [40]**	**Saprolite [50]**	**Transition [60]**	**Rock [70]**
Gold Price	$/oz	n/a	1,500	1,500	1,500
Mining Cost (Ore)	$/ t milled	n/a	2.24	2.24	2.24
Mining Cost (Waste)	$/ t milled	2.24	2.24	2.24	2.24
Incremental Mining Cost (Ore)	$/ t milled	0	0	0	0
Incremental Mining Cost (Waste)	$/ t milled	0	0	0	0
Slope Angle	°	21.4-30.8	21.4-30.8	23.6-36.5	41.6
Mining Throughput	Mtpa	n/a	2.75	2.75	2.75
Processing Cost	$/ t milled	n/a	10.09	10.73	14.35
Metallurgical Recovery	%	n/a	92.3	93.8	95.5
G&A	$/ t milled	4.35	4.35	4.35	4.35
Cut-off Grade	g/t Au	n/a	0.38	0.39	0.48

Note: n/a = not applicable

The cut-off grades established for the Boto 5 deposit by alteration zone are:

- Saprolite 0.38 g/t Au.
- Transition 0.39 g/t Au.
- Rock 0.48 g/t Au.

14.6.5 Mineral Resources – Boto 6

Optimized Constraining Shell and Cut-off Grades

In order to demonstrate 'reasonable prospects for eventual economic extraction' an optimized constraining shell was used to report mineral resources for the Boto 6 deposit. The constraining shell was developed using Hexagon Mining MineSight® 3D. Table 14.47 lists the parameters used for the optimized constraining shell by alteration zone.

Table 14.47 Parameters for the Optimized Constraining Shell for the Boto 6 Deposit

		Boto 6 Parameters			
Description	**Units**	**Laterite [40]**	**Saprolite [50]**	**Transition [60]**	**Rock [70]**
Gold Price	$/oz	n/a	1,500	1,500	1,500
Mining Cost (Ore)	$/ t milled	1.41	1.41	1.75	1.92
Mining Cost (Waste)	$/ t milled	1.34	1.34	1.67	1.84
Incremental Mining Cost (Ore)	$/ t milled	0.018	0.018	0.018	0.018
Incremental Mining Cost (Waste)	$/ t milled	0.022	0.022	0.022	0.022
Slope Angle	°	31.0	31.0	36.5	41.6
Mining Throughput	Mtpa	n/a	2.75	2.75	2.75
Processing Cost	$/ t milled	n/a	10.09	10.73	14.35
Metallurgical Recovery	%	n/a	92.3	93.8	95.5
G&A	$/ t milled	4.35	4.35	4.35	4.35
Cut-off Grade	g/t Au	n/a	0.38	0.39	0.48

Note: n/a = not applicable

The cut-off grades established for the Boto 6 Deposit by alteration zone are:

- Saprolite 0.38 g/t Au.
- Transition 0.39 g/t Au.
- Rock 0.48 g/t Au.

Mineral Resources

The mineral resources are reported inclusive of mineral reserves. The mineral resources for the Boto 6 deposit are: An Indicated Resource of 3.7 Mt at 0.84 g/t Au; and an Inferred Resource of 0.5 Mt at 1.06 g/t Au. Table 14.48 presents the mineral resources for the Boto 6 deposit.

Table 14.48 Mineral Resources for the Boto 6 Deposit; Reported within the Optimized Constraining Shell

Classification	Tonnes ('000 t)	Grade (g/t Au)	Contained Gold ('000 oz Au)
Indicated	3,661	0.84	99
Inferred	475	1.06	16

***Notes**:*

1. *The mineral resources are reported within an optimized constraining shell using a gold price of US$1,500/oz.*
2. *Summation errors may occur due to rounding.*
3. *Mineral Resources are reported inclusive of Mineral Reserves.*
4. *Mineral Resources are classified in accordance with the CIM (2014) Standards and Definitions of mineral resources.*
5. *Cut-off grades used to report mineral resources vary from 0.38 g/t Au to 0.48 g/t Au depending on alteration zone.*
6. *Capping of grade outliers at 10.00 g/t Au (MZ-Main) and no capping (MZ-W).*
7. *The density varies between 1.70 g/cm^3 and 2.76 g/cm^3 depending on alteration zone.*

AGP is not aware of any information not already discussed in this report, which would affect their interpretation or conclusions regarding the subject property. AGP is required to inform the public that the quantity and grade of reported Inferred resources in this estimation must be regarded as conceptual in nature and are based on limited geological evidence and sampling. The geological evidence is sufficient to imply, but not verify, geological grade or quality of continuity. For these reasons, an Inferred resource has a lower level of confidence than an Indicated resource. It is reasonably expected that most of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. The rounding of values, as required by the reporting guidelines, may result in apparent differences between tonnes, grade, and metal content.

It is also noted that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.6.6 Mineral Resources – Boto 4

There are no reported Mineral Resources for the Boto 4 deposit due to the proximity to the Balinko River within the 500 m exclusion zone from the river and the situation of the village of Guémédji above the deposit. Should the 500 m offset limit change or be lifted, the block model and mineral resources for the Boto 4 deposit will be re-evaluated.

14.6.7 Mineral Resources – Summary

The following is a summary of the mineral resources for the Project with an effective date of 8 May 2018. The resources were estimated using Geovia GEMS 6.8.1 resource estimation software. Mineral resources are reported within optimized constraining shells using Hexagon Mining MineSight 3D software using a gold price of $1500/oz. Mineralized zones were captured using solid 3D solid wireframes and gold grades were estimated within these zones using the ID3 interpolation method. Cut-off grades vary between 0.37 and 0.51 g/t Au, and densities vary between 1.70 and 2.76, depending on alteration zone. Mineral resources are classified as Indicated Resources and Inferred Resources in accordance with the CIM (2014) Standards and Definitions of Mineral Resources and Mineral Reserves.

Mineral resources are reported inclusive of mineral reserves.

Table 14.49 presents the Mineral Resources for the Project.

Table 14.49 Mineral Resources for the Boto Project; effective date 8 May 2018

Zone	Classification	Tonnes (,000 t)	Grade (g/t Au)	Contained Gold (,000 oz)
Malikoundi/Boto 2	Indicated	41,915	1.66	2,240
	Inferred	1,974	2.00	127
Boto 5	Indicated	2,469	1.86	148
	Inferred	34	0.75	1
Boto 6	Indicated	3,661	0.84	99
	Inferred	475	1.06	16
Boto 4	Indicated	-	-	-
	Inferred	-	-	-
Total	**Indicated**	48,045	1.61	2,487
	Inferred	2,483	1.80	144

***Notes**:*

1. *The mineral resources are reported within optimized constraining shells using a gold price of US$1,500/oz.*
2. *Summation errors may occur due to rounding.*
3. *Mineral Resources are reported inclusive of Mineral Reserves.*
4. *Mineral Resources are classified in accordance with the CIM (2014) Standards and Definitions of mineral resources.*
5. *Cut-off grades used to report mineral resources vary from 0.37 g/t Au and 0.51 g/t Au depending on alteration zone.*
6. *Capping of grade outliers varies between 1.71 g/t Au and 42.02 g/t Au depending on interpreted mineralized zone and sub-domain.*
7. *The density varies between 1.70 g/cm^3 and 2.76 g/cm^3 depending on alteration zone.*

AGP is not aware of any information not already discussed in this report, which would affect their interpretation or conclusions regarding the subject property. AGP is required to inform the public that the quantity and grade of reported Inferred resources in this estimation must be regarded as conceptual in nature and are based on limited geological evidence and sampling. The geological evidence is sufficient to imply, but not verify, geological grade or quality of continuity. For these reasons, an Inferred resource has a lower level of confidence than an Indicated resource. It is reasonably expected that most of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. The rounding of values, as required by the reporting guidelines, may result in apparent differences between tonnes, grade, and metal content.

It is also noted that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Cut-off Grade Sensitivity

To illustrate the sensitivity to cut-off grade, the Indicated and Inferred mineral resources for the Malikoundi/Boto 2, Boto 5 and Boto 6 deposits, within their constraining shells, were combined and reported as grade-tonnage curves (plots) at incremental cut-off grades of 0.1 g/t Au. The reader is cautioned that these grade-tonnage curves are not be taken as the reported mineral resources and serve only to show the sensitivity of the block model to selected cut-off grades.

Figure 14.43 and Figure 14.44 show the grade tonnage curve for the Indicated Resources and the Inferred Resources within constraining shells, respectively, at various cut-off grades for the Project.

Figure 14.43 Grade-Tonnage Plot for the Boto Project; for Indicated Resources within Constraining Shells

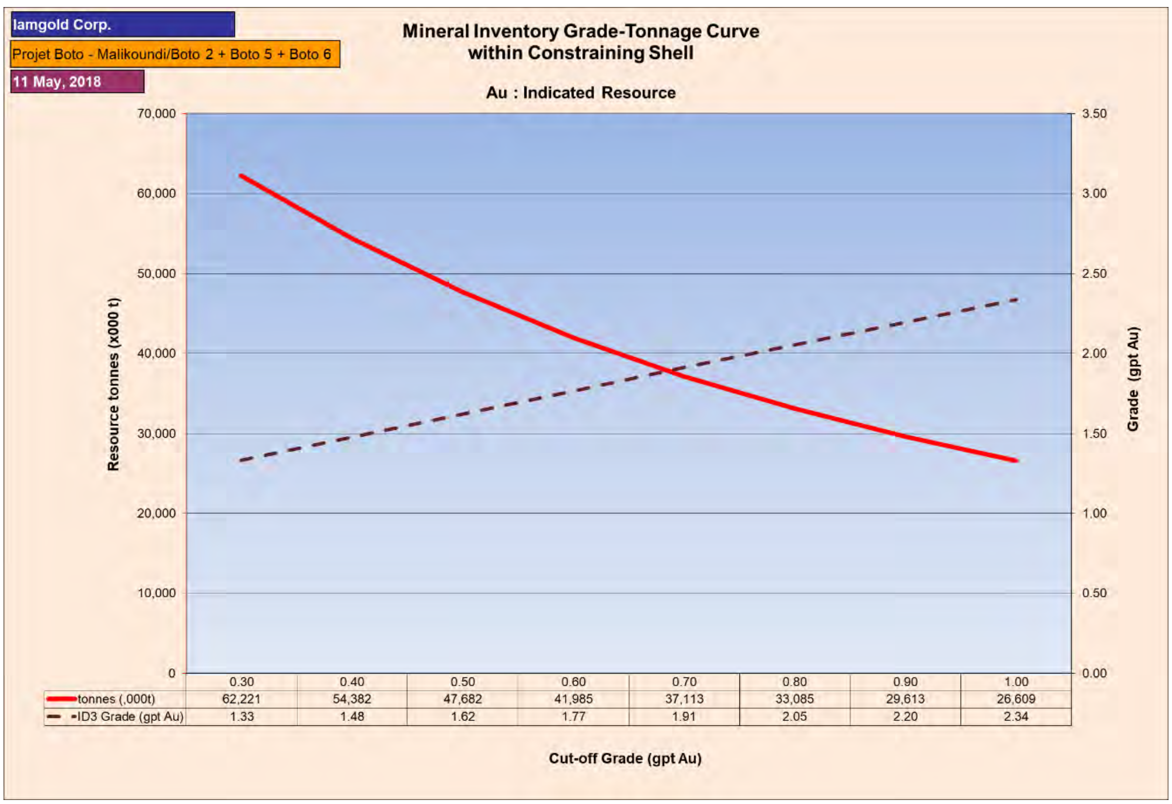
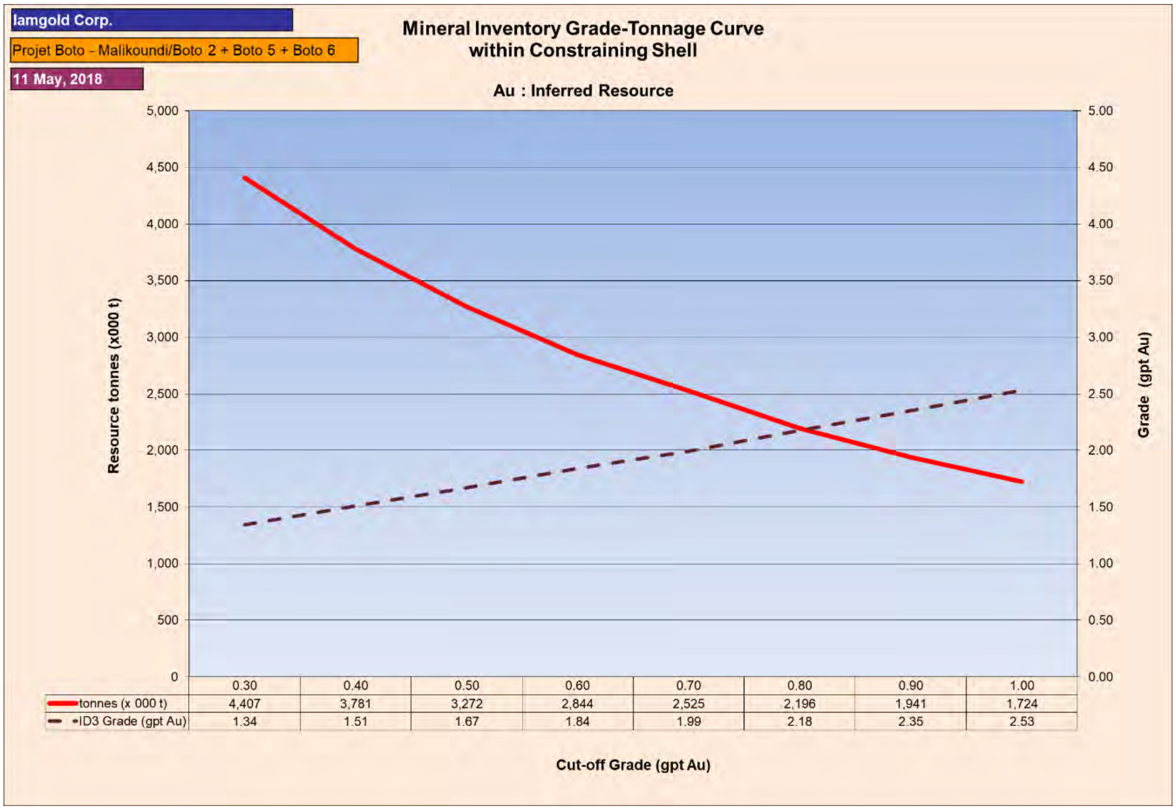


Figure 14.44 Grade-Tonnage Plot for the Boto Project; for Inferred Resources within Constraining Shells



14.7 PREVIOUS MINERAL RESOURCES

14.7.1 Previous Mineral Resources

Since the initial mineral resources were reported on the Project, there has been an overall increase in resource tonnes and contained metal. In the current mineral resource estimate there has been a significant increase in overall tonnes and contained metal. This may be partially attributed to the additional drilling along strike in Malikoundi/Boto 2, Boto 5, and Boto 6 deposits, and at depth at the Malikoundi/Boto 2 and Boto 5 deposits. Additionally, in Malikoundi/Boto 2, several mineralized zones have been re-interpreted as being slightly wider than the previous resource interpretation and some narrower zones have been included into larger zones. The additional drill information allows for upgrading of more material into the Indicated category from previously Inferred resources and increasing the resource tonnes captured by larger optimized constraining shells. It should also be noted that the cut-off grades for the current mineral resources are lower than the previous mineral resource.

Table 14.50 summarizes the mineral resources from 2013 to present.

Table 14.50 Summary of Previous Mineral Resources for the Boto Project

Date	Indicated			Inferred		
	Tonnage	Grade	Contained Metal	Tonnage	Grade	Contained Metal
	(Mt)	(g/t Au)	(Moz Au)	(Mt)	(g/t Au)	(Moz Au)
July 2013	22.0	1.62	1.14	1.9	1.35	0.08
December 2014	22.8	1.68	1.23	11.0	1.80	0.63
July 2015	27.7	1.76	1.60	2.9	1.34	0.12
July 2017	37.4	1.60	1.90	11.0	1.66	0.59
May 2018	48.0	1.61	2.49	2.5	1.80	0.14

14.7.2 Malikoundi/Boto 2

Compared to the previous mineral resource estimate (July 2017), the Malikoundi/Boto 2 mineral resources show an increase in overall tonnages and contained metal. This is due in part to the re-interpretation of the mineralized zones, where zones are slightly wider or larger; and have included narrower zones from the previous interpretation. Additional drill hole data from the 2017 and 2018 drill campaigns have also allowed for greater confidence and better continuity of geology and grade to allow previously categorized Inferred resources to be upgraded to Indicated resources. The change to a lower set of cut-off grades have also had the effect of increasing the mineral resources.

Table 14.51 presents the comparison of the July 2017 mineral resources to the current mineral resources for the Malikoundi/Boto deposit.

Table 14.51 Comparison of Mineral Resources for Malikoundi/Boto2; February 2017 vs May 2018; different cut-off grades and different constraining shells

Classification	Malikoundi/Boto 2 (30 July 2017) in 2017 Resource Shell COG > 0.44 g/t Au (Saprolite) and > 0.45 g/t Au (Transition) and > 0.53 g/t Au (Rock)			Malikoundi (8 May 2018) in updated 2018 Resource Shell COG > 0.37 g/t Au (Saprolite) and > 0.40 g/t Au (Transition) and > 0.51 g/t Au (Rock)					
	Tonnage ('000 t)	Au (g/t Au)	Contained Au ('000 oz)	Tonnage ('000 t)	Au (g/t Au)	Contained Au ('000 oz)	Tonnage (% diff)	Grade (% diff)	Contained Au (% diff)
Indicated	33,866	1.61	1751	40,421	1.69	2199	19%	5%	26%
Inferred	8,188	1.84	484	3,512	2.03	229	-57%	10%	-53%

14.7.3 Boto 5

Compared to the previous Mineral Resource Estimate (July 2017), the Boto 5 mineral resources show an increase in resource tonnages and contained metal. This is partially due to the re-interpretation of the mineralized zones but also in part due to the addition of drill hole information from the 2017 and 2018 drill campaigns that have allowed for greater confidence and better continuity of geology and grade to allow previously categorized Inferred resources to be upgraded to Indicated resources. The change to a lower set of cut-off grades have also had the effect of increasing the mineral resources.

Table 14.52 presents the comparison of the July 2017 mineral resources to the current mineral resources for Boto 5 deposit.

Table 14.52 Comparison of Mineral Resources for Boto 5: July 2017 vs. May 2018 Resources; different cut-off grades and different constraining shells

Classification	Boto 5 (30 July 2017) in 2017 Resource Shell COG > 0.44 g/t Au (Saprolite) and > 0.45 g/t Au (Transition) and > 0.53 g/t Au (Rock)			Boto 5 (8 May 2018) in updated 2018 Resource Shell COG > 0.38 g/t Au (Saprolite) and > 0.39 g/t Au (Transition) and > 0.48 g/t Au (Rock)					
	Tonnage ('000 t)	Au (g/t Au)	Contained Au ('000 oz)	Tonnage ('000 t)	Au (g/t Au)	Contained Au ('000 oz)	Tonnage (% diff)	Grade (% diff)	Contained Au (% diff)
Indicated	1,617	2.17	113	2,469	1.86	148	53%	-14%	31%
Inferred	0	0.00	0	34	0.75	1	-	-	-

14.7.4 Boto 6

Compared to the previous Mineral Resource Estimate (July 2017), the Boto 6 mineral resources show a significant increase in resource tonnages and contained metal. This is partially due to the additional drill hole information that have allowed for greater confidence and better continuity of geology and grade to allow previously categorized Inferred resources to be upgraded to Indicated resources. The change to a lower set of cut-off grades have also had the effect of increasing the mineral resources.

Table 14.53 presents the comparison of the July 2017 mineral resources to the current mineral resources for Boto 6 deposit.

Table 14.53 Comparison of Mineral Resources for Boto 6; July 2017 vs May 2018 Resources

Classification	Boto 6 (30 July 2017) in 2017 Resource Shell COG > 0.42 g/t Au (Saprolite) and > 0.45 g/t Au (Transition) and > 0.52 g/t Au (Rock)			Boto 6 (8 May2018) in updated 2018 Resource Shell COG > 0.38 g/t Au (Saprolite) and > 0.39 g/t Au (Transition) and > 0.48 g/t Au (Rock)					
	Tonnage ('000 t)	Au (g/t Au)	Contained Au ('000 oz)	Tonnage ('000 t)	Au (g/t Au)	Contained Au ('000 oz)	Tonnage (% diff)	Grade (% diff)	Contained Au (% diff)
Indicated	1,925	0.95	59	3,661	0.84	99	90%	-12%	68%
Inferred	396	1.14	14	475	1.06	16	20%	-7%	12%

Note: the July 2017 Inferred Resource at Boto 6 was erroneously reported at 184,000 tonnes when it was actually 184,000 $m^{3.}$ The correct resource tonnes are shown in this Table

15.0 MINERAL RESERVE ESTIMATES

15.1 Summary

The reserves for the Project are based on the conversion of the Indicated resources within the current Technical Report mine plan. No Measured resources are currently part of the model. Indicated resources are converted directly to Probable Reserves. The total reserves for the Project are shown in Table 15.1.

Table 15.1 Proven and Probable Reserves

	Proven			Probable			Total		
Ore Type	Tonnes (kt)	Grade (g/t)	Gold (oz)	Tonnes (kt)	Grade (g/t)	Gold (oz)	Tonnes (kt)	Grade (g/t)	Gold (oz)
Saprolite	-	-	-	2,910	1.85	173,000	2,910	1.85	173,000
Transition	-	-	-	2,160	2.01	139,000	2,160	2.01	139,000
Fresh Rock	-	-	-	29,990	1.67	1,614,000	29,990	1.67	1,614,000
Total	**-**	**-**	**-**	**35,060**	**1.71**	**1,926,000**	**35,060**	**1.71**	**1,926,000**

Note: This mineral reserve estimate is as of Aug 30, 2018 and is based on the new mineral resource estimate dated May 8, 2018 for Malikoundi and Boto 5 by AGP. The mineral reserve calculation was completed under the supervision of Gordon Zurowski, P.Eng of AGP., who is a Qualified Person as defined under NI 43-101. Mineral reserves are stated within the final design pit based on a $1,044/oz gold price pit shell with a $1,200/oz gold price for revenue for Malikoundi, $960/oz for Malikoundi North and $900 /ounce for Boto 5. The cut-off grade varied by material type from 0.46 g/t Au in saprolite, 0.50 g/t Au in transition and 0.63 g/t Au in fresh rock for the Malikoundi and Malikoundi North pit areas. The cut-off was 0.48 g/t Au in saprolite, 0.49 g/t Au in transition and 0.59 g/t in fresh rock for the Boto 5 area. The mining cost varied by rock type and area but averaged $2.11/t, processing costs vary by rock type but averaged $13.83/t milled and G&A was $4.15/t milled. The process recovery averaged 89.5%. The Technical Report scope only considers the Malikoundi, Malikoundi North and Boto 5 open pit mineralized zones.

The reserves are based solely on the Malikoundi, Malikoundi North and Boto 5 areas.

The QP has not identified any known legal, political, environmental, or other risks that would materially affect the potential development of the Mineral Reserves. The risk of not being able to secure the necessary permits from the government for development and operation of the project exist but the QP is not aware of any issues that would prevent those permits from being withheld per the normal permitting process.

15.2 Mining Method and Mining Costs

The Project is amenable to extraction by open pit methods. Costs were developed from base principles and with current equipment quotations from local vendors.

All design work is based on the Malikoundi and Boto 5 models generated by AGP with an effective date of May 8, 2018. Only Indicated Resources were used for the Feasibility Study and all Inferred resources are considered as waste. No Measured resources exist in the current models.

This section discussed the development and parameters employed to declare reserves for the current FS pit design.

15.2.1 Geotechnical Considerations

A geotechnical study was completed on the Malikoundi and Boto 5 deposits for use in the FS study. This work was completed by AG.

The Malikoundi deposit consists predominantly of pelite, with sandstone units present in the southeast of the pit, and at depth. The mineralisation is aligned to the north-south structural trend and is constrained within two Limestone/marble units, dipping at ~60° to the west, which are interpreted to have formed impermeable barriers to the flow of mineralising fluids. The Malikoundi North deposit lies on the extension along strike of the eastern Limestone unit. The geology of the Boto 5 deposit is more poorly understood, partly due to the deep (>100m) saprolitic weathering profile. A saprolitic profile overlies fresh rock at Malikoundi and is generally <40m thick.

Data from previous phases of geotechnical and hydrogeological study have been collated and used within this assessment. Geotechnical and hydrogeological investigations targeted data gaps and areas of greater uncertainty within conceptual models. Geotechnical data collection for this phase of study focused on the hangingwall of Malikoundi (additional drilling into the footwall was undertaken subsequent to the PFS), Malikoundi North and Boto 5. Hydrogeological assessment was designed to refine the characterisation of the low permeability conditions inferred from previous phases of hydrogeological testing. A packer testing programme was undertaken to supplement available groundwater monitoring data, and results of previous phases of downhole testing.

Geotechnical domains have been defined based on similar geological, structural, and rock mass conditions. Both Malikoundi and Malikoundi North pits were subdivided initially by the main Limestone units, and then by lithology as warranted. It has been recognised that the stability of the footwall within the Pelite is likely to be controlled by the orientation of the major structural fabric of the deposit. The stability of the hangingwall is likely to be controlled by a combination of instability through the rock mass and structural instabilities at the batter scale. Subdivision of Boto 5 was based primarily on the thickness of the various weathered units. Due to the low material strengths, instability through the rock mass is the controlling mechanism, although structural instability on relic structure can still occur.

Detailed information for the recommendations can be found in Section 16.3. For the final design, the pit slopes discussed in Section 16.6 were used. These are shown in Table 15.2.

Table 15.2 Pit Slope Parameters for Detail Design

Rock Domains Horizon	Sector	Sector Code (GEOT1)	Slope Domain (GEOT)	Face Angle (degrees)	Height between berms (m)	Catch Bench Width (m)	Inter-Ramp Angle (IRA) (degrees)
Malikoundi Pit							
Saprolite/Laterite	HW	3,6	11	60	5	4.8	33
Saprolite/Laterite	FW	4,5,7	12	60	5	4.8	33
Transition	HW	3,6	13	60	10	6.5	39.2
Transition	FW	4,5,7	14	55	10	6.5	36.5
Fresh	HW + south wall	3,6	15	75	20	8.5	55.3
Fresh	FWA(int)+FWB(pelite)	4,7	16	60	20	8.5	44.9
Fresh	FWC(sandstone)	5	17	70	20	8.5	51.7
Malikoundi North Pit							
Saprolite/Laterite	HW	1	1	60	5	4.8	33.0
Saprolite/Laterite	FW	2	2	60	5	4.8	33.0
Transition	HW	1	3	60	10	6.5	39.2
Transition	FW	2	4	55	10	6.5	36.5
Fresh	HW	1	5	75	20	8.5	55.3
Fresh	FW	2	6	55	20	8.5	41.6
Boto 5 Pit							
Saprolite/Laterite	FW		1	55	5	4.9	30.8
Transition	FW		2	55	10	6.5	36.5
Saprolite/Laterite	West Wall		3	55	5	4.9	30.8
Transition	West Wall		4	55	10	5.5	39.3
Saprolite/Laterite	HW West		5	55	5	5.5	29.1
Transition	HW West		6	55	10	10	30.5
Saprolite/Laterite	HW East		7	55	5	5.5	29.1
Transition	HW East		8	55	10	6.5	36.5
Fresh	All		9	55	20	8.5	41.6

Drained conditions are assumed because horizontal drain holes are planned every 20 m vertically as mining progresses. The initial design is to have drill stations every 200 m horizontally on the 20 m levels and three drillholes drilled 50 m from each station radiating out as a fan.

15.2.2 Economic Pit Shell Development

The final pit designs were based on pit shells using the Lerch-Grossman procedure in MineSight. The parameters for the shells are shown in Table 15.3 and 15.4.

Table 15.3 Pit Optimization Parameters

Parameter	Units	Malikoundi Pit	Boto 5 Pit
Metal Prices			
Gold Price	$/oz	1200	1200
Payable	%	99%	99%
Participation (on profits)	%	90%	90%
Transportation & Refining	$/oz	3.04	3.04
Royalty	%	4%	4%
General			
Resources blocks used		M+I	M+I
General & Administration Cost	$/t ore	4.35	4.35
Process Recovery			
Laterite (ROCK=40)	%	0.0%	0.0%
Saprolite (ROCK=50)	%	94.8%	92.3%
Transition (ROCK=60)	%	92.1%	93.8%
Fresh Rock (ROCK=70)	%	89.8%	95.5%
Process Costs *			
Laterite Process Cost	$/t	-	-
Saprolite Process Cost	$/t	10.09	10.09
Transition Process Cost	$/t	10.73	10.73
Fresh Rock Process Costs	$/t	14.34	14.34
Mining Costs **			
Incremental haul cost - waste	$/5m bench	0.022	-
Incremental haul cost - ore	$/5m bench	0.028	-
Waste		Ref. elev = 160	Ref. elev = 200
Laterite	$/t	1.34	2.24
Saprolite	$/t	1.34	2.24
Transition	$/t	1.67	2.24
Fresh Rock	$/t	1.84	2.24
Ore			
Laterite	$/t	1.41	2.24
Saprolite	$/t	1.41	2.24
Transition	$/t	1.75	2.24
Fresh Rock	$/t	1.92	2.24

** process costs based on 2.75 Mt/y dry*

*** mining costs based on using 95 t haul trucks, and contractor in Boto 5 with 40 t trucks*

Table 15.4 Pit Optimization Slope Angles

Rock Domains Horizon	Sector	Face Angle (degrees)	Height between berms (m)	Catch Bench Width (m)	Inter-Ramp Angle (IRA) (degrees)	# Ramps in Slope	Slope Height (m)	Overall Slope Angle (degrees)
Malikoundi Pit								
Saprolite/Laterite	HW	60	5	4.8	33	0	30	33
Saprolite/Laterite	FW	60	5	4.8	33	0	30	33
Transition	HW	60	10	6.5	39.2	0	20	39.2
Transition	FW	55	10	6.5	36.5	0	20	36.5
Fresh	HW + south wall	75	20	8.5	55.3	3	280	44.3
Fresh	FWA(int)+FWB(pelite)	60	20	8.5	44.9	3	300	37.3
Fresh	FWC(sandstone)	70	20	8.5	51.7	3	300	42.3
Malikoundi North Pit								
Saprolite/Laterite	HW	60	5	4.8	33.0	0	40	33
Saprolite/Laterite	FW	60	5	4.8	33.0	0	40	33
Transition	HW	60	10	6.5	39.2	0	20	39.2
Transition	FW	55	10	6.5	36.5	0	20	36.5
Fresh	HW	75	20	8.5	55.3	0	30	55.3
Fresh	FW	55	20	8.5	41.6	1	30	25.8
Boto 5 Pit								
Saprolite/Laterite	FW	55	5	4.9	30.8	0	50	30.8
Transition	FW	55	10	6.5	36.5	1	30	23.6
Saprolite/Laterite	West Wall	55	5	4.9	30.8	0	40	30.8
Transition	West Wall	55	10	5.2	39.3	1	40	27.4
Saprolite/Laterite	HW West	55	5	5.5	29.1	1	70	21.4
Transition	HW West	55	10	10	30.5	0	40	30.5
Saprolite/Laterite	HW East	55	5	5.5	29.1	1	55	23.4
Transition	HW East	55	10	6.5	36.5	0	40	36.5
Fresh	All	55	20	8.5	41.6	0	40	41.6

The final pit design was based on pit shells developed using the Lerch-Grossman procedure in MineSight.

A series of nested shells were generated using a revenue factor (rf). Initially these were varied between a gold price of \$300/oz (rf=0.25) and \$1440/oz (rf=1.2) to examine the deposit sensitivity to gold prices and outline the higher value areas. This information was graphed, and the various phases and final shell determined based on a net revenue curve.

The final pits are based on the \$1044/oz gold price shell for Malikoundi, \$960/oz gold price shell for Malikoundi North and \$900/oz shell for Boto 5.

15.2.3 Cut-off Grade

For determining the tonnes and grade in the pit, cut-offs were varied by pit area and weathering type. The cut-offs used are shown in Table 15.5.

Table 15.5 Marginal Cut-off Grades

Weathered Material	Malikoundi Au (g/t)	Boto 5 Au (g/t)
Laterite	n/a	n/a
Saprolite	0.46	0.48
Transition	0.50	0.49
Fresh	0.63	0.59

For scheduling purposes, a higher-grade cut-off of 1 g/t was used to bin the higher-grade material while the marginal cut-off used to define the low-grade bin. All reserves are reported based on the marginal cut-off grades.

15.2.4 Dilution

The geologic models developed for the FS were whole block fully diluted models. This means the grade from the wire frames was diluted over the full volume of the block to arrive at a diluted block grade. The geologic model had been created with grade wireframes prior to assigning the grade into a whole block. AGP believed that this did not adequately reflect the amount of dilution that would be expected with normal mining practice, even with more selective equipment.

AGP also believes that contact dilution would play a role in material sent to the mill. To determine the amount of dilution and the grade of the dilution the size of the block in the model was examined. The Malikoundi block size within the model was 5 m in the dip direction, 10 m along strike, and 5 m high.

The percentage of dilution is calculated for each contact side using an assumed 0.5 m contact dilution distance. If one side of the block is touching waste, then it is estimated that dilution of 10% would result. If two sides are contacting, it would rise to 15%. Three sides would be 20%, and four sides 30%. Four sides represent an isolated block of ore. This assumes a development of the block on the hangingwall side first, then the two sides and finally the footwall.

For Boto 5, the block model was equally sized at 5 m on each side (dip, strike and height). The dilution calculation used the same 0.5 m of contract dilution. The resulting dilution percentages were different with one side equal to 10%, two sides at 20%, three sides at 30% and four sides at 40%. The same development sequence was assumed.

Since the model contained whole blocks already, the percentage of dilution could be estimated and then included in the block ore percentage item. The mining model was modified to include an ore percent item, and any blocks with a grade above the marginal cut-off grade were assigned an ore percent of 100% (deemed entirely ore).

MineSight has a routine that enables the user to query surrounding blocks against a set of conditions. For the dilution percentage calculation, the procedure was run to determine how many ore blocks contacted a waste block, which determined the dilution percentage to apply. This was stored in the waste block and the waste block grade used as the diluting value. If a waste block was only surrounded by other waste blocks, the dilution percentage was zero.

In this manner, the contact blocks could be included in the tonnage and grade calculation of ore tonnes. The ore tonnage was then run with the block model DORE% item to report out the proper tonnes and grade.

Comparing the in-situ to the diluted values for the designed pit phases showed an increase in ore tonnage along with a lowering of gold grade. For the Malikoundi pit (phases 1-3 and Malikoundi North), the diluted ore contained 4.5% more tonnes and 4.1% lower gold grade than the in-situ ore summary. For the Boto 5 pit, the diluted ore contained 7.8% more tonnes and 6.7% lower gold grade than the in-situ ore summary. The grade dilution is lower due to the waste blocks containing some mineralization. The ore tonnage contact dilution is lower than the one contact block dilution percentage due to the thickness of the ore where not all the ore blocks are diluted with waste. The “internal dilution” from the conversion of wireframe to whole block was not reported separately within the reported in-situ summaries otherwise the dilution tonnages would have been higher.

Tonnes and grade for the pit designs are reported with the diluted tonnes and grade.

15.2.5 Pit Design

The detailed pit design utilized the pit shells developed to provide guidance on the phasing and final pit. Wall slopes for the inter-ramp were per the geotechnical recommendations.

Equipment sizing for ramps and working benches is based on the use of 95 t rigid frame haul trucks. The operating width used for the truck is 6.9 m. This means that single lane access is 21.4 m (2x operating width plus berm and ditch) and double lane widths are 28.3 m (3x operating width plus berm and ditch). Ramp gradients are 10% in the pit for uphill gradients and 8% uphill on the dump access roads. Working benches were designed for 35 m to 40 m minimum on pushbacks, although some pushbacks in the Malikoundi pit did work in a retreat manner to facilitate access.

The Malikoundi pit is designed as 4 phases within the main pit. Phase 0 as the initial pit is called as a subset of Phase 1 to drive quickly to fresh rock for tailings dam construction purposes. Malikoundi North is a single-phase pit as is Boto 5.

The mine schedule delivers 35.1 Mt of ore grading 1.71 g/t Au to the mill over a mine life of 12.8 years. Waste tonnage totalling 204.3 Mt will be placed into waste rock management facilities. The overall strip ratio is 5.8:1 life of mine.

The current mine life includes two years of pre-stripping and 12.8 years of mining. The final year will complete the pit and clear the stockpiled ore. The stockpiled ore, together with pit phasing will be utilized to ensure sufficient mill feed is available in the rainy season and for blending purposes. This will also be coupled with in-pit sumps and surface ditches around the pits. Phases will be advanced quickly in the dry season to provide temporary water storage after a rainfall event that pumping will remove in the wet season.

15.2.6 Mine Reserves Statement

The reserves for the Project are based on the conversion of the Indicated resources within the current Technical Report mine plan. No Measured resources are present in the current models. Indicated resources are converted directly to Probable Reserves. These were prepared under the supervision of Gordon Zurowski, P. Eng. of AGP who is a qualified person as defined under NI 43-101. The reserves are based solely on the Malikoundi, Malikoundi North and Boto 5 open pits.

Cut-offs for the Malikoundi and Malikoundi North areas varied by material type from 0.46 – 0.63 g/t Au. For Boto 5 the cut-offs were between 0.48 and 0.59 g/t Au.

This estimate is as of August 30, 2018. The total reserves for the Project are shown in Table 15.6 and Table 15.7.

Table 15.6 Proven and Probable Reserves – Summary for Project

	Proven			Probable			Total		
Ore Type	Tonnes (kt)	Grade (g/t)	Gold (oz)	Tonnes (kt)	Grade (g/t)	Gold (oz)	Tonnes (kt)	Grade (g/t)	Gold (oz)
Saprolite	-	-	-	2,910	1.85	173,000	2,910	1.85	173,000
Transition	-	-	-	2,160	2.01	139,000	2,160	2.01	139,000
Fresh Rock	-	-	-	29,990	1.67	1,614,000	29,990	1.67	1,614,000
Total	**-**	**-**	**-**	**35,060**	**1.71**	**1,926,000**	**35,060**	**1.71**	**1,926,000**

Note: Mineral Reserves are included within Mineral Resources

Table 15.7 Proven and Probable Reserves – By Pit Area

	Proven			Probable			Total		
Pit Area (Cutoff g/t)	Tonnes (kt)	Grade (g/t)	Gold (oz)	Tonnes (kt)	Grade (g/t)	Gold (oz)	Tonnes (kt)	Grade (g/t)	Gold (oz)
Malikoundi (0.46-0.63 g/t)									
Saprolite (0.46 g/t)	-	-	-	819,000	1.63	43,000	819,000	1.63	43,000
Transition (0.50 g/t)	-	-	-	1,184,000	2.00	76,000	1,184,000	2.00	76,000
Fresh Rock (0.63 g/t)	-	-	-	29,081,000	1.67	1,557,000	29,081,000	1.67	1,557,000
Total Malikoundi	**-**	**-**	**-**	**31,084,000**	**1.68**	**1,676,000**	**31,084,000**	**1.68**	**1,676,000**
Malikoundi North (0.46-0.63 g/t)									
Saprolite (0.46 g/t)	-	-	-	728,000	1.90	44,000	728,000	1.90	44,000
Transition (0.50 g/t)	-	-	-	326,000	1.65	17,000	326,000	1.65	17,000
Fresh Rock (0.63 g/t)	-	-	-	909,000	1.94	57,000	909,000	1.94	57,000
Total Malikoundi North	**-**	**-**	**-**	**1,963,000**	**1.88**	**118,000**	**1,963,000**	**1.88**	**118,000**
Boto 5 (0.48 – 0.59 g/t)									
Saprolite (0.48 g/t)	-	-	-	1,363,000	1.96	86,000	1,363,000	1.96	86,000
Transition (0.49 g/t)	-	-	-	650,000	2.19	46,000	650,000	2.19	46,000
Fresh Rock (0.59 g/t)	-	-	-	-	-	-	-	-	-
Total Boto 5	**-**	**-**	**-**	**2,013,000**	**2.03**	**132,000**	**2,013,000**	**2.03**	**132,000**
Boto Gold Project									
Saprolite	-	-	-	2,910,000	1.85	173,000	2,910,000	1.85	173,000
Transition	-	-	-	2,160,000	2.01	139,000	2,160,000	2.01	139,000
Fresh Rock	-	-	-	29,990,000	1.67	1,614,000	29,990,000	1.67	1,614,000
Total Boto Gold Project	**-**	**-**	**-**	**35,060,000**	**1.71**	**1,926,000**	**35,060,000**	**1.71**	**1,926,000**

16.0 MINING METHODS

16.1 Introduction

Open pit mining was selected as the method to examine the development of the Project located in Senegal. This is based on the size of the resource, tenor of the grade, grade distribution and proximity to topography for the Malikoundi and Boto 5 deposits.

No mining has been conducted on the Malikoundi part of the project but artisanal mining is ongoing at Boto 5. AGP's opinion is that with current metal pricing levels and knowledge of the mineralization, open pit mining offers the most reasonable approach for development.

The Project is located to the west of the Falémé River, which also represents a border with Mali. A 500 m buffer zone was kept with the river and surrounding villages for all infrastructure and waste dump facilities, while 200 m was observed for the pits.

The potential for underground development beneath the open pit has not been examined as part of this technical report.

16.2 Geologic Model Importation

AGP worked with the IAMGOLD team to prepare the Project resource models. While resource models were developed for Malikoundi, Boto 4, Boto 5 and Boto 6 only the Malikoundi and Boto 5 models were used for the Feasibility Study mine plan.

The resource estimates were created using Geovia GEMS™ 6.8.1 resource estimation software. The blocks models were estimated using inverse distance cubed. These models were exported as a comma-separated values (CSV) file and imported into the mining software, MineSight©.

Details of the different block models are provided in Table 16.1.

Table 16.1 Geologic Model Details

Framework Description	Malikoundi Value	Boto 5 Value
GEMS® workspace file	nov16Boto2	June15Boto5
MineSight® file 10 (control file)	bot210.dat	bot510.dat
MineSight® file 15 (model file)	bot215.dat	bot515.dat
X origin (m)	241,000	239,400
Y origin (m)	1,378,000	1,375,600
Z origin (m)(max)	225	225
Rotation (degrees clockwise)	No rotation	28.5
Number of blocks in X direction	350	220
Number of blocks in Y direction	301	230
Number of blocks in Z direction	130	90
X block size (m)	5	5
Y block size (m)	10	5
Z block size (m)	5	5

The resource block models contain rock type, density, classification weathering and gold grade. The resource models are whole block models. The mining model created by AGP in MineSight uses the same model dimensions as the original resource model with added items used for mine planning purposes. MineSight was used for the mining portion of the FS to take advantage of the included Lerchs-Grossman routine for economic pit shell development.

The grade in each block of the resource models is considered to be fully diluted except for any contact dilution of the block. No ore percentages are considered in the block model provided. All of the block model items in the mining model remain the same as in the geologic model.

Only Measured and Indicated Resources were used for the FS with no reported Measured in the model provided. All Inferred Resources were considered as waste.

Resources for the various models are shown in Table 16.2. The effective date of the resources is May 8, 2018.

Table 16.2 Mineral Resources – Boto Gold Project

Zone	Tonnes (,000 t)	Grade (g/t Au)	Contained Gold (,000oz)
INDICATED			
Malikoundi (including North)	41,915	1.66	2,240
Boto 4	-	-	-
Boto 5	2,469	1.86	148
Boto 6	3,661	0.84	99
Total INDICATED	**48,045**	**1.61**	**2,487**
INFERRED			
Malikoundi (including North)	1,974	2.00	127
Boto 4	-	-	-
Boto 5	34	0.75	1
Boto 6	475	1.06	16
Total INFERRED	**2,483**	**1.80**	**144**

Notes:

1. *The mineral resources are reported within an optimized constraining shell using a gold price of $1,500/oz*
2. *Summation errors may occur due to rounding.*
3. *Mineral Resources are reported inclusive of Mineral Reserves.*
4. *Mineral Resources are classified in accordance with the CIM (2014) Standards and Definitions of mineral resources.*
5. *Cut-off grades used to report mineral resources vary from 0.37 g/t Au and 0.51 g/t Au depending on alteration zone.*
6. *Capping of grade outliers varies between 1.71 g/t Au and 42.02 g/t Au depending on interpreted mineralized zone and sub-domain.*
7. *The density varies between 1.70 g/cm^3 and 2.76 g/cm^3 depending on alteration zone.*

16.3 Geotechnical Information

AG has been engaged to undertake a geotechnical assessment of open pit slopes for IAMGOLD's Project Feasibility Study (FS). The Project is located in Senegal, and consist of 3 proposed open pits –Malikoundi, Malikoundi North, and Boto 5. The Malikoundi design is proposed for mining in 3 staged designs. This geotechnical assessment has been undertaken in line with CSIRO's best practice document Guidelines for open pit slope design (Read and Stacey, 2009).

The Malikoundi pit is approximately 335 m deep, and 1250 m long by 800 m wide and is understood to be the ultimate pit of a potential 4 phase development. The Malikoundi North and Boto 5 pits extend to depths of approximately 80 m and 140 m, respectively. The Malikoundi pit has been previously assessed to prefeasibility study level, but geotechnical assessment of Boto 5 and Malikoundi North has not previously been undertaken.

The Malikoundi deposit consists predominantly of Pelite, with Sandstone units present in the southeast of the pit, and at depth. The mineralization is aligned to the north-south structural trend and is constrained within two Limestone/marble units, dipping at ~60° to the west, which are interpreted to have formed impermeable barriers to the flow of mineralising fluids. The Malikoundi North deposit lies on the extension along strike of the eastern Limestone unit. The geology of the Boto 5 deposit is more poorly understood, partly due to the deep

(>100m) saprolitic weathering profile. A saprolitic profile overlies fresh rock at Malikoundi and is generally <40 m thick.

Data from previous phases of geotechnical and hydrogeological study have been collated and used within this assessment. Geotechnical and hydrogeological investigation has targeted data gaps and areas of greater uncertainty within conceptual models. Geotechnical data collection for this phase of study focused on the hangingwall of Malikoundi (additional drilling into the footwall was undertaken subsequent to the PFS), Malikoundi North and Boto 5. Hydrogeological assessment was designed to refine the characterization of the low permeability conditions inferred from previous phases of hydrogeological testing. A packer testing programme was undertaken to supplement available groundwater monitoring data, and results of previous phases of downhole testing.

Geotechnical logging and laboratory test data has been analyzed to allow characterization of intact, rock mass, and defect plane strengths for materials subdivided by weathering grade and lithological unit. Weathered materials have been classified using a weak rock classification scheme in conjunction with a programme of laboratory classification and strength testing. Fresh rock has been characterized using the RMR89 and GSI classification systems (Bieniawski, 1989, Hoek et al, 2013).

Structural orientation data from both the geotechnical dataset and the resource definition drilling has been processed and used in structural analysis. A range of structure types were encountered, with foliation, bedding and joints dominating. The foliation and bedding are generally sub-parallel and form the dominant anisotropy of the deposits. Structural domaining has been undertaken based primarily on the subdivision of deposit by major structures.

Pore pressure modelling was undertaken to provide groundwater inputs to stability modelling and highlighted the requirement for active depressurisation measures at Boto 5, due to the low permeability of the Saprolite units. Such materials are very sensitive to adverse ground and surface water conditions and the modelling re-iterates the requirement for pro-active management of both.

Geotechnical domains have been defined based on similar geological, structural, and rock mass conditions. Both Malikoundi and Malikoundi North pits were subdivided initially by the main Limestone units, and then by lithology as warranted. It has been recognised that the stability of the footwall within the Pelite is likely to be controlled by the orientation of the major structural fabric of the deposit. The stability of the hangingwall is likely to be controlled by a combination of instability through the rock mass and structural instabilities at the batter scale. Subdivision of Boto 5 was based primarily on the thickness of the various weathered units. Due to the low material strengths, instability through the rock mass is the controlling mechanism, although structural instability on relic structure can still occur.

Stability analyses of the batter, inter-ramp and overall slope scales have been undertaken. Sensitivity analyses have also been performed to assess the potential for seismic effects, toppling instability modes, potential step-path mechanisms and the influence of groundwater. Results of sensitivity analyses to groundwater conditions highlight the need for pore pressure monitoring through a network of vibrating wire piezometers. High level rock fall analysis has been undertaken to check the suitability of design berm widths.

The overall and inter-ramp stability analysis was undertaken primarily by two-dimensional limit equilibrium stability analyses, with large scale kinematic analysis and 3D limit equilibrium analysis used to augment this in the footwall domains of Malikoundi. The location and orientation of the stability analysis sections were chosen to reflect the critical sections based on pit depth, geological conditions and wall orientations.

The following tables present the design recommendations for the overall slope heights appropriate to the pit designs provided. Design options are presented in places where exceeding acceptance criteria may be considered appropriate. Design sectors and boundaries are shown in the following figures.

Dewatering estimates have been undertaken to allow consideration dewatering infrastructure requirements. Based on the predicted groundwater inflow rates and direct rainfall estimates it is anticipated that by completion of excavation required dewatering capacity would be in the region of:

- Malikoundi: 15,000 m^3/day.
- Malikoundi North: 2,500 m^3/day.
- Boto 5: 8,000 m^3/day.

The dewatering requirement for Boto 5 is based on a higher design rainfall estimate given the geotechnical sensitivity of the pit slope material to saturated conditions at the base of pit. The pumps would operate at these capacities for a few weeks each year. However, continuous pumping at these rates would not be expected for most of the wet season.

A geotechnical risk assessment is provided, and the greatest un-managed risks are considered to be for slope failure in the weathered slopes at Boto 5 in the absence of groundwater and surface water management. Residual risks are all reduced to low and moderate through mitigation measures as outlined in the recommendations for design implementation. These recommendations are an important part of the design and will assist in early identification of conditions which could lead to instability (as highlighted by the presented sensitivity analyses). Given the relatively small proportion of material sampled as part of this study relative to the volume of material which will constitute the pit walls, validation of the geotechnical conditions during the early stages of development will be important.

Table 16.3 Geotechnical Slope Design parameters – Malikoundi (constraining design criteria emboldened)

Horizon	Domain	Approximate height (m)/ Deign option	Maximum batter/bench face angle (BFA, °)	Maximum batter height / bench height (m)	Minimum berm / safety bench widths (m)	Inter-ramp angle (toe to toe, °)	Maximum Overall Slope Angle for Horizon (°)	Comment
Laterite caprock - ferricrete	All	Up to 5m	Vertical	-	-	-	-	Likely to require blasting, has the potential to "slab" on erosion of underlying lateritic soils
Transported and Saprolite	All	Up to 35m, without ramp	FW – 55 HW – 60	5	FW - 4.2 HW – 4.8	**33**	-	A BFA of 55 to 60° is recommended to promote surface water run-off. Inter-ramp angles constrain design.
		Up to 35m, with ramp	FW – 55 HW – 60	5	FW - 5.2 HW – 5.75	**30**	-	
Transition	Footwall	Up to ~12m	**55**	**10**	**6.5**	36.5	-	Double batter height option in Transition material could be trialled in phased designs at Malikoundi.
	Hangingwall	Up to ~20m for phase 3 (Phase 1 & 2 - 15 to 30m)	**60**	**10**	**6.5**	39.2	-	
Geotechnical berm at base of weathering	All	FW ~ 130mRL HW ~ 115mRL	-	-	**12 to 20m**	-	-	Recommended.
Fresh (~130 to -171m)	Hangingwall A and C, South Wall	With ramps	**75**	**20**	8.5 **(~10 to fit OSA in Pelite)**	55.3 **(52.5 with 10m berms)**	**46° within Fresh Pelite, including ramps**	Maximum overall slope angle within Pelite of 46° (including ramps) constrains slope angle in fresh. Allowing for two passes of ramp (2x23m), this equates to a required berm width of ~10m. Batter scale geometry constrains OSA within Sandstone unit at toe of slope.
	Hangingwall B	With ramps	**75**	**20**	8.5 **(~11 to fit OSA in Pelite)**	55.3 **(50.7 with 11m berms)**	**44° within Fresh Pelite, including ramps**	Reduced slope angle due to extended depth of Pelite (no Sandstone in lower slope)
	Footwall A and B	With ramps	**60**	**20**	**8.5**	44.9	**40° within Fresh**	
	Footwall C (Sandstone)	With ramps	**70**	**20**	**8.5**	51.7	-	Current design not geotechnically - constrained at inter-ramp to overall slope scales.

Note: Recommended that batter heights in Saprolite are limited to between 6 and 8m to limit erosion. The 5m quoted is based on preferred batter height combinations. Incorporation of drainage / surface water control on berms in weathered and transported materials is recommended. A geotechnical berm is recommended for incorporation at the base of the weathered material for

Malikoundi. The berm should have maintained access throughout the life of the pit, to allow access by mechanical equipment for clean-up of ravelling / eroded material, accommodation of surface water control systems, and other required in pit infrastructure.

Table 16.4 Geotechnical slope design parameters – Malikoundi North (constraining design criteria emboldened)

Horizon	Domain	Approximate height (m)/ Deign option	Maximum batter/bench face angle (BFA, °)	Maximum batter height / bench height (m)	Minimum berm / safety bench widths (m)	Inter-ramp angle (toe to toe, °)	Comment
Laterite caprock - ferricrete	All	Up to 5m	Vertical	-	-	-	Likely to require blasting, has the potential to "slab" on erosion of underlying lateritic soils
Transported and Saprolite	All	Up to 35m, without ramp	FW – 55 HW – 60	5	FW - 4.2 HW – 4.8	**33**	A BFA of 55 to 60° is recommended to promote surface water run-off. Inter-ramp angles constrain design.
		Up to 35m, with ramp	FW – 55 HW – 60	5	FW - 5.2 HW – 5.75	**30**	
Transition	Footwall	Up to ~20m	**55**	**10**	**6.5**	36.5	
	Hangingwall	Up to ~20m	**60**	**10**	**6.5**	39.2	
Geotechnical berm at base of weathering	All	~129mRL	-	-	**12m**	-	Optional for Malikoundi North given limited depth extent of pit and operational timeframe.
Fresh	Hangingwall (Int)	~30m, with or without ramp	**75**	**20**	**8.5**	55.3	
	Footwall	~30m, with or without ramp	**55**	**20**	**8.5**	41.6	Compliant design
		~30m, with or without ramp - **steeper option**	**60**	**20**	**8.5**	44.9	Exceeds PoF on batter scale (38%). Elevated risk may be considered appropriate given limited slope height.

Notes: - Recommended that batter heights in Saprolite are limited to between 6 and 8m to limit erosion. The 5m quoted is based on preferred batter height combinations. Incorporation of drainage / surface water control on berms in weathered and transported materials is recommended.

Table 16.5 Geotechnical Slope design parameters – Boto 5 (constraining design criteria emboldened)

Design sector	Horizon	Approximate height (m)/ Deign option	Maximum batter/bench face angle (BFA, °)	Maximum batter height / bench height (m)	Minimum berm / safety bench widths (m)	Inter-ramp angle (toe to toe, °)	Comment
All	Laterite caprock - ferricrete	Up to 5m	Vertical	-	-	-	Likely to require blasting, has the potential to "slab" on erosion of underlying lateritic soils
Footwall	Saprolite, up to ~40m	With ramp	50	5	4.1	**29**	A BFA of 50 is recommended to promote surface water run-off while minimising potential undercutting from relic structure. Inter-ramp angles constrain design.
		Without ramp	50	5	4.8	**31**	
	Transition	With or without ramps	**55**	**10**	**6.5**	36.5	BFA constrained by kinematic analysis
	Fresh		**55**	**20**	**8.5**	41.6	Compliant design
			60	**20**	**8.5**	44.9	Exceeds PoF on batter scale (43%). Elevated risk may be considered appropriate given limited slope height.
West wall	Saprolite, up to ~40m	With ramp	55	5	5.5	**29**	A BFA of 55 is recommended to promote surface water run-off, with batter heights of 6 to 8m. These BFAs for 8m high batters would require berm widths of 7.5 to 9m to maintain IRAs.
		Without ramp	55	5	4.8	**31**	
	Transition	Without ramp	**60**	**10**	**6.5**	39.2	
Hangingwall East	Saprolite – Soft to 50m	With ramp	55	5	6.75	**26**	A BFA of 55 is recommended to promote surface water run-off. Inter-ramp angles constrain design.
		Without ramp	55	5	5.5	**29**	
	Saprolite – Hard to base of pit (69mRL)	With ramp	60	5	4	**36**	Inter-ramp angles constrain design.
Hangingwall West	Saprolite – Soft to 50m	With ramp	55	5	6.75	**26**	A BFA of 55 to 60° is recommended to promote surface water run-off, with batter heights of 6 to 8m. These BFAs for 8m high batters would require berm widths of 9 to 12m to maintain IRAs.
		Without ramp	55	5	5.5	**29**	
	Geotechnical berm at 150m RL	All	-	-	**30**		Required for overall slope scale stability
	Saprolite – Hard to base of pit (69mRL)	With ramp	60	5	5.5	**31**	Berm width of 6.5m required not to exceed IRA.
	Geotechnical berm at 100mRL	All	-	-	**30**	**-**	Required for overall slope scale stability of pit below 90mRL. Not required where pit bottom is above 90mRL

Notes: - Recommended that batter heights in Saprolite are limited to between 6 and 8m to limit erosion. The 5m quoted is based on preferred batter height combinations. Incorporation of drainage/surface water control on berms in weathered and transported materials is recommended.

Figure 16.1 **Malikoundi (Phase 3) PFS Pit design coloured by weathering grade/geology and showing Slope Design Sectors**

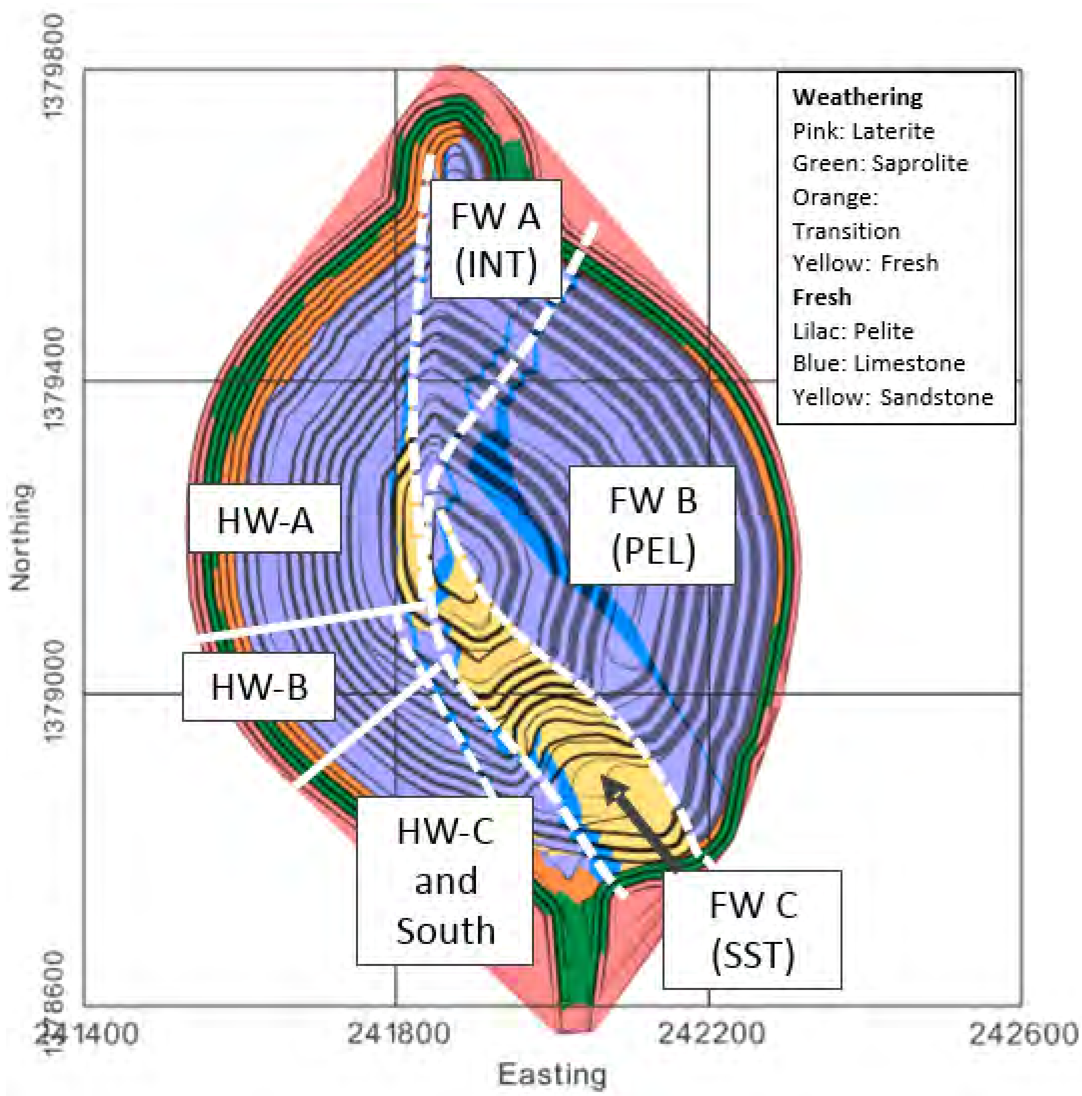


Figure 16.2 Malikoundi North PFS Pit Design Coloured by Weathering Grade/Geology and showing Slope Design Sectors

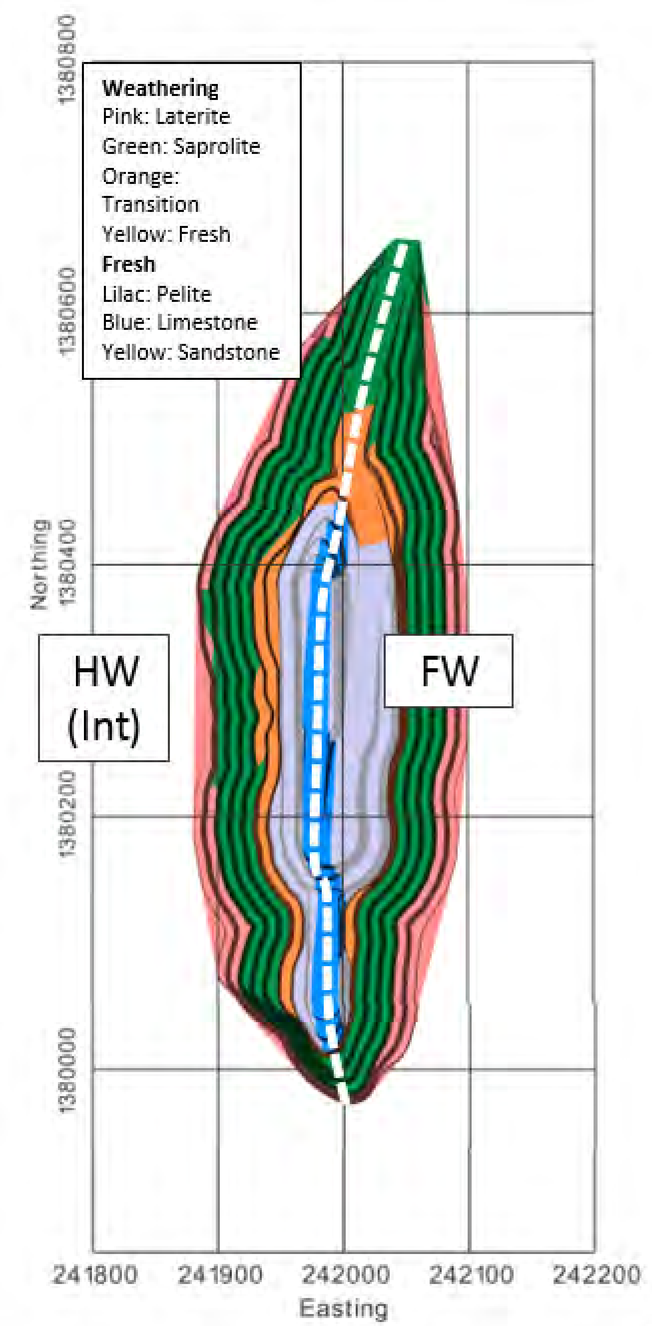
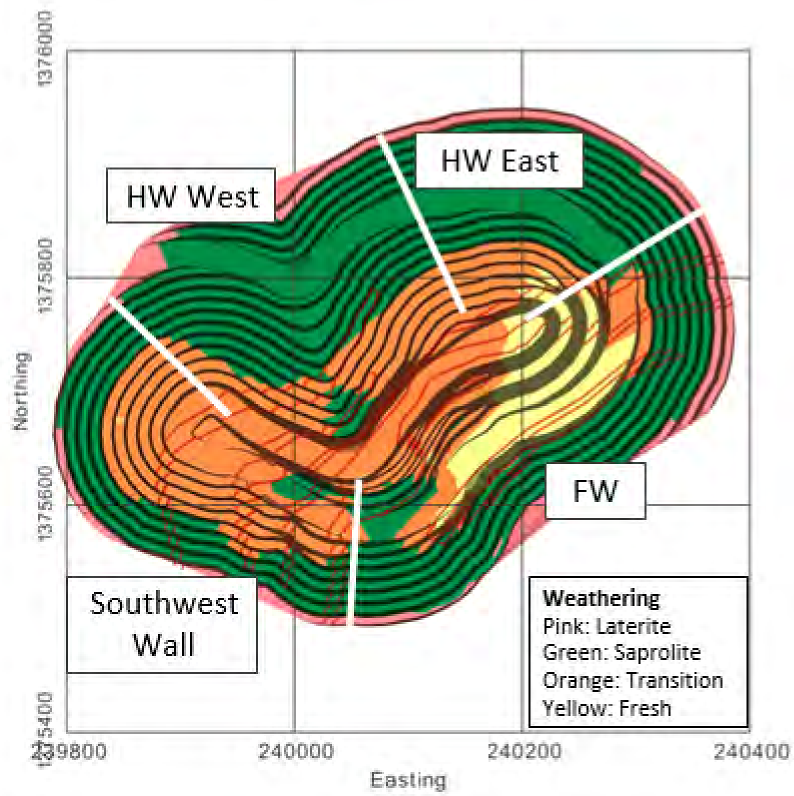


Figure 16.3 Boto 5 PFS Pit Design Coloured by Weathering Grade/Geology and showing Slope Design Sectors



16.4 Economic Pit Shell Development

The open pit ultimate size and phasing requirements were determined with various input parameters including estimates of the expected mining, processing and G&A costs, as well as metallurgical recoveries, pit slopes and reasonable long-term metal price assumptions. AGP worked together with IAMGOLD personnel to select appropriate operating cost parameters for both Boto 5 and Malikoundi. The mining costs are estimates based on cost estimates for equipment from vendors and the previous PFS completed by AGP. The costs represent what is expected as a blended cost over the life of the mine for all material types to the various dump locations. Process and G&A costs were provided by Lycopodium based on Lycopodium's own study work.

The parameters used are shown in Table 16.6.

Table 16.6 Pit Optimization Parameters

Parameter	Units	Malikoundi Pit	Boto 5 Pit
Metal Prices			
Gold Price	$/oz	1200	1200
Payable	%	99%	99%
Participation (on profits)	%	90%	90%
Transportation & Refining	$/oz	3.04	3.04
Royalty	%	4%	4%
General			
Resources blocks used		M+I	M+I
General & Administration Cost	$/t ore	4.35	4.35
Process Recovery			
Laterite (ROCK=40)	%	0.0%	0.0%
Saprolite (ROCK=50)	%	94.8%	92.3%
Transition (ROCK=60)	%	92.1%	93.8%
Fresh Rock (ROCK=70)	%	89.8%	95.5%
Process Costs *			
Laterite Process Cost	$/t	-	-
Saprolite Process Cost	$/t	10.09	10.09
Transition Process Cost	$/t	10.73	10.73
Fresh Rock Process Costs	$/t	14.34	14.34
Mining Costs **			
Incremental haul cost - waste	$/5m bench	0.022	-
Incremental haul cost - ore	$/5m bench	0.028	-
Waste		Ref. elev = 160	Ref. elev = 200
Laterite	$/t	1.34	2.24
Saprolite	$/t	1.34	2.24
Transition	$/t	1.67	2.24
Fresh Rock	$/t	1.84	2.24
Ore			
Laterite	$/t	1.41	2.24
Saprolite	$/t	1.41	2.24
Transition	$/t	1.75	2.24
Fresh Rock	$/t	1.92	2.24

** process costs based on 2.75 Mt/y dry throughput*
*** mining costs based on using 95 t haul trucks, and contractor in Boto 5 using 40 t trucks*

All values are in United States dollars unless otherwise noted.

The mining cost estimates are based on the use of 95 t trucks using the PFS study waste dump configuration to determine incremental hauls for ore and waste. The Boto 5 cost is based on mining only the saprolite and transition material using a contractor. It was a fixed cost estimate to cover the mining of the entire pit.

Wall slopes for pit optimization were based on the work completed by AG and discussed in Section 16.3. Allowances were made for ramps in the slopes to determine an overall angle for use in the Lerch-Grossman routine. The overall slope angle calculations are shown in Table 16.7.

Table 16.7 Pit Optimization Slope Angles

Rock Domains Horizon	Sector	Sector Code (GEOT1)	Slope Domain (GEOT)	Face Angle (degrees)	Height between berms (m)	Catch Bench Width (m)	Inter-Ramp Angle (IRA) (degrees)	# Ramps in Slope	Slope Height (m)	Overall Slope Angle (degrees)
Malikoundi Pit										
Saprolite/Laterite	HW	3,6	11	60	5	4.8	33	0	30	33
Saprolite/Laterite	FW	4,5,7	12	60	5	4.8	33	0	30	33
Transition	HW	3,6	13	60	10	6.5	39.2	0	20	39.2
Transition	FW	4,5,7	14	55	10	6.5	36.5	0	20	36.5
Fresh	HW + south wall	3,6	15	75	20	8.5	55.3	3	280	44.3
Fresh	FWA(int)+FWB(pelite)	4,7	16	60	20	8.5	44.9	3	300	37.3
Fresh	FWC(sandstone)	5	17	70	20	8.5	51.7	3	300	42.3
Malikoundi North Pit										
Saprolite/Laterite	HW	1	1	60	5	4.8	33.0	0	40	33
Saprolite/Laterite	FW	2	2	60	5	4.8	33.0	0	40	33
Transition	HW	1	3	60	10	6.5	39.2	0	20	39.2
Transition	FW	2	4	55	10	6.5	36.5	0	20	36.5
Fresh	HW	1	5	75	20	8.5	55.3	0	30	55.3
Fresh	FW	2	6	55	20	8.5	41.6	1	30	25.8
Boto 5 Pit										
Saprolite/Laterite	FW		1	55	5	4.9	30.8	0	50	30.8
Transition	FW		2	55	10	6.5	36.5	1	30	23.6
Saprolite/Laterite	West Wall		3	55	5	4.9	30.8	0	40	30.8
Transition	West Wall		4	55	10	5.5	39.3	1	40	27.4
Saprolite/Laterite	HW West		5	55	5	5.5	29.1	1	70	21.4
Transition	HW West		6	55	10	10	30.5	0	40	30.5
Saprolite/Laterite	HW East		7	55	5	5.5	29.1	1	55	23.4
Transition	HW East		8	55	10	6.5	36.5	0	40	36.5
Fresh	All		9	55	20	8.5	41.6	0	40	41.6

The pit design sectors by pit are shown in the following figures.

Figure 16.4 Malikoundi Slope Sectors

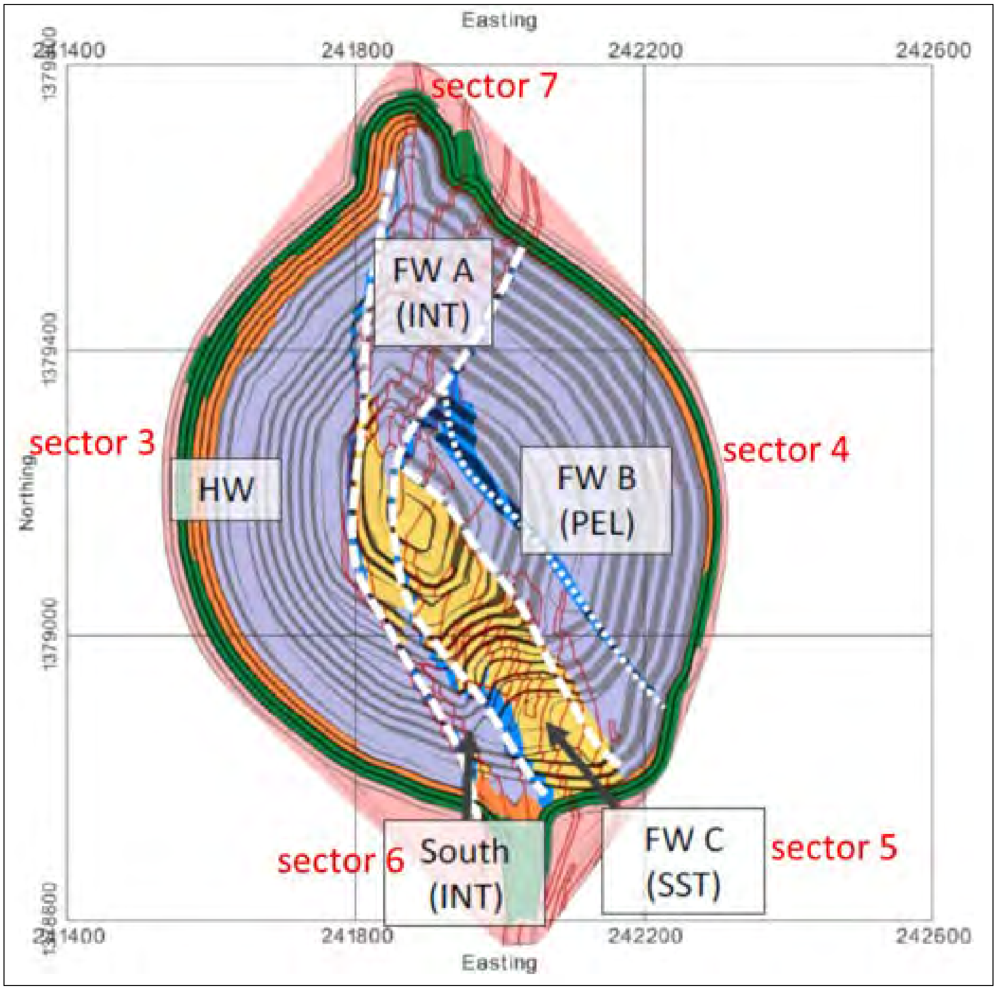


Figure 16.5 Malikoundi North Slope Sectors

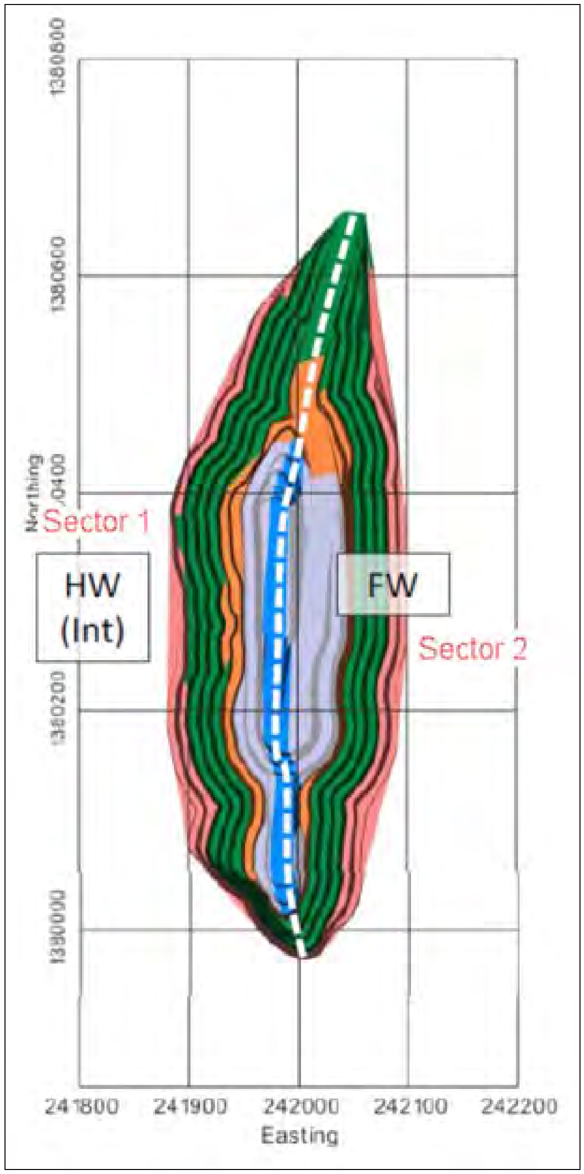
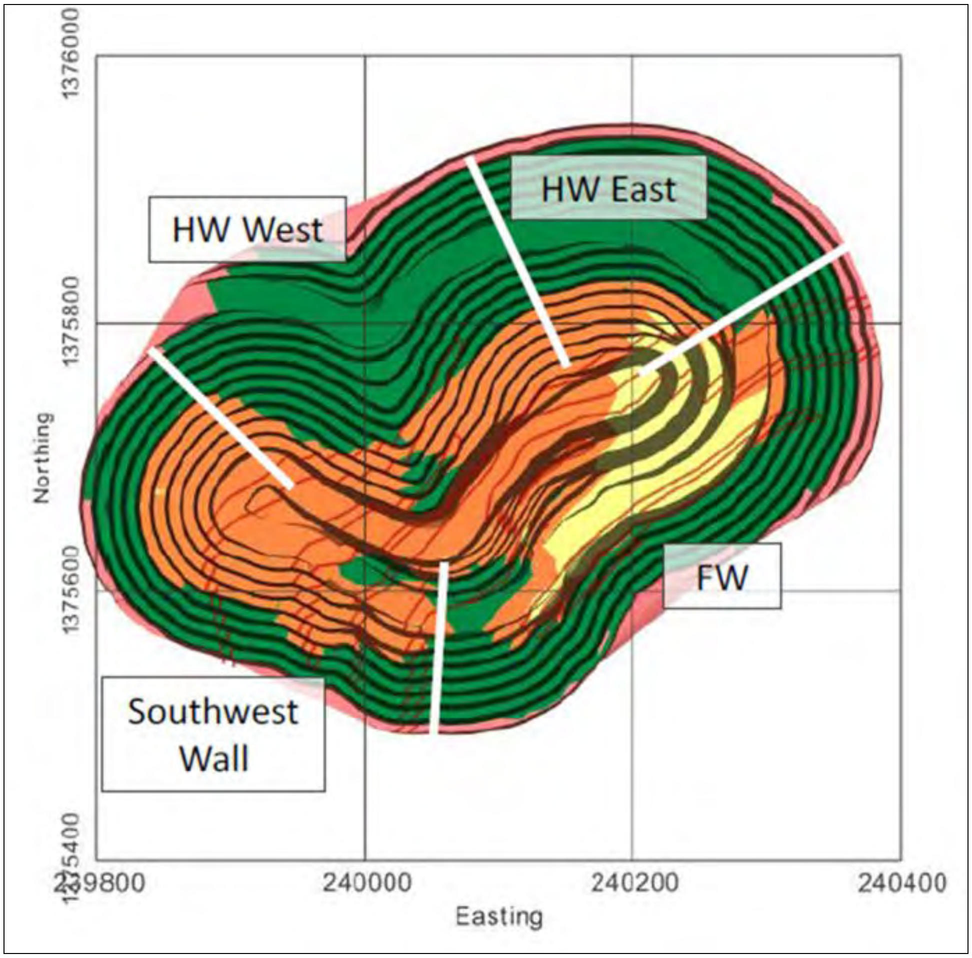


Figure 16.6 Boto 5 Slope Sectors



Pit shells for Malikoundi and Boto 5 were generated to examine sensitivity to the gold price with a target of \$1,200/oz. This was to gain an understanding of the deposit and highlight potential opportunities in the design process to follow. Only undiluted Measured and Indicated material was used in the analysis. The gold price was varied from \$300/oz up to \$1,440/oz to determine the size of the shell. All other parameters were fixed. This was intended to visualize any natural breakpoints in the deposit and assist in phase development. The net profit before capital for each pit was calculated on an undiscounted basis for each pit shell using \$1,200/oz as the revenue. Ore/waste t and net profit were plotted against gold price. Graphs of the two areas are shown in the following figures.

Figure 16.7 Malikoundi Profit vs. Price by Pit Shell

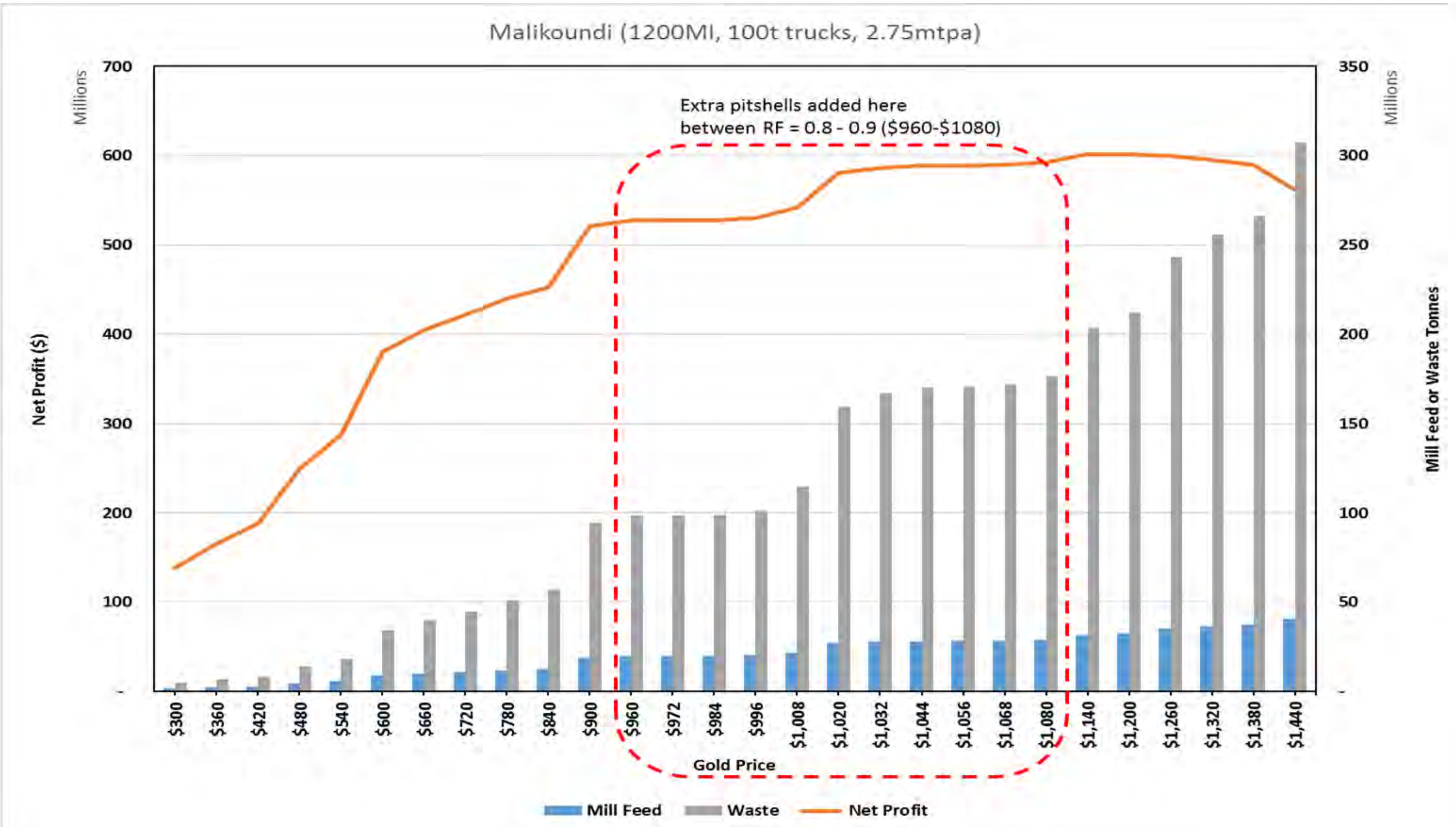


Figure 16.8 Boto 5 Profit vs. Price by Pit Shell

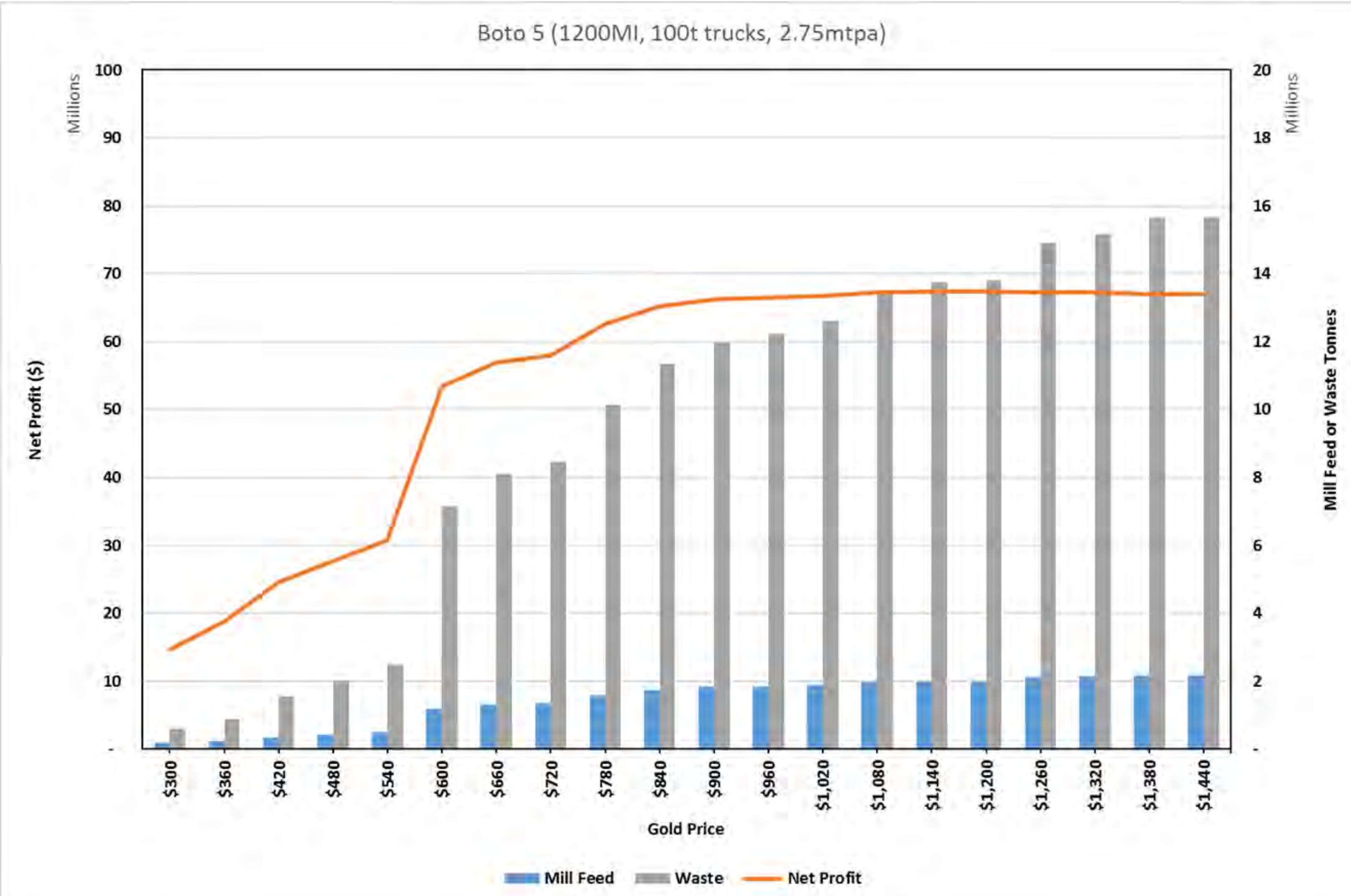


Figure 16.7 illustrates various break points in the pit shells. With the increase in the waste tonnage, and to a lesser degree the mill tonnage, the net profit also increased. In the case of the first break point shown at \$540/oz Au, the cumulative waste tonnage is 18.1 Mt, with a corresponding mill feed tonnage of 5.9 Mt or a strip ratio of 3.07:1. The net profit also increased beyond this point showing that there was still value to be obtained by going with a higher metal price or an additional phase. This break point represented 48% of the net value of a \$1,200/oz pit but with only 9% of the waste of the larger pit shell.

The second break point was at \$840/oz. The incremental waste tonnage from the first break point is 38.6 Mt, with a corresponding increase in mill feed tonnage of 6.9 Mt or a strip ratio of 5.6:1. The net profit also increased beyond this point showing that there was still value to be obtained by going with a higher metal price. This pit shell was used for the design of the Malikoundi phase 2 pit. There is a significant waste tonnage in the next higher pit price to achieve the next increase in profit. The cumulative value of the two break points was 75% of the \$1,200/oz pit shell but with only 27% of the waste movement of the larger pit required.

The third and final major break point was at \$1,044/oz gold. This resulted in a substantial jump in the waste tonnage from the second break point by 113.3 Mt with a gain of 15.3 Mt of ore for an incremental strip ratio of 7.39:1. The net profit continues to increase beyond the third break point, although at a flatter rate than in earlier breakpoints. The additional potential pit value was considered negligible when considering schedule discounting and the accuracy of the analysis. This pit shell was used for the design of the Malikoundi phase 3 pit. The cumulative strip ratio of material within the \$1,044/oz gold pit shell is 6.03:1. This pit mines 98% of the value of the larger \$1,200/oz pit shell and only 80% of the waste. Increasing the pit size further only gains another 2% in net revenue for 20% of the waste. This was not considered to be a reasonable exchange.

Figure 16.8 shows the result for the Boto 5 pit. A single pit due to the small size of Boto 5 was selected with the mining costs associated with the 40t haul trucks and process costs associated with a 2.75 Mtpa throughput. The \$900/oz shell pit was used for design purposes. The cumulative waste tonnage for the selected pit shell is 12.0 Mt, with a corresponding mill feed tonnage of 1.8 Mt or a strip ratio of 6.6:1. This represented 98% of the net value of the \$1,200/oz pit with 87% of the contained waste.

The Malikoundi North ultimate pit used was the \$960/oz shell. Deeper shells required flatter slopes to accommodate the required road lengths and switchbacks. This was not considered to be a reasonable trade-off, so the lower value pit shell was used.

16.5 Dilution Calculation

The geologic model provided was a whole block, fully diluted grade model. This means the grade from the wire frames was diluted over the full volume of the block to arrive at a diluted block grade.

The geologic model had been created with grade wireframes prior to assigning the grade into a whole block. AGP believed that this did not adequately reflect the amount of dilution that would be expected with normal mining practice, even with more selective equipment.

AGP also believed that contact dilution would play a role in material sent to the mill. To determine the amount of dilution and the grade of the dilution the size of the block in the model was examined. The Malikoundi block size within the model was 5 m in the dip direction, 10 m along strike, and 5 m high.

The percentage of dilution is calculated for each contact side using an assumed 0.5 m contact dilution distance. If one side of the block is touching waste, then it is estimated that dilution of 10% would result. If two sides are contacting, it would rise to 15%. Three sides would be 20%, and four sides 30%. Four sides represent an isolated block of ore. This assumes a development of the block on the hangingwall side first, then the two sides and finally the footwall.

For Boto 5, the block model was equally sized at 5 m on each side (dip, strike and height). The dilution calculation used the same 0.5 m of contract dilution. The resulting dilution percentages were different with one side equal to 10%, two sides at 20%, three sides at 30% and four sides at 40%. The same development sequence was assumed.

Because the model contained whole blocks already, the percentage of dilution could be estimated and then included in the block ore percentage item. The mining model was modified to include an ore percent item, and any blocks with a grade above the marginal cut-off grade were assigned an ore percent of 100% (deemed entirely ore). The marginal cut-off grades are shown for each weathered material type and pit in Table 16.8.

Table 16.8 Marginal Cut-off Grades

Weathered Material	Malikoundi Au (g/t)	Boto 5 Au (g/t)
Laterite	n/a	n/a
Saprolite	0.46	0.48
Transition	0.50	0.49
Fresh	0.63	0.59

MineSight has a routine that enables the user to query surrounding blocks against a set of conditions. For the dilution percentage calculation, the procedure was run to determine how many ore blocks contacted a waste block, which determined the dilution percentage to apply. This was stored in the waste block and the waste block grade used as the diluting value. If a waste block was only surrounded by other waste blocks, the dilution percentage was zero.

In this manner, the contact blocks could be included in the tonnage and grade calculation of ore tonnes. The ore tonnage was then run with the block model DORE% item to report out the proper tonnes and grade.

Comparing the in-situ to the diluted values for the designed pit phases showed an increase in ore tonnage along with a lowering of gold grade. For the Malikoundi pit (phases 1-3 and Malikoundi North), the diluted ore contained 4.5% more t and 4.1% lower gold grade than the in-situ ore summary. For the Boto 5 pit, the diluted ore contained 7.8% more t and 6.7% lower gold grade than the in-situ ore summary. The grade dilution is lower due to the waste blocks containing some mineralization. The ore tonnage contact dilution is lower than the one contact block dilution percentage due to the thickness of the ore where not all the ore blocks are diluted with waste. The “internal dilution” from the conversion of wireframe to whole block was not reported separately within the reported in-situ summaries otherwise the dilution tonnages would have been higher.

Tonnes and grade for the pit designs are reported with the diluted tonnes and grade calculated with this method.

16.6 Pit Design

Pit designs were developed for the Boto pit areas – Malikoundi, Malikoundi North and Boto 5 pits. The pit optimization shells used to determine the ultimate pits were also used to outline areas of higher value for targeted early mining and phase development.

Geotechnical parameters outlined in Section 16.3 were used for each of the weathering zones and are shown again in Table 16.9.

Table 16.9 Pit Slope Parameters for Detail Design

Rock Domains Horizon	Sector	Sector Code (GEOT1)	Slope Domain (GEOT)	Face Angle (degrees)	Height between berms (m)	Catch Bench Width (m)	Inter-Ramp Angle (IRA) (degrees)
Malikoundi Pit							
Saprolite/Laterite	HW	3,6	11	60	5	4.8	33
Saprolite/Laterite	FW	4,5,7	12	60	5	4.8	33
Transition	HW	3,6	13	60	10	6.5	39.2
Transition	FW	4,5,7	14	55	10	6.5	36.5
Fresh	HW + south wall	3,6	15	75	20	8.5	55.3
Fresh	FWA(int)+FWB(pelite)	4,7	16	60	20	8.5	44.9
Fresh	FWC(sandstone)	5	17	70	20	8.5	51.7
Malikoundi North Pit							
Saprolite/Laterite	HW	1	1	60	5	4.8	33.0
Saprolite/Laterite	FW	2	2	60	5	4.8	33.0
Transition	HW	1	3	60	10	6.5	39.2
Transition	FW	2	4	55	10	6.5	36.5
Fresh	HW	1	5	75	20	8.5	55.3
Fresh	FW	2	6	55	20	8.5	41.6
Boto 5 Pit							
Saprolite/Laterite	FW		1	55	5	4.9	30.8
Transition	FW		2	55	10	6.5	36.5
Saprolite/Laterite	West Wall		3	55	5	4.9	30.8
Transition	West Wall		4	55	10	5.2	39.3
Saprolite/Laterite	HW West		5	55	5	5.5	29.1
Transition	HW West		6	55	10	10	30.5
Saprolite/Laterite	HW East		7	55	5	5.5	29.1
Transition	HW East		8	55	10	6.5	36.5
Fresh	All		9	55	20	8.5	41.6

Geotechnical berms of 12 m to 20 m in width were designed in the Malikoundi and Malikoundi North pits at the base of the weathering (transition) zone. For Boto 5, flatter slopes on the hangingwall material were recommended and ramps were incorporated in this material to act as geotechnical berms. In addition, a geotechnical berm was recommended at the 150 level in Boto 5 with a width of 30 m.

Equipment sizing for ramps and working benches is based on the use of 95 tonne rigid frame haul trucks. The operating width used for the truck is 6.9 m. This means that single lane access is 21.4 m (2x operating width plus berm and ditch) and double lane widths are 28.3 m (3x operating width plus berm and ditch). Ramp gradients are 10% in the pit for uphill gradients and 8% uphill on the dump access roads. Working benches were designed for 35 m to 40 m minimum on pushbacks, although some pushbacks in the Malikoundi pit did work in a retreat manner to facilitate access.

Boto 5 was envisaged as a contract mining operation using 40 tonne trucks, but the same road and ramp requirements were applied as for Malikoundi.

The Malikoundi pit is designed as 4 phases within the main pit. Phase 0 as the initial pit is called is a subset of Phase 1 to drive quickly to fresh rock for tailings dam construction purposes. Malikoundi North is a single-phase pit as is Boto 5. The final design phase tonnages and grades are shown in Table 16.10.

Table 16.10 Final Design – Phase Tonnages and Grades

Pit	Ore (Mt)	Au (g/t)	Waste (Mt)	Total (Mt)	Strip Ratio
Malikoundi Phase 0	1.6	2.2	9.0	10.6	5.45
Malikoundi Phase 1	3.8	2.14	8.7	12.5	2.29
Malikoundi Phase 2	7.8	1.71	46.4	54.2	5.93
Malikoundi Phase 3	17.8	1.52	111.2	129.0	6.24
Malikoundi North	2.0	1.88	14.8	16.8	7.55
Boto 5	2.0	2.03	14.3	16.3	7.09
Total	**35.1**	**1.71**	**204.3**	**239.4**	**5.83**

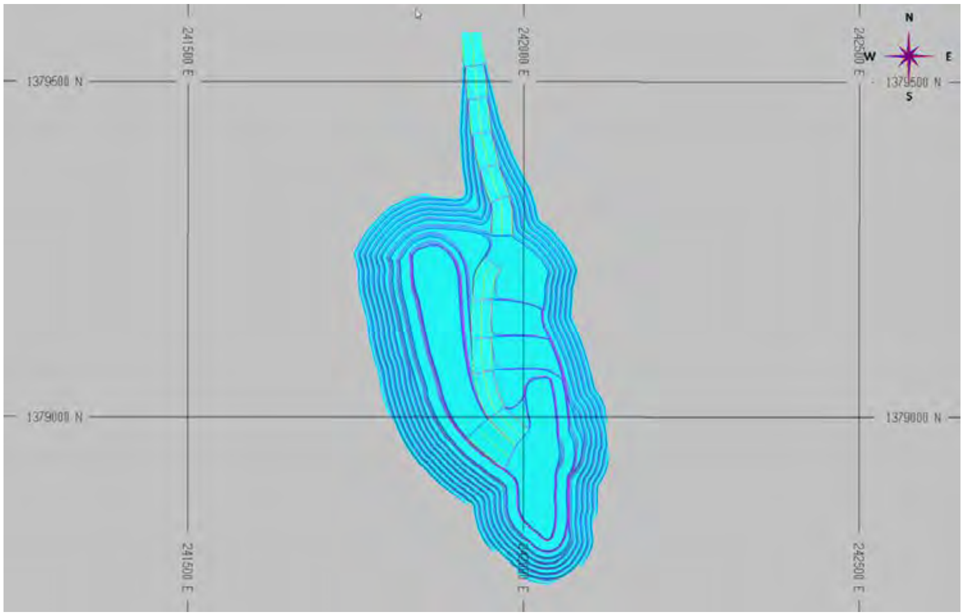
The resources for the Malikoundi pit phases are based on lower cut-off grades of 0.46 g/t Au for saprolite, 0.50 g/t Au for transition material, and 0.63 g/t Au for rock. The resources for the Boto 5 pit are based on lower cut-off grades of 0.48 g/t Au for saprolite, 0.49 g/t Au for transition material, and 0.59 g/t Au for rock. The schedule has a high-grade cut-off of 1 g/t Au for all material types to separate high-grade from low-grade.

The phase designs are described in further detail below:

16.6.1 Malikoundi – Phase 0

Phase 0 is a subset of Phase 1 with the purpose of driving quickly to fresh rock for construction purposes in addition to material for road and other infrastructure items. It is the first phase mined in the Project. The Phase elevations range from 165 masl to the pit bottom of 105 masl. The design is shown in Figure 16.9.

Figure 16.9 Malikoundi – Phase 0



16.6.2 Malikoundi – Phase 1

Phase 1 expands on Phase 0 and is also developed from the north and advanced south. It has elevations ranging from 165 masl to a pit bottom of 60 masl. This is primary ore source early in the mine life. Phase 0 is shown with Phase 1 in Figure 16.10 and Phase 1 alone in Figure 16.11. As is shown in the Figures, the ramp starts further to the north to avoid disrupting material flow from Phase 0.

Figure 16.10 Malikoundi - Phase 0 and Phase 1 Together

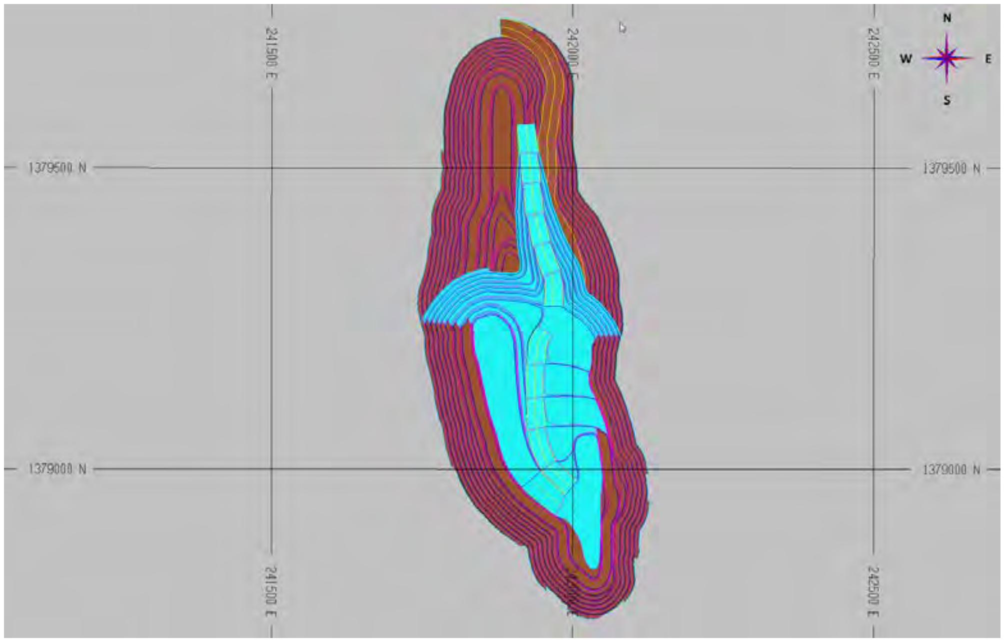
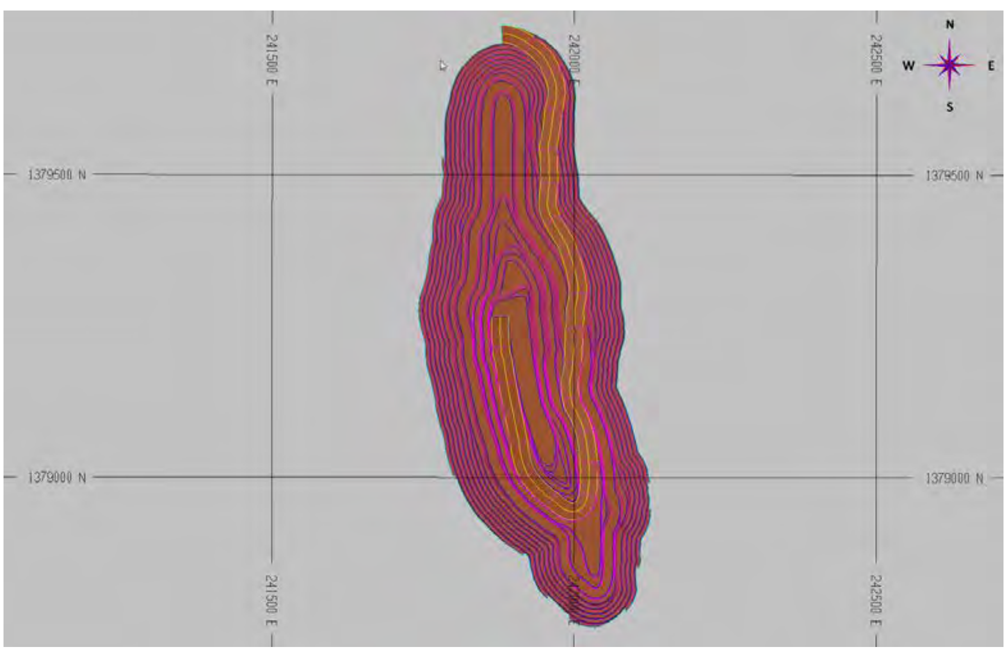


Figure 16.11 Malikoundi – Phase 1



16.6.3 Malikoundi – Phase 2

Phase 2 is also accessed from the north but slightly further than Phase 1. Phase 2 is unique in that it has multiple exits to allow mining from both the hangingwall and footwall at the same time as mining in Phase 1 is advancing. This provides the mine planning team the flexibility to advance one side over the other quickly to accommodate rainy seasons and also the future development of Phase 2.

Two figures are shown to shown Phase 1 and 2 together and how they interact and then a figure of Phase 2 alone to show the various access points. Phase 2 has elevation limits of 165 masl to a depth of -40 masl.

Figure 16.12 Malikoundi - Phase 1 and Phase 2

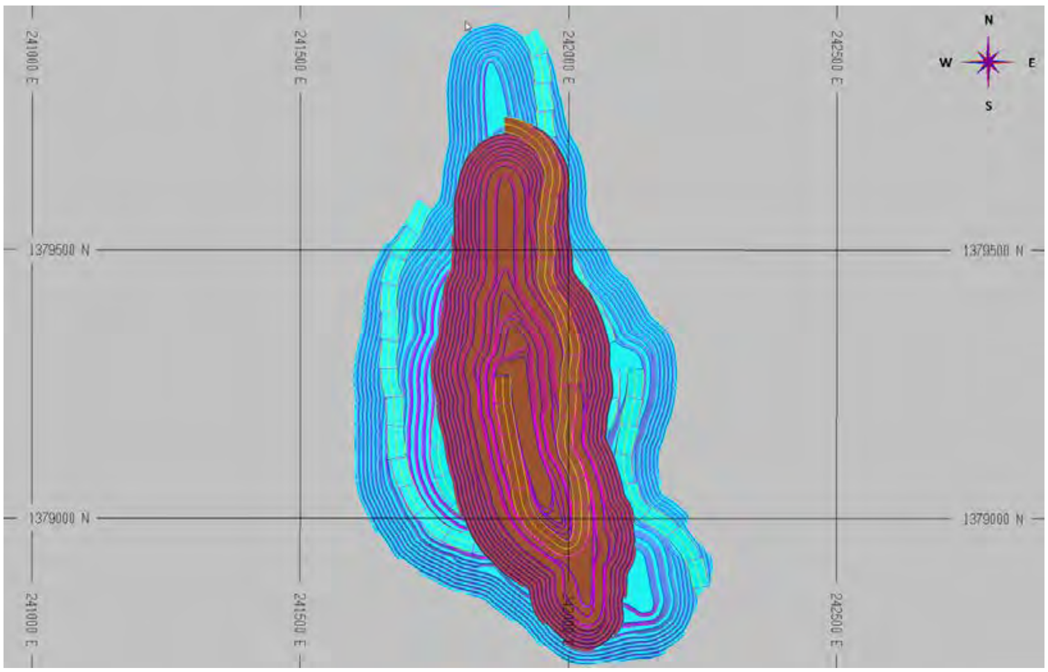
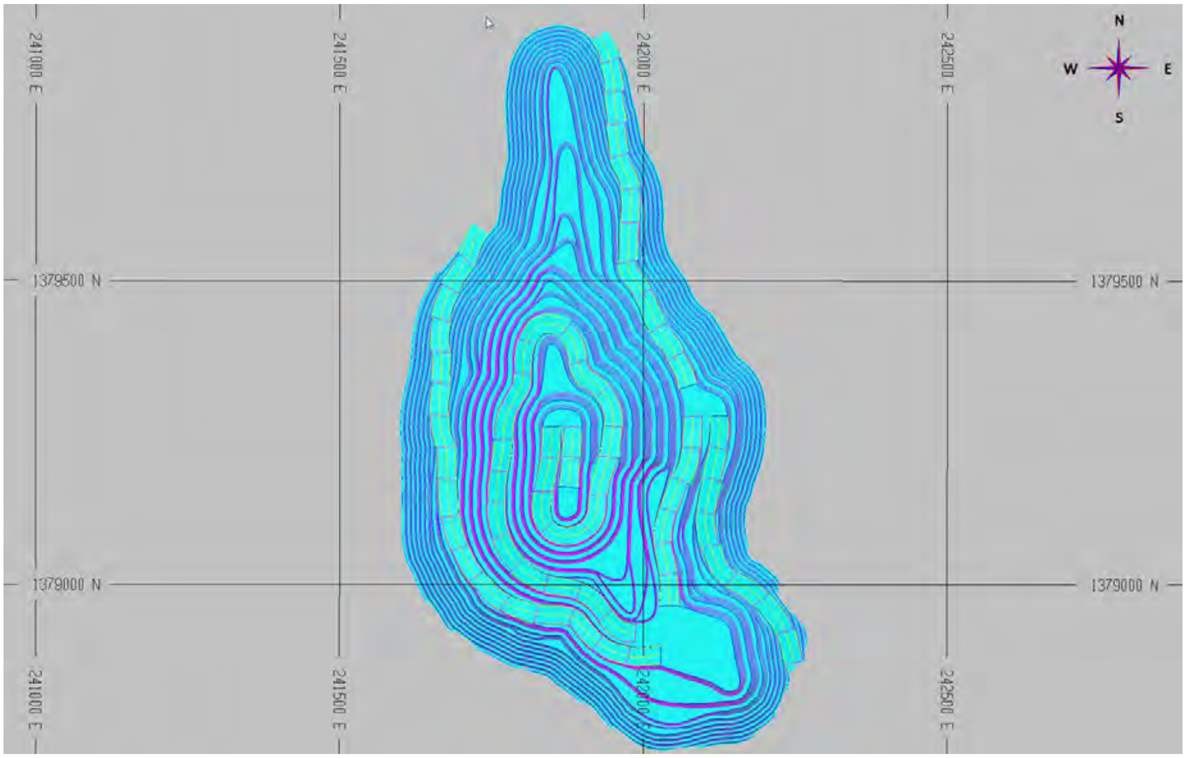


Figure 16.13 Malikoundi - Phase 2



16.6.4 Malikoundi – Phase 3

Phase 3 is a deepening of the southern area of Phase 2. This will be a single access phase to save on waste and exits to topography on the south side. The geotechnical berm is evident in the design at the base of the transition zone. The multiple access points of Phase 2 assist Phase 3 by not disrupting material movement from Phase 2 and offer the ability to shorten haul profiles at different times in the mine life. Phase 3 extends from the same 165 masl level to a final depth of -160 masl or 325 m at the deepest point. Phase 2 and Phase 3 are shown together in Figure 16.14 and Phase 3 is shown alone in Figure 16.15.

Figure 16.14 Malikoundi – Phase 2 and Phase 3

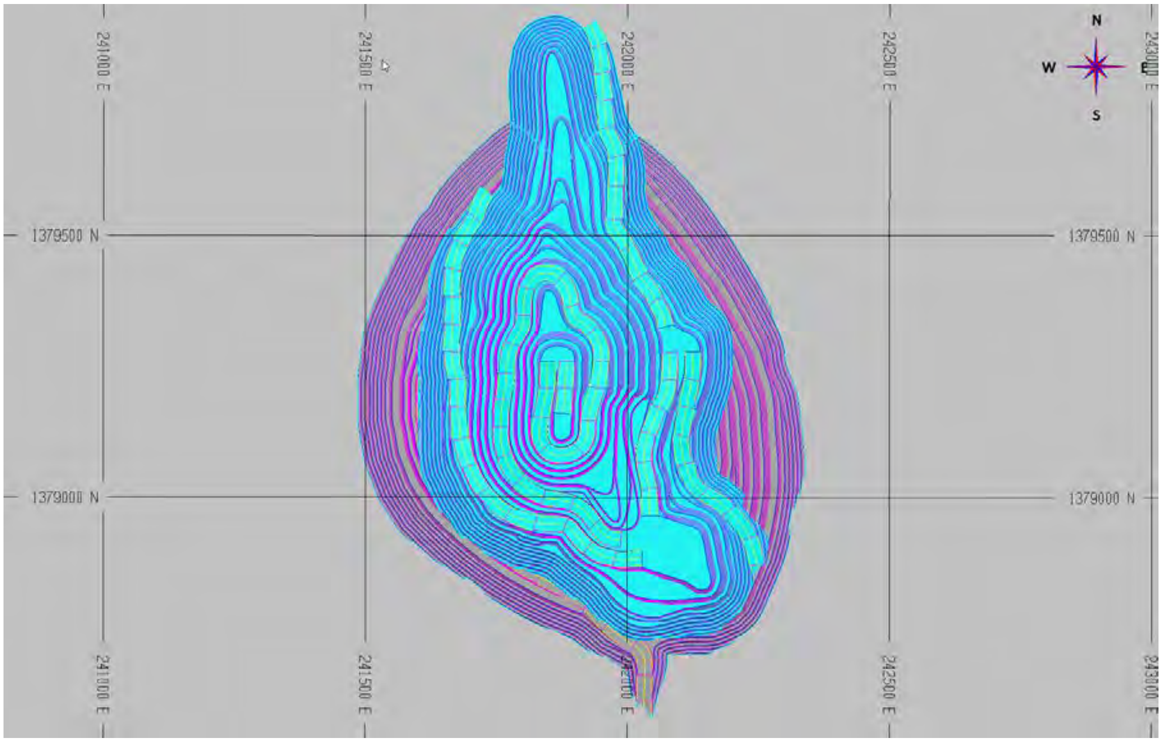
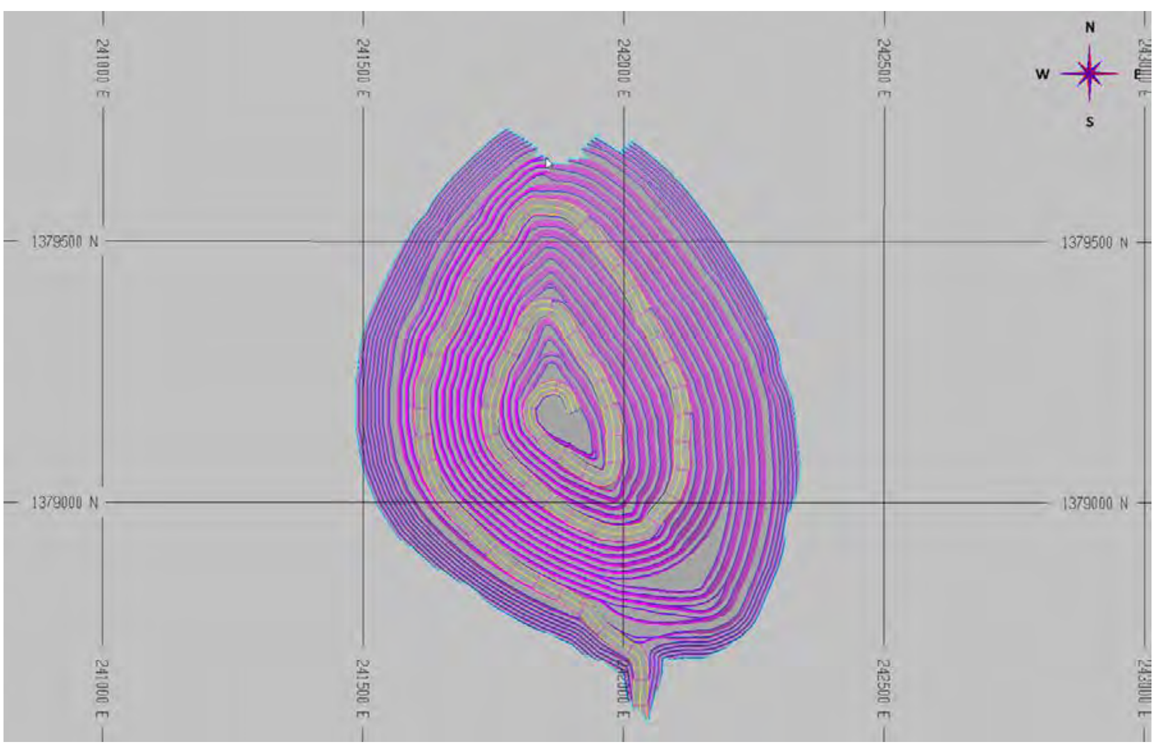


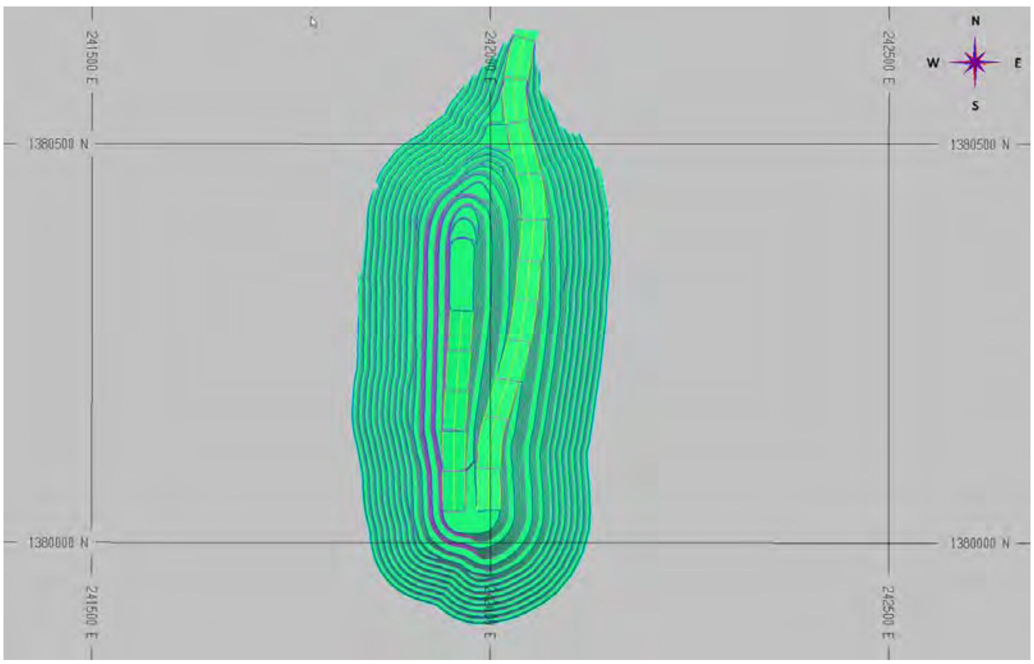
Figure 16.15 Malikoundi – Phase 3



16.6.5 Malikoundi North

The Malikoundi North pit is designed as a separate satellite pit to the north of the main Malikoundi pit. It consists of a single small, narrow mining phase that is accessed with a slot from the north end of the pit. The haul road will be to full operating width, except for the single lane for the bottom benches. This phase is located immediately to the east of the proposed water storage facility. Malikoundi North has an elevation variation from 160 masl to 50 masl. The ultimate pit is shown in Figure 16.16.

Figure 16.16 Malikoundi North



16.6.6 Boto 5

The Boto 5 phase was designed as a separate satellite pit to the south of the main Malikoundi pit. It consists of a single mining phase that is accessed with a wrap-around ramp access from the northwest corner of the pit. The haul road will be to full operating width, except for the single lane for the bottom benches. The ultimate pit is shown in Figure 16.17. The pit ranges in elevation from 205 masl to 85 masl.

Figure 16.17 Boto 5

16.7 Mine Schedule

The mine schedule delivers 35.1 Mt of ore grading 1.71 g/t gold to the mill over a mine life of 12.8 years. Waste tonnage totalling 204.3 Mt will be placed into waste rock management facilities. The overall strip ratio is 5.83:1 life of mine.

The mine schedule utilizes the pit and phase designs described previously to send a maximum of 2.75 Mtpa of ore to the mill facility. Ore is a mixture of saprolite, transition and hard rock.

The current mine life includes 15 months of pre-stripping and 12.8 years of mining. The final year will clear the stockpiled ore. The stockpiled ore, together with pit phasing will be utilized to ensure sufficient mill feed is available in the rainy season. This will also be coupled with in-pit sumps and surface ditches around the pits. Phases will be advanced quickly in the dry season to provide temporary water storage after a rainfall event that pumping will remove in the wet season.

When mining starts, various infrastructure items are required. These include; establishing proper roads to the TMF, FWP, to the plant and to the various waste management facilities. Boto 5 will require a 4.7 km access road/ore haulroad to be developed plus access to the pit area. The TMF areas require another 4.8 km of road and mine roads connecting these various areas total 2.8 km.

A fresh water storage facility will be built in the valley to the north of the plant in the preproduction period. Operationally, ditching around the pits to intercept surface run-off and wider ramps with rock capping foundations will help to minimize reductions in mine production.

Year -2 is only a three-month period beginning at the end of the rainy season. In this time period, a total of 2.3 Mt of material will be moved as the project ramps up. Year -1 mining brings the total material movement to a total of 13.7 Mt moved. This includes the development of a 2.2 Mt ore stockpile grading 2.25 g/t Au in anticipation of plant commissioning and operation. Phase 0 and Phase 1 will be the only active phases in the preproduction period.

Project activities in the pre-production period include:

- Haul road construction.
- FWP construction.
- TMF material placement for the starter portions of the west and central embankments.
- Initiation of mining in Malikoundi Phase 0 and Phase 1
- Development of an ore stockpile at plant for commissioning and operations.

Year 1 production is based on the assumption that the plant will require 3 months to achieve full production levels. The first month the plant will be capable of 60% of capacity. The second month 80% and the third month 90%. Subsequent months will be at 100% of nameplate capacity in the mill. This ramp-up schedule requires the Year 1 production to be 2.59 Mt. Ore mining will be from stockpile, Boto 5 and Malikoundi Phase 0, 1 and 2. Mining will be initiated in Malikoundi Phase 2, Malikoundi North and Boto 5. Boto 5 will be mined by contractor using smaller 40 t trucks.

Year 2 production is at the full 2.75 Mt of ore to the mill and continued stripping of Boto 5, and Malikoundi Phases 1 and 2 and Malikoundi North. Malikoundi Phase 3 will also start pre-strip activity. Boto 5 will advance to the 125 level. Malikoundi Phase 1 will be completed this year with a final pit elevation of 60 masl. Malikoundi Phase 2 will be at the 105 level and Phase 3 at 145 level. Malikoundi North finishes the year at the 115 level. Waste from Phase 3 is directed to the dump to the east of the Malikoundi Pit.

Year 3 production is the final year for Boto 5 with its final level being 85 masl. Malikoundi Phase 2 is the dominant phase of mining at this time driving to a depth of 75 masl. Phase 3 continues to advance behind Phase 2 to the 125 level. Malikoundi North is used intermittently and only drops 10 m to the 105 level.

Year 4 maintains mining in Malikoundi Phase 2 and 3 plus Malikoundi North. They each reach the following levels respectively; 40 masl, 100 masl, and 75 masl.

Year 5 sees Malikoundi North completed to its final design elevation of 50 masl. Phase 2 remains the dominant source of mill feed, but Phase 3 is developing significantly. Phase 2 completes the year at 15 masl and Phase 3 is at 65 masl.

Year 6 has Phase 2 and 3 mining in Malikoundi. Malikoundi Phase 2 is at the -20 level and Phase 3 is at the 25 masl level.

Year 7 Phase 2 is completed part way through the year to its final depth of -40 masl and Phase 3 is the only mining phase being mined by year end. Phase 3 ends the year at the -10 level.

Years 8 thru 12 have Phase 3 providing the 2.75 Mtpa mill requirement. In Year 12, Phase 3 finishes at its final level of -160 masl. The stockpile created over the mine life is being rehandled in earnest in Year 12 with 2.1 Mt of ore reclaimed

Year 13 reclaims the remaining 2.2 Mt of ore in the stockpile. There is no mining in the pit this year.

The mine schedule was completed on a monthly basis for Years -2 and -1, quarterly for Year 1 and annually for Years 2 onwards. This has been summarized to an annual schedule for conciseness. The mine is scheduled to deliver 35.1 Mt of ore to the mill grading 1.71 g/t Au. Waste totalling 204.3 Mt will be stored in waste management facilities in the Boto 5, Malikoundi West, Malikoundi East, TMF and FWP areas. The overall strip ratio is 5.83:1. The mine schedule is shown in Table 16.11.

Table 16.11 Feasibility Mine Schedule

Period	Ore to Plant kt	g/t Au	Direct to Mill kt	To Stockpile kt	From Stockpile kt	Waste kt	Total Material Mined kt
Pre-production (Yr-2)	-	-	-	-	-	2,296	2,296
Pre-production (Yr-1)	-	-	-	2,245	-	11,460	13,705
Year 1	2,590	2.34	1,409	799	1,181	21,876	24,084
Year 2	2,750	2.27	2,163	1,312	587	29,952	33,426
Year 3	2,750	1.67	2,475	789	275	26,360	29,624
Year 4	2,750	1.93	2,022	841	728	24,686	27,550
Year 5	2,750	1.81	2,475	557	275	24,548	27,580
Year 6	2,750	2.14	2,375	1,626	376	23,358	27,359
Year 7	2,750	1.58	2,431	290	320	15,536	18,257
Year 8	2,750	1.39	2,475	269	275	10,567	13,311
Year 9	2,750	1.79	2,475	387	275	7,302	10,164
Year 10	2,750	1.74	2,475	260	275	4,152	6,887
Year 11	2,750	1.59	2,221	-	529	1,916	4,137
Year 12	2,750	1.08	692	-	2,058	326	1,017
Year 13	2,222	0.72	-	-	2,222	-	-
Total	35,062	1.71	25,686	9,376	9,376	204,335	239,397

Figure 16.18 to Figure 16.20 show the variation of the mill feed over the life of mine by ore type, grade, contained ounces and mine production by open pit phase.

Figure 16.18 Mill Feed by Type

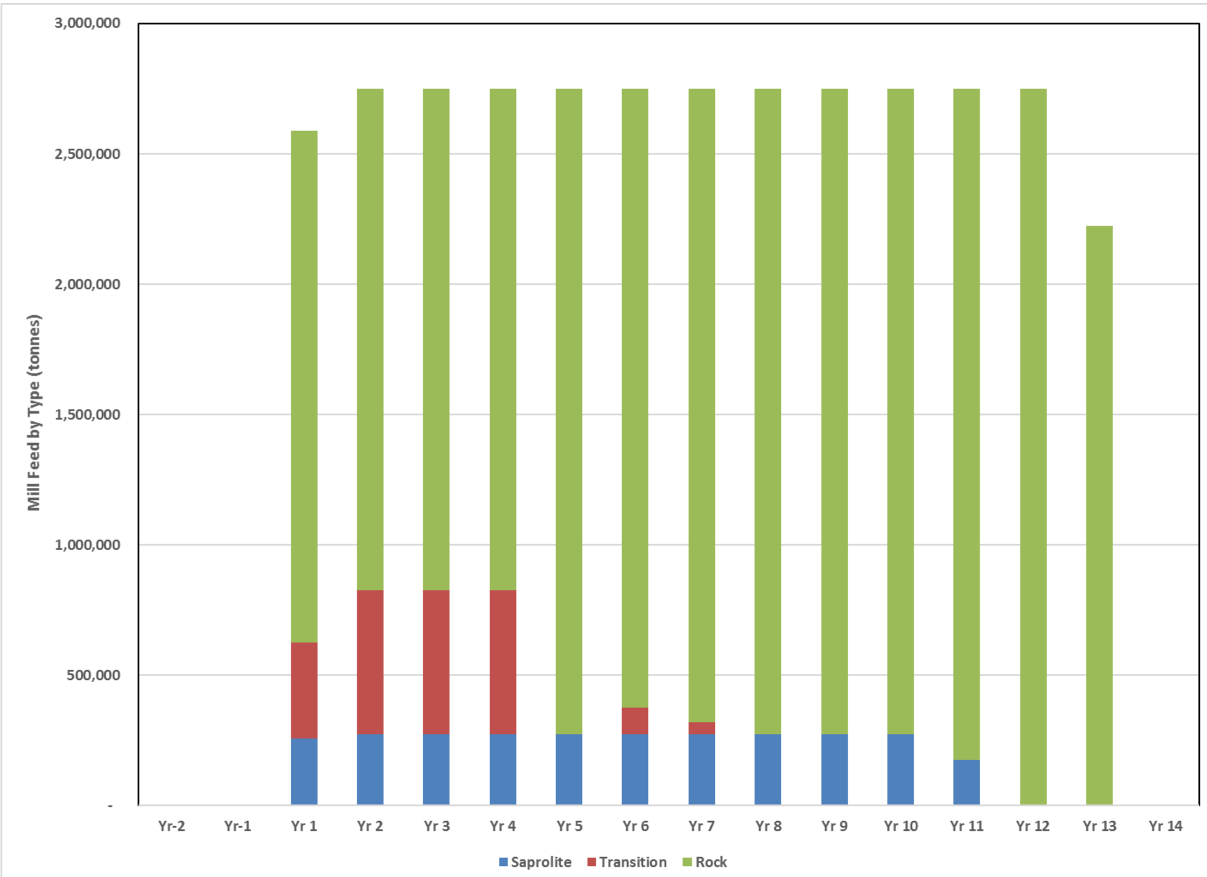
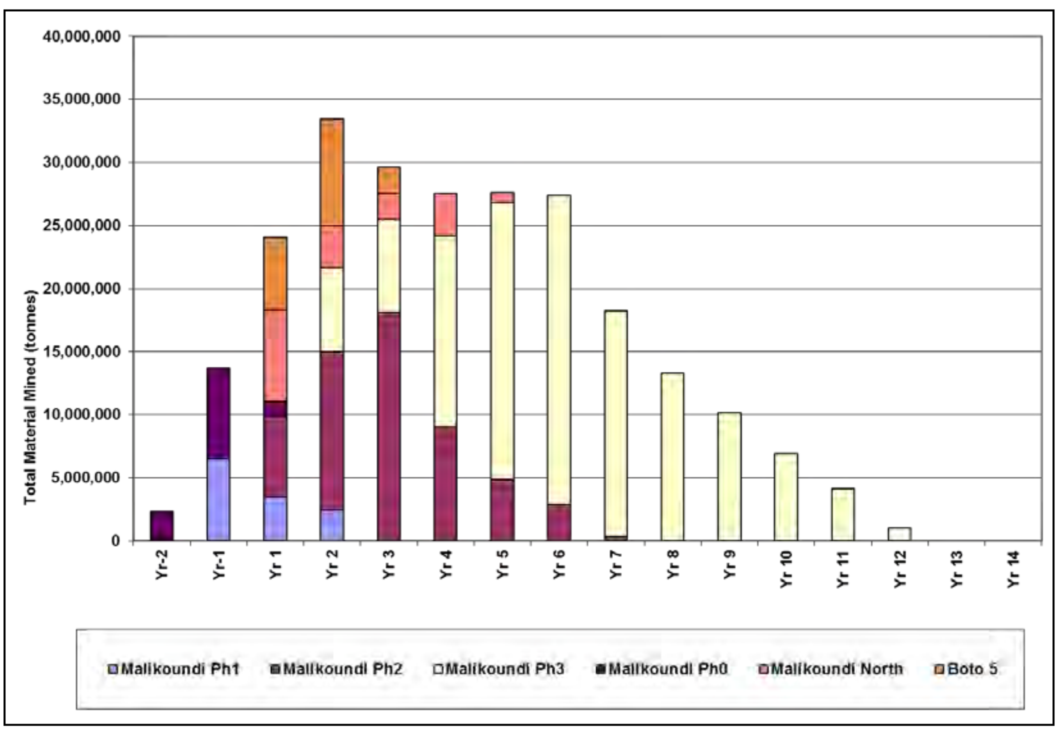


Figure 16.19 Ore Grade and Ounces to the Process Plant

Figure 16.20 Mined Tonnage by Year and Phase



16.8 Grade Control

Grade control is an item that was considered from the beginning of the mine planning sequence. Blasthole sampling was considered but due to the selectivity required was excluded. Other operations in the area utilize reverse circulation drilling as a method of ore definition. This involves using a reverse circulation drill in advance of mining on tight inclined drillhole spacing, to accurately define the ore/waste contacts. This is typically done when the mineralized zone is more dispersed or inclined towards the horizontal which is similar to Malikoundi. This information is then built into the short-range models and used to guide the loading equipment. This practice is widespread in Australia with great success as well as in Canada and Africa.

The method involves using a dedicated grade control drill rig and crew in the pit to drill a series of shallow vertical holes drilled in a pattern similar to the blast hole pattern. For this study, that is assumed to be a contract drill rig and team with experience in this type of drilling. The grade control drill will be drilling the saprolite where blast holes are not required. The pattern for drilling will be a 10 m spacing and a 5 m burden with samples taken every 1 m in presumed mineralized zones as outlined by both previous ore control drilling and the exploration drilling. The volume of the sample to be assayed and sample intervals will need be determined in a grade deportation study in subsequent studies. An additional 25% will be drilled along the contacts to ensure that unknown structures are not missed in the saprolite. These “waste” samples will be drilled with a 20 m spacing and 10 m burden and sampled over 6m.

The amount of reverse circulation drilling peaks in Year 2 at 33,200 m then drops off after that averaging 22,300 m/a from Year 2 until Year 12. No ore control drilling is required after Year 12.

The reverse circulation drills will operate in advance of mining for 24 h/d minimizing disturbance and be in advance of mine operations with the information. A three-man crew per drill is required; one driller and two drill helpers. In addition, geologists will provide guidance throughout the day and be on call if unknown issues arise.

The drill penetration rate is estimated at 10 m/h with setups, sampling, etc. Overall, the cost for the RC drill program with contract labour is estimated at $63/m. Sampling and assay cost is estimated from laboratory quotes at $14.25/sample.

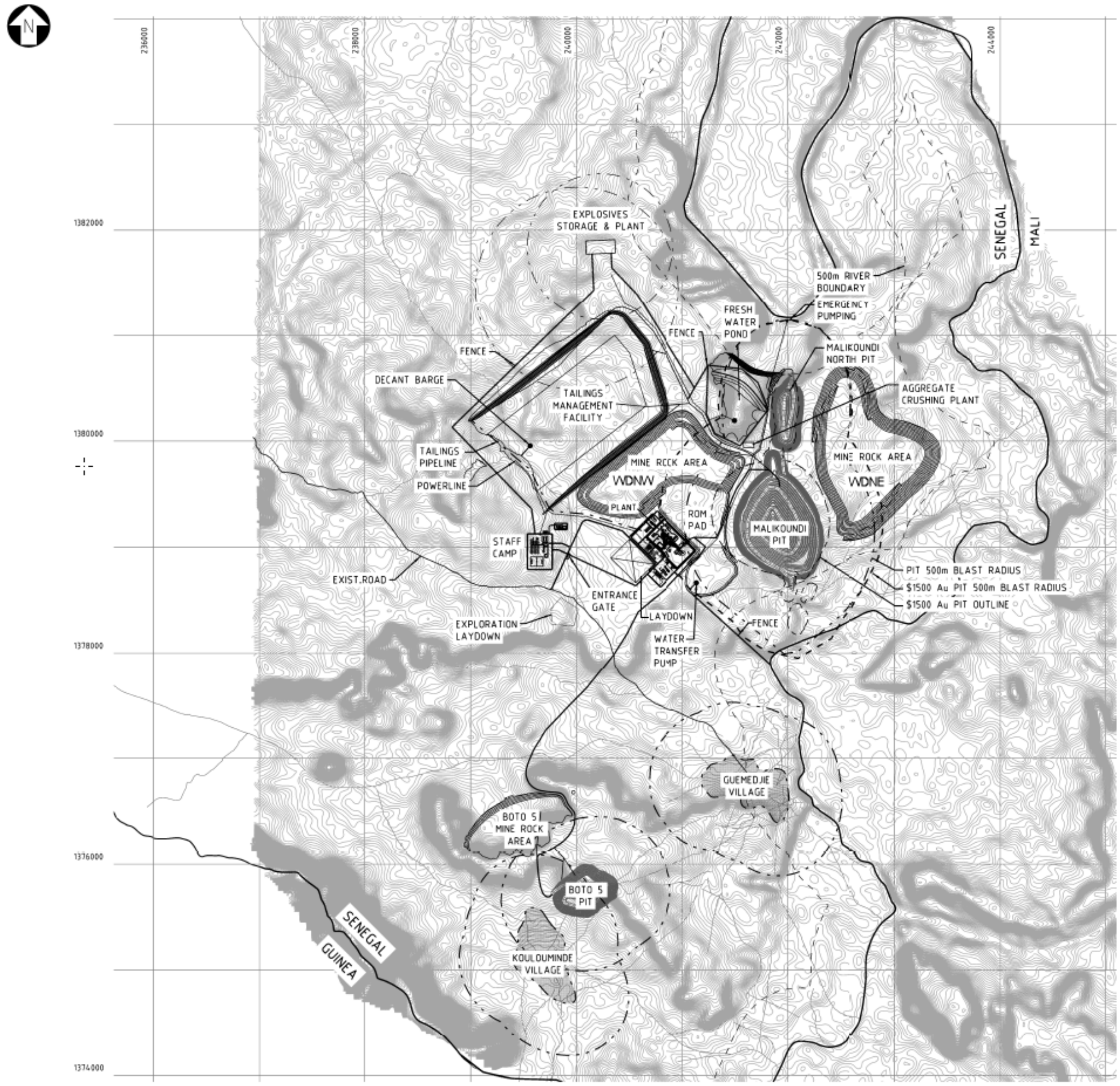
The data from the grade control drilling is then interpreted by the geologist and the ore is then remodelled. The production drilling and blasting will then be designed to mine the ore material separately from the waste.

16.9 Waste Management Facility Design

Various rock types are present in the material mined within the final pits. They include the weathering profile of laterite, saprolite, transition and hard rock. The percentages vary by pit and phase. Ferricrete is present in some areas and will be utilized for construction material and roads. All material types will be co-mingled in the waste management facilities.

Certain portions of the material will be directed to the TMF for the embankment construction. In addition, there will be four waste storage areas. These are shown in Figure 16.21.

Figure 16.21 Waste Management Facilities



These facilities include:

1) TMF
   a. Material types include ferricrete, saprolite and fresh rock.
   b. Saprolite will be placed in the tailings basin for bedding of the TMF liner.
   c. Material will be provided by the mine, placed and final compaction and liner placement will be completed by contractors.
   d. Constructed of Phase 0, 1, and 2 material from Year -1 to Year 4.
2) FWP
   a. Material includes laterite, saprolite and mixed fill.
   b. Saprolite will be placed in the basin for a liner in the facility.
   c. Constructed from Phase 0 material in Year -2 and Year -1.

3) WDNW
   a. Contains mixed materials including laterite, saprolite, transition and fresh rock.
   b. Forms the SE side of the TMF.
   c. The platform is for ore stockpiles.
   d. The lower level is at the level of the crusher feeding the plant.
   e. Constructed from material out of Phases 0, 1, 2, 3, and Malikoundi North.
   f. Active from Year -2 to Year 12.
4) WDNE
   a. Contains material from Phase 3 – laterite, saprolite, transition and fresh rock.
   b. Is designed to stay 500 m from the river.
   c. Active from Year 2 to Year 11.
5) Boto 5
   a. Is composed of laterite, saprolite and transition material entirely from Boto 5

The design capacities and actual capacities used are shown in Table 16.12.

Table 16.12 Waste Management Facility (WMF) Parameters

Parameter	Units	WDNW	WDNE	FWP	WMF Boto 5	Total WMF
Design Capacity	Mm^3	40.8	53.8	1.3	9.0	104.9
Capacity Utilized	Mm^3	36.5	53.8	1.3	9.0	100.6
Maximum Height	Masl	220	210	157	195	
	m	55	70	32	41	

The design of the WMF's considers variable swell factors dependent on material types. For laterite and saprolite a swell factor of 15% was applied. Transition material is expected to be in the 20% range and fresh rock swell at 30%.

The facilities are built in different manners dependent on the dominant material type. For the Malikoundi facilities lift height will be 10 m with a bench width of 10 m. Assuming a 37.6° face slope, the overall slope will be 23.5°.

Boto 5 is constructed of lower strength material (laterite, saprolite and transition). This requires more time for pore pressure dissipation and staging of the various lifts. Initially the Boto 5 facility will be built in 5 m lifts with a bench slope of 30.5 degrees and berm width of 10 m. Later stages of the facility development will be placed in 10 m lifts with 25 m benches. This results in an overall slope of 15° for long term stability.

Waste management facilities will be actively reclaimed as they are developed. Dozers will re-slope them as they are advanced to allow revegetation to occur as soon as possible.

Drainage from each of the WMF's will be diverted to sedimentation ponds. This is to ensure the sediment washed down from the facilities is captured before it escapes from the mine property. The sediment ponds will be cleaned annually or more frequently if required to ensure storage capacity in the ponds is not compromised.

16.10 Mine Plan Sequence

End of year positions for the open pits are shown in the following figures. The initial set is around the Malikoundi area with the later part showing the three years of mining at Boto 5.

Figure 16.22 End of Preproduction Period - Year -2

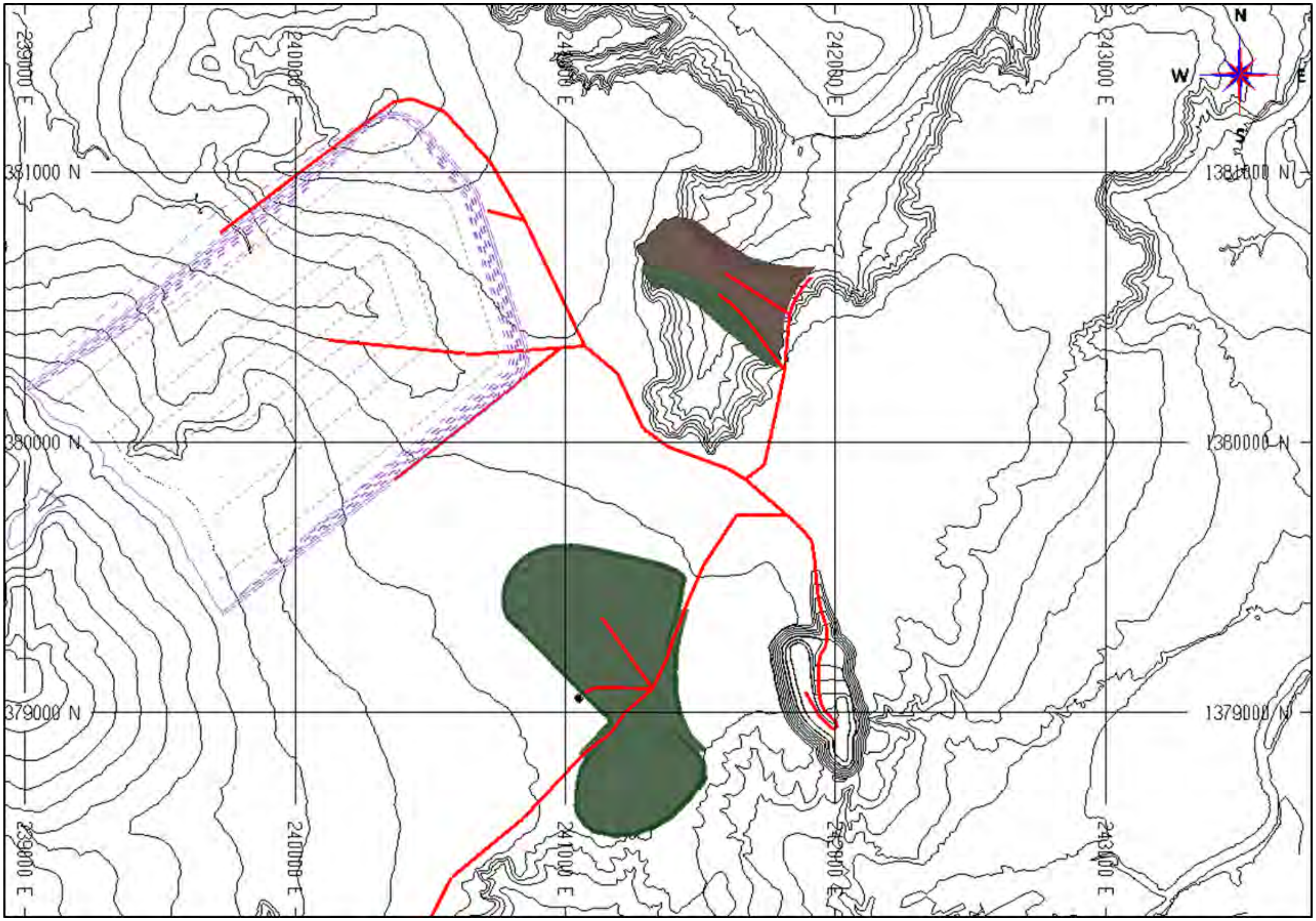


Figure 16.23 End of Preproduction Period - Year -1

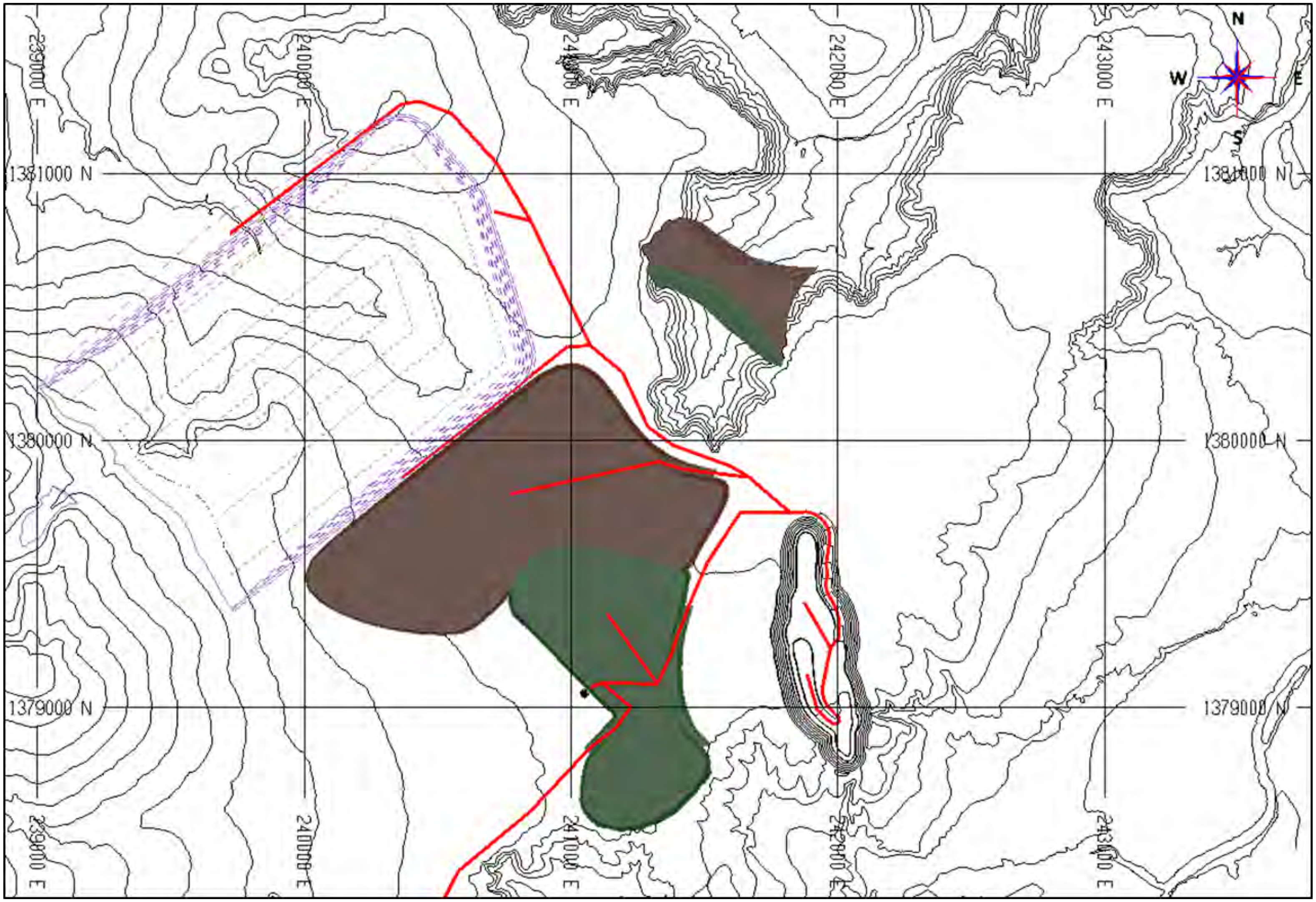


Figure 16.24 End of Year 1

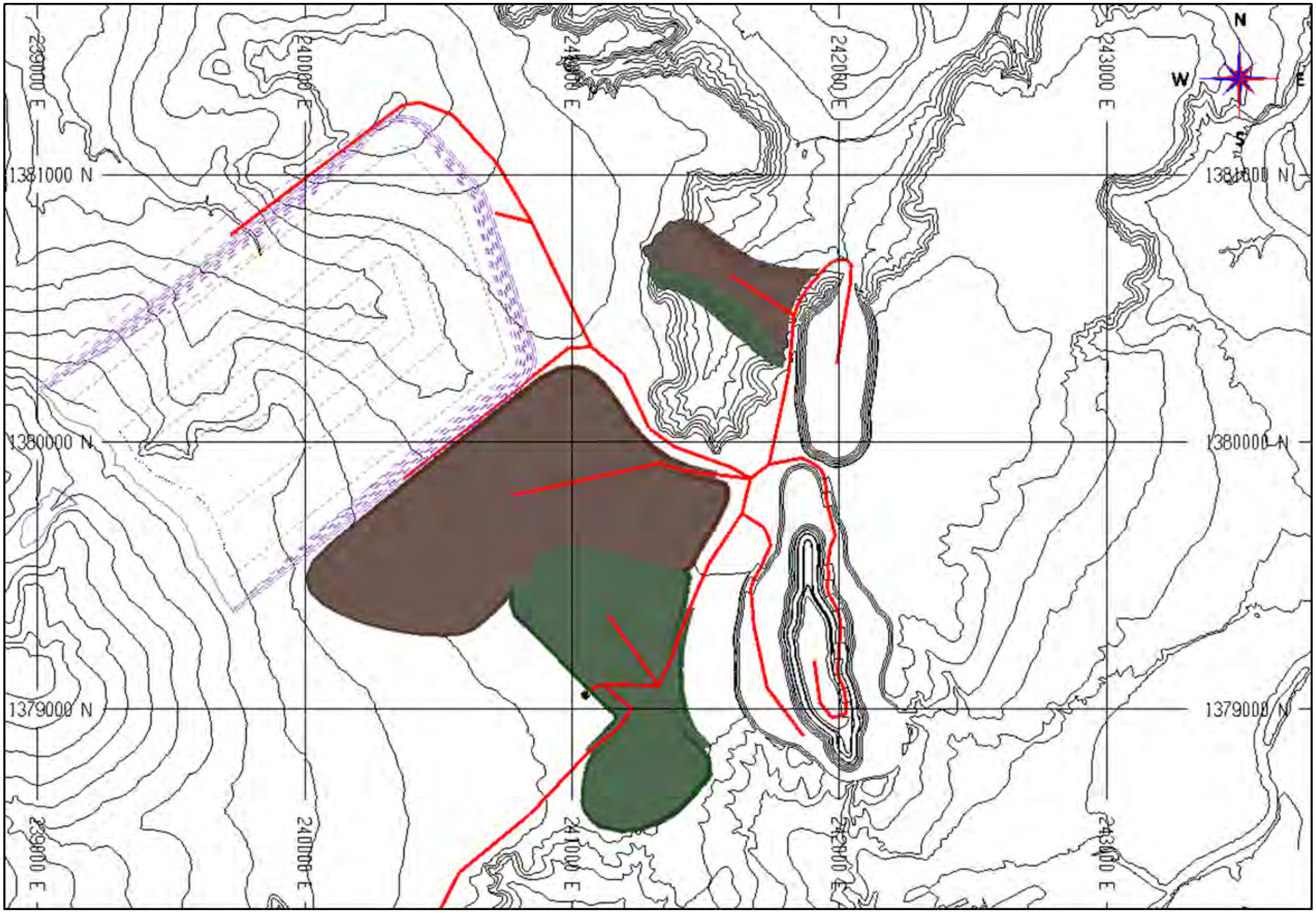


Figure 16.25 End of Year 2

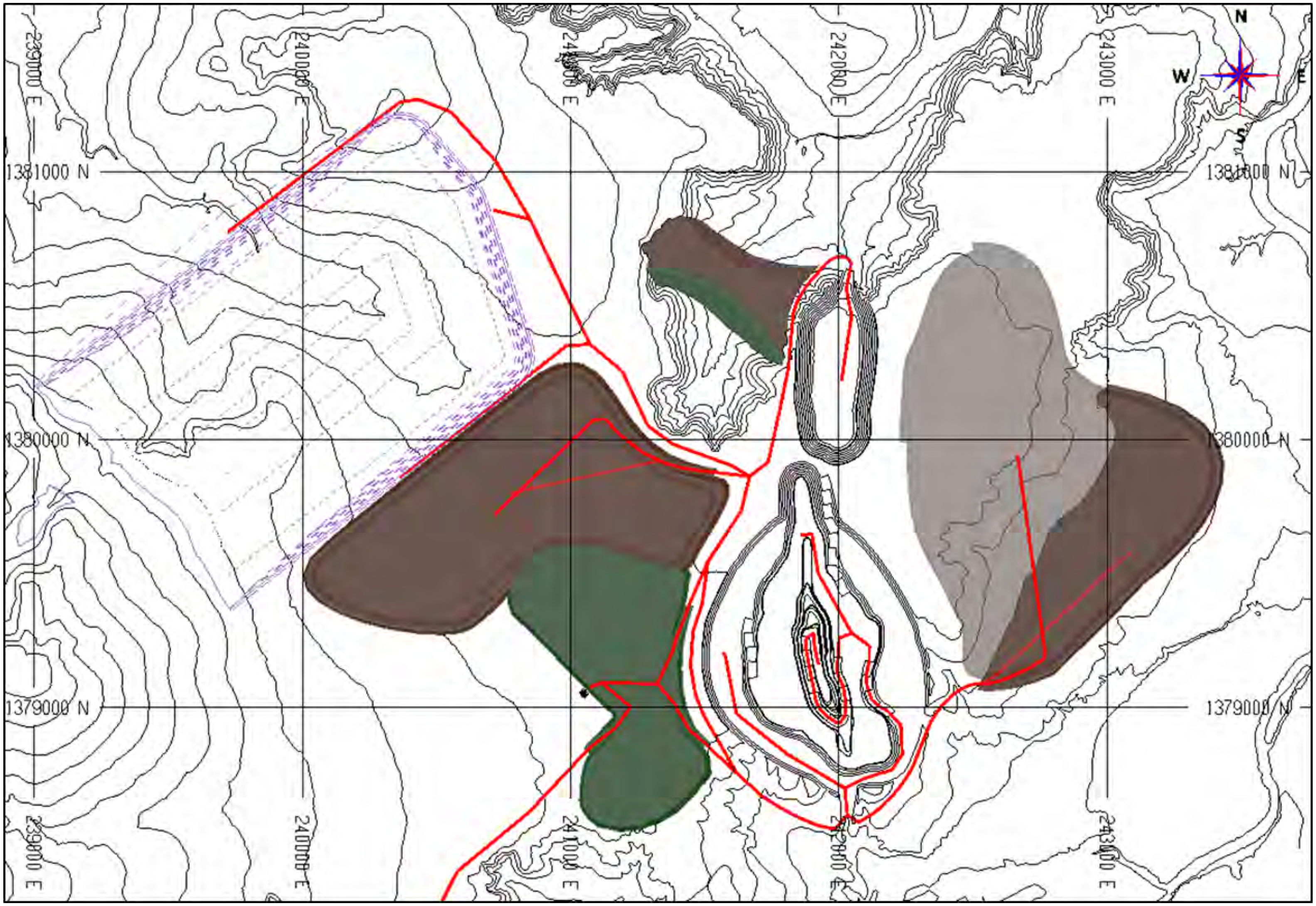


Figure 16.26 End of Year 3

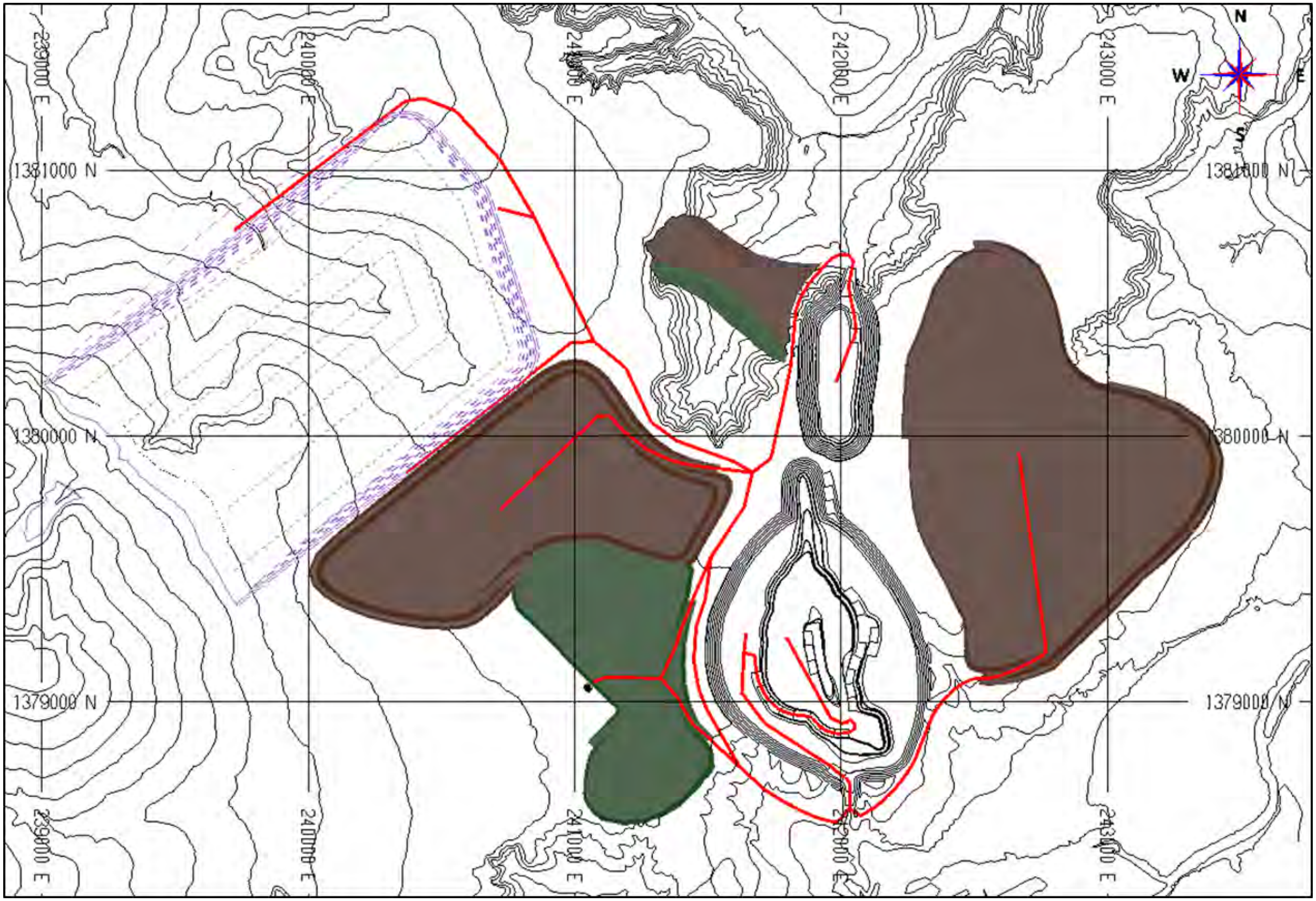


Figure 16.27 End of Year 4

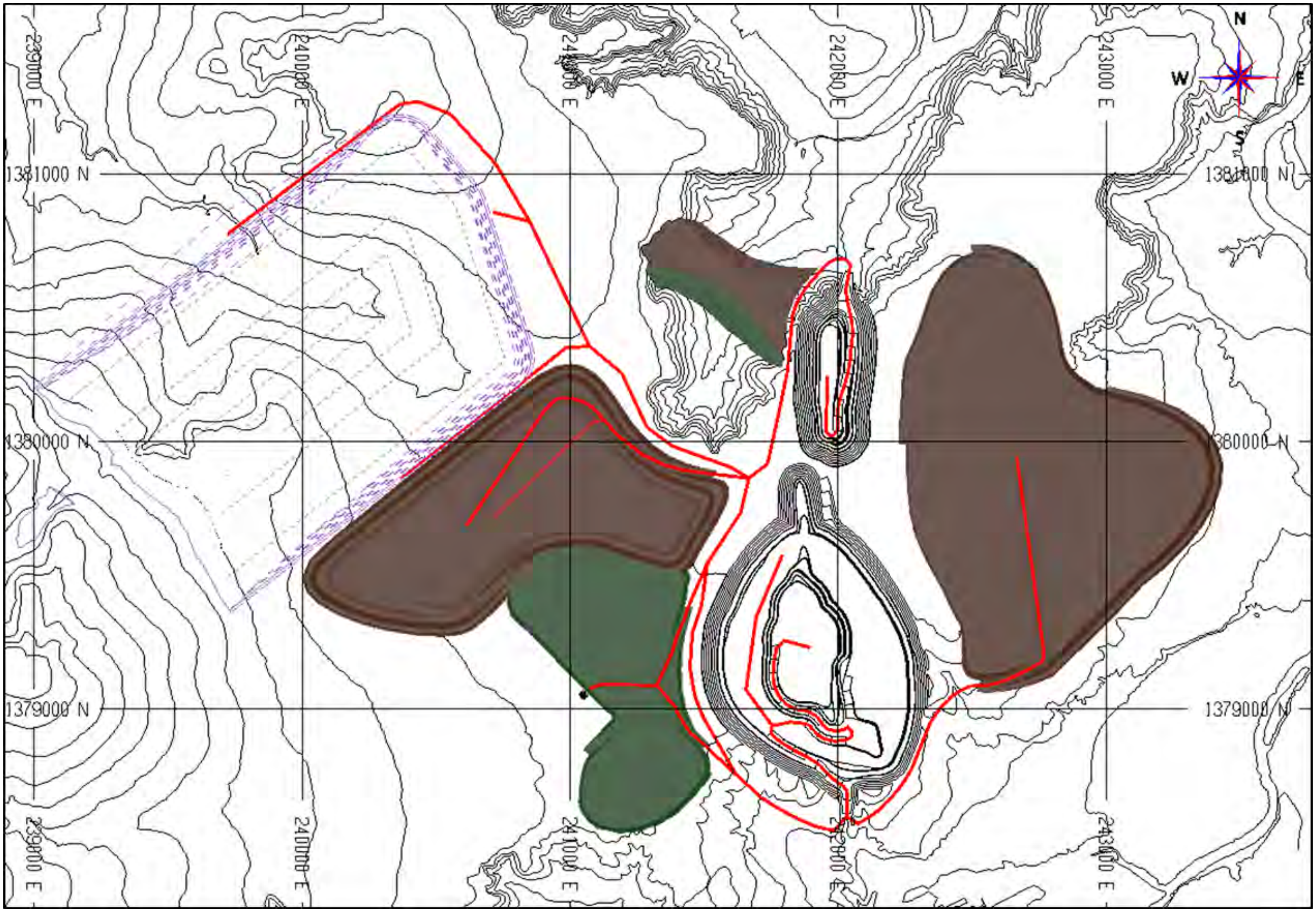


Figure 16.28 End of Year 5

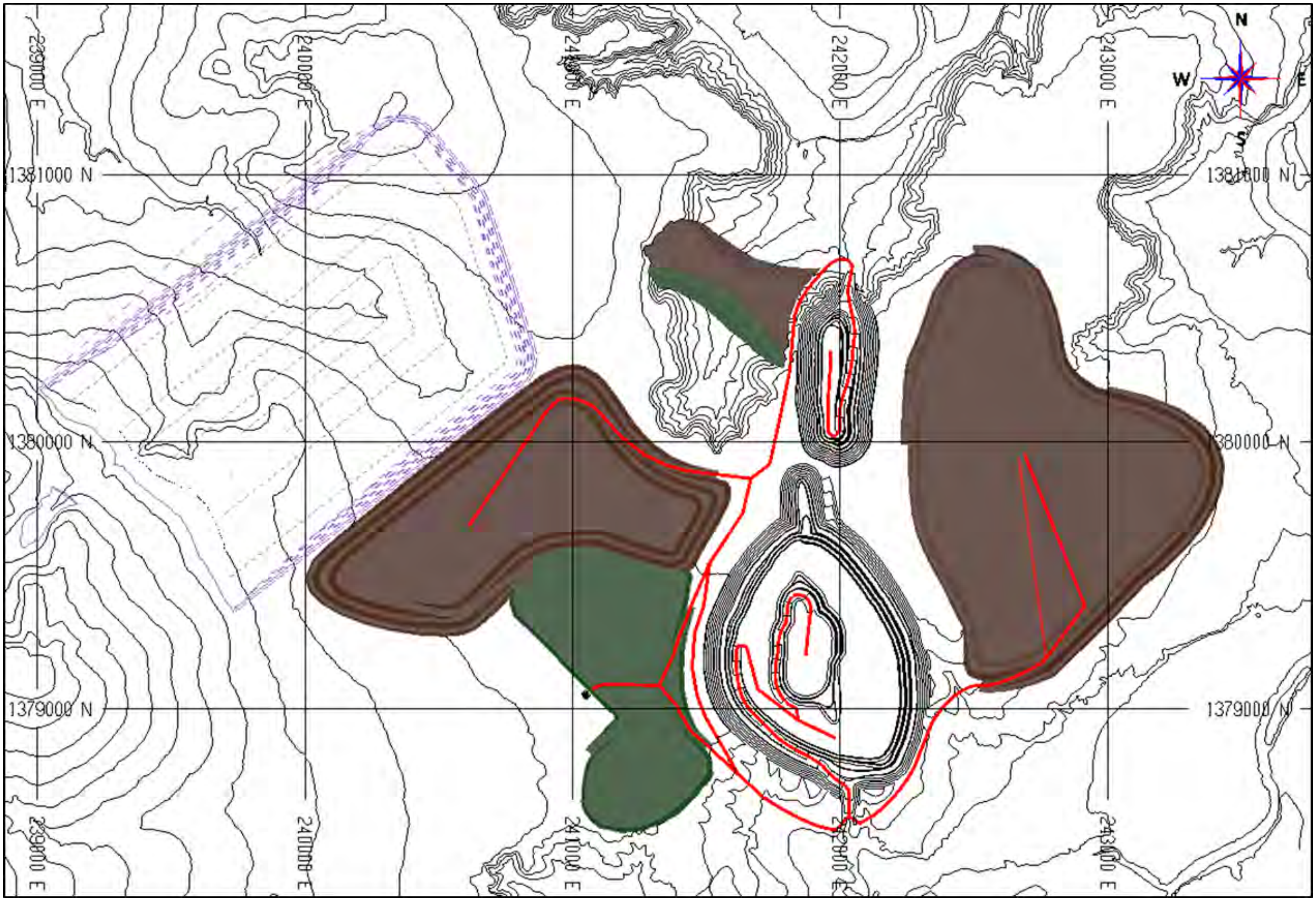


Figure 16.29 End of Year 6

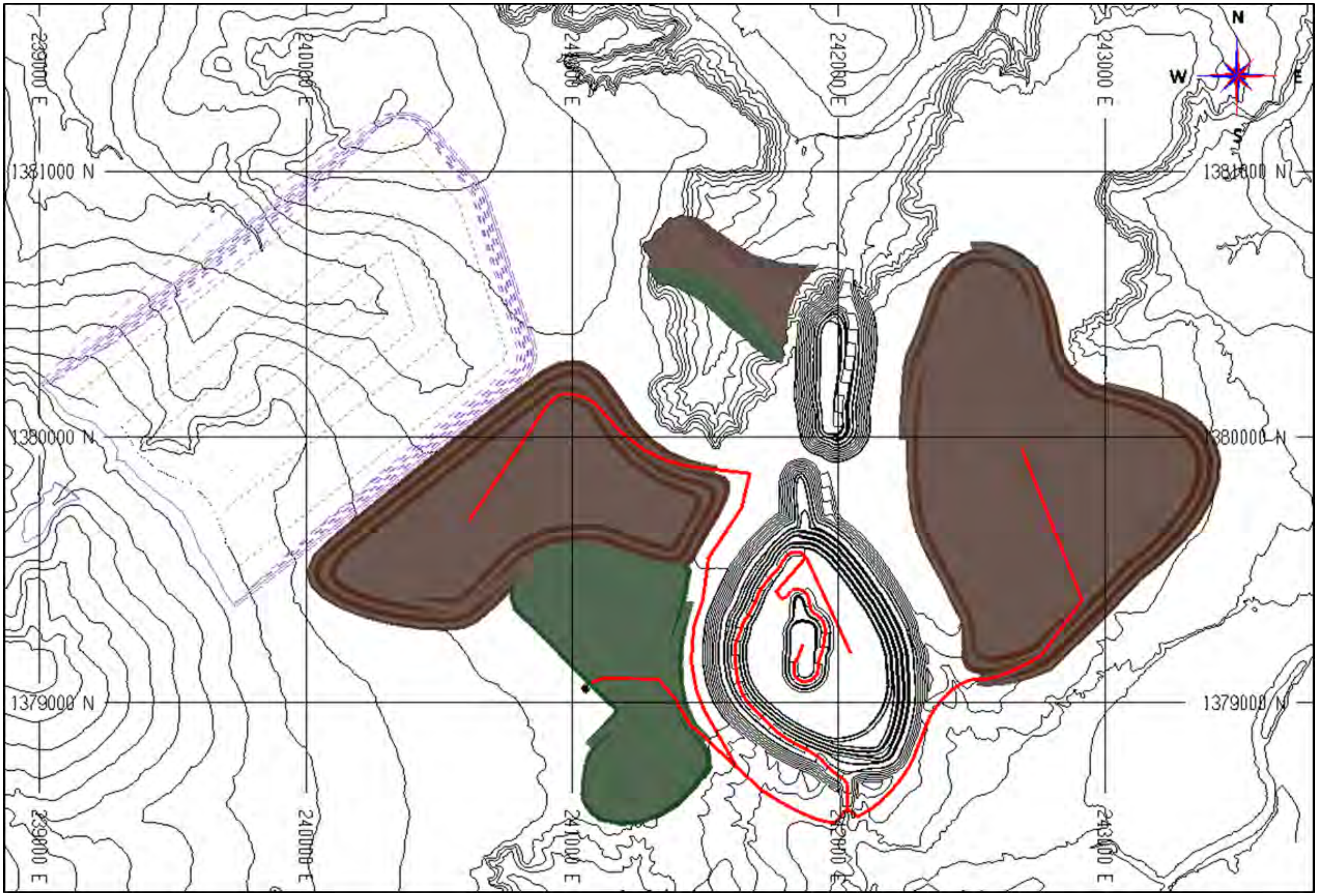


Figure 16.30 End of Year 7

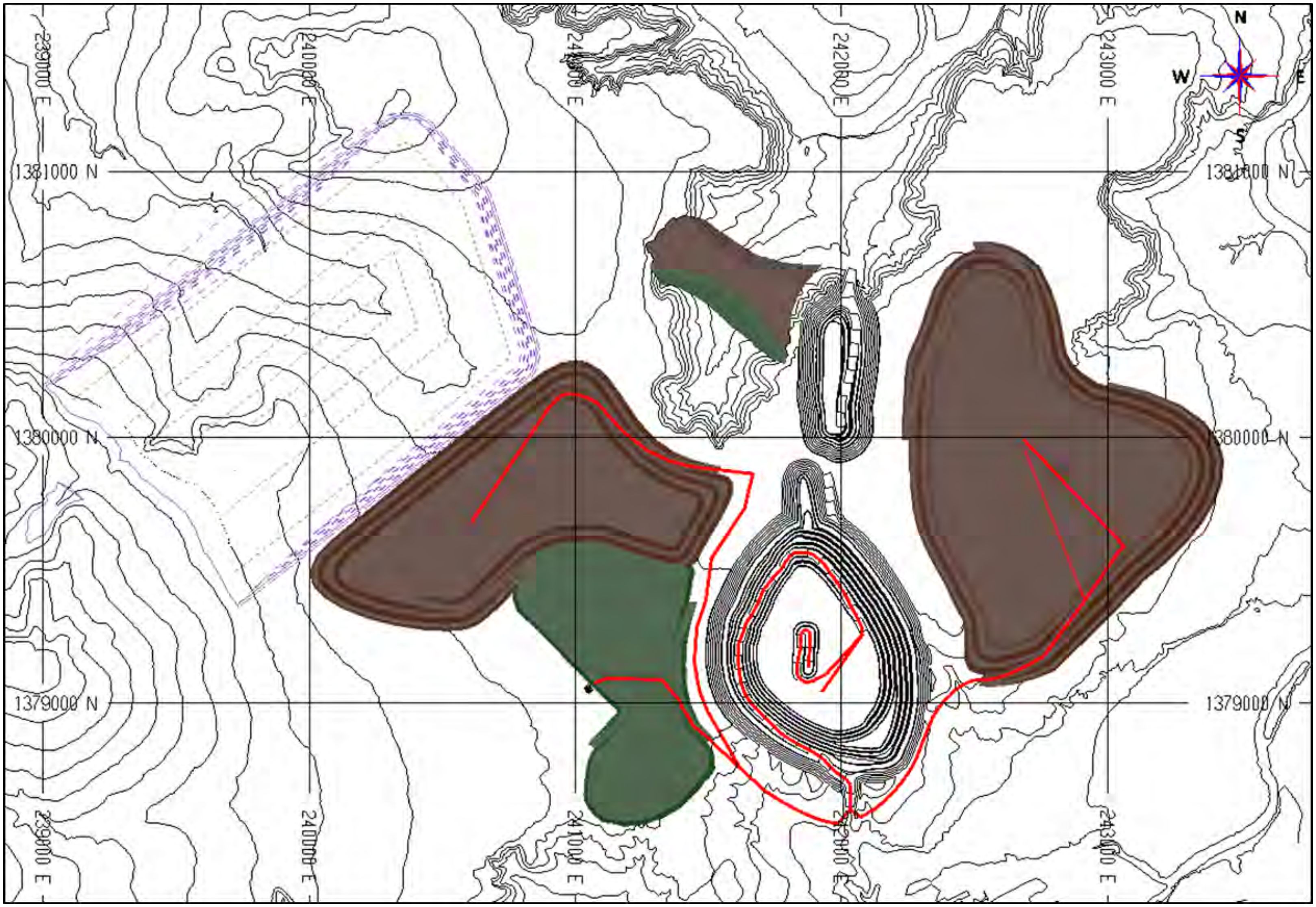


Figure 16.31 End of Year 8

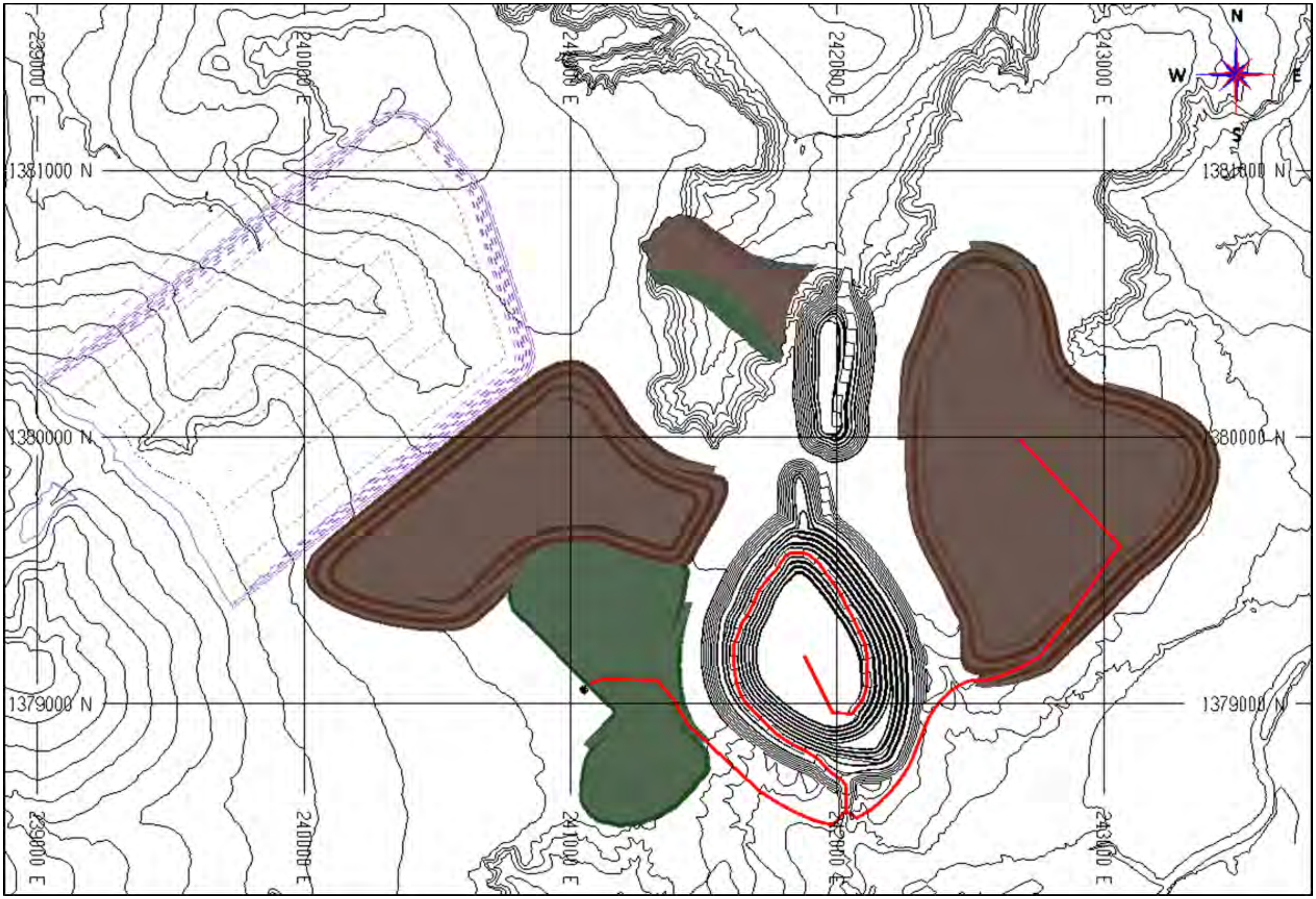


Figure 16.32 End of Year 9

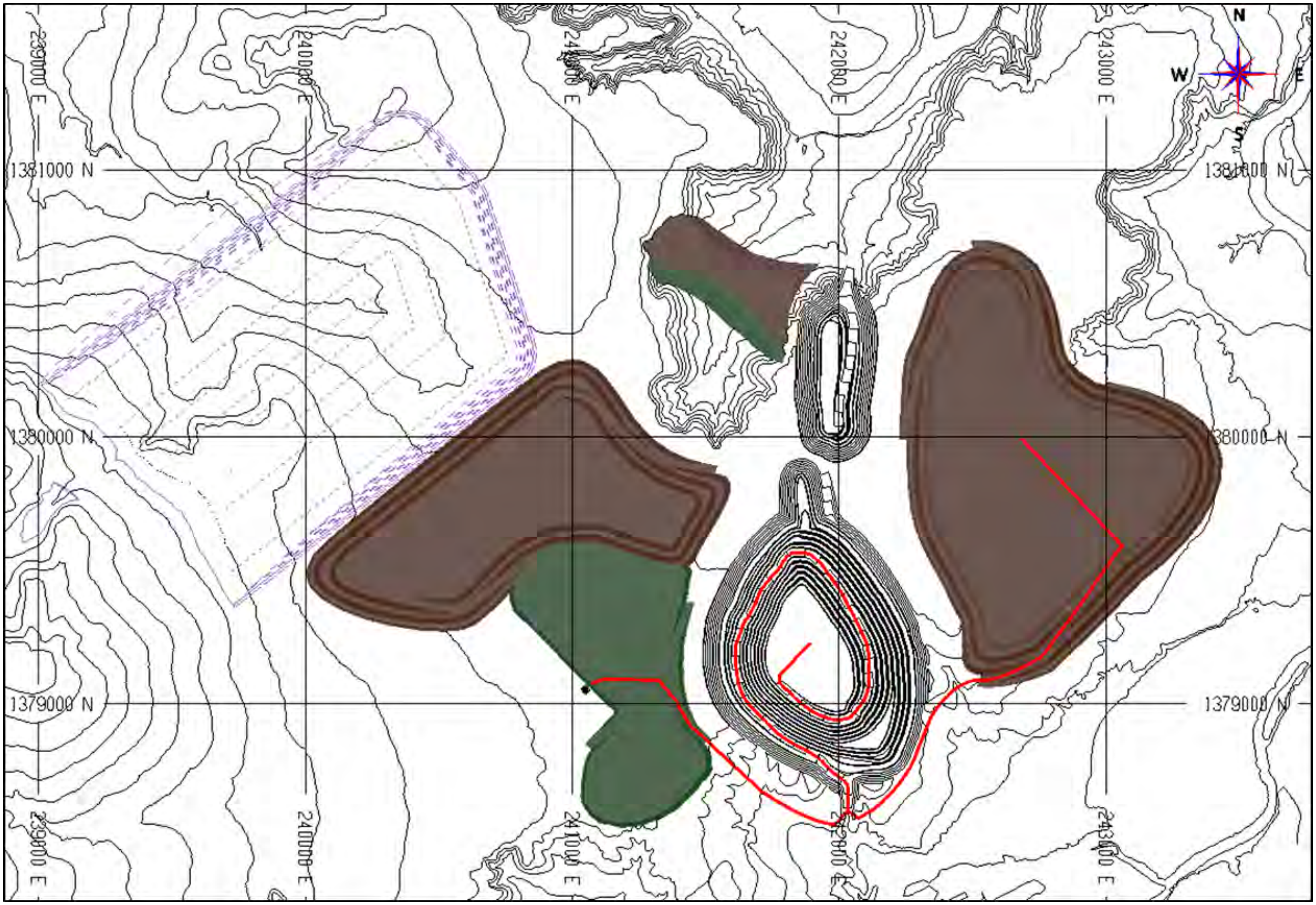


Figure 16.33 End of Year 10

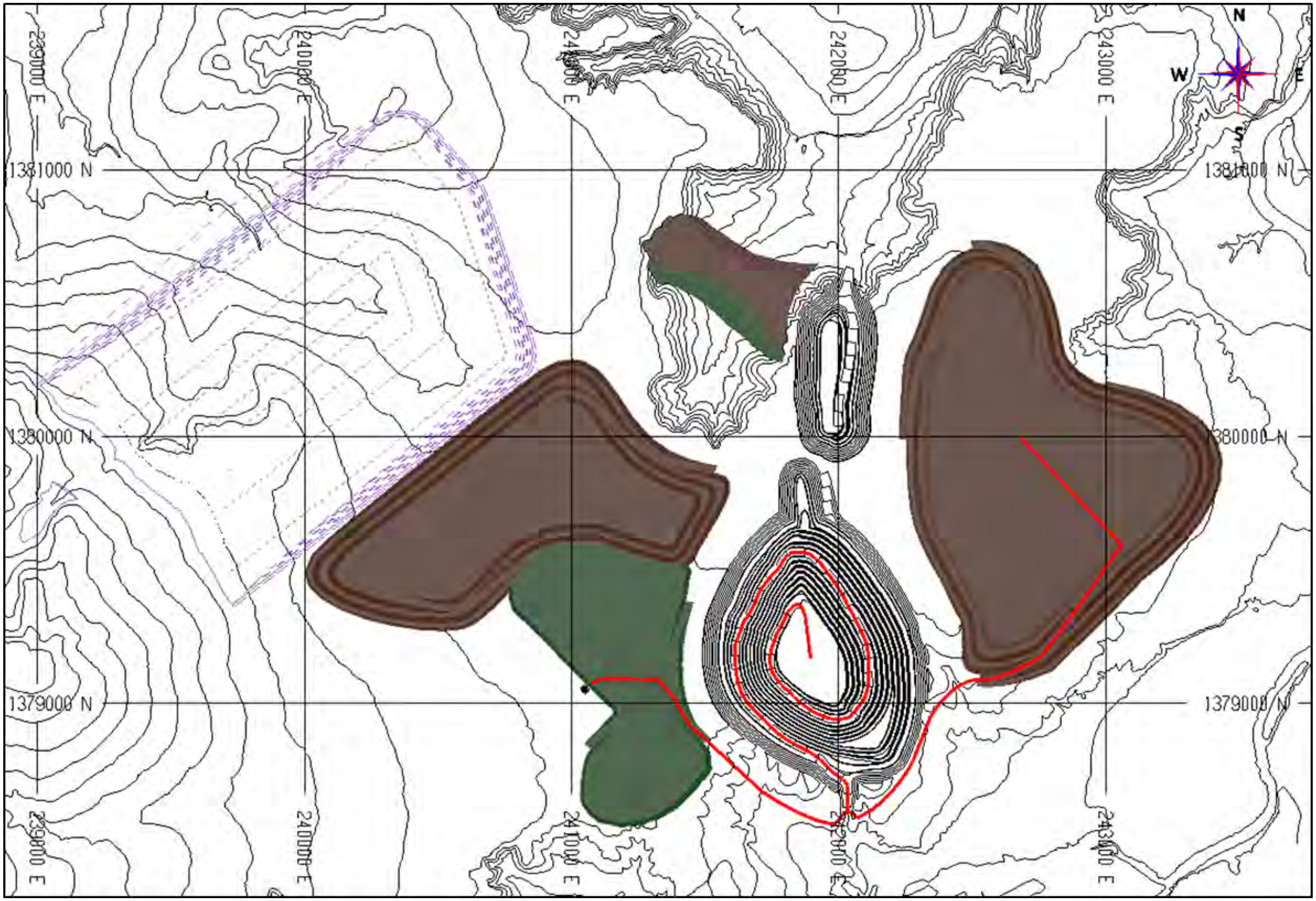


Figure 16.34 **End of Year 11**

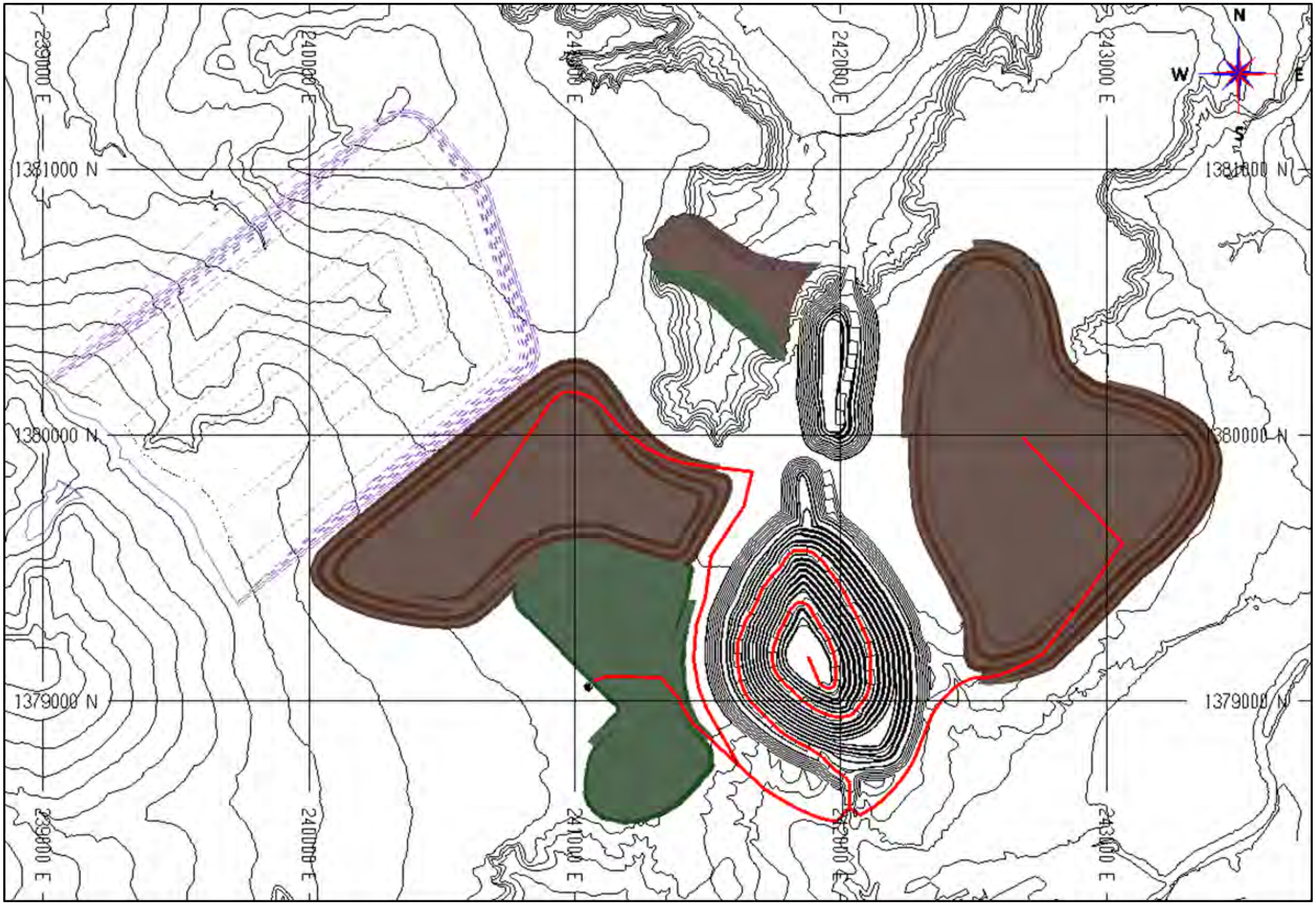


Figure 16.35 End of Year 12 (End of Mining)

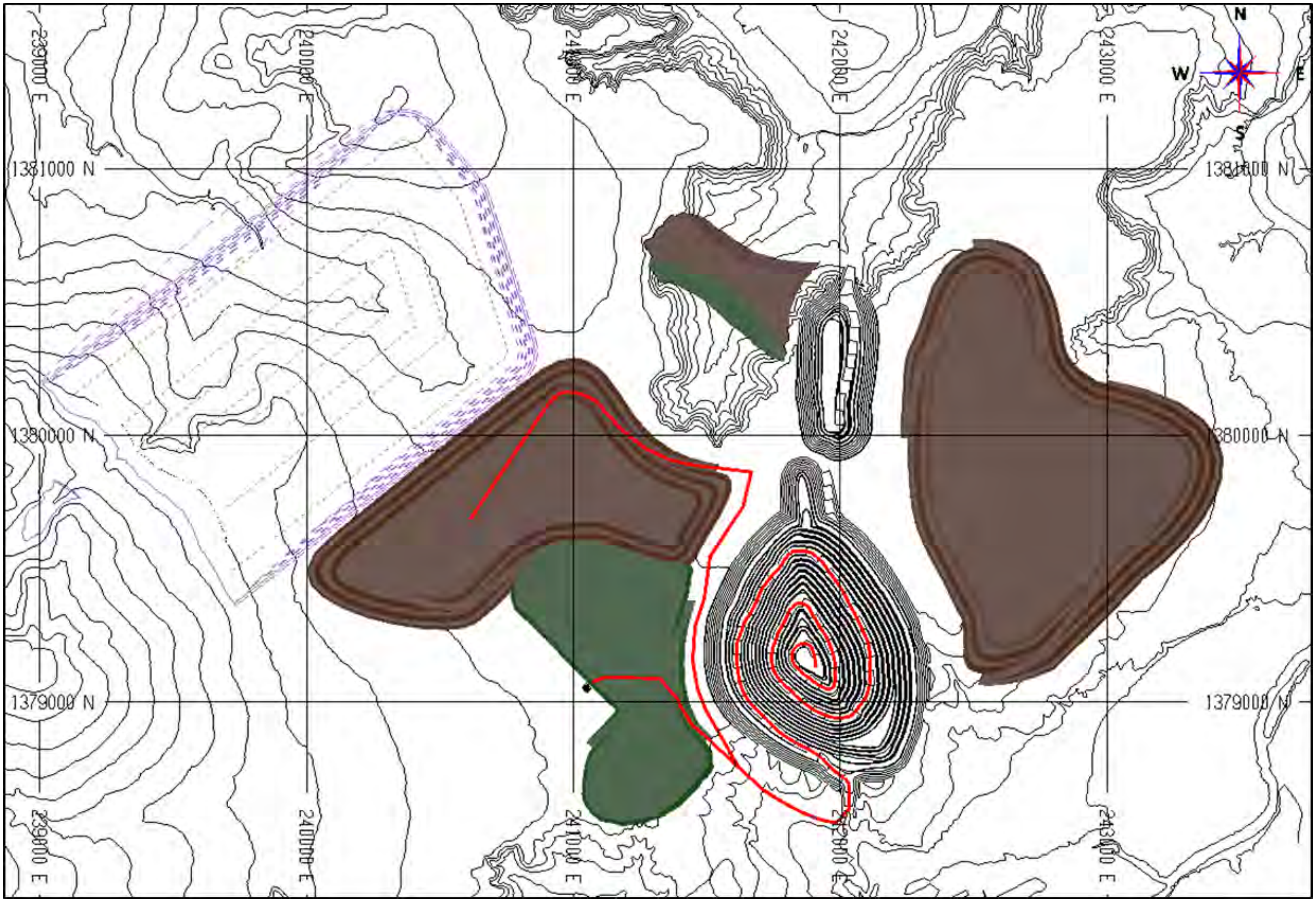


Figure 16.36 End of Year 1 – Boto 5

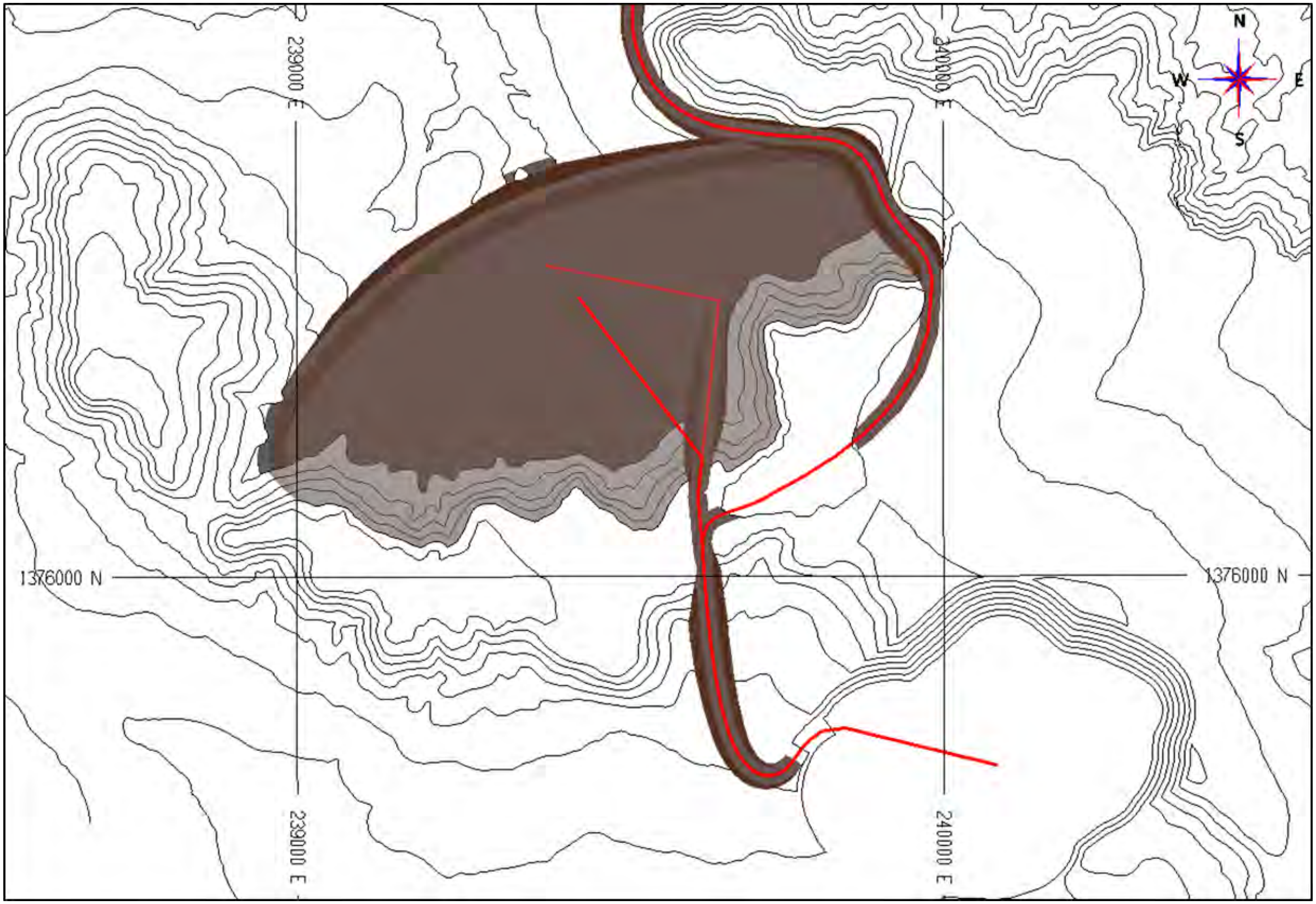


Figure 16.37 End of Year 2 – Boto 5

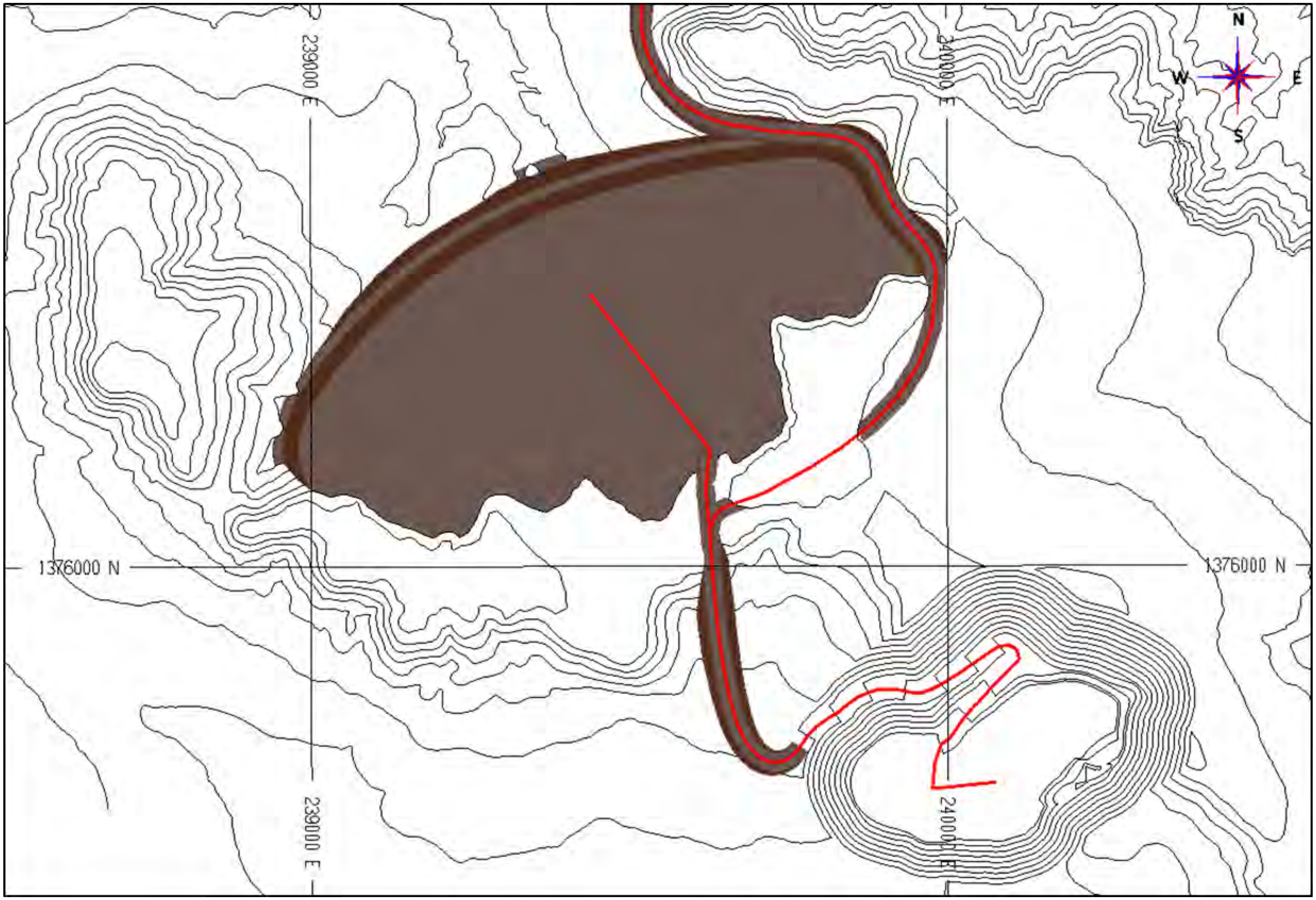
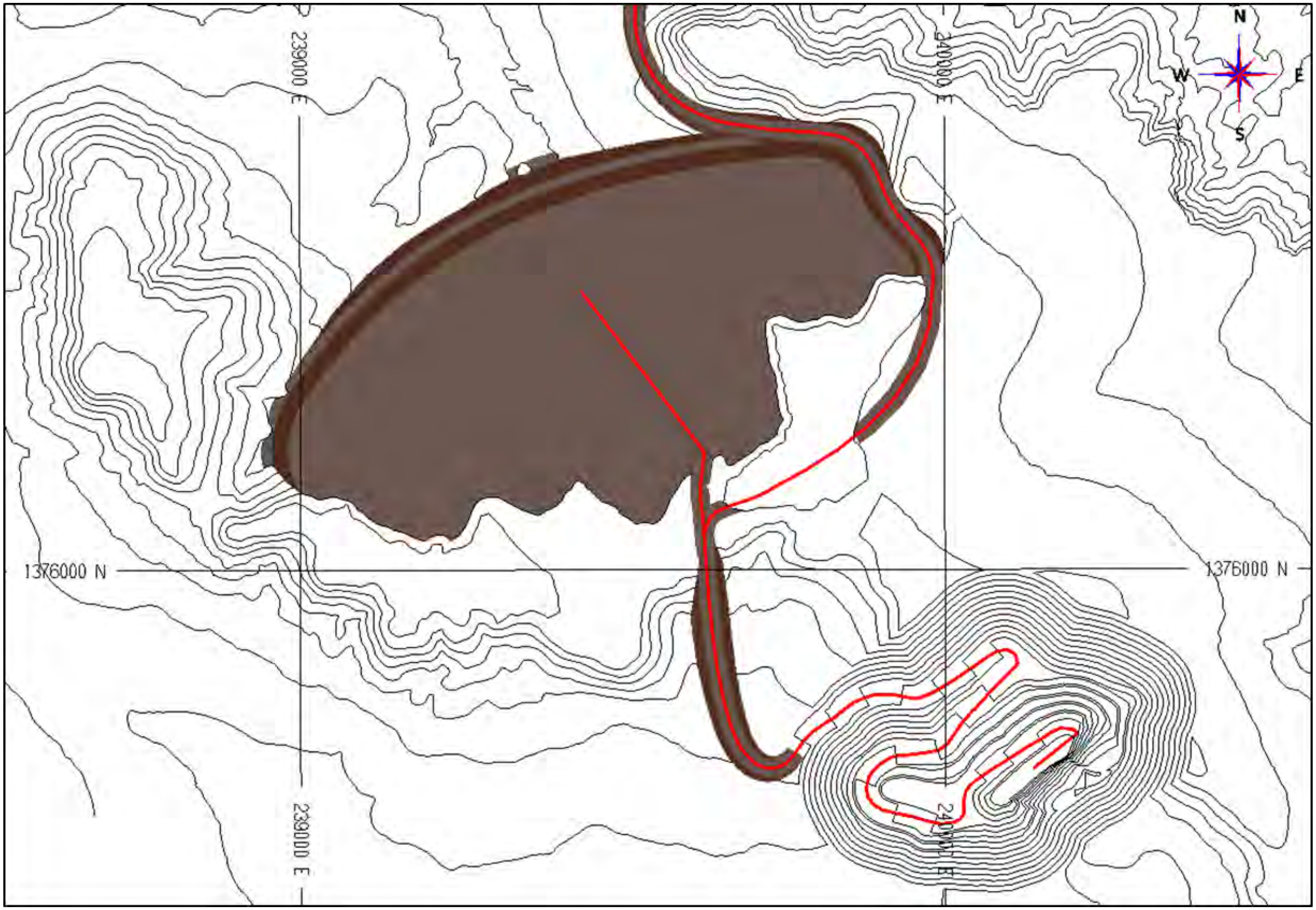


Figure 16.38 End of Year 3 – Boto 5



16.10.1 Comments on Section 16

- The open pit designs are comprised of three areas: Boto 5, Malikoundi, and Malikoundi North.
- Malikoundi is divided into four phases with Phase 1 subdivided into a Phase 0 to drive to fresh rock sooner for construction purposes.
- Malikoundi North is mined starting in Year 1.
- Boto 5 is mined from Year 1 until Year 3 to minimize the amount of saprolite in the feed initially.
- Mill feed totals 35.1 Mt grading 1.71 g/t Au diluted.
- Waste tonnage over the mine life will total 204.3 Mt for a strip ratio of 5.83:1 LOM.
- Contact dilution for Malikoundi and Malikoundi North added 4.5% more tonnes and lowered the gold grade by 4.1% based on 0.5 m of contact dilution.
- Boto 5 contact dilution added 7.8% more tonnes and lowered the gold grade by 6.7% with the 0.5 m of contact dilution.
- The open pit mine life is 12.8 years after 2 years of preproduction stripping.
- The final ore stockpile reclaim is in Year 13.
- Mine production will be preceded by two years of pre-production work that will be used to establish roads, create an ore stockpile, build initial stages of the tailings embankments and prepare the pits for full production.
- Waste Management Facilities (WMF) are located to the north-west, north and north-east of Malikoundi and one to the north-west of Boto 5.
- Owner operated mining was considered Malikoundi and Malikoundi North.
- Boto 5 is contract mined with 40 t trucks to minimize disturbance to the local populace. No blasting is planned or required for Boto 5 as only soft material is mined.

17.0 RECOVERY METHODS

17.1 Process Design

The process plant design for the Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet is based on well proven unit operations 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 2.75 Mtpa ore based on the 85th percentile of the bedrock or hard rock hardness.
- Crushing plant availability of 75%.
- Process plant availability of 92% supported by the design of the crushing plant, surge capacity where required and standby equipment in critical areas.
- Sufficient automated plant control to minimize the need for continuous operator interface and allow manual override and control if and when required.

Study design documents have been prepared incorporating engineering design criteria and key metallurgical design criteria derived from the results of the metallurgical testwork.

17.1.1 Selected Process Flowsheet

The treatment plant design incorporates the following unit process operations:

- Single stage primary crushing with a jaw crusher to produce a crushed product size of P_{80} of 138 mm.
- Mill feed surge/overflow bin that overflows to an approximately 5,000 t stockpile to provide 14.5 hours of surge capacity. During extended periods of up to two days for primary crusher equipment maintenance, additional crushed ore inventory can be generated in the weeks leading up to the planned shutdown by dozing crushed ore from this stockpile to the area adjacent to the stockpile. This ore can then be reclaimed during the shutdown by front-end loader to feed the grinding circuit.
- The grinding circuit is a SSAG type, which consists of a closed circuit single stage SAG mill, producing a P_{80} of 75 µm.

- Hydrocyclones are operated to achieve a hydrocyclone overflow slurry density of 28.1%w/w solids to promote better particle size separation efficiency. The overflow stream passes through a trash screen to remove foreign materials prior to downstream processing. Subsequently, a pre-leach thickener is included to increase slurry density to the leach circuit, minimize leach tank volume requirements and reduce overall reagent consumption.

- Leach circuit with five tanks to achieve the required 34.4 hours of residence time at nominal plant throughput. A pre-oxidation step is included ahead of leaching to minimize cyanide consumption and improve downstream leach kinetics.

- CIP carousel circuit consisting of six stages for recovery of gold dissolved in the leaching circuit.

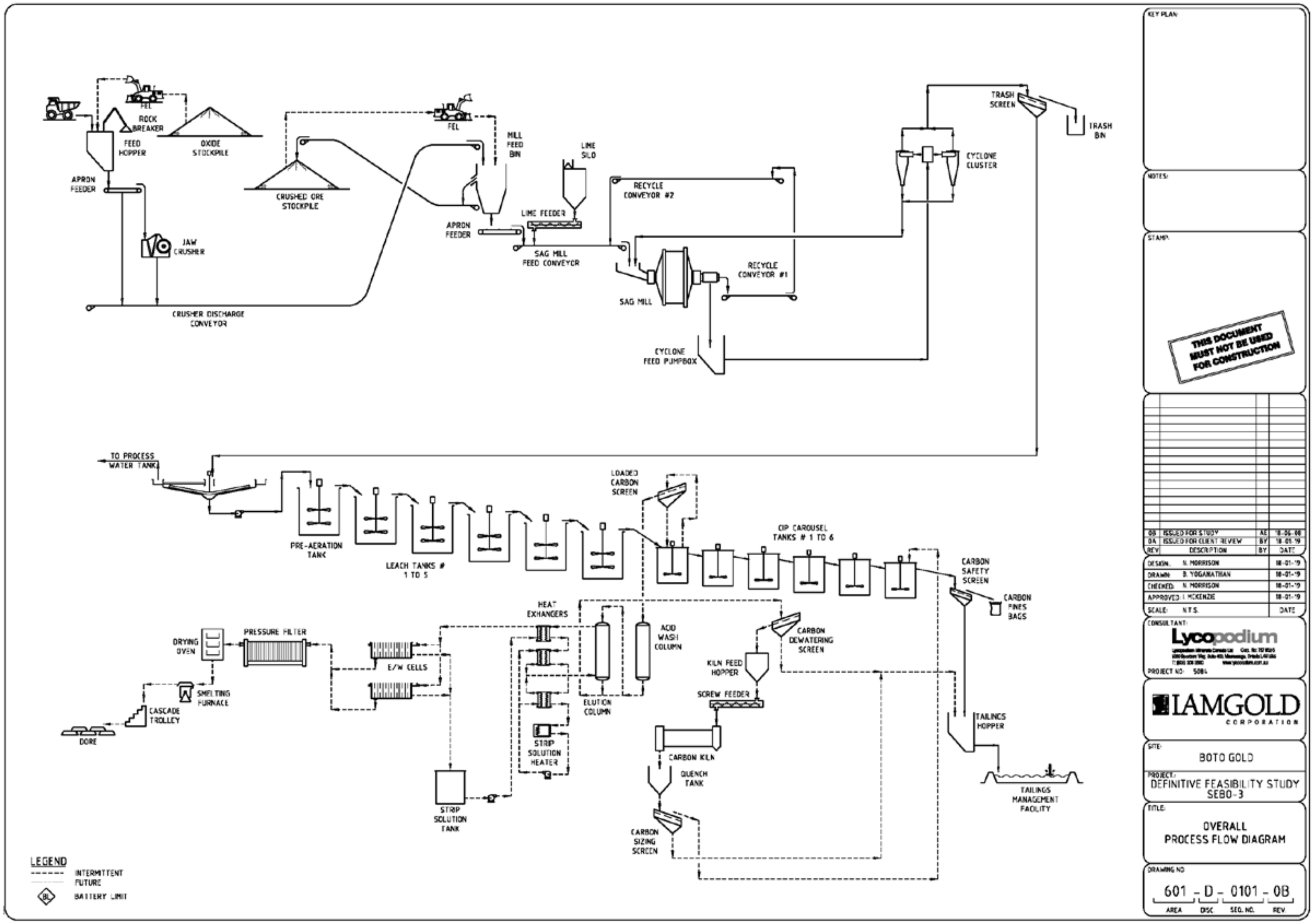
- Pressure Zadra elution circuit with gold recovery to doré. The circuit includes an acid wash column to remove inorganic foulants from the carbon with hydrochloric acid, followed by an elution column.

- Carbon regeneration kiln to remove organic foulants from the carbon and reactivate the adsorption sites on the activated carbon with heat.

An overall process flow diagram depicting the unit operations incorporated in the selected process flowsheet is presented in Figure 17.1.

The key issues considered in process and equipment selection are outlined in the following section.

Figure 17.1 Overall Process Flow Diagram



17.1.2 Key Process Design Criteria

The key process design criteria listed in Table 17.1 form the basis of the detailed process design criteria and mechanical equipment list.

Table 17.1 Summary of Key Process Design Criteria

	Units	Design	Source*
Plant Throughput	tpa	2,750,000	IAMGOLD
Head Grade	g/t Au	1.71	IAMGOLD
Crushing Plant Availability	%	75.0	Lycopodium
Plant Availability	%	92.0	Lycopodium
Crushing Work Index (CWi, ave)	kWh/t	13.5	Testwork
Bond Ball Mill Work Index (BWi, ave)	kWh/t	18.6	Testwork
SMC Axb[1] (ave)		50.0	Testwork
Bond Abrasion Index (Ai, ave)		0.433	Testwork
Grind Size	µm	75	Testwork
Pre-Leach Thickener Solids Loading	$t/m^2.h$	1.4	Testwork
Leach Circuit Residence Time	hrs	34.4	Testwork
Leach Slurry Density	% solids (w/w)	50.3	Lycopodium
Number of Pre-Oxidation Tanks		1	Testwork
Number of Leach Tanks		5	Lycopodium
Number of Adsorption Tanks (Stages)		6	Lycopodium
Sodium Cyanide Addition	kg NaCN/t ore	0.274	Testwork
Lead Nitrate Addition	Kg $PbNO_3$/t ore	0.2	Testwork
Dissolved Oxygen Level in Leach	ppm	>15	Testwork
Quicklime Addition[2]	kg/t	1.93	Testwork
Elution Circuit Type		Zadra	Lycopodium
Elution Circuit Size	T	5.0	Lycopodium
Frequency of Elution	strips/week	7.0	Lycopodium
Cyanide destruction		Passive	IAMGOLD

Notes:

1. *A x b value derived from the 50th percentile ranking of specific energies determined for bedrock*
2. *Lime addition based on 85% CaO.*

17.2 Process and Plant Description

17.2.1 Introduction

The Boto mineralization is predominantly hosted in quartz veins. Sulphide minerals comprise pyrite, pyrrhotite and traces of arsenopyrite and chalcopyrite. The Boto deposits are considered free milling. The ore body consists of approximately 5% saprolite overlaying a layer of approximately 5% transition material (also referred to as “saprock”) followed by the remaining 90% fresh rock at depth. The proposed process facility will consist of the following process areas:

- Primary crushing and coarse ore storage.
- Grinding, utilizing a SSAG circuit.
- Leach-CIP Carousel circuit.
- Gold recovery and carbon handling circuit (consisting of a cold acid wash followed by a pressure Zadra elution circuit and horizontal carbon regeneration kiln).
- Tailings disposal in a lined TMF with natural degradation of residual cyanide.

The process plant will be designed to process 2.75 Mtpa (7,534 tpd) ore with an average gold head grade of 1.71 g/t Au. ROM feed to the Plant will be a blend of 90% bedrock and 10% saprolite/transition material. The process plant was designed to be fit-for-purpose with no allowances for future expansion.

The primary crusher installation will, from the onset of production, have the full capacity of 2.75 Mtpa available. The primary crushing circuit will operate for 365 days per annum, for a nominal 24 hrs/day. On this basis, and at a design operating availability of 75%, the crushing circuit will operate for a nominal 6,570 hrs per annum. This equates to a nominal crusher circuit throughput of 419 t/h.

Downstream of the crushing circuit, the grinding, leach, adsorption and tails disposal circuits will operate for 365 days per annum, for a nominal 24 hrs/day. On this basis, and at a design operating availability of 92%, these circuits will operate for a nominal 8,059 hrs/annum. This equates to a nominal circuit throughput of 341 t/h.

The grinding circuit will consist of a single 32’ (9.75 m) diameter by 20.9’ (5.97 m) length (EGL) SAG mill. The SAG milling circuit is closed out by hydro-cyclones with the cyclone overflows reporting to a vibrating trash screen ahead of the pre-leach thickener.

The gold leach circuit will be comprised of a single pre-oxygenation tank followed by five leach tanks, providing a nominal leach residence time of 34 hrs. The CIP carousel adsorption circuit will be comprised of six adsorption contactors. These contactors will operate with a carbon concentration of 50 g/L on a one day cycle. The total slurry residence time in the CIP section is 1.3 hrs.

The gold recovery and carbon handling circuit will operate on a batch basis at a carbon throughput rate of 5 tonnes of carbon daily.

17.2.2 Ore Receiving and Crushing

Run-of-mine (ROM) ore from the open pit will be transported to the plant by 95 tonne capacity rear dump trucks. The trucks will tip directly into the ROM bin. However, allowance will be made for a ROM stockpile to blend material per grade and hardness. The ROM stockpile will be primarily utilised for ore blending to optimize mill power consumption, grade and to ensure a maximum saprolite content not exceeding 20% in the feed to the crusher. Ore will be reclaimed, from the stockpile, to the ROM bin by a front-end loader.

A static grizzly (600 x 600 mm), mounted above the ROM bin, will prevent the ingress of oversize material. A mobile rock breaker will be used to break oversize material retained on the static grizzly. Ore will be withdrawn from the ROM bin, by a variable speed apron feeder, directly into a jaw crusher, which will operate in open circuit. Crushed ore from the crusher will discharge directly onto the primary crusher discharge conveyor, which will convey the crusher product to the mill feed bin.

The crusher discharge conveyor will be fitted with a weightometer, to monitor and control the crushing area throughput by adjusting the output of the apron feeder variable speed drive.

The crushing circuit will be serviced by a single dust collection system, comprised of a series of extraction hoods, ducting and a bag house. Dust collected from this system will be discharged onto the crusher product conveyor.

A static tramp metal magnet will be installed at the discharge end of the primary crusher discharge conveyor. Tramp metal will be manually removed from the magnet when necessary.

Any spillage generated, within the crushing area, will be manually recovered and transported to the mill feed bin.

Auxiliary equipment for the crushing circuit will include:

- Crushing area control station.
- Primary crusher maintenance hoist.
- Primary crusher lube pack.
- Primary crusher area camera.

17.2.3 Coarse Ore Storage

The mill ore feed bin will have a live capacity of 108 t (equivalent to approximately 18 mins plant feed at the instantaneous feed rate to the SAG mill). The mill feed bin includes an overflow facility, with excess crushed ore conveyed to the crushed ore stockpile. The crushed ore stockpile will have a capacity of approximately 5,000 tonnes (providing 14.5 hrs of plant feed). Crushed ore will be reclaimed from the stockpile, to the ore bin, via a front-end loader.

Crushed ore will be withdrawn from the ore bin, by a variable speed apron feeder. The feeder will discharge onto the SAG mill feed conveyor, which will convey the crushed ore to the SAG mill feed chute. The SAG mill feed conveyor will be fitted with a weightometer, used for controlling the speed of the apron feeder and to account for the mass of feed presented to the grinding circuit.

Quicklime will be added directly to the SAG mill feed conveyor, via the lime variable speed rotary feeder. The Quicklime storage silo will have a storage capacity of 54 t, equivalent to 3.4 days storage. To mitigate the risk of lime shortages a strategic inventory of at least 102 1t lime bags will be kept in storage, providing a 7-day contingency supply.

Grinding media (a mix of 125 mm and 38 mm balls) will be added to the crushed ore bin using a front-end loader.

Any spillage generated, within the reclaim area, will be manually recovered and transported to the mill feed bin, utilizing a dedicated drive-in sump and sump pump. Supernatant solutions from this sump will be pumped to the mill discharge pumpbox while the solids will be transported to the ore bin using a front-end loader.

Auxiliary equipment for the reclaim area will include:

- Crushed ore stockpile dust suppression sprays.
- Lime silo and lime hopper dust collector.
- Lime hoist, lifting frame.
- Weightometer calibration chain.

17.2.4 Grinding and Classification

The grinding circuit will be a SSAG circuit, comprised of a single, variable speed, semi-autogenous grinding (SAG) mill. The SAG mill will operate in closed circuit with hydro-cyclones while pebbles will be removed by a Trommel screen and recycled back to the SAG feed conveyor via two conveyors. The product particles exiting the grinding circuit (cyclone overflow) will contain 80% passing 75µm material.

To achieve the required leach product size when treating ore at the 85th percentile of hardness, a 9.75 m x 6.38 m SAG mill (32 ft x 20.9 ft; 11.60 MW) will be required.

Crushed ore, reclaimed from the ore bin, will be conveyed to the SAG mill feed chute via the SAG mill feed conveyor. Process water will be added to the SAG mill feed chute, to control the in-mill pulp density. The SAG mill will be fitted with discharge grates, which will allow slurry to pass through the mill and will also relieve the mill of pebble build-up. The SAG mill product will discharge to a Trommel screen for size classification.

SAG mill trommel screen oversize will be recycled back to the SAG Mill feed conveyor via two conveyor belts. Undersize from the discharge screen will flow by gravity to the SAG mill discharge pumpbox, prior to being pumped to the classification cyclone cluster by a single variable speed cyclone feed pump. The classification cyclone cluster overflow will flow by gravity, via a trash screen, to the pre-leach thickener feed distribution box. This overflow stream will be sampled for metallurgical accounting before reporting to the trash screen. Trash screen undersize will gravitate to the pre-leach thickener, whilst trash screen oversize will be discharged to a trash bin. Underflow slurry, from the classification cyclone underflow launder, will flow by gravity back to the SAG mill feed chute.

Spillage within the grinding circuit will be managed through a slanted floor draining to a central sump fitted with a sump pump. Slurry from this sump will be discharged into the mill discharge pumpbox. During flooding events the excess water will flow via trenches to an event pond.

Auxiliary equipment within the grinding area will include:

- SAG mill drive lubrication system.
- SAG mill liner handler and relining monorail.
- Mill area mobile crane.
- Cyclone maintenance hoist.

17.2.5 Pre-Leach Thickening

Trash screen undersize will flow by gravity directly to the pre-leach thickener feed box, where flocculant will be added to aid with particle settling. Overflow from the pre-leach thickener will flow to the process water tank. Underflow from the pre-leach thickener, at 50.7% solids, will be pumped by dedicated thickener underflow pumps to the leach circuit feed distribution box.

The pre-leach thickener area will be serviced by a dedicated sump pump. Spillage and wash down collected by the sump pump will be returned to the pre-leach thickener distribution box. Excess water will overflow from this bunded area via trenches to the event pond.

Auxiliary equipment within the pre-leach thickener area will include:

- Pre-leach thickener flocculant in-line mixer for diluting mixed flocculant at 0.25% strength down to a final feed concentration of 0.025%.
- De-aeration feed box.

17.2.6 Leach Circuit

Pre-leach thickener underflow will be pumped to the leach feed distribution box. The slurry from the leach feed distribution box will discharge into the pre-oxygenation tank. If the pre-oxygenation tank is offline, the slurry will be diverted to the first leach tank, via an internal dart plug distribution system.

Compressed oxygen will be bubbled through the slurry in the pre-oxygenation tank to oxidize cyanide consuming species and to improve downstream leach kinetics.

The leach circuit will consist of five, mechanically agitated, leach tanks operating in series. This equates to a nominal residence time of 34.4 hrs at a slurry feed rate of 458 m^3/h. Each leach tank will have a live volume of 3,150 m^3.

Cyanide, for gold dissolution, will be added to the leach circuit by dedicated cyanide dosing pumps. The primary cyanide dosing point will be the first leach tank, with further addition points located down the leach train. The operating pH of the leach circuit will be maintained above 10.5 to maintain the protective alkalinity of the circuit and prevent the loss of cyanide to gaseous hydrogen cyanide. Protective alkalinity will be maintained by the addition of quicklime to the SAG mill feed conveyor. Because of the slow response time between the lime addition point and the leach tanks a secondary means of emergency pH control is provided in the form of caustic addition points into each leach tank. Note that this will be used in emergencies only and will need to be monitored to limit unnecessary or accidental caustic consumption.

To aid with gold dissolution, oxygen will be added to the leach circuit. Oxygen will be supplied from the PSA system's oxygen storage tank and injected into each CIL tank via dedicated oxygen spargers. To sustain the desired dissolved oxygen levels, external oxygen contactors will be installed on the first two leach tanks. Lead nitrate will also be added to the leach feed distribution box.

Following dissolution, the solubilised gold will be recovered by carbon adsorption, within the dedicated carbon-in-pulp carousel (Pumpcell®) circuit. Leach circuit tails slurry will gravitate to the carousel circuit feed distribution launder.

Should a leach tank be off-line for maintenance, it will be possible to bypass any of the leach tanks. The ability to bypass tanks will be made possible by the installation of two pneumatic gates located within the leach inter-stage launders. One gate will divert slurry to the following leach tank while the second gate will allow slurry diversion to the subsequent leach tank.

The leach area will be serviced by two sump pumps. The sump pump closer to the front-end will return spillage to the leach feed distribution box. The sump pump closer to the back-end will discharge spillage to either the final leach tank or to the trash screen as an alternative. The leach bund area will overflow in case of emergencies to the event pond.

Auxiliary equipment within the leach area will include:

- Cyanide analyser.
- Hydrogen Cyanide (HCN) monitor.
- Control and titration room.

17.2.7 Carbon Absorption Circuit

The slurry from the leach circuit will report to the Pumpcell® CIP plant feed launder. The feed launder will distribute the slurry to the first tank, within the carousel adsorption sequence.

The Pumpcell® circuit will consist of six, mechanically agitated, tanks operating in series each with a live volume of 100 m^3. This equates to a total slurry residence time of 1.3 hrs at a feed rate of 461 m^3/h. The tanks will operate with a carbon concentration of 50 g/L. The adsorption tanks will operate on a daily cycle with the single stage inventory of 5 t carbon recovered and discharged to the acid wash circuit every day.

Activated carbon will be retained in each of the tanks, by an inter-tank screen, which is integral to the tank agitator mechanism. The inter-tank screen will be a stainless steel wedge wire cylinder equipped with an internal agitator and external rotating wiper blade mechanism to prevent screen blinding.

The Pumpcell® circuit operates as a carousel, with carbon retained within the tanks and the slurry advanced counter current to the carbon adsorption stage. Slurry will nominally flow from Tank 1 through to Tank 6, with the gravitational flow between tanks induced by the pumping action of the tank agitator/screen mechanism.

Periodically (1 hour every day), a complete batch of loaded carbon, from the first contactor, will be pumped by the Loaded Carbon Recovery Pump to the Loaded Carbon Screen, where it will be washed with spray water to remove excess slurry. The excess slurry (screen underflow) will return to the Pumpcell® feed launder whilst the loaded carbon will discharge to the acid wash column.

Regenerated carbon (or fresh carbon) will be hydraulically added to the Pumpcell® circuit, from the carbon regeneration circuit. The regenerated carbon (or fresh carbon) will be pumped, to the Pumpcell® circuit, via the carbon sizing screen. The sizing screen will remove excess water and carbon fines. The dewatered carbon will discharge into the last adsorption tank, with excess water and carbon fines flowing to the carbon safety screen.

Slurry discharging the last adsorption tank will flow by gravity to a transfer hopper, from where it will be pumped to a carbon safety screen via the carbon safety screen feed box. The carbon safety screen will capture and recover any carbon exiting the adsorption circuit. The safety screen oversize will report to a fine carbon bin while the undersize will flow to the tails pumpbox. A sampler, installed on the carbon safety screen feed will periodically collect a sample of the adsorption tail stream. This sample will be used for circuit monitoring and for metal accounting.

The adsorption circuit will be serviced by a dedicated sump pump. The sump pump will return spillage to the Pumpcell® CIP feed distribution launder.

Auxiliary equipment within the Pumpcell® circuit area will include:

- CIP overhead gantry crane.
- Spare Pumpcell® mechanism.
- High pressure cleaner.

17.2.8 Elution and Carbon Regeneration

The Elution circuit will consist of separate acid wash and elution columns. A cold acid wash will be utilized. Following acid wash, gold will be eluted from the carbon, utilizing a pressure Zadra elution process. With the CIP circuit operating on a daily cycle, one elution will be completed every day. The elution circuit will be designed to complete one strip per 20-hour period. At a carbon gold loading of 2,659 g/t the required daily carbon movement equates to 5 t.

Acid Wash

The cold acid wash sequence will be required to remove accumulated, calcified scale, from the carbon surface. The acid wash column fill sequence will be initiated by taking the first carbon tank offline and pumping its entire content to the Loaded Carbon Screen. Carbon will gravitate from the Loaded Carbon Recovery Screen directly into the Acid Wash Column while the underflow slurry from this screen will return by gravity back into the carousel feed launder. Once the Acid Wash Column is filled to the required level, the carbon fill sequence will be stopped.

The acid wash cycle will utilize a 3% w/v hydrochloric acid solution. This dilute acid will be prepared by the addition of raw water and neat (32%) hydrochloric acid, into the Hydrochloric Acid Dilution Tank. The acid wash sequence will involve the injection of the dilute acid solution into the column, by the Hydrochloric Acid Dosing Pump, via the feed manifold located beneath the column. Once the required amount of acid has been added to the column, the HCl Acid Pump will be stopped and the carbon will be allowed to soak for a period of one hour.

Upon completion of the acid soak, the acid wash cycle will be initiated by pumping dilute acid solution through the column for a period of 2 hrs. The acid solution will be recycled back to the dilute acid tank. After completion of this step the acid rinse/neutralization step will be initiated. During the rinse cycle, four bed volumes (4 BV) of water, at 2 BV/h, will be pumped through the column. The first 2 BV will include a caustic injection, to neutralize the acid waste, whilst the last 2 BV are comprised of a fresh water rinse only. Acid waste and displaced solution from both the acid rinse and wash steps will pass through the Acid Wash Discharge Strainer before discharging to the Tails Collection Hopper.

The sequence will conclude with carbon being hydraulically transferred to the Elution Column. Water, for carbon transfer between the acid wash and elution columns, will be supplied from the Transfer Water Tank via the Transfer Water Pump.

Elution

The elution sequence will commence with the injection of make-up raw water into the Strip Solution Tank, along with the simultaneous injection of cyanide and caustic solution. A set amount of cyanide and sodium hydroxide (caustic) will be added to achieve a 1% w/w NaOH and 0.2% w/w NaCN strip solution. Both reagent additions will be automatically stopped once the prescribed volume has been added. The pre–heating period will then commence. During this period, the strip solution will be circulated through the first heat exchanger to pre-heat it to 95°C. Upon completion of pre-heating, the elution sequence will commence and gold will be stripped from the carbon. During this stripping time barren eluate, from the strip solution tank, will be pumped, through the heat recovery heat exchanger, picking up residual heat from the eluate exiting the elution column.

The pre-heated, incoming eluate, will then pass through the Primary Heat Exchanger to elevate the eluate temperature to 135°C prior to entering the base of the column. A diesel fired elution heater will provide the heat to the heat exchangers. A temperature probe will monitor the temperature of eluate exiting the column, which will be used to control the heater output. Eluate will flow up through the carbon bed and out of the top of the column, passing through the recovery heat exchanger via the elution discharge strainers to the flash tank. Initially, eluate emerging from the heat exchanger will be directed to the pregnant eluate tank. Pregnant eluate will flow by gravity from the Flash Tank through the electrowinning (EW) cells, with the barren eluate exiting the EW cells being pumped back to the strip solution tank.

A total of 32 bed volumes of strip solution will be cycled through this closed-circuit comprising of the strip column and EW cells. Upon completion, heating will cease and cooling water will be injected into the circulating stream for a period of 1 hr. This cooling water will displace a portion of the strip solution, which will be bled from the circuit to the CIP feed launder. Upon completion of the cool down sequence, the eluted carbon will be hydraulically transferred to the carbon regeneration kiln de-watering screen.

The elution area will be serviced by the Elution Area Sump Pump. Elution area spillage will be pumped to the CIP feed launder.

Auxiliary equipment within the Acid Wash and Elution circuits will include:

- Elution Heater Pump.
- Strip Solution Heater.
- Acid wash and elution column discharge strainers.

Carbon Regeneration

After elution, the carbon will be hydraulically transferred from the elution column to the carbon regeneration circuit by pressurizing the column with transfer water. The carbon and transfer water will be directed to the carbon dewatering screen, allowing excess water to be removed prior to the carbon discharging into the carbon regeneration kiln feed hopper. Dewatering screen undersize will gravitate to the carbon safety screen.

Carbon will be withdrawn from the Kiln Feed Hopper, via the Kiln Screw Feeder, and discharged directly to the carbon regeneration kiln, at a rate of 250 kg/h. Within the diesel fired, horizontal rotary kiln, the carbon will be heated to 700°C, to remove volatile organic foulants from the carbon surface, thereby restoring the carbon activity.

Re-activated carbon exiting the kiln will discharge directly to the carbon quench vessel, where it will be submerged into water and rapidly cooled. From the quench tank, carbon will be pumped, by the regenerated carbon transfer pump, to the carbon sizing screen. Sizing screen oversize will flow to the CIP Feed Launder. Sizing screen undersize will discharge to the carbon safety screen feed box. Fresh carbon will also be added to the CIP circuit via the Quench Tank.

Auxiliary equipment within the Carbon Regeneration circuit will include:

- Fresh Carbon Loading Hoist.
- Carbon Transfer Water Pump.

17.2.9 Electrowinning and Gold Room

Soluble gold and silver recovery, from the pregnant eluate, will be achieved by electrowinning onto stainless steel cathodes. The electrowinning circuit will consist of two parallel electrowinning cells, each containing 12 cathodes and 13 anodes. A dedicated rectifier, per electrowinning cell, will supply the necessary current, to electroplate the dissolved gold onto the cathode.

Once the elution pre-heating cycle has been completed, the electrowinning sequence will be initiated by diverting strip solution through the closed loop of the elution column and EW cells. During the electrowinning cycle, the electrowinning cell discharge will be continuously returned to the strip solution tank, via gravity.

Upon completion of electrowinning, gold sludge on the plated cathodes will be washed off the cathodes, with a high pressure cathode washer. The gold bearing sludge will be recovered to a sludge hopper, from where it will be filtered, via a pressure filter.

The gold bearing filter cake will be thermally dried in an electric drying oven. Dried filter cake 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 react with base metal oxides to form a slag, whilst the gold and silver 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. The slag produced 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.

Auxiliary equipment for the Electrowinning and Gold Room circuit will include:

- Furnace bag house, extraction fan and stack.
- Electrowinning cell fume hood.
- High pressure cathode cleaner.
- Smelting furnace cascade trolley and slag cart.
- Doré moulds and doré wash table.
- Flux bin, platform scale, flux mixing table.
- Doré balance.
- Doré safe and strongroom.

17.2.10 Tailings Disposal

Slurry from the CIP circuit will be pumped to a carbon safety screen by the CIP Tails Transfer Pump. The carbon safety screen will capture and recover any carbon exiting the adsorption circuit. The safety screen oversize will report to a fine carbon bin while the undersize will flow by gravity to the Tails Collection Hopper. A sampler, installed on the carbon safety screen feed will periodically collect a sample of the adsorption tail stream. This sample will be used for circuit monitoring and for metal accounting. Tailings will be pumped from this hopper to the lined tailings management facility for permanent storage.

Residual cyanide will degrade naturally through hydrolysis and UV irradiation in the TMF.

17.2.11 Reagents Mixing and Storage

The major reagents utilized within the process plant will include:

- Quicklime (85% CaO) for pH control.
- Sodium cyanide ($NaCN$) for gold dissolution and desorption.
- Lead nitrate ($PbNO_3$) for gold dissolution (leach accelerant).
- Caustic soda ($NaOH$) for neutralization and desorption.
- Hydrochloric acid (HCl) for carbon acid washing.
- Flocculant for thickening.
- Antiscalant to reduce fouling in the process water distribution, carbon wash and stripping circuit.

- Fluxes for smelting charge preparation.

Quicklime

Quicklime will be delivered to site in bulk 36 t tankers or 1,000 kg bulk bags. Bulk tankers will off-load directly to the lime storage silo via a dedicated pneumatic transfer system. The bulk bags will be lifted, by the lime hoist, to the lime storage silo. The lime storage silo will have a storage capacity of 3.4 days or 54 t.

Quicklime will be withdrawn from the silo by a lime screw feeder and deposited directly onto the SAG Mill feed conveyor by a variable speed screw feeder.

Sodium Cyanide (NaCN)

Sodium cyanide will be delivered to site in 1 t boxes of briquettes. The box will be lifted, by the cyanide hoist, to a platform above the cyanide mixing tank.

Process water will be added to the mixing tank to achieve a solution with the desired cyanide concentration (20% w/w). The mixing tank will be mechanically agitated to assist with cyanide dissolution. Sodium hydroxide will be added to the cyanide mixing tank, to maintain the pH above 10.5 during the mixing process. Control of the pH above 10.5 is required to prevent the formation of gaseous hydrogen cyanide (HCN) during the mixing process.

Briquettes will be discharged into the partially filled mixing tank from the platform. After complete dissolution of a batch of cyanide the mixed solution will be transferred into the cyanide storage tank. From here cyanide solution will be distributed, via a cyanide ring main, to the leach circuit. A dedicated metering pump will deliver cyanide solution to the elution column.

The sodium cyanide and sodium hydroxide mixing and storage areas will be placed in a common bunded area serviced by a common sump pump. Any spillage generated within this area will be pumped to the cyanide mixing tank or leach feed distribution box, or, alternatively, to the tails collection hopper.

Sodium Hydroxide (NaOH)

Sodium hydroxide (caustic) will be delivered to site in 1,200 kg bulk bags. The bulk bag will be lifted, by the caustic hoist, to the bulk bag splitter mounted above the caustic mixing tank.

Caustic will be released, from the bulk bag, by the bag splitter. Raw water will be added to the mixing tank to achieve a solution with the desired caustic concentration (20% w/v). The mixing tank will be mechanically agitated to assist with caustic dissolution. Dedicated metering pumps will deliver caustic solution to the acid wash column strip solution tank and cyanide mixing tank. A caustic line will also deliver caustic to the leach section for emergency adjustments to the operating pH.

Hydrochloric Acid (HCl)

Hydrochloric acid (32% w/w) will be delivered to site in 1,000 L bulk containers (IBC). The acid drum pump will transfer acid, from the bulk containers, to the acid mixing and storage tank. From the acid mixing and storage tank, the hydrochloric acid dosing pump will transfer the requisite amount of acid, to the acid wash column.

The hydrochloric acid storage area will be serviced by a sump pump. Any spillage generated within this area will be pumped to the tails collection hopper or returned to the acid mixing tank.

Lead Nitrate

Lead nitrate will be delivered to site in 1,000 kg bulk bags. The bulk bag will be lifted, by the caustic hoist, to the bulk bag splitter mounted above the lead nitrate mixing/storage tank.

Lead nitrate will be released, from the bulk bag, by the bag splitter. Raw water will be added to the mixing tank to achieve a solution with the desired lead nitrate concentration (20% w/v). The mixing/storage tank will be mechanically agitated to assist with lead nitrate dissolution. Dedicated metering pumps will deliver lead nitrate solution to the leach feed distribution box.

The lead nitrate mixing and storage area will be serviced by a dedicated sump pump. Any spillage generated within this area will be pumped to the leach feed distribution box or returned to the lead nitrate mixing/storage tank.

Flocculant

Flocculant powder will be delivered to site in 750 kg bags and mixed in a proprietary mixing system, comprised of a bag breaker, feed hopper, eductor, mixing tank and storage tank. The Flocculant Plant will mix flocculant powder with raw water to achieve the required storage concentration (0.25% w/w).

Upon completion of the mixing cycle, the flocculant will be transferred to the flocculant storage tank, by the flocculant transfer pump.

From the storage tank, flocculant will be distributed to the pre-leach thickener (via an in-line mixer) by the flocculant dosing pumps. Additional water is added to the in-line mixers to dilute the flocculant to 0.025% w/w prior to its discharge into the pre-leach thickener feed launder.

The flocculant area will be serviced by a sump pump. Any spillage generated within this area will be pumped to the flocculant mixing tank or the tails collection hopper.

Antiscalant

Antiscalant will be delivered to the plant in bulk containers (IBC). Dosing pumps will distribute antiscalant, directly from the IBC, to the elution and process water circuits.

Fluxes

The following fluxes, will be delivered to the plant in 25 kg bags, and used in the gold room; Borax ($Na_2B_4O_2$), Sodium Nitrate ($NaNO_3$), Sodium Carbonate (Na_2CO_3) and Silica (SiO_2).

17.2.12 Water Services

The process plant will utilize fresh water, filtered water and process water. Any shortfall of process water will be made up, preferentially, from water contained within the FWP.

Run-off Water, Ponds and Water Management

Dedicated ponds will be provided to manage site water and environmental impacts. Water from the open pit de-watering station will be pumped to the FWP. Waste dump run-off (via waste rock dump toe drains) will be collected within dedicated sediment ponds. From these sediment ponds, water will be pumped to the FWP. From the FWP, water will be pumped to the Raw Water Tank.

Process Water

Process water will predominantly consist of pre-leach thickener overflow and TMF reclaim water. Process water will be stored in a 1,600 m^3 process water tank (PWT), which provides almost two hours of surge capacity. Provision has been made to top-up the PWT from the raw water tank should this be required. From the process water tank, process water will be reticulated via headers by a duty and standby configuration of single stage process water pumps, with off-takes supplied for the predominant user points, namely:

- Grinding (dilution and screen spray water).
- Leaching and CIP circuit (washing and screen spray water).

Fresh Water and Fire Water

Fresh water, for the process plant and mining operation, will be harvested from various runoff collection ponds throughout the Operation. It could be supplemented by river water intake in emergency cases only. Fresh water from the various sources will be stored within the 2.5 Mm^3 FWP, which would provide 58 months of water storage capacity. From here water will be pumped into a raw water tank (RWT) at the plant site. The live volume of this RWT will be 718 m^3, which comprises of 288 m^3 fire water reserve and 430 m^3 raw water storage (4 hrs surge). The raw water suction take-off will be placed such as to ensure a minimum volume of fire water reserve is always available at the bottom of the RWT.

Raw water from the RWT will be reticulated through the plant by dedicated raw water pumps, to the predominant user points, namely:

- Dust suppression (crushing area).
- Carbon regeneration.

- Flocculant dilution at the pre-leach thickener.
- Reagent make-up.
- Workshops and Mine Services.

Firewater will be supplied from the plant raw water tank, via a dedicated suction manifold. The firewater system will comprise of:

- An electrical jockey pump.
- An electrical firewater pump.
- A diesel standby firewater pump.

The firewater system pressure will be maintained by the jockey water pump. An electric fire water pump will automatically start on a drop in line pressure. The diesel fire water pump will automatically start if the line pressure continues to drop below the target supply pressure or during a power failure.

Filtered and Gland Seal Water

Some raw water users require water with a low suspended solids content (cooling water circuit, acid wash circuit and elution circuit, gland seal water). To satisfy this need, a portion of the raw water will be subjected to water treatment by filtration. Treated water, from the water filtration plant, will be stored within a dedicated filtered water storage tank from where it will be pumped to the various end users by dedicated duty and standby pumps. A closed circuit cooling water circuit is utilized, with cooling water being returned to the water storage tank.

Gland water will be supplied from the filtered water storage tank by a dedicated high pressure gland water pump (duty and standby).

Potable Water

Potable water will be sourced from dedicated boreholes and subjected to water treatment including a reverse osmosis (RO) step for calcium, magnesium and chloride removal. Treated potable water will be pumped to separate camp and site storage tanks from where it will be distributed to various end users. Potable water will be distributed for human consumption and to the safety showers and eye wash stations.

17.2.13 Air and Oxygen Services

Plant air at 700 kPa will be provided by two high pressure air compressors, operating in a lead-lag configuration. The entire high pressure air supply will be dried. Dried air will be fed to plant air receivers from where it will be distributed to the required plant areas, via three dedicated air receivers servicing the crushing, grinding and elution areas respectively.

Oxygen, for use within the leach circuit, will be supplied from a dedicated PSA oxygen plant. Oxygen consumption is estimated at 11 tpd (at 90% purity)

17.3 Plant Consumption

17.3.1 Water Consumption

A water balance for the process plant has been completed. Water from the pre-leach thickener overflow stream is recycled within the process plant to reduce external water requirements. During an average rainfall year approximately 295 m^3/h of decant return water is expected to be recycled from the TMF to the process plant. This would fully satisfy the process water requirements i.e. there would be no make-up water required from the raw water system.

Fresh water consumption is estimated at 59 m^3/h. The water balance shows that there will be an excess of 201 m^3/h in an average year (in addition to the excess from the TMF) that would need to be stored in the FWP. Given the very large positive water balance, no extraction from the river is anticipated.

17.3.2 Energy Consumption

The power demand for the plant, along with the rest of the site and camp, will be provided by on-site thermal/solar power plant. The average power demand is summarized in Table 17.2 and utilized for the operating cost estimate. The average power demand does not reflect the instantaneous power demand for equipment start-up and power plant capacity sizing.

Table 17.2 Average Power Demand Summary

Plant Areas	Installed Power (kW)	Average Continuous Draw (kW)	Annual Power Consumption (kWh / year)
Area 550 – Potable Water	78	25	214,620
Area 551 – Fire Protection	92	2.4	21,199
Area 552 – Sewage Treatment and Disposal	79	47	410,581
Area 604 - Crushing	325	192	1,683,584
Area 605 – Ore Handling	180	95	835,879
Area 610 - Grinding	12,896	10,171	89,101,464
Area 620 – Pre-Leach Thickener	241	114	995,224
Area 625 - Leaching & Adsorption	1,084	703	6,159,769
Area 630 – Acid wash and Elution	57	20	173,273
Area 631 – Carbon regeneration	25	2	17,345
Area 632 – Electrowinning and Refining	104	40	349,787
Area 645 – Tails Pumping	264	104	907,448
Area 650 – Reagent Preparation and Storage	129	27	240,112
Area 655 - Air Services	1,019	399	3,497,255
Area 660 - Water Services	666	265	2,319,648
Area 805 - Tailings Management Facility	15	10	90,929
Area 810 – Reclaim Water	90	71	620,646
Area 820 - Raw Water	310	89	779,728
Area 830 - Event Pond	19	1	8,410
Area 420 – On-site Power Distribution - LSP	703	220	1,926,149
Area 500 - Infrastructures and Buildings	505	167	1,460,730
Area 300 - Mining	315	159	1,394,329
Area 515 - Permanent Camp	335	154	1,349,040
Total	**19,529**	**13,077**	**114,557,148**

17.3.3 Reagent and Consumable Consumption

Reagent storage, mixing and pumping facilities will be provided for all reagents for the process plant. Table 17.3 provides a summary of reagents and consumables that will be used at the process plant at the design consumption rate for a plant throughput of 2.75 Mtpa.

Table 17.3 Annual Reagent and Major Consumable Consumption

Reagent/Consumable	Annual Consumption
Jaw Crusher Liners (fixed and swing jaw)	16 sets
SAG Mill Liners	2.4 sets
SAG Mill Grinding Media	4,046 t
Quicklime	5,303 t
Sodium Cyanide	786 t
Lead Nitrate	550 t
Activated Carbon	96 t
Sodium Hydroxide (Caustic)	163 t
Hydrochloric Acid	219 t
Flocculant	41 t
Borax	4.7 t
Sodium Nitrate (Nitre)	0.64 t
Soda Ash	0.64 t
Silica Sand	1.98 t
Smelting Furnace Crucibles	3.7 units
Diesel Fuel (plant usage only)	851m^3

17.4 Plant Control System

17.4.1 General Overview

The following provides a broad overview of the control strategy that will be employed for the plant.

The general control philosophy for the plant will be one with a moderate level of automation and remote control facilities, to allow process critical functions to be carried out with minimal operator intervention. Instrumentation will be provided within the plant to measure and control key process parameters.

The main control room, located in the Mill Office, will house two PC based operator interface terminals (OIT) and a single server. These workstations will act as the control system supervisory control and data acquisition (SCADA) terminals. The control room is intended to provide a central area from where the plant is operated and monitored and from which the regulatory control loops can be monitored and adjusted. All key process and maintenance parameters will be available for trending and alarming on the process control system (PCS).

Two additional OITs will be provided for data logging and engineering/programming functions.

A field touch panel will be installed in the feed preparation area to allow local operator control of the crushing plant to facilitate ease of operation for rock breaking and stockpiling. A second field touch panel will be installed in the elution area to allow local operator control of the elution sequence. A third field touch panel will be supplied for the grinding area.

The process control system that will be used for the plant will be a programmable logic controller (PLC) and SCADA based system. The PCS will control the process interlocks and PID control loops for non-packaged equipment. Control loop set-point changes for non-packaged equipment will be made at the OIT.

In general, the plant process drives will report their ready, run and start pushbutton status to the PCS and will be displayed on the OIT. Local control stations will be located in the field in proximity to the relevant drives. These will, as a minimum, contain start and latch-off-stop (LOS) pushbuttons which will be hard-wired to the drive starter. Plant drives will predominantly be started by the control room operator, after inspection of equipment by an operator in the field.

The OITs will allow drives to be selected to Auto, Local, Remote and Maintenance or Out-of-Service modes via the drive control popup. Statutory interlocks such as emergency stops and thermal protection will be hardwired and will apply in all modes of operation. All PLC generated process interlocks will apply in Auto, Local and Remote modes. Process interlocks will be disabled or bypassed in Maintenance mode with the exception of safety related and critical interlocks such as lubrication systems on the mill.

Local selection will allow each drive to be operated by the operator in the field via the local start pushbutton which is connected to a PLC input. Remote selection will allow the equipment to be started from the control room via the drive control popup. Maintenance selection will allow each drive to be operated by maintenance personnel in the field via the local start pushbutton which is connected to a PLC input. A PLC output will be wired to each drive starter circuit for starting and stopping drives. Status indication of process interlocks as well as the selected mode of operation will be displayed on the OIT.

Vendor supplied packages will use vendor standard control systems as required throughout the Project. Vendor packages will generally be operated locally with limited control or set-point changes from the PCS. General equipment fault alarms from each vendor package will be monitored by the PCS and displayed on the OIT. Fault diagnostics and troubleshooting of vendor packages will be performed locally.

Vendor package control will be implemented for the following equipment:

- Pre-leach thickener rake mechanism.
- CIP carousel system.
- Carbon regeneration kiln.
- Flocculant mixing system.
- Compressed air system.
- Instrument air dryers.
- Oxygen plant.

The use of actuated isolation or control valves will be implemented around the plant for automatic control loops or sequencing as part of the plant control or the elution sequence. All actuated valves and control valves will be operated from the OITs with remote position indication available. Automatic control valves will be controlled by PID loops within the PCS.

The PCS will perform all digital and analogue control functions, including PID control, for all non-packaged plant. Faceplates on the PCS displays will facilitate the entry of set-points, readout of process variables (PVs) and controlled variables (CVs) and entry of the three PID parameters (Proportional, Integral and Derivative).

The majority of equipment interlocks will be software configurable. However, selected drives will be hard wired to provide the required level of personal safety protection e.g. the emergency stop buttons associated with each and every motor and the pull wire switches associated with conveyors.

All alarm and trip circuits from field or local panel mounted contacts will be based on fail-safe activation. Alarm and trip contacts will open on abnormal or fault condition. If equipment shutdown occurs due to loss of mains power supply, the equipment will return to a de-energised state and will not automatically restart upon restoration of power.

Sequential group starts and sequential group stops will not be incorporated for non-packaged plant equipment, with the exception of the elution circuit. However, in any process, critical safety and equipment protection interlocks will cause a cascade stop in the event of interlocked downstream equipment stopping (e.g. trip of SAG mill feed conveyor will result in stop of apron feeder). Standard vendor packages may include automatic sequence start/stop controls within the vendor package only.

17.4.2 Control System Configuration and Communications

The process control system will be distributed throughout the plant, with a PLC installed in each of the following locations:

- Feed preparation.
- Grinding circuit.
- Leach/CIP.
- Elution/reagents/gold room
- Water services.
- Raw water dam.
- Sedimentation ponds.
- TMF.

The process plant PLCs will be interlinked via fibre optic cables and will all report back to the main control room. All field instrument and controls will be cabled back to their relevant switchroom utilizing field marshalling panels where appropriate. Owing to the site topography and location of waste dumps, remote pumping stations will utilize fibre optic communications installed as optical fibre ground wire (OPGW) onto the HV power lines.

17.4.3 Drive Controls

Drives will be powered from starters installed in a Motor Control Centre (MCC) switchboard located in the electrical substation. Each drive MCC will present a ‘Running’ indication and a ‘Fault’ alarm to the PLC system and will have provision for a PLC output contact for ‘Process Interlocks’.

Variable Speed Drive (VSD) units will be of a Variable Voltage Variable Frequency (VVVF) type utilising Pulse Width Modulated (PWM) technology. The drive will be mounted in a free standing cubicle. The drive will be provided with an integral control panel for programming and operation at the VVVF unit for commissioning and emergency running.

The starting of conveyors and rotating equipment, such as mills, will be preceded with a start siren. Interlocks will not be provided to stop large loads starting simultaneously.

De-contactor connectors will be adopted for sump pumps. Sump pumps will have low current trip relays installed in the MCC.

18.0 PROJECT INFRASTRUCTURE

18.1 Overall Site

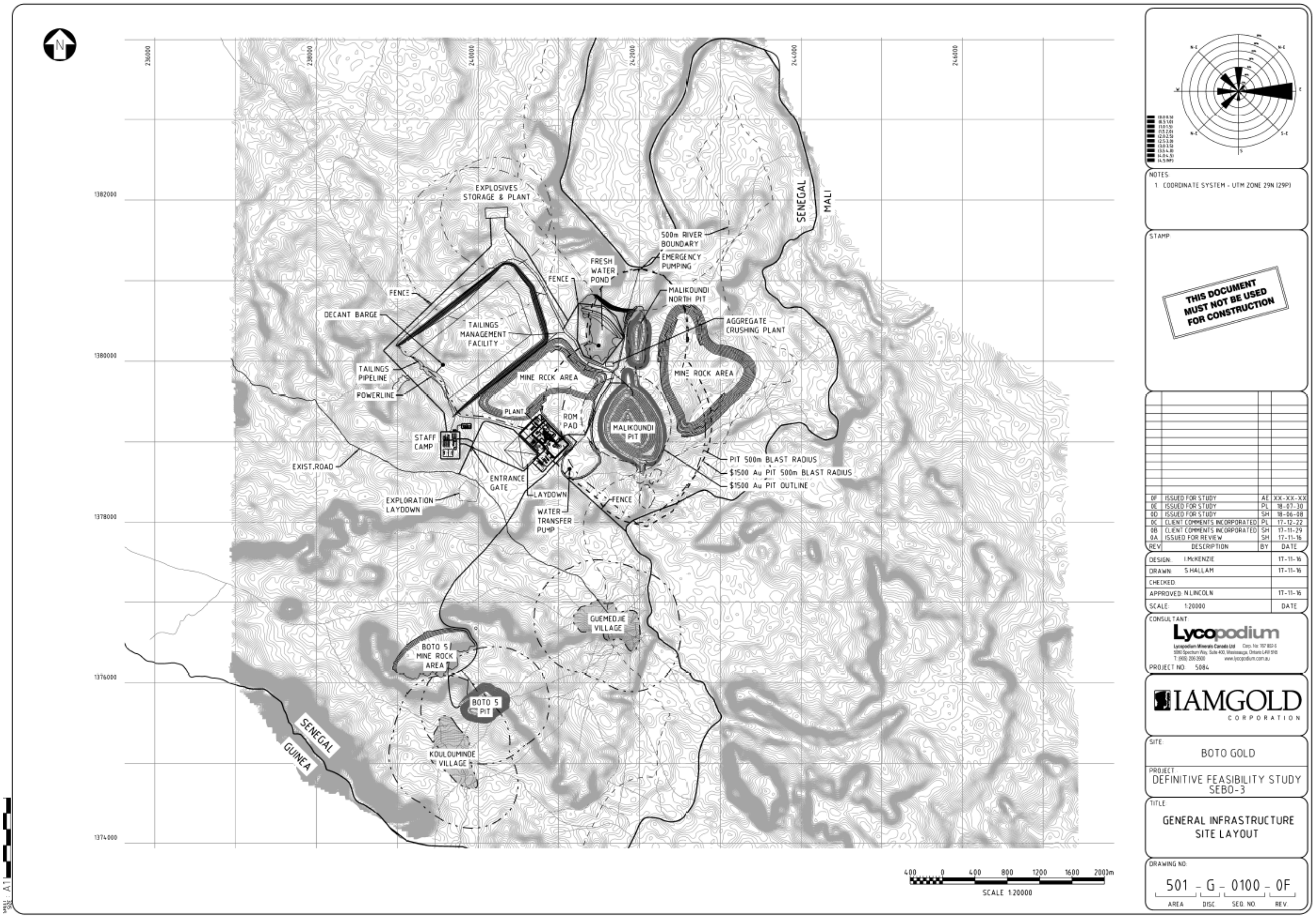
The overall site plan for the Project (refer to Figure 18.1) includes the main facilities such as the open pit mines (Malikoundi, Malikoundi North and Boto 5), waste dumps, process plant, TMF, FWP, staff camp and main access road. An onsite power plant and bulk fuel storage not shown on Figure 18.1 is also provided. An airstrip not shown on Figure 18.1 is provided approximately 10 km from the mine site.

The mine site is adjacent to the Falémé River to the east and the Balinko River to the North.

The process plant, associated buildings, onsite power plant and bulk fuel storage are located west of the Malikoundi mine. The TMF is located north west of the process plant. The staff camp is located near the main access road and west of the process plant for ease of personnel access. The main access road approaches the site from the west.

The site as a whole except for the open pit mines will be fenced to clearly delineate the area, prevent animal access and deter access by unauthorized persons. Road access into the fenced area will be through a manned security checkpoint. Security fencing will surround the accommodation camp. High security fencing will surround the process plant.

Figure 18.1 Overall Site Plan



18.2 Roads

18.2.1 Access to Site

Main access to the mine site is approximately 67 km away via a lateritic road from the village of Saraya. As part of the development of the Project, the main access road will be upgraded and new bridges constructed to access the mine site year-round. Materials and consumables will be transported to the site via an upgraded access road. Culverts will also be installed or upgraded as appropriate on creek crossings. Fill material for the road will be obtained from borrow pits alongside the road where possible. The main access road will be designed for an 80 km/h speed limit. Construction of the access road will commence as part of project early works and at a minimum the bridges will repaired and/or replaced prior to the 2019 wet season to enable continued all weather access to site for construction.

18.2.2 Project Site Roads

Plant internal roads will provide access between the administration area, process plant facilities, bulk fuel storage, power plant, mine services area, and staff camp. These roads will generally be 6 m wide and will be constructed flush with bulk earthworks pads to ensure that storm water sheet flow is achieved across the site, thereby avoiding the need for deep surface drains and culvert crossings within the plant area.

18.2.3 Haul Roads

A network of mine haul roads will be constructed and maintained by the mining department and used to access the pits and waste dumps and for the transport of ore to the process plant and waste to the TMF.

18.2.4 Access Tracks

Several new tracks will be constructed to access infrastructure such as the TMF, sediment ponds, water storage pond, and water bore pumps remote from the plant site. The access tracks will be cleared and graded natural earth tracks. Exact routes will be determined during construction of the Project to best fit local terrain and vegetation density.

18.3 Airstrip

An existing, certified but unused airstrip located approximately 10 km from the mine site will be extended and upgraded to enable turboprop aircraft to access the site. The existing road track to the airstrip will be upgraded to a single lane laterite road. There is no provision for aircraft refuelling.

18.4 Power

18.4.1 Power Supply

Due to the remote location of the Boto site, power will be provided from an onsite power plant located adjacent to the process plant under a ‘Build, Own, Operate’ (BOO) contract arrangement with an independent power provider (IPP). The power plant will supply 11 kV to the process plant from which power will be distributed. Power will also be distributed to infrastructure outside the process plant via overhead pole lines. The power plant has been sized at 19.1 MW connected load to accommodate a peak load of 16.9 MW, and average running load of 13.1 MW with the following configuration:

- 4 x 4.4 MW medium speed HFO units (Wartsila 9L32 or equivalent machines).
- 3 x 2.2 MW high speed diesel units (CAT D3516 or equivalent machines).
- 7.5 MWp solar units.

Three HFO units will satisfy the average running load and the diesel units will supplement the power required during start-up, peak load and when one of the HFO unit is offline for maintenance. The solar units will operate during the day and the thermal generators will provide spinning reserve.

The SAG mill at the process plant is the largest load and has been specified with a variable speed drive to provide a 'soft start' capability to reduce the load surge during start-up and minimize the need for spinning reserve at the power plant.

18.4.2 Electrical Distribution

The electrical system is based on 11 kV distribution and 410 V, 50 Hz working voltage. The 11 kV feeder from the power plant will feed the site distribution 11 kV switchboard. For the process plant the 11 kV supply will be stepped down from 11 kV to 415 V at each switchroom using separate 11 kV/415 V distribution transformers fed from the HV distribution board.

The following switchrooms will be provided in the plant:

- Primary crushing area.
- Grinding area.
- Thickener and water services area.
- Leaching and CIP area.
- Refining, reagents and air services area.
- Mine services area.

Switchrooms will house 415 V motor control centres (MCCs), area VVVF drives, plant control system cabinets, plant lighting transformers, various distribution boards and UPS power distribution.

11 kV overhead power lines will provide power to various remote facilities (TMF pumps, bore pumps, FWP pump, explosive storage area, etc.). Pole mounted transformers will step down the voltage at each location and supply an outdoor 415 V switchboard local to each equipment area.

The staff camp power will be supplied from a local MCC/transformer fed from the 11 kV overhead line.

18.4.3 Electrical Buildings

Electrical buildings will be pre-fabricated 'flat pack' panel buildings to minimize installation time on site. Buildings will be installed on a structural framework over 2 m above ground level to allow for bottom entry of cables into electrical cabinets. The electrical buildings will be installed with air-conditioners and suitably sealed to prevent ingress of dust.

18.4.4 Transformers and Compounds

All the 11 kV/415 V distribution transformers will be of ONAN cooling configuration and vector group Dyn11.

Fire rated concrete walls will be constructed around the pad mounted transformers.

Outdoor rated 11 kV/415 V kiosk substations will be used to provide power to the TMF pumps, bore pumps, FWP pump and explosive storage area.

18.5 Fuel Supply

Bulk fuel supply will be provided by an onsite fuel storage facility and will store HFO and diesel for the power plant, mine trucks, light vehicles and users at the process plant. Day storage tanks are provided at the power plant and in the process plant. Bulk lubrication and diesel fuel dispensing is provided for the mine trucks and light vehicles. The fuel supply and facilities will be under a BOO contract arrangement with an independent fuel provider.

18.6 Potable Water

A common potable water system will be provided for the staff camp, process plant and mine services usage and will be located at the staff camp and distributed to the various users. A vendor package modular potable water treatment plant including filtration, ultra-violet sterilisation and chlorination will be installed. Water will be delivered via a reticulation system using a constant pressure variable flow pump system. The pump skid will include a UV disinfection unit to provide additional security against contamination.

18.7 Sewage and Solid Waste Management

18.7.1 Sewage Treatment

Effluent from all water fixtures in the process plant, mine services area, staff camp and administration areas will be pumped to a common sewage treatment plant vendor package located near the staff camp. Treated effluent will be discharged to the TMF. Treatment plant sludge will be suitable for direct landfill burial.

18.7.2 Solid Wastes

Wastes will be sorted and reused or recycled as much as possible. Waste lubricating oils and general non-hazardous solid wastes will be removed and disposed of by the fuel provider. Dangerous or hazardous waste will be collected and stored briefly before being transferred to a suitable permitted facility, either on-site or off-site depending on the specific materials and requirements.

18.8 Accommodation Camps

18.8.1 Staff Camp

A 132-room staff camp will be located west of the process plant under a BOO contract arrangement with an independent camp provider. The staff camp will provide accommodation for senior staff not originating from the local area. The camp provider will also provide meals for all personnel during the day. All camp buildings will be of modular design for ease of transport to site and relatively quick installation.

The staff camp will be fenced and consist of the following:

- 6 x blocks consisting of 22 single rooms with en-suite bathrooms
- Guard house.
- Recreation facilities.
- Mess
- Kitchen offices and storage area.
- Laundry.
- Soccer field

18.8.2 Construction Camp

During construction, the permanent staff camp will be erected early and used by the owner’s team, EPCM contractor’s team and vendor representatives.

Majority of semi-skilled and unskilled labour required for project implementation will be sourced from Saraya and surrounding villages. Contractors will be required to make their own accommodation arrangements with local businesses. Contractors will also be required to make arrangements for bussing their employees to and from site but the Project will provide for the midday meal.

18.9 Mine/Plant Site Facilities

18.9.1 General

Site buildings will be 'fit for purpose' industrial type structures. Workshops, warehouses and reagent storage sheds will primarily consist of sea containers. Structural steel frames connecting the sea containers, with concrete flooring will be used for workshops. Offices and amenity buildings will be modular type.

18.9.2 Outside Plant Area

The following building and facilities will be provided outside the plant area:

- Metallurgical and assay laboratory (provided as a contract laboratory).
- Site administration building.
- Medical centre.
- Long term reagents storage.
- Main gatehouse.

18.9.3 Inside Plant Area

The following building and facilities will be provided inside the plant area:

- Plant office with control room.
- Plant workshop and maintenance.
- Plant warehouse
- Plant mess.
- Male and female ablutions.
- Short term reagent storage.
- Plant gatehouse.

18.9.4 Mine Services

The following building and facilities will be provided for the mine services area:

- Mine office.
- Mine mess.
- Mine shift change building.
- Mine vehicle workshop.
- Mine warehouse.
- Truck-wash down facility.

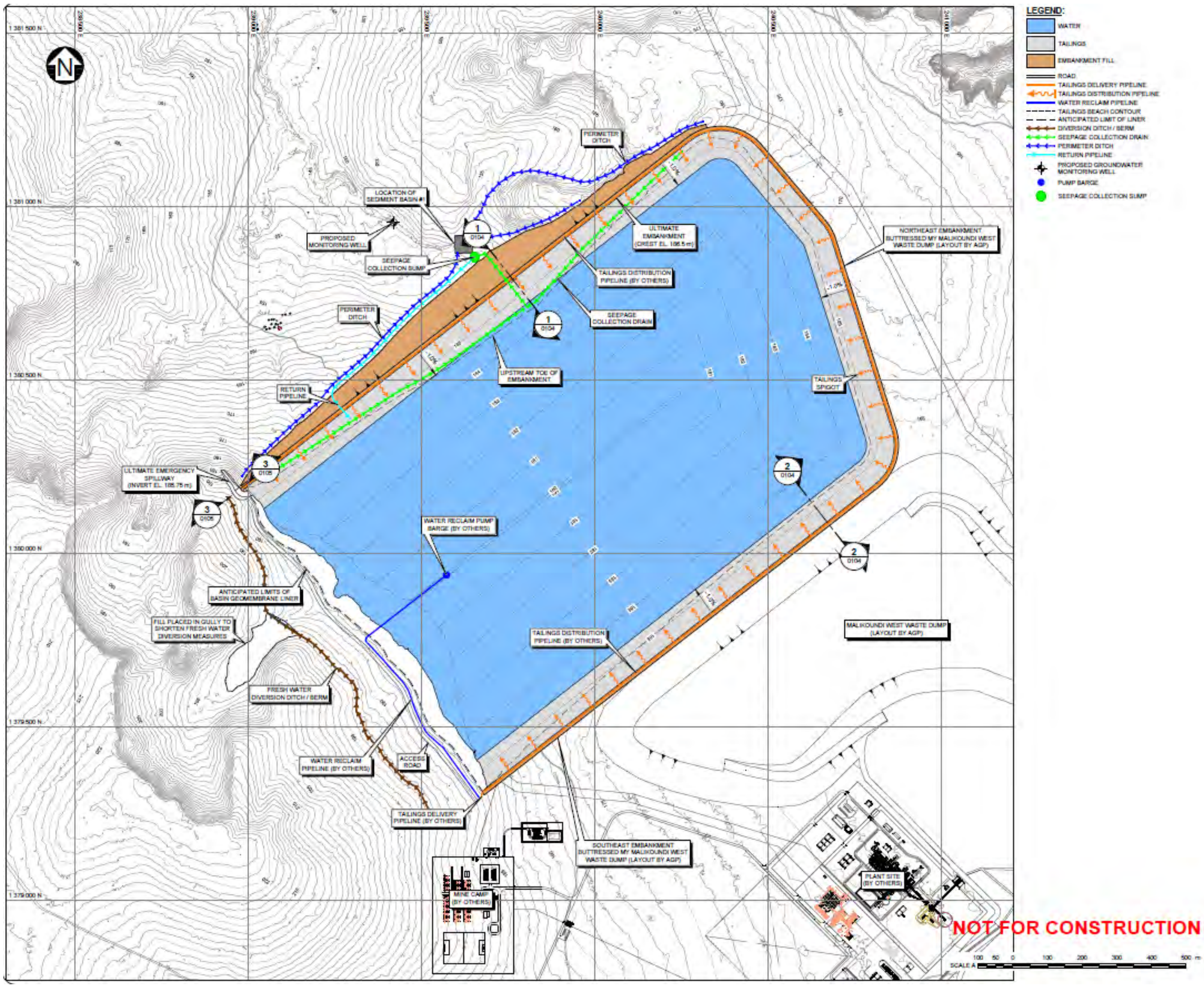
18.10 Tailings Management Facility

18.10.1 Introduction

KP completed a feasibility level design for the TMF for the Project. The TMF will provide secure storage for tailings and process water, and protect groundwater and surface waters during operations and post closure. The feasibility level design is based on a projected 12.8-year mine life at a nominal processing rate of approximately 2.75 Mtpa. The TMF has been sized to permanently store approximately 35.06 Mt of tailings, or 26.0 Mm^3 at an average settled dry density of 1.35 t/m^3.

The TMF is located northwest of the process plant. The ultimate configuration of the TMF is shown on Figure 18.2.

Figure 18.2 Tailings Management Facility – General Arrangement – Ultimate Facility



18.10.2 Tailings Characteristics

Physical Properties

Physical testwork was completed on samples of tailings to estimate the geotechnical properties to support the design of the TMF. The samples were obtained from composite sample MC2 gravity separation testwork and tested by SGS Minerals Services (SGS).

The tailings samples underwent a range of physical testwork to characterize the settling and consolidation properties of the tailings. The results are summarized as follows:

- **Specific Gravity**: 2.81
- **Grain Size Distribution**: Sandy (26%) SILT (66%), trace clay (8%). The P_{80} was measured to be approximately 90 µm.
- **Atterberg Limits**: The liquid limit was measured to be 19% and the plastic limit was not obtainable. The tailings are non-plastic.
- **Classification**: Based on the results described above and the unified soil classification system (USCS) the tailings are classified as ML (inorganic sandy silt; trace clay with no plasticity)
- **Consolidation**: The Coefficient of Consolidation (Cv) values obtained from the consolidation testing range from 26 m^2/year at low effective stresses to 36 m^2/year at high effective stresses. The results indicate that the sample will consolidate slowly with loading.
- **Permeability**: The coefficient of vertical permeability (kv) of the tailings was measured to range from 4.6×10^{-6} m/s at low effective stresses to 1.9×10^{-8} m/s at high effective stresses.
- **Settled Dry Density Estimate**: Tests were completed at 43%, 48%, 53%, and 62% solids content by weight under undrained, drained, and drained and air-dried conditions. The average settled dry density of the tailings to be deposited into the TMF has been estimated to be 1.2 t/m^3 in the early stages, increasing to an overall density of around 1.3 to 1.35 t/m^3. It is expected that more extensive tailings beaches will develop with time for later stages of operation that will allow for sub-aerial (above water surface) tailings deposition to promote drainage, air drying, and consolidation of the tailings.

  Field results are depended on the deposition strategy, pond size and height of the storage facilities, area available for drying, thickness of deposited layers, climatic conditions at site and operating parameters of the processing plant. A suitable deposition plan and efficient operation of the facility can improve settled density for the project

Geochemical Properties

The geochemical characterization program, including total metals and acid-base accounting, completed by KP at SGS for IAMGOLD, has indicated that the majority of the rock contained in and surrounding the ore body will not have Acid Rock Drainage (ARD) or Metal Leaching (ML) potential.

18.10.3 Design Basis Overview

Objectives

The principal objectives of the TMF design are to provide secure storage for tailings and process water, and protect groundwater and surface waters during operations and post closure. The feasibility level design for the TMF has taken the following requirements into account:

- Permanent, secure, and total confinement of all solid waste materials within an engineered facility.
- Control and collection of potential seepage from the TMF basin and runoff from the embankments during operations.
- Control, collection, and recycling of process water and runoff within the TMF basin to maintain a suitable operating pond volume and maintain the operating pond well away from the embankment.
- The inclusion of monitoring features for all aspects of the TMF to compare actual facility performance against design expectations and help verify the ongoing safe operation of the facility.

Embankment construction will be scheduled to provide sufficient storage capacity and freeboard in the TMF to temporarily store runoff resulting from the Inflow Design Flood (IDF). The design basis and operating criteria are based upon accepted national and international standards for mining dam design and operation (Canadian Dam Association - CDA, 2014, Mining Association of Canada, 2017).

Dam Hazard Classification (DHC)

The DHC has been determined based on the criteria below.

- Population at risk and loss of life.
- Environmental and cultural values.
- Infrastructure and economics.

The criteria were assessed based on the assumption that failure of the embankment would release all stored water and a portion of the tailings to the environment in an uncontrolled manner. A DHC was assigned for each of the individual categories listed above. The DHC was selected by taking the highest DHC from the individual categories.

A hypothetical failure of the TMF embankment could potentially cause incremental losses along an inundation route downstream of the TMF. Water and sediments (from subsequent erosion) from a hypothetical failure of the embankment would be directed by the natural topography to the Falémé River. The Falémé River is the primary contributing factor to the DHC. Restoration or compensation of damage to the Falémé River from a potential breach in the embankment is suspected to be extremely difficult due to the consequences associated with remediation of an international river. The Falémé River borders Mali (immediately downstream) and Mauritania (further downstream).

The TMF has been identified as having a DHC of Extreme based on a review of the criteria outlined above. The Earthquake Design Ground Motion (EDGM) and Inflow Design Flood (IDF) thresholds used for the design reflect this classification.

Seismic Design Criteria

The EDGM for a facility with a DHC of EXTREME during operations is specified as the estimated ground acceleration generated during the 1 in 10,000-year earthquake or the MCE, whichever is higher. The peak ground acceleration (PGA) for the 1 in 10,000-year event is 0.069g assuming rock site conditions (Site class B; ASCE/SCI 7-10).

Hydrologic Design Criteria

The IDF for a facility with a DHC of EXTREME is specified as the runoff generated from the Probable Maximum Precipitation (PMP) event. The TMF is required to provide sufficient wet freeboard to temporarily store the IDF above the maximum operating level during operations. This event for the TMF equates to 524 mm of rainfall over 24 hours. The runoff that would report to the TMF during this rainfall event is estimated to be approximately 1,189,000 m^3. For added security, the facility will also be equipped with a spillway capable of passing the PMP.

18.10.4 TMF Design

General

Tailings will be pumped as a slurry from the process plant to the TMF via a tailings pipeline. The TMF will be constructed as a single cell facility within the valley located directly northwest of the plant site location. Initially constructed as a valley impoundment, the embankments will be raised to form a three-sided paddock style impoundment. Two (2) embankments will be constructed to establish the TMF, including the Northwest Embankment that straddles the natural valley, and the East Embankment that runs along the northeast and southeast side of the basin adjacent to the Malikoundi West Waste Dump. The TMF design will include an initial starter embankment (Stage 1) with ongoing raises using downstream construction methods throughout the life of the facility.

The feasibility level design of the TMF includes an initial Stage 1 (starter) embankment with five subsequent embankment raises (i.e. Stage 2 to Stage 6) over the projected operational life of the facility. Staged development of the TMF offers the following advantages:

- Reduction of initial capital expenditures.
- Refining of design and construction methods as experience is gained with local conditions and/or as operating criteria change.

- Adjustment of plans at a future date in order to remain current with “state-of-the-art” engineering and environmental practices, etc.

This staged approach will be used for the future design, construction, and operation of the facility as part of a continuous and integrated process to identify cost savings and enhance safety. The approach requires construction controls, monitoring, and review to improve the understanding of site-specific conditions.

Storage Capacity and Filling Schedule

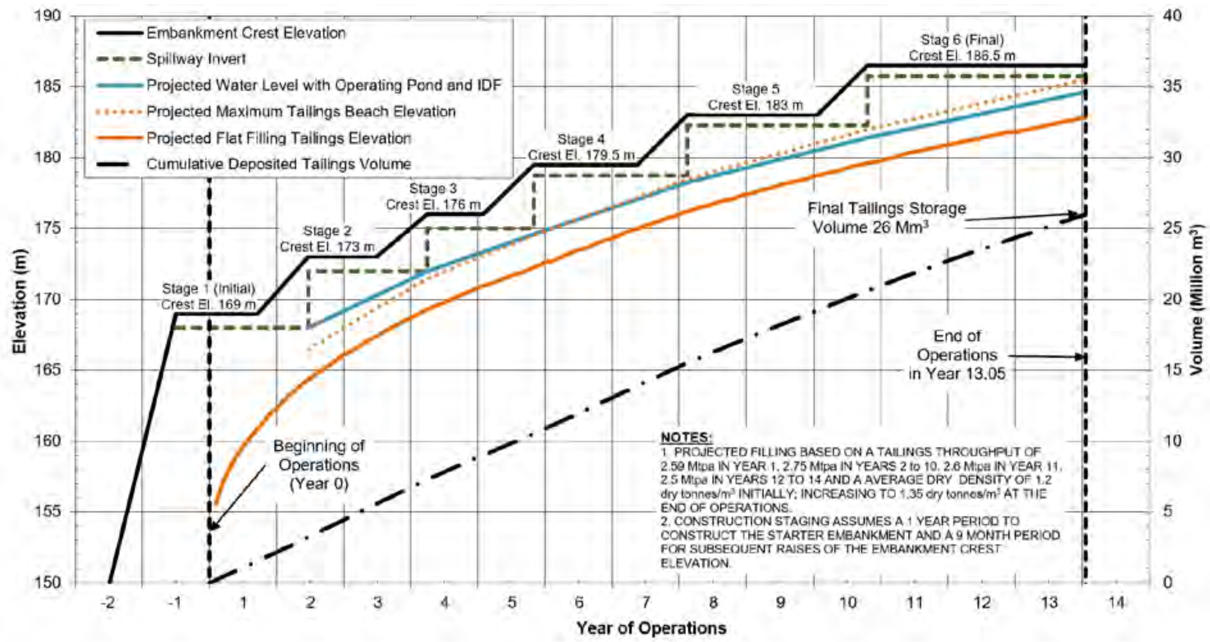
The capacity and filling schedule of the TMF are based on the following:

- Mill throughput data provided by Lycopodium (2018).
- Reduction of initial capital expenditures.
- Local topography, as provided by Lycopodium (2018).
- The storage basin filling characteristics.
- The estimated final average settled dry density of the tailings.
- The supernatant pond volume and stormwater runoff reporting to the TMF during operations.
- Temporary storage of runoff generated from storm events, up to and including the IDF.
- Provision of overtopping protection for wave run-up.

The filling schedule for the TMF is provided on Figure 18.3.

The available storage capacity for each stage is based on the embankment layouts, crest elevations, and freeboard requirements. The actual filling schedule will be updated as required, based on the actual tailings dry density achieved and the amount of tailings deposited. The current design includes for the storage of all runoff under average precipitation conditions, temporary storage of the IDF, and a wave run-up allowance.

Figure 18.3 TMF Filling Schedule



Embankment Cross-section

Ferrocrete/duracrust and colluviual deposits are typically present near surface within the TMF basin foundation. The embankment foundations and TMF basin will be cleared, grubbed, and stripped of surface soils and unsuitable materials to expose stiff to hard ferrocrete/duracrust or compact to dense colluvial.

Two embankments will be constructed to establish the TMF, including the Northwest Embankment that straddles the natural valley, and the East Embankment that runs along the northeast and southeast side of the basin adjacent to the Malikoundi West Waste Dump.

The embankments will be constructed of zoned earthfill and rockfill with a composite liner installed on the upstream slope. Transition/filter zones will be established between the liner and the embankment rockfill to ensure internal stability.

The downstream slopes will be 2.5H:1V. The upstream slopes will be 2.5H:1V with a 3 m wide mid-slope bench at each stage to facilitate the installation and tie-in of the geomembrane liner. The composite lining system will be installed on the upstream face of the embankment and within the basin to minimize seepage. The bedding material will be prepared on the slope and within the basin to support the liner.

The crest width will be 10 m and the maximum embankment height above original ground varies from approximately 21.5 m along the plateau for the East Embankment that runs along the northeast and southwest side of the basin to 36.5 m in the valley along the Northwest Embankment.

The fill for the embankments will come from the open pits. Basin shaping will primary include development of the bench around the basin to facilitate installation of the geosynthetic lining system over the basin footprint and general dozer shaping within the basin to achieve the grades and surface required for the installation of the composite liner. Following removal of the surface soils from within the basin and the dozer shaping, the subgrade will be proof rolled with a smooth drum compactor followed by fill placement for the low permeability liner bedding layer for the composite lining system installation.

Embankment Fill Zones

The materials to be used for construction of the various components of the embankments are described below.

- **Liner Bedding - Soil (i.e. saprolite) for Embankment Fill (300 mm thick layer) -** The liner bedding material shall consist of soil (i.e. saprolite) with a maximum particle size of 1 mm and placed and compacted in a 300 mm thick layer on the upstream face of the embankment to form a composite liner with the HDPE geomembrane.

- **Transition - Soil (i.e. laterite) for Embankment Fill (300 mm lifts) -** The Zone A material shall consist of soil (i.e. laterite) with a maximum particle size of 30 mm and placed and compacted in 300 mm thick lifts in the upstream zone. The width of this zone will be 3 m.

- **Zone A - Soil (i.e. ferrocrete) for Embankment Fill (300 mm lifts) -** The Zone A material shall consist of soil (i.e. ferricrete) with a maximum particle size of 75 mm and placed and compacted in 300 mm thick lifts in the upstream zone. The width of this zone will be 10 m.

- **Zone B - Rockfill for Embankment Fill (300 mm lifts) -** The Zone B material shall consist of hard durable processed rockfill with a maximum particle size of 150 mm and placed and compacted in 300 mm thick lifts in the upstream zone. The width of this zone will be 10 m. It is anticipated that production of this material will require selective excavation, closer blast hole spacing or processing (screening) to produce the specified gradation envelop.

- **Zone C - Rockfill for Embankment Fill (600 mm lifts) -** The Zone C material shall consist of hard durable rockfill with a maximum particle size of 400 mm and placed in 600 mm thick lifts in the upstream zone. It is expected that the mine haul fleet traffic will provide the majority of the compaction for the Zone C material. The actual compactive effort and number of equipment passes will be confirmed with a test pad constructed prior fill placement. It is anticipated that production of this material may require selective excavation and/or closer blast hole spacing to produce the specified gradation envelop.

- **Zone D - Rockfill for Embankment Fill (2,000 mm lifts) -** The Zone D material shall consist of hard durable fresh or non-weathered rockfill with a maximum particle size of 1,200 mm and placed in 2,000 mm lifts in the downstream zone of the embankments. It is expected that the mine haul fleet traffic will provide the majority of compaction for the Zone D material. The material is not expected to require processing, except for the removal of oversized particles.

Geosynthetic Lining System

The entire impoundment will be lined to contain the tailings solids and process water, and to reduce seepage from the facility. A composite liner consisting of a 60 mil HDPE geomembrane overlying a low permeability soil (saprolite) layer will be placed on the upstream slope of the embankments and over the entire basin footprint. The lining system will be installed over the prepared subgrade surface.

The geosynthetic lining system will be installed in six stages. The lining system will be installed on the basin floor, basin side slopes, and upstream face of the Stage 1 embankment as part the Stage 1 construction. The lining system will be installed on the upper basin side slopes and upstream face of the remaining stages as part of the staged construction of the embankments.

Seepage Collection Drains

A Seepage Collection Drain will be installed in the foundation of the Northwest TMF embankment to collect potential seepage from the TMF. The seepage collection drains will consist of drainage sand and gravel and 100 mm dia. corrugated perforated CPT pipe in an excavated trench. A non-woven geotextile separation layer will be installed between the drain sand and gravel and the foundation soils. The collected seepage will drain by gravity to a Seepage Collection Sump located downstream of the TMF basin where it can be monitored and pumped back to the TMF if required.

Instrumentation

Instrumentation will be installed in the embankment, embankment foundation, and Embankment Seepage Collection Drain to confirm that the TMF is performing as designed. The instrumentation will include:

- Three vibrating wire piezometers (VWPs) installed in the Embankment Seepage Collection Drain and 11 VWPs installed in the embankment fill will be used to monitor for a potential phreatic surface within the embankment.

- One slope inclinometer installed during construction of Stage 1 at the toe of the Stage 6 Northwest Embankment, surface movement monuments installed every 250 m along the embankment crest following the completion of each raise of the Northwest Embankment (91 in total), and 3 slope inclinometers installed in the Stage 6 embankment to monitor for potential movement in the foundation and embankment fill.

- Additional groundwater monitoring wells may need to be installed downstream of the TMF.

The instrumentation will provide an early warning if the phreatic surface in the embankment or potential movement exceeds allowable levels. Trigger limits for the instrumentation will be defined in later stages of design as part of the Operations, Maintenance, and Surveillance (OMS) Manual for the TMF.

Stability

The TMF is required to be stable under the design loading conditions. The required Factors of Safety (FoS) against slope instability as per CDA guidelines (2014) are:

- Static stability:
    - 1.3 immediately following construction (undrained or total stress conditions) and prior to filling
    - 1.5 during operations and at closure (drained or effective stress conditions)
- Pseudo-Static stability: 1.0
- Post- earthquake (residual strengths) stability: 1.2

Stability analyses for static loading during normal operating conditions were completed using SLOPE/W©, a two-dimensional Limit Equilibrium stability analysis software package. The stability models incorporated the proposed embankment configuration, estimated strengths of the fill and foundation materials, the projected tailings level, and the projected water levels. A review of the available site investigation data and observations made during the site investigations indicate that the embankment fill and foundation materials are not expected to liquefy. A 20% strength reduction was applied to the minimum undrained shear strength and the undrained shear strength ratio for the saprolite foundation unit for the post-earthquake loading analyses (Makdisi and Seed, 1978) to account for the potential for cyclic softening following the design earthquake.

Two representative cross sections were selected for analysis based on the embankment design, geometry, height, and foundation conditions. The analyses considered the in-situ foundation conditions and the final tailings elevation for Stage 2, Stage 3, and Stage 6 (Ultimate) embankments. The influence of the abutting Malikoundi West Waste Dump was included in the analysis for Section 2 (East Embankment). It was assumed that Malikoundi West Waste Dump would be constructed to El. 193 m (i.e. Stage 3) prior to construction of the TMF and that the waste dump would be extended to adjoin the TMF following completion of Stages 4, 5, and 6. The target FoS are met or exceeded for all sections and loading conditions evaluated. The TMF embankments may be constructed to Stage 3 (El. 176 m) prior to filling. The adjacent Malikoundi West Waste Dump does not negatively impact the stability of the TMF.

Seepage

A composite liner consisting of a 60 mil HDPE geomembrane overlying a 300 mm thick fine grained soil layer (i.e. saprolite) will be placed on the upstream slope of the embankments and over the entire basin footprint to reduce seepage from the facility.

The potential leakage through the lining system was estimated with the supernatant pond at the Stage 6 maximum filling level. The seepage analyses considered leakage due to the presence of geomembrane defects. Leakage due to permeation was not considered, as this leakage rate is estimated to be several orders of magnitude less than the leakage due to geomembrane defects. The total estimated leakage for the Stage 6 TMF was calculated for the following:

- Leakage through the basin floor.
- Leakage through the basin side slopes and embankment for Stage 6.

The total seepage from the TMF is estimated to be approximately 6.5 m^3/hr.

Tailings Management

Tailings will be conveyed to the TMF as a conventional slurry and deposited from the upstream face of the TMF embankment. This deposition strategy will develop a low permeability tailings deposit adjacent to the embankment and maintain the supernatant pond away from the embankments and toward the central, southwest portion of the basin. Tailings will be deposited from additional locations around the perimeter of the TMF basin to optimize the basin filling and manage the location of the supernatant pond.

The deposition plan will include for rotational discharge of tailings from several discharge locations to develop an exposed tailings beach. Following development of the tailing beach, the tailings discharge will be rotated to adjacent discharge locations to continue to develop the lateral extent of the tailings beach. The development of a tailings beach will allow for sub-aerial (above water surface) tailings deposition to achieve the following objectives:

- Optimize the basin filling by depositing tailings in relatively thin layers around the perimeter of the facility above the supernatant pond surface.
- Maintain the supernatant pond location well away from the embankments, while maintaining adequate depth adjacent to the water reclaim barge.
- Maximize the settled density and strength of the tailings by promoting drainage of process water and air drying of the tailings.

The conventional tailings slurry will be approximately 48% solids by weight. The tailings slurry will be conveyed to the TMF from the plant site via a HDPE pipeline. The pipeline will extend to the farthest discharge point along the embankment crest during Stage 1 operations with discharge spigots at approximately 25 m spacing along the embankment crest. The pipelines will be extended around the basin perimeter as required during operations. During subsequent staged construction, the pipelines will need to be raised to the staged embankment crest and perimeter bench around the TMF basin.

18.10.5 TMF Water Management

Objectives

The primary water management objectives for the TMF are as follows:

- Maximize the recycle of process water and runoff water from the TMF to the plant site.
- Divert run-off water reporting to the TMF from the upstream catchment areas.
- Provide temporary containment of the IDF within the TMF basin during operations.

The process limitations, available precipitation, and extreme storm event data were used to estimate the water reclaim, water removal and diversion requirements. The Water Reclaim System, Diversion System, Water Balance, and Stormwater Management for the TMF are summarized below.

Water Reclaim System

The Water Reclaim System at the TMF will reclaim water from the TMF to the plant site for use in the process. The system will include a barge, pump, and HDPE pipeline. The system will convey reclaim water to the plant site. The supernatant pond in the TMF basin will be managed to provide adequate draft for the barge.

Diversion System

Diversion ditches will be constructed upstream of the TMF basin during each stage of construction to reduce the amount of runoff entering the basin. The ditches were designed to divert runoff from storms up to and including the 1 in 200 year, 24-hour storm event. The ditches will be excavated into natural ground and will be trapezoidal in shape. The ditches in the southwest will be approximately 1.2 m deep with a 6 m wide base and 2H:1V side slopes. The ditches in the northeast will be approximately 0.65 m deep with a 1 m wide base and 2H:1V side slopes.

Water Balance

The water balance model for the TMF was developed to estimate the monthly pond volume in the TMF, quantify any reclaim water shortfalls that would require additional make-up water from the Fresh Water Pond (FWP), and determine if there is any period of excess water in the TMF that would potentially require treatment and release.

The monthly operational water balance estimates were calculated using GoldSim software. The findings of the water balance are summarized as follows:

- The estimated TMF pond volume will range from 39,000 m^3 to 1.96 Mm^3 under average precipitation conditions.

- The results from the TMF water balance indicate that the TMF will generally operate with a monthly net surplus of water based on average precipitation conditions. It is estimated that the TMF will be able to provide sufficient reclaim water during the wet season. In the dry season, a portion of the required reclaim water will need to be taken from the FWP.

Storm Water Management

The TMF has been sized to provide temporary storage of the IDF during operations. In addition, during operations, a spillway will be installed during each stage to convey the IDF from the TMF.

The inflow to the TMF basin resulting from the IDF was estimated using HydroCAD© software. The estimated inflow was used to determine the required storage volume to temporarily contain the IDF and to size the spillways during operations. The analysis assumed that the diversion ditches would fail during the IDF and that all runoff within the upstream catchment area of the TMF would report to the TMF basin.

The estimated IDF volume ranges from 928,400 m^3 for Stage 1 to 1,189,000 m^3 for Stage 6. Approximately 0.8 m to 1.0 m of wet freeboard is required during all stages of operation to temporarily store the IDF. A dry freeboard of 0.5 m is also required to provide overtopping protection for wave run-up. Although the TMF is planned to contain sufficient freeboard for the operating pond and containment of the IDF, for added security, emergency overflow spillways have been incorporated into the design for each stage to safely pass the IDF (in addition to the IDF temporarily stored in the TMF). The emergency overflow spillways will be located at the northwest abutments of the embankment. The emergency overflow spillways meet the recommendations by CDA (CDA, 2014). The spillway invert ranges from 0.75 m to 1 m below the embankment crest and is designed for conveyance of the peak inflow resulting from the IDF storm event. The total freeboard above the maximum operating levels range from 1.8 m to 2.8 m for the 6 stages.

18.10.6 Monitoring and Surveillance

General

The facilities will be operated in compliance with applicable international and national guidelines and standards. An OMS Manual and Emergency Preparedness Plan (EPP) for the TMF will be developed prior to operations. These documents will be used for operator training and support for the management of the TMF.

A TMF manager shall be designated for the TMF. The TMF manager will have overall responsibility for the TMF, including the review of operational, monitoring, and surveillance data. The general monitoring and inspection protocols for the operation of the TMF are summarized in the following sections.

Monitoring

Monitoring of the TMF will be carried out at specified regular intervals to evaluate the performance of the TMF and to refine the operating practices. Key monitoring requirements will include:

- Daily recording of the supernatant pond level.

- Daily recording of the tailings discharge location.
- Monitoring of pump and pipeline performance for pressure fluctuations and potential leaks.
- Equipping all water pumps and pipelines with devices to measure flows and volumes. Measurements will be used to calibrate the water balance and to adjust the water management strategy, as required.
- Collection of site-specific meteorological data. The data will be used to calibrate the water balance and to adjust the water management strategy, as required.
- Daily monitoring of piezometers during embankment construction.
- Weekly monitoring of piezometers during operations.
- Monthly monitoring of the embankment crest, surface movement monument surveys, and review of slope inclinometer data to confirm that embankment displacement has not occurred.
- Quarterly surveys of the deposited tailings surface and supernatant pond extents and depth to estimate the tailings in situ settled dry density.

The monitoring data will be reviewed by the TMF manager and submitted to the Design Engineer for review.

Inspections

Regular inspections of the TMF will be completed as part of the TMF operations to confirm that the TMF is being operated in accordance with the design intent. The inspections will include:

- Visual inspection of the embankment to check for evidence of displacement and/or instability.
- Visual inspection of the tailings beaches to identify situations that may require adjustments to tailings deposition practices.
- Visual Inspection of the supernatant pond location and water level, and water levels in Sediment Basin #1, and the Seepage Collection Drain Sump.
- Visual inspection of the pipelines and pumps to identify any damage, leaks, and other operational issues will need to be addressed.

The following inspections will be completed in addition to the regular inspections by the TMF operators:

- Detailed monthly inspections of all the TMF components by the TMF Manager.
- Detailed inspections by the TMF Manager following any extreme precipitation or seismic event.
- An annual inspection by the Design Engineer will be completed to verify that all components are performing as designed and that the facilities are being operated as intended.

- A formal Dam Safety Review should be completed every 5 years as recommended by CDA (2014) for a TMF with a DHC of EXTREME.

Documentation

Studies, maps, reports, Record Documentation, and any other technical and scientific evidences used as criteria for the construction and operation of the TMF shall be kept on site and available for review by authorities on short notice.

18.10.7 Reclamation and Closure

Reclamation and closure will be based on the following general goals and objectives:

- Reclamation goals and objectives will be considered during the design of the TMF.
- Reclamation goals and objectives will be periodically updated during construction and operations.
- Progressive reclamation will be implemented wherever possible.
- Upon cessation of operations, the TMF will be decommissioned and reclaimed to allow for future land use as guided by local regulators.
- Reclamation and closure construction will be designed to meet long-term physical and chemical stability objectives.

Generally, the closure work will consist of the following:

- Removal of all ponded water.
- Decommissioning and dismantling of all tailings delivery and distribution pipework.
- Decommissioning and dismantling of all water reclaim and the pump barge.
- Decommissioning and dismantling of all seepage recycle pipework and pumps, assuming that the seepage water meets water quality objectives.
- Placement of a soil cover and vegetation on the tailings to minimize water infiltration and to improve site aesthetics.
- Construction of final water management measures (spillways, ditches, berms, etc.) to convey run-off from the IDF to the environment.

18.11 Fresh Water Pond

18.11.1 Introduction

KP completed a FS level design for the FWP for the Project. The FWP will be constructed to store water to support mine operations. A storage volume of 2.5 Mm^3 has been selected by IAMGOLD to ensure adequate capacity to provide water to the process under all operating and climatic scenarios. The FWP will be constructed within the valley feature located directly west of the Malikoundi North Open Pit and northeast of the Malikoundi West Waste Dump.

18.11.2 Design Basis Overview

The design basis and operating criteria are based upon accepted national and international standards for mining dam design and operation, available site information, and the operational requirements developed in consultation with Lycopodium and IAMGOLD. The FWP has been designed to meet the Canadian Dam Association Technical Guidelines for Mining Dams (CDA, 2014), and includes freeboard and design earthquake ground motion (DEGM) considerations to minimize operational risks. The FWP has been identified as having a Dam Hazard Classification of Extreme based on the foreseeable consequences.

18.11.3 Embankment Section and Basin Lining System

General

The typical embankment section has been developed based on the results of the stability assessment (KP, 2018b) and the construction staging and methodology proposed by AGP. The FWP has been designed as a lined valley impoundment confined by an earth embankment. The embankment has been designed using available materials from mining activities and local borrow sources as required. The embankment will be buttressed to the downstream by waste overburden.

The embankment has been designed to crest El. 157.0 m to provide 2.5 Mm^3 of fresh water storage in addition to freeboard contingencies for storm water runoff management (under normal operating conditions), excess water discharge, wave run-up, and conveyance of the IDF through the Overflow Spillway.

Embankment Section

The overall downstream slope of the FWP embankment will be 3H:1V with a 20 m wide buttress constructed to El. 135 m along the downstream toe. The overall upstream slope will be 3.5H:1V with a 3 m internal bench at El. 147 m to accommodate for the geomembrane liner installation and staged embankment construction. A 0.3 m thick liner bedding layer will be placed along the upstream slope of the embankment to reduce the risk of damage to the HDPE liner and to reduce the potential for seepage and excess pore pressures to develop within the embankment fill during a rapid drawdown scenario. A seepage drain will be located along the toe of the liner bedding material to convey any potential seepage from the basin below the embankment.

Basin Lining System

The FWP basin and upstream embankment face will be lined with a geosynthetic lining system to reduce seepage from the facility. The lining system will include a 60 mil HDPE geomembrane installed over a 12 oz./yd^2 non-woven geotextile and a sand and gravel liner bedding cushion layer on the upstream face of the embankment. A basin underdrain will be installed below the liner system along the floor of the FWP and the toe of the embankment at the base of the liner bedding layer to collect seepage.

18.11.4 Water Management

During operations, fresh water will be obtained from direct precipitation on the pond (when present) and runoff from the surrounding catchment area, Malikoundi West Waste Dump, Malikoundi and Malikoundi North Pits, and groundwater inflow into the Malikoundi and Malikoundi North Pits. In addition, the facility will serve as a settling basin for the surface water runoff originating from the contributing catchment areas, and reduce the total suspended solids in the runoff prior to discharge to the environment (KP, 2018c).

The FWP will include an Overflow Spillway to route excess water and flows through the FWP basin. The spillway will consist of a trapezoidal channel. The spillway will discharge downstream of the FWP and the toes of the embankment and buttress will be lined as required with geotextile and rip rap to minimize erosion. The Overflow Spillway has been designed as a two-stage channel, with the lower portion (Stage 1) designed to discharge excess water from normal operating conditions up to the EDS, and the upper portion (Stage 2) designed to convey the peak flow resulting from the IDF.

Details regarding the site wide surface water management plan, and estimation of normal operating surface water flows to the FWP are summarized in the Site Wide Surface Water Management Design (KP, 2018c), and Site Wide Water Balance (KP, 2018d), respectively.

18.11.5 Monitoring and Instrumentation

The FWP will require the installation of geotechnical instrumentation to support the monitoring of the FWP during the initial construction of the embankment, and long term operations of the facility. The instrumentation is required to ensure the facility is meeting or exceeding the design intent, and to detect for potential changes in the facility’s performance. The instrumentation will be installed during the initial construction of the facility and will include vibrating wire piezometers, an inclinometer and survey monuments. Regular data collection from the instrumentation will be completed to ensure the facility is meeting the design intent.

18.12 Site Wide Water Management Plan

18.12.1 Introduction

KP completed a FS level site wide water management plan for the Project. The site water management plan and feasibility design of the associated water management measures have been developed based on the FS level site arrangement, operational requirements and environmental site conditions. The proposed plan and measures will allow for the management of runoff from disturbed areas while minimizing the runoff from undisturbed areas reporting to the mine site.

Following a precipitation event, the runoff will be managed to reduce the total suspended solids prior to discharge to the environment. This is a requirement of the Project's operating conditions.

The primary objectives of the site water management plan are as follows:

- To collect runoff originating within disturbed areas, through a series of ditches and berms, and convey the runoff to one of a number of sediment basins, where the majority of the total suspended solids can settle out prior to sending the water to the FWP (for potential use in the mining process) or releasing it to the environment.
- To minimize the volume of runoff entering the mine site from undisturbed areas by diverting upstream runoff around mine infrastructure through a series of diversion berms and ditches.

18.12.2 Catchment Areas

The development of the site water management plan was based on dividing the project area into two main types of catchment areas (Undisturbed and Disturbed Catchment Areas). The two main types of catchment areas are described as follows:

- **Undisturbed Catchment Areas** - Areas within (or adjacent to) the main project footprint that will not be disturbed by the project operations. In general, the undisturbed catchment areas consist of desert shrub terrain that is flat to gently sloping. Where feasible, surface water runoff from the undisturbed catchment areas will be diverted around mine infrastructure areas, reporting directly to the environment.
- **Disturbed Catchment Areas** - Areas within the project footprint that are disturbed by project activities and whose runoff may require treatment prior to discharge (i.e. settling of suspended solids). Surface runoff from the disturbed catchment areas will be conveyed by ditches and temporarily stored in sediment basins. Disturbed areas include the pits, waste dumps, and TMF and FWP embankments, and the Plant Site.

18.12.3 Methodology

The runoff volumes and peak runoff flows resulting from the design storm events were estimated using HydroCAD® (HydroCAD®, 2015). The runoff volumes and peak flows were then used to determine the storage capacity of the sediment basins and typical details of the water management measures (collection/diversion ditches, diversion berms and sediment basin overflow channels).

18.12.4 Water Management Measures

General

Water management measures for the Project will include a series of diversion berms, collection and diversion ditches, sediment basins, and water transfer pipelines. In general, runoff from the various catchment areas will be directed towards the water management structures. If required, site specific regrading may be conducted in order to accomplish this.

Diversion Berms

Diversion berms will be constructed to facilitate and direct runoff away from the pits, and to help direct site runoff to collection ditches or sediment basins. The berms will have a minimum height of 0.5 m, a crest width of approximately 0.5 m and side slopes of 1.5H:1V. Berms will be constructed from durable processed waste rock excavated from the pits and nominally compacted. Berms will be used where necessary to suit site specific conditions.

Diversion and Collection Ditches

In order to convey surface runoff around and away from the mine site infrastructure, or to the sediment basins (including the FWP), diversion/collection ditches will be constructed. The diversion/collection ditches have been designed to safely convey the 1 in 200 year, 24-hour storm event.

Each diversion/collection ditch will be trapezoidal in section with 2H:1V side slopes. Ditches will be excavated into the existing overburden or formed by grading surface material to achieve the required ditch geometry. Excavated or graded material will be used to form a berm alongside the ditch in order to prevent runoff from adjacent areas from flowing into the ditch. It is suspected that the majority of the ditches will be constructed in overburden and will require erosion protection along their base and side slopes. Erosion protection will likely consist of riprap overlying non-woven geotextile (if required).

Sediment Basins

Runoff from disturbed areas will be conveyed to one of the sediment basins, to the FWP or to the Event Pond at the Plant Site prior to discharge or use as process water at the Plant Site. The main components of the sediment basins include the basin geometry (retention capacity for runoff), discharge structure(s) (to facilitate drainage) and overflow channel (to safely release extreme event runoff).

- **Basin Geometry** - The sediment basins have been sized for temporary storage of runoff resulting from the 1 in 10 year, 24-hour storm event. Temporary storage of this runoff will allow the majority of the suspended solids to settle out prior to the water being discharged to the environment or pumped to the FWP for eventual use in the milling process. Sediment basins have been sized based on site specific conditions and inputs from industry standard recommendations that include:

- Sediment Storage: 0.5 m above the basin floor for storage of accumulated sediment
- Operating Pond Volume: Varies but equals volume of runoff reporting to basin from the 1 in 10 year, 24-hour storm event
- Wet Freeboard: Varies but equals peak flow depth into the overflow channel, resulting from the 1 in 200 year, 24-hour storm event
- Dry Freeboard: 0.5 m above the peak spillway flow depth

- **Decant Structures** - Each sediment basin will be equipped with a floating decant system to discharge water from the basin to the environment during normal operating conditions. It is estimated that the water stored in these basins will be released over a period of approximately one to five days following the design storm event (1 in 10 year, 24-hour event).

- **Overflow Channel** - Each sediment basin will be equipped with an overflow channel to convey flows resulting from storm events greater than the 1 in 10 year, 24-hour event, and up to and including the 1 in 200 year, 24-hour event, to the downstream environment. The peak flows were calculated to determine the required channel width, depth and erosion protection requirements. The overflow channels will be trapezoidal in section and lined with riprap overlying non-woven geotextile.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No formal market studies have been undertaken.

A gold price of $1,200/oz has been used for the Mineral Reserve estimate and $1,250/oz has been used for the economic analysis.

The Project will produce gold doré which is readily marketable on an 'ex-works' or 'delivered' basis to a number of international refineries. There are no indications of the presence of penalty elements that may impact the price or render the product unsalable.

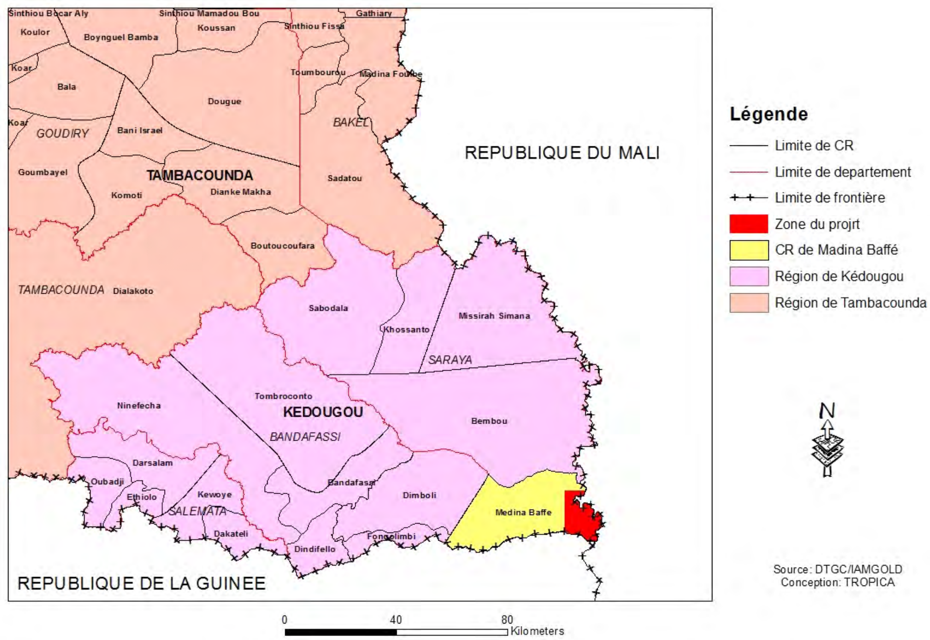
19.2 Contracts

There are no material contracts or agreements in place as of the effective date of this report. Refining contracts are typically put in place with well recognized international refineries and sales are made based on spot gold prices. These contracts typically include fees for transportation of the product from the site, insurance, assaying, refining and an allowance for metal losses during refining. The ability to get a contract in place for the sale of doré prior to start of production is not seen as a risk to the Project. IAMGOLD has contracts in place for sale of gold from its producing mines and it has been assumed that Boto doré will attract similar terms.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The Project is located within the Boto concession block, in the south-east of Senegal. The Boto sector of the Daorala-Boto exploration permit covers 148 km² and is bounded to the east by the Malian border, and to the east by the Guinean border. The Project is located near the Guémédji village in the Madina Baffé commune, Saraya Department, Kédougou region (Figure 20.1).

Figure 20.1 Kédougou Region Administrative Map



20.1 Legal Requirements Relevant to Environmental and Social Aspects

Many legal and regulatory requirements relevant to environmental and social aspects apply to mining projects. The key applicable legal texts in Senegal for developing a mining project are:

- Act No. 2003-36 of November 24, 2003, on the Mining Code.
- Act No. 2001-01 of January 15, 2001, on the Environment Code.
- Act No. 1998-03 of January 8, 1998, on the Forestry Code.
- Decree No. 98-164 of February 20, 1998, on the application of the Forestry Code.
- Decree No. 2001-282 of April 12, 2001, on the application of the Environment Code.

- Decree No. 2004-647 of May 17, 2004, establishing the terms of application of Act No. 2003-36 of November 24, 2003, on the Mining Code.
- Act No. 2009-24 of July 8, 2009, on the Sanitation Code.
- Act 81-13 of March 4, 1981, on the Water Code.

The main environmental and social requirements in accordance with the Mining Code are:

- Completing an ESIA in compliance with the Environment Code and its regulations (Section 83-CM).
- Creating a mine site reclamation fund at the *Caisse de dépôt et de consignation* (Deposit and Consignment Office) (Section 84-CM).

As for the main environmental and social requirements in accordance with the Environment Code, they are:

- The necessity of completing an impact study and implementing an Environmental and Social Management Plan (ESMP) (Section L. 44 et seq. of the Code and its regulations).
- The acquisition of an environmental compliance certification granted by the Directorate of Environment and Classified Establishments (DEEC) of Senegal after: i) completion of the ESIA by a certified studies office (by the DEEC); ii) validation of the ESIA report by the technical committee; and iii) public hearing. The ESIA's Terms of Reference must be approved in advance by the DEEC.
- As part of the environmental assessment, there is a requirement to notify the authorities of neighbouring countries of a mining operation i) if operations are liable to have a cross-border impact (section L44 of the Environment Code), or ii) if the mining operation must use shared infrastructures or resources (e.g. drawing water from a river on the border).
- Compliance with safe distance rules: a mining operation is a classified establishment, i.e. it can include facilities deemed classified for the protection of the environment (grinder, crusher, hydrocarbon or chemical depot, etc.). Section L 13 stipulates that a first class classified facility (applicable to the mining project) must be located at least 500 m from a watercourse, habitations, thoroughfares, and water catchment areas. After verification with the Directorate of Classified Facilities, it was determined that pits are also considered classified facilities, but their distance to watercourses and habitations can be discussed.
- A requirement to consult local communities as part of the ESIA procedure for a mining project, because the Environmental and Social Management Plan (ESMP), which is part of the ESIA, must take into account their concerns (Section L 52-CE). Public consultation is done upstream and downstream (Section 2 – Order No. 9468 MJEHP-DEEC of November 28, 2001, on the regulation of public involvement in the environmental impact study).

Upstream procedure (Section 1, Order No. 9468 MJEHP-DEEC of November 28, 2001, on the regulation of public involvement in the environmental impact study):

- Announcement of the initiative via posting at the town hall and/or regional office.
- Press release (written or spoken).
- Filing of documents at the town hall or local community involved.
- Organization of an information meeting.
- Collection of written and spoken comments.
- Negotiations, if needed.
- Development of the report.

Downstream procedure (Section 6 et seq., Order No. 9468 MJEHP-DEEC of November 28, 2001, on the regulation of public involvement in the environmental impact study):

- Preparation of public hearing with DEEC and the local community.
- Information in the media and any means appropriate to inform the public.
- Public hearing in the local community that will host the project.
- Collection of comments from the populations.

For a first class classified facility (applicable to the mining project), the region's Governor may ask for a 15-day public inquiry (Section R6 et seq. of Decree No. 2001-282 of April 12, 2001, on the application of the Environment Code).

In addition to the legislative requirements above, IAMGOLD is committed to complying with a number of existing texts, such as:

- IAMGOLD's Health and Safety Policy.
- IAMGOLD's Sustainability Policy.
- The International Finance Corporation's Performance Standards.
- The Voluntary Principles on Security and Human Rights.
- The World Bank Group Environmental, Health and Safety Guidelines.

- The World Bank Group Environmental, Health and Safety Guidelines for mining.
- The World Health Organization Guidelines on the quality of drinking water.

20.2 Environmental and Social Impact Study

A summary analysis of the initial environmental and social status of the exploration permit was carried out in 2014 by TROPICA Environmental Consultants and was completed during the preliminary study by field investigations.

In order to comply with these legal and regulatory requirements as well as the World Bank Group Guidelines, an environmental and social impact study process was initiated in June 2015 and to be completed in 2018 for the Project. The mandate to complete the impact study was awarded to Norda Stelo, who collaborated with the Senegalese study firm Synergie Environnement and the Canadian firm BBA to carry out the study.

To properly understand the Project's human, physical and biological context, baseline environmental studies on social sanitation conditions, public health, fauna, flora and biodiversity, surface water and ground water quality, the water regime, and the cultural heritage were completed in 2015, in the first half of 2016, and in the second half of 2017. Tailings and waste geochemical characterization studies were also conducted during these periods.

The upstream public consultation process took place in 2016, and a public inquiry was made in May and June 2016, at the request of the Kédougou region Governor.

The complete ESIA report, including the ESMP and the closure and reclamation framework, were submitted to the authorities in 2016, on the basis of the Project as developed as part of the original prefeasibility study. At the request of IAMGOLD, the impact study validation procedure was suspended due to the continuation of technical studies.

Following the publication of the optimized prefeasibility study and the launch of the FS, the ESIA report was updated with new data at the end of the first half of 2018 and submitted to the Ministry of Environment for instruction and validation. The report was reviewed in April 2018 by the technical committee, representing all key and administrative stakeholders, and additional information was requested. The amended report taking into consideration this feedback was submitted to the Ministry of Environment in May 2018.

The highlights of the baseline environmental studies and the impact study are presented in the following sections. The complete ESIA report contains more detailed information. It must be noted that the information presented in the following sections are from the ESIA filed in 2018.

20.2.1 Physical Environment

Landscape

In the Boto permit zone, the landscape's altitude varies between less than 100 m and more than 250 m, compared with the flat, low elevation geography of the rest of the country, where the altitude is rarely over

50 m in the uplands, plains and alluvial valleys of the sedimentary basin. There are four main landscape entities: hills, cuirass plateaus, plains, and riverbanks.

Each of these entities is characterized by a specific vegetation cover type, use, or human activities. Hills are covered with shrubs with a rather limited human presence, while riverbanks, which are densely wooded, remain bio-corridors and major biodiversity areas with significant human activities. Plains are the preferred areas for habitation. Cuirass plateaus often host a grass savannah type vegetation used for pasture.

Figure 20.2 Hills in the Permit Zone

Figure 20.3 Riverbank Landscape

Perennial Riverbanks (Boféto) Temporary Riverbanks

Figure 20.4 Plain and Plateau Landscape

View from a top the Hills | View from the Boféto Village

Climate

Climate data for the Boto mine site comes from the closest weather station, which is at Kédougou. For the purposes of the study, IAMGOLD acquired the most recent data from the National Agency of Civil Aviation and Meteorology (ANACIM) over the 1985-2014 period. This data comes from the Kédougou station. The climate of the Kédougou region which hosts the Project is of the Sudano-Guinean type. It is located between 7° and 12° north latitude and constitutes a transitional zone toward Guinean humid subtropical climate. The Boto zone's climate is characterized by seasonal variations (a four-month humid season and an eight-month dry season), reinforced by a continental touch with effects discernible late in the dry season. To obtain climate data specific to the Project's site, a latest generation meteorological station was installed in Spring 2016, which provides real-time data.

Temperature

The analysis of monthly temperature averages charts for the 1968-2014 period shows that temperatures are at their lowest at the end and beginning of the year, and they gradually increase during the year. There is a relative drop in temperature in July, August and September and the highest temperatures are observed in April and May, as shown in the figure below.

Figure 20.5 Heat Regime at the Kédougou Weather Station (1968- 2014)

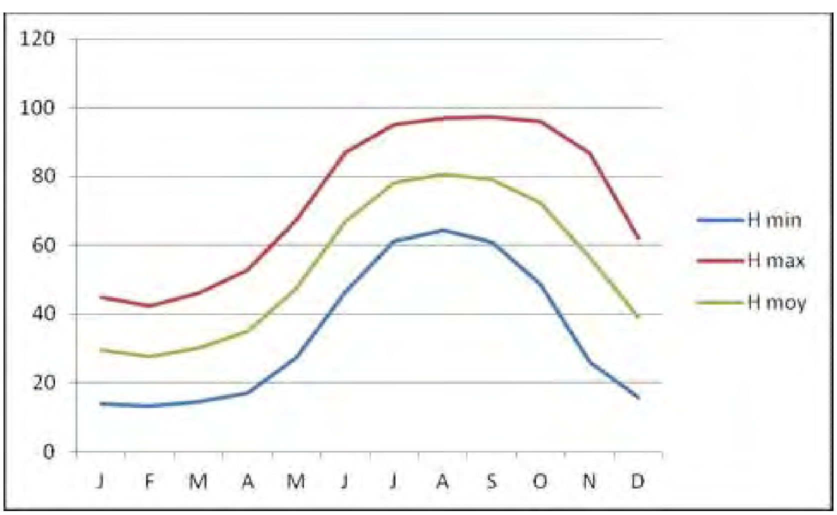
Kédougou's annual temperature average is 28.4°C, and the annual temperature range average is 14°C. Diurnal variations are between 14°C (April) and 17°C (December). Kédougou's temperature extremes during the 1978-2007 period are 41.4°C (April 2007) and 14.6°C (December 2002).

Relative Humidity

Relative humidity is the ratio of water vapour in the air to the air's absorption capacity at a given temperature. It varies based on seasons. During the rainy season (June to October), it is affected by the monsoon, while during the dry season, it is affected by continental trade winds.

In the region studied, average annual relative humidity is at its highest in August (81%) and at its lowest in February (29.1%). Maximum relative humidity is at its highest in September (97.5%) and at its lowest in February (44.9%). As for minimum relative humidity, it varies between 64% in August and 13.4% in February.

Figure 20.6 Monthly Relative Humidity (%) in the Kédougou Region (1978-2007)



Winds

Wind conditions are characterized by seasonal variations in prevailing wind directions, with easterly winds or the harmattan, which blows for practically the entire year and the maritime trade winds from the west (December, January). Wind speed rarely exceeds 3 m/s. But the wind can reach speeds of 3.3 m/s in May and 2.8 m/s in June, decreasing to 1.5 m/s in August.

The harmattan can be felt in this zone starting in February. In May-June, it meets the monsoon from the Gulf of Guinea, which brings humidity. This is the period of tornadoes and storms, followed by the rainy season until October. In November, the monsoon is replaced with westerly winds (maritime trade winds).

Insolation

Insolation is the factor that directly affects temperature. In the region studied, over the period of 1968 to 2005, average insolation is at its highest in April (9.1 h/d) and May (8.3 h/d), and drops to its lowest in August (6.2 h/d), July (6.7 h/d) and September (7.0 h/d). The monthly average rarely exceeds 10 h, especially during the dry season.

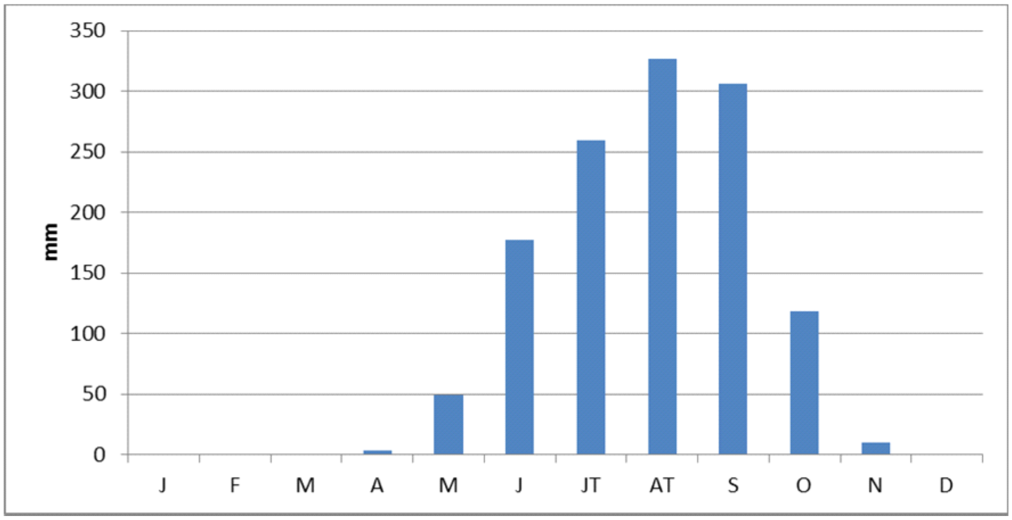
Evaporation

Evaporation depends on temperature, insolation, wind speed and relative humidity. It increases with temperature, insolation and wind speed, but decreases with relative humidity. In Kédougou, over the period of 1985 to 2014, the monthly average for evaporated water varies similarly to insolation and temperature. It is at its highest in March with 307 mm and at its lowest in September with 49 mm. The annual average during this period is 2,089 mm. There seems to be a slight increase in evaporation between the period of 1968-2005 and the period of 1985-2014. This could be partially attributable to the higher temperatures recorded in the past few years.

Rainfall

Rainfall in the region hosting the permit is distinctive of the Sudanese zone with a two-season regime: a humid season from June to September-October, and a dry non-rainy season from November to April. Rainfalls are only observed during five or six months (May to September or October), and can continue until November in the event of a late season. Annual pluviometry varies between 600 mm and 1,900 mm. The average pluviometry for the period of 1923-2006 is 1,247 mm, while the one for the period of 1985-2014 is 1,191 mm. The highest pluviometry was recorded in 2006 (1,966 mm), and the lowest, in 2007 (685 mm). There seems to have been a decrease in average annual rainfall from the period of 1923-2006 to the period of 1985-2014. This decrease is mainly observed in June and October. However, pluviometry is characterized by high internal variability, as the average annual total can be below average by over 400 mm (1990, 1992, and 2007), or above average by nearly 950 mm (1954).

Figure 20.7 Unimodal Rainfall Regime at the Kédougou Station (1930- 2014)



Surface Water Hydrology

The Boto exploration area is located in the hydrological basin of the Falémé River, which is a tributary to the Senegal River, and it is the largest watercourse in the zone. This perennial river, whose water regime is closely related to the rainfall regime, constitutes a natural border between Guinea, Mali, and Senegal.

The river drains a hydrological basin which extends from latitudes 12°11 N to 14°27 N, and longitudes 11°10 W to 12°13 W, and covers an area of 28,900 km² in Kidira, spread over Mali (47.8%), Guinea (12.5%), and Senegal (39.7%). The Senegalese part of the basin is centred in the Bakel and Saraya Departments, respectively located in regions of Tambacounda and Kédougou. The river's total length is 650 km.

Falémé's water regime combines a rising stage during the rainy period, and falling stage during the dry period. The rising stage lasts four months (May to August), including three months of high water (July, August, and September). Hydrology is at its highest in September, out of sync with the highest rainfall, which is in August. This one-month difference is due to periods of capillary retention and runoff organization. The falling stage starts in September and continues until May, which is the beginning of the rainy season on the basin.

The Falémé River is essentially fed by rain water, and its drainage is characterized by high irregularity and high interannual variability. Drainage is generally observed throughout the year. The table below shows the Falémé River's flow for three hydrometric stations located on the river, based on recurrences of dry and humid years. It must be noted that the Fadougou station is located directly downstream from the location where it is planned to draw fresh water to feed the mine site.

Table 20.1 Falémé River Flow Frequency at the Hydrometric Stations of Kidira, Gourbassi, and Fadougou (m^3/sec)

	Récurrences sèches					Médiane	Récurrences humides				
Fréquence	0,01	0,02	0,05	0,1	0,2	0,5	0,8	0,9	0,95	0,98	0,99
Récurrence (ans)	100	50	20	10	5	2	5	10	20	50	100
Station de Kidira	33,1	35,6	42,8	54,6	78,9	168,9	332,4	451,2	567,3	717,7	829,7
Station de Gourbassi	17,9	21,6	29,4	39,0	54,3	94,1	145,6	176,2	203,0	234,6	256,5
Station de Fadougou	13,5	15,7	20,6	26,9	37,2	65,7	104,6	128,4	149,6	175,0	192,7

The main tributary to the Falémé River is the Balinko River from Guinea and acting as a natural border with Mali, including the eastern part of the permit near Guémédji.

There is also another tributary in the northern part of the permit, the Koila Kobé, which is crossed by the Boféto Bridge and is a temporary river originating in Guinea.

Other temporary and less important watercourses include the Sondogna, the Kiriboung, and the Boto, which are also tributaries to the Falémé River.

Surface water resources in the project zone are mainly used to meet the local populations' domestic needs. These mainly include consumption by the populations and cattle, laundry and swimming. Surface water resources are also used by local populations for gold washing at small-scale mining sites.

Two sampling campaigns (June 2015 and February 2016) were carried out to establish the current surface water quality status. Broadly, surface waters are of poor quality due to the generalized presence of fecal and total coliforms. However, chemical metrics such as cyanide and heavy metals are generally present in low or undetectable concentrations.

Hydrogeology

The permit zone is located in the hydrogeological basin of the Falémé River and the ground water resources depend of the formations in place, of their weathering, of tectonic accidents such as faults and of food conditions, which depend on the climate. Two types of ground water formations are observed: high ground water contained in perched colluvial-alluvial water tables and in clay or sand alterites, and deep ground water from the fissure or fault zone of the crystalline or foliated crystalline bedrock.

Ground water is used for the local populations' domestic needs and provided by the few existing wells and boreholes in the area.

A series of piezometers were installed in the Boto sector at the location of the future mining facilities to establish ground water quality before conducting the project. These piezometers will be used to monitor the water table's quality based on the sensitivity of each type of ground water formation.

Two sampling campaigns were completed in 2015 and 2016 to collect data on ground water quality before conducting the project. Generally, ground waters are of poor quality, as most samples are contaminated with coliforms and streptococci, except at a few points. Arsenic was detected in 17 samples from the first campaign, but only one sample exceeded the WHO’s standard (0.03 mg/l). In the second campaign, the pH of traditional wells exceeded the standard; only the boreholes complied with the standard, except for one.

20.2.2 Biological Environment

Flora

Existing plant formations vary based on topographical units. They include savannah woodland, as well as grassland savannah, on cuirass plateaus, and dry woodland and wooded savannah on hills. On more extended slope biotopes, there are wooded savannahs and open forests, while the thalweg hosts gallery forests with arborescence around the stream system.

Two main strata are observed: the grass cover made up of the species *Andropogon pseudapricus* and *Andropogon gayanus* characterized by its vulnerability to bush fires and the wood population including Sudanese and Sudan-Sahelian species.

Recent studies in the Boto sector list 205 plant species, including 80 woody species and 125 herbaceous species. In addition to the ecosystemic diversity, there is also intraspecific diversity. The vegetation cover rate is important because the average density is 446 individuals per hectare, but it is not homogenous as it varies between 266 and 136 individuals per hectare, Plant formations are mainly represented by the shrub to wooded savannah (74.1%) and the grass savannah (13.5%). Gallery forest and open forest cover respectively 5.2% and 3.6% of the total area of the study zone.

Out of the identified species, 11 are threatened, 9 are partially protected, and 3 are fully protected, under the national legislation.

A few habitats with an ecological potential were also identified as part of the ESIA. They are:

- The silty banks of watercourses, which include threatened and rare species, such as *Borassus aethiopum, Celtis toka, Cola laurifoli, Diospyros mespiliformis, and Saba senegalensis.*
- Bamboo groves, which are the specific habitats of a threatened species of the Senegalese flora, Oxytenanthera abyssinica.
- The open forest, which includes *Holarrhena floribunda.*
- Wooded savannahs of low plains, which include *Acalypha senensis.*
- Shrub savannahs with boval, which can host *Lepidagathis capilliformis, Indigofera leptoclada and Ozoroa pulcherrima.*

Gallery forests include threatened or rare species, such as *Cassia sieberiana*, *Diospyros mespiliformis*, *Khaya senegalensis*, *Saba senegalensis*, and *Pavetta cinereifolia*. These forest resources are exploited, namely by the local communities, for various uses: food (edible fruits, culinary usage), medication, construction, source of energy, etc.

Regarding protected areas, the permit is located in part in the *Zone d'Intérêt Cynégétique* (ZIC) (hunting zone) of Falémé. It covers 177.5 ha of this ZIC, which has a total area of 1,360,000 ha. In a ZIC, the fauna has a partial protection for its development, but certain species have particular statuses (full protection, partial protection, protection of endemic species, etc.). Additionally, the permit zone is located at more than 100 km south-west of Niokolo-Koba National Park (PNNK).

Figure 20.5 and Figure 20.6 show the various plant formations and locations where special status plant species were identified in the study zone.

Figure 20.8 Plant Formations

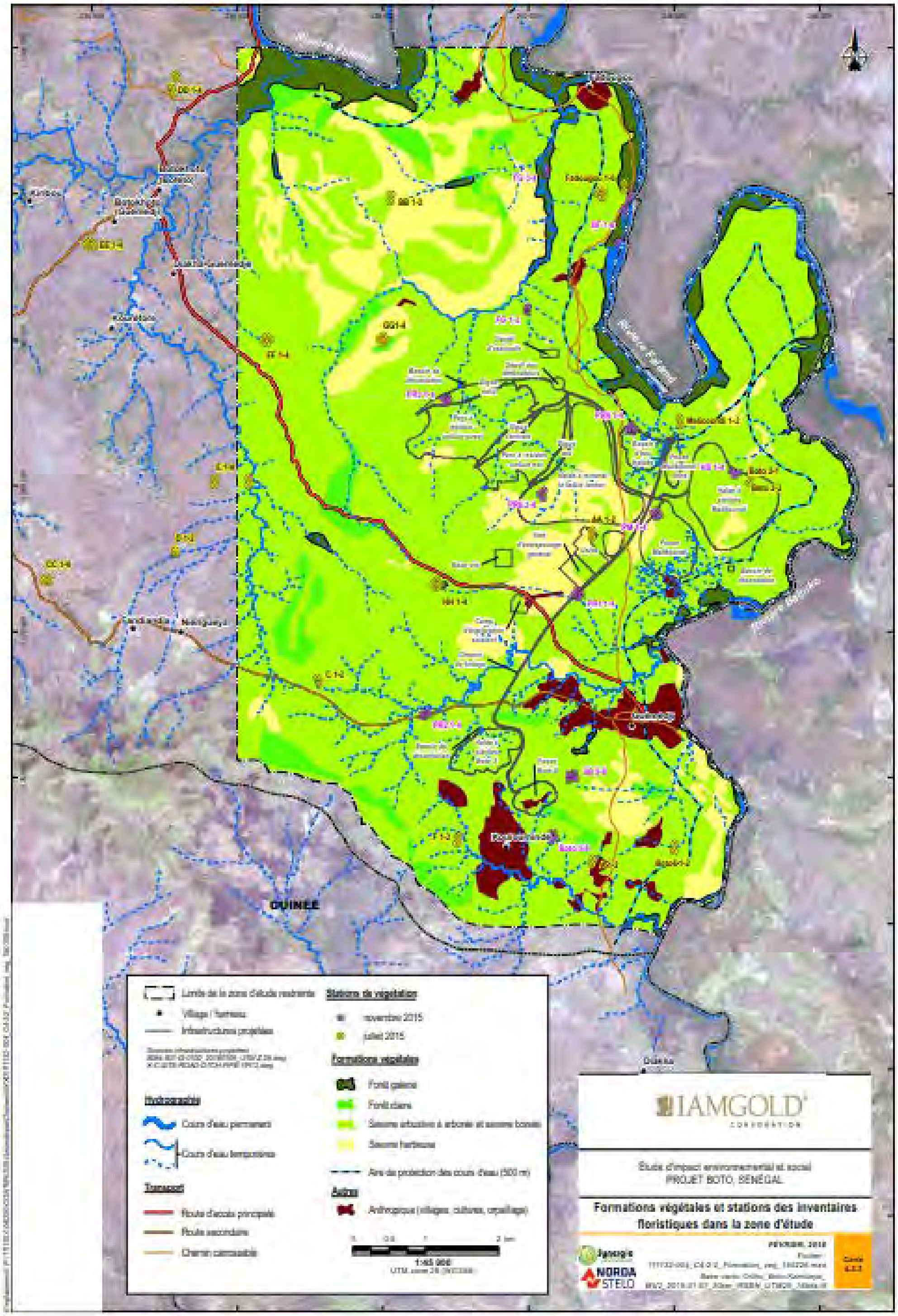
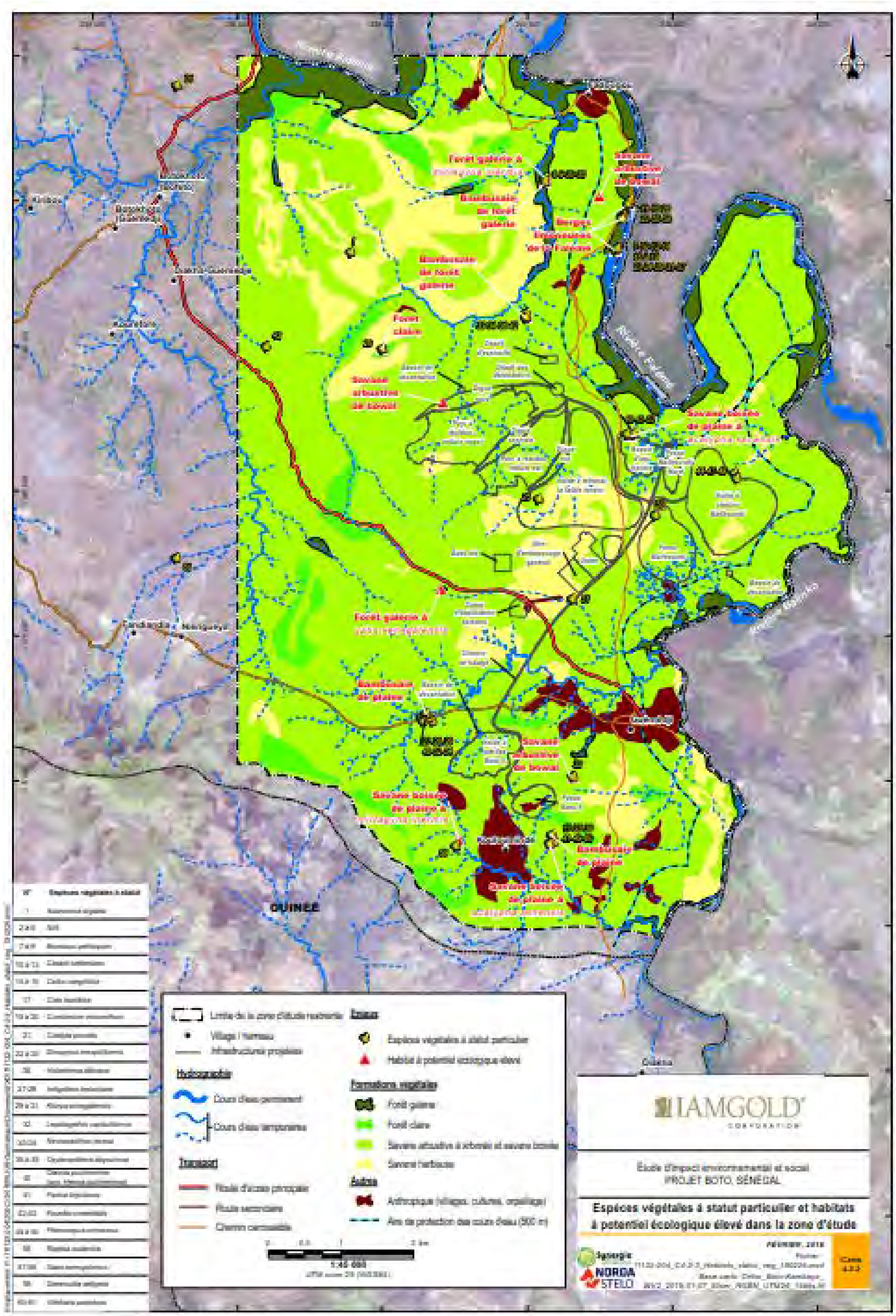


Figure 20.9 Locations where Special Status Plant Species were Identified



Fauna

The diversity of biotopes or habitats in the zone affects the diversity of the fauna. This fauna, more specifically mammals, has been subject to a regression during the past decades due to several natural or human factors, including climate change, increased rural population, poaching, and disruptive activities such as small-scale mining.

The wildlife is subject to various threats and constraints, the most significant ones being small-scale mining activities, which progressively developing in the Kédougou region, namely near the Boto 5 deposit, in Fadougou, Guémédji, and Diakha-Sénégal, the forest agrarian system in which new lands are progressively being cleared, logging conducted as part of the exploitation of forest resources, bush fires, transhumance, the early depletion of ponds and poaching.

The latest inventories from July and November 2015 identified four reptile species: a garter snake species (Psammophis sp.), the red-sided skink (Trachylepis (Mabuya) perrotetii), the African spurred tortoise (Centrochelys (Geochelone) sulcata) and the Nile monitor (Varanus niloticus).

The number of bird species listed in Senegal varies based on the sources. According to Bird Life international (2016), there are 548 bird species in Senegal, including 135 species of water birds. Migratory species, i.e. those with populations that move to other regions on a seasonal basis, account for 44% of avian species observed in Senegal, or 242 species (Bird Life International, 2016). More than half of these migratory species are of Palearctic origin and migrate to Senegal during the northern winter (Coulthard, 2001).

16 bird species in Senegal are internationally considered critically endangered, endangered, or vulnerable by the International Union for Conservation of Nature (IUCN, 2016). 17 additional species are considered threatened by the IUCN (2016). Based on the known distribution area, 13 of these species could be observed in the Project’s study zone. Out of these, nine are resident species in the region and three are Palearctic species that winter in Africa. The red-footed falcon is a Palearctic species that could be observed during its migration to Southern Africa.

A total of 38 fish species from 12 families were listed during the two experimental fishery campaigns carried out in the study zone by Norda Stelo and Synergie in July 2015 (high water period) and in December 2015 (recession period). The most species-rich families are the Mochokidae (nine species), representing nearly 25% of all captured species, followed by the Cichlidae (five species), the Claroteidae (five species), the Cyprinidae (five species), the Alestidae (five species) and the Mormyridae (three species). The other families are represented by one or two species. By comparison, as part of a study performed in 2011 in the Boto permit zone, a total of 21 species were collected in the Falémé and Koïla Kabé rivers (Tropica, 2013). The Cichlidae family was the most represented, with four species, following by the Alestidae, with three species. The number of fish species captured was greater during the second fishery campaign, with 30 species from 11 families, compared with 22 species divided into 10 families during the first campaign, in July 2015.

The composition of the fish communities differs markedly between the two listing seasons. Out of the 38 fish species listed in total for both listings, less than half, or 14 species, are common to both campaigns. Combining the fishery results for both seasons shows that fish communities are richer in the Falémé (24 species) and

Balinko (23 species) rivers than in the Koïla Kabé River (13 species). The communities of the three rivers are relatively more diverse, with only nine common species out of a total of 38 listed species. Finally, the communities of the Falémé and Balinko rivers are the ones with the most similarities, with 16 fish species in common.

According to IUCN (2016), Senegal has 187 mammal species, including sea mammals (whales, dolphins, manatees). Out of those species, one is extinct in the wild (scimitar oryx), one is critically endangered (dama gazelle), four are endangered, twelve are vulnerable and five are near threatened, based on the IUCN Red List.

The 2015 field inventories identified twelve mammal species in the Project's study zone. The most common species are the green monkey and the striped ground squirrel. Two of these observed species, the Cape bushbuck and the Gambian sun squirrel, were not reported as present in the region during the inquiry conducted among local communities. None of species observed have a special status on the IUCN Red List. However, the hippopotamus is fully protected in accordance with Decree No. 86-844 on Hunting and Fauna Protection Code.

Meetings with the local communities in 2017 confirmed the presence of chimpanzees around the majority of villages visited. The presence of chimpanzees was also confirmed by observing several indications of indirect presence along the transects made in 2017 on the concession: nests (140 in total), fecies, and scraps of food. Additionally, images obtained from camera traps confirmed the presence of 19 species of mammals in the study zone, including the chimpanzee and two other species with a precarious status: the common hippopotamus and the Guinea baboon. Based on the results from camera traps, gallery forests represent the habitat with the richest biodiversity. In total, 20 species of mammals were identified during the 2017 inventories, which brings the total number of confirmed species in the study zone to 22 mammals, including the results from the 2015 inventories (small mammals included). Several additional species were added to the list of large and medium mammal species present in the study zone, including the chimpanzee, the jackal, the greater cane rat, the Guinea baboon and the civet. Out of the 20 mammal species still observed today, five have a special status in the IUCN Red List: the chimpanzee (endangered), the common hippopotamus and the lion (vulnerable), and the leopard and the Guinea baboon (near threatened).

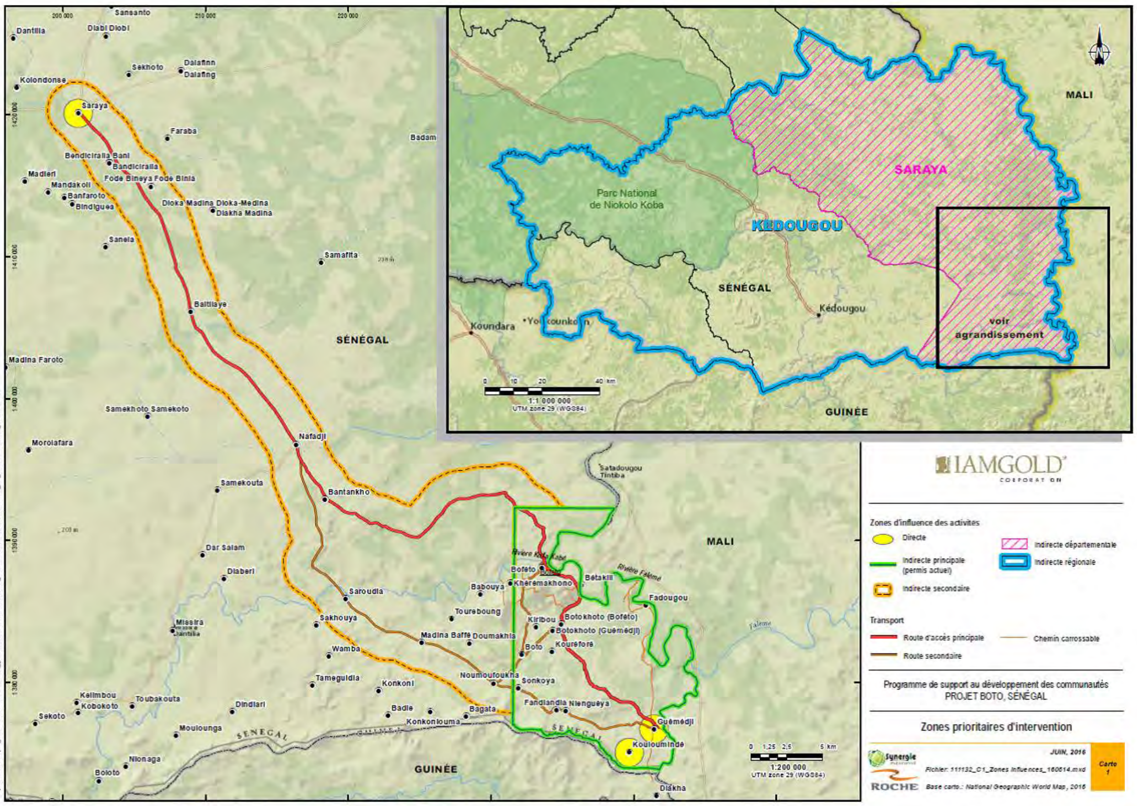
20.2.3 Human Environment

Based on the 2013 census, the population of the Saraya Department was 50,724, or 33.5% of the total population of the Kédougou region. The Saraya commune, administrative centre of the department, had a population of barely over 2,700 inhabitants, far less than the rural communities of Madina Baffé (6,782 inhabitants) and Bembou (13,646 inhabitants).

The rural community of Madina Baffé, where the permit zone is located, covers an area of 965.8 km^2, for an average population density of approximately seven inhabitants per km^2.

In the permit zone, there are two major villages and six hamlets with a total population of more than 3,000 inhabitants, or nearly 50% of the population of the rural community. The three main villages are Guémédji, Noumoufoukha and Madina Baffé. Many other villages and hamlets are located near the Project zone or access road connecting Boto to the town of Saraya. Figure 20.7 below shows the location of the villages and hamlets in the zone.

Figure 20.10 Map showing the Location of the Villages and Hamlets in the Boto Zone



The population present in the Project zone is relatively young and is 98% Muslim; majority ethnic groups are, respectively, the Malinkés and the Djallonkés and Peuls. The settlement of the villages and hamlets was done through the movement of populations looking for fertile lands.

Human settlements are very dispersed, certainly due to the logic of being close to fertile lands. Accommodations are mainly huts built in adobe, bamboo and straw.

Agriculture is the main socio-economic activity, followed by small-scale mining, husbandry – which has clearly declined as a result of cattle theft and recurrent animal diseases – and fishing, which is done scarcely on the Falémé River or the Koila Kobé. Agriculture is of the subsistence type, and it barely provides self-sufficiency due to constraints related to its rudimentary nature, the lack of inputs and its abandonment in favour of small-scale mining, which unfortunately does not guarantee gains to compensate agricultural shortfalls.

In terms of health, the predominant diseases are malaria, pulmonary diseases, infectious diarrhea, and malnutrition and they appear on a seasonal basis. The main health facility is the Madina Baffé health centre, which faces certain challenges during the rainy season due to its remote location. In addition, there are challenges related to the mobility of communities, whose access to the facility is particularly hindered during the rainy season.

In terms of education, the multitude of temporary shelters and/or multigrade classes, the insufficient equipment, the lack of support infrastructures, such as toilets and waterworks, and the difficulty of accessing school supplies due to poverty are all obstacles to schooling in the zone.

Regarding access to potable water, some villages have boreholes (Noumoufoukha, Guémédji, Madina Bafé, Saroudia, Nafadji and Boféto), but traditional wells, marshes and rivers are the most frequently used sources of water.

It must be noted that the zone, due to its geographical position (borders with Guinea and Mali) and the existence of small-scale mining sites, which are points of convergence for foreigners from the region or the bordering countries, is faced with serious, constantly growing insecurity.

The Project is located near Guémédji village and its Kouloumindé hamlet. Guémédji village is one of the largest official villages of the rural community of Madina Baffé. Established approximately two centuries ago, it has the distinction of being the most easterly village of its administrative district. Guémédji has five hamlets: Kouloumindé, Diakha-Guémédji (commonly called Diakha-Sénégal), Fadougou, Fandiandia and Botokhoto-Guémédji. Guémédji's population was estimated at 1,700 inhabitants in 2015.

The Diakha-Sénégal hamlet, located in the zone that is subject to a dispute between Senegal and Guinea-Conakry, was relocated along the road connecting Boféto and Guémedji, just outside the limited study zone defined for this project. It was also renamed Diakha-Guémédji (or, according to certain inhabitants, Diakha-Macky).

The Kouloumindé hamlet, created around the 2000s due to the presence of pasturelands and farmland, is one of the three main hamlets of Guémédji village. It is located approximately 4 km south-west from the village, on a site wedged between the range of hills forming the border between Guinea to the west and south, the cuirasse plateau to the east and a small-scale mining site and the forest to the north. According to the hamlet's chief, the population was evaluated at 272 inhabitants in 2014 and currently includes 30 households.

Figure 20.11 Guémédji village

Small-Scale Mining

Small-scale mining, an income-generating activity, is ubiquitous on the Boto permit zone. A specific study on the small-scale mining situation was completed as part of the ESIA. The main sites currently used by small-scale miners are the deposit Boto 5 and Fadougou. Intense activities of processing the ore mined the small-scale miners take place along the Balinko River, in Guémédji village. A draft small-scale miner engagement plan was developed as part of the ESIA. For safety reasons, small-scale mining activities currently in progress at Boto 5 will have to stop.

Archaeology and Cultural Heritage

The permit zone fits perfectly in the culture of eastern Senegal, where one of the most ancient stone industries in Senegal was developed.

Essentially, the populations of the villages in the Koudékourou sector come from Guinea, except for the inhabitants of Boféto, which are purportedly from Bétékhill, a Malian village, and the inhabitants of Samécouta. Most villages maintain that their founder left his country or land of origin to escape conflicts (Madina Baffé, Noumoufoukha), retaliation that could result from loss of power, or taxation (Saroudia). However, other villages state that their founder came to their current land in search of more fertile lands (Boféto, Babouya).

Sacred sites including cemeteries, sacred trees, rock shelters and rocks were listed. Most villages have mosques, but aside from Saroudia, a marabout village that seems well resourced, the mosques of the other villages are built in bamboo.

Archaeological sites and isolated finds in the zone show a highly variable material culture and includes lithic industries and pottery industries, predominant in the assemblages. Grinding stones collected on several sites suggest activities related to processing grains, leaves, etc., for food or medical purposes.

A literature review of the archaeological potential was conducted as part of the ESIA in progress. In consultation with the relevant authorities, it will be decided whether excavations are necessary on the site before the start of operations.

20.3 Waste and Tailings Geochemistry

Tailings

The geochemical characterization of 43 tailings samples from the treatment of samples representative of the ore was completed in 2016. The main components of the tailings are iron, calcium, magnesium, aluminium and potassium. Tailings have low levels of heavy metals (nickel, copper, chrome, zinc and cobalt).

According to Price's classification (2009) used in Canada to determine acid generating potential (AGP), the materials that show an acid neutralizing capacity (ANC) twice as significant as the acidifying potential (AP) (ANC/AP > 2) are not likely to generate acid mine water. As a whole, samples presented an average AP of 26.1 kg $CaCO_3$/t and an average ANC of 128 kg $CaCO_3$/t, for an average ANC/AP ratio of 4.9. Tailings sent to the tailings storage facility are not likely to generate acid water.

Leaching kinetic tests conducted over a period of 20 weeks on 3 tailings samples corroborated the absence of a mine drainage potential. Indeed, leachates collected were still alkaline (pH > 7.0), and sulphate concentrations (indicator of the extent of sulphide oxidation) at the end of the tests were low (< 50 mg/l). Regarding heavy metals, nickel and copper concentrations remained low after the initial leaching period.

Waste

Geochemical characterizations were carried out from 2015 to 2017 on 44 samples from the Malikoundi deposit and 27 samples from the Boto 5 deposit. Four composite samples were also formed from a selection of individual samples to complete the leaching kinetic tests. These four samples were also characterized. However, the Project currently only involves mining saprolite and a portion of the Boto-5 deposit's transition zone. Only 14 waste samples from the Boto 5 deposit come from mined weathering profiles.

Compared with the Boto 5 deposit, the Malikoundi deposit is characterized by high aluminium, iron, calcium and magnesium contents. The Boto 5 deposit, which has been more weathered, shows higher arsenic, cobalt and copper contents. In fact, the Boto 5 deposit's calcium and magnesium content suggest that the carbonates were consumed over time.

As a whole, the Malikoundi project's waste does not present an acid generating potential. Both composite samples used for the leaching kinetic tests also showed an absence of metal leaching potential.

None of Boto 5's eleven individual samples from non-consolidated horizons (saprolite and transition) and collected at less than 30 m showed an acid generating potential. Three non-consolidated material samples collected between 30 m and 40 m from the surface showed an acid generating potential. However, the average sulphide content of these three samples is 1.36%, which is not very high compared with many other acid generating mine materials. But these three samples were from the same drill core. Therefore, one can suspect that these samples are representative of a very limited portion of the total non-consolidated materials of the deposit and that they constitute a nugget effect. In fact, samples collected closer to the surface (19 m -20 m) and deeper (50 m 51 m) in the same drill core showed very low sulphide contents (<0.01% and 0.03%, respectively), which tends to confirm the very localized and limited presence of sulphide materials.

In this context for the Boto 5 deposit, an in-depth study of mineralogical analyzes completed on all drill cores will be carried out.

If an acid mine drainage issue were to be identified, mitigation measures would be implemented to manage potentially acid generating waste, in accordance with the best environmental practices recognized in the mining industry.

It must be noted that additional static and kinetic tests are currently in progress, and the results will be used to better document the Boto 5 deposit's surface horizon geochemical behaviour.

20.4 Potential Impacts of the Project and Mitigation Measures

The ESIA resulted in the identification of the main potential impacts as well as the benefits the Project could have on the environment and the social environment. The main potential negative impacts are the following:

- Reduced area for lands that could be used by the community for the purposes of agriculture, husbandry, market gardens and other uses, due to land occupation by infrastructures and various components of the Project.
- Loss of cropland to build certain infrastructures and to establish a safety radius around components that present a risk to the population.
- Disruption of plant and wildlife habitats by construction activities and mining operations.
- Modification of the sector's hydrological and hydrogeological regime due to land occupation by infrastructures and components of the Project, the development of ditches and drainage channels, the development of water storage ponds, the excavation and dewatering of open pits, etc.
- Increased ambient noise level due to blasting and ore and waste handling activities, as well as the equipment used in the industrial sector.
- Disruption of ambient air quality due to handling of material, ore and waste, operation of the thermal power plant and of the ore processing mill, etc.
- Disruption of surface and ground water quality as a result of deforestation exposing the land to erosion, the potential discharge of contaminated water by the septic waste water treatment plant, waste dumps and tailings storage facility, potential discharges of hazardous material or petroleum products, etc.

- Increased pressure on already limited services related to health, education, and water and food supply, potential increase in crime rate and cases of communicable diseases, caused by the influx of migrants, namely crossing the borders from Mali and Guinea, seeking job and economic opportunities in the sector.

On the other hand, the Project will bring several benefits for the Senegalese State and the communities in the Kédougou and Saraya regions. Indeed, the mine operations will result in significant revenues for the Senegalese Treasury, create hundreds of direct and indirect jobs, provide business opportunities for Senegalese service providers and suppliers, and offer possibilities of professional development and training to local populations. In addition, the Project will contribute to the development of local communities through its development support program, whose outlines are presented below.

20.4.1 Mitigation Measures

IAMGOLD's Zero Harm vision guides all of the company's operations and activities. It is the company's commitment to continually strive to reach the highest standards in health and safety, minimize impact on the environment, and work co-operatively with host communities.

The Zero Harm vision is not only a corporate commitment, it is also the vision that guides the development of the Project and the design of its various components. Thus, the Project was developed in order to:

- Promote a beneficial, harmonious and respectful coexistence between local communities and mining operations.

- Comply with national and international regulations in terms of environment, occupational health and safety and corporate social responsibility (CSR).

- Protect the communities' health and safety.

- Effectively mitigate impacts, nuisances and inconveniences caused by the activities on local communities and the environment.

- Minimize encroachment of project components on lands used by the communities and on protected or important habitats.

The Project was designed to minimize impacts on the population and the environment. First, the 500 m safety distance prescribed by Senegal's Environmental Code will be complied with for all project components. Indeed, the Project's various infrastructures and components were positioned so that they are at more than 500 m from permanent watercourses and population centres. These include waste dumps, TMF, explosive magazines, the process plant, the FWP, etc. The only exception is the Boto 5 pit, which is located at less than 500 m from some of the Kouloumindé hamlet residences.

For safety and security reasons, it was proposed to plan significant buffer zones around the main infrastructures that could present a risk for the community as well as their cattle. As part of the ESIA, it was proposed to the Senegalese authorities that a 500 m wide buffer zone be implemented around the tailings storage facilities, the explosives storage area, the ore processing mill, the open pits and the waste dumps. Additionally, a linear 50 m wide buffer zone was proposed around the ore hauling road connecting the Boto 5 pit to the processing plant. Measures will be taken to ensure the safety of community members who cross this road at the two points where it crosses paths currently used by the inhabitants. Figure 20.12 shows the location of infrastructure planned in the original feasibility study and the 500 m safety radii around them.

Various common mitigation measures were integrated to the Project during its design. The storage and handling areas for petroleum products and reagents will have a secondary spill retention capacity, run-off water from waste dumps and drainage water from pits will be collected and directed to settling ponds to reduce its concentration of suspended solids and septic waste water will be treated by a proven technology before it is released in the environment.

In addition, the TMF’s foundation will be designed and developed to minimize the risk of exfiltration. A watertight geomembrane will be installed at the bottom of the TMF to protect the quality of ground water. The TMF will also be designed and operated so as to prevent process waste from being released into the environment, except in the event of extreme weather events. In such a situation, the emergency spillway developed for that purpose would become functional to reduce risks of damage and breach in the facility’s dam.

Many other mitigation measures will be implemented. They are presented in detail in the ESIA report.

Kouloumindé Hamlet

Since a part of the Kouloumindé hamlet is located at less than 500 m from the Boto 5 pit, there will be no blasting operations in the Boto 5 pit.

A model of the noise level of ground vibration and excess air pressure resulting from blasts was completed. The noise at the Kouloumindé hamlet will reach levels of 35 to 42 dBA during the day. These levels comply with the 55 dBA limit for daytime. Operations at the northern portion of the Boto 5 pit during the night will produce maximum noise levels of 30 to 39 dBA in the Kouloumindé village. These levels comply with Senegal’s 40 dBA limit for night time as well as with the IFC/World Bank’s guidelines (45 dB for night time).

20.4.2 Relocation and Compensation Strategy

While the Project had been designed to minimize encroachment of infrastructures and components on inhabited sectors and fields, its completion will inevitably lead to impacts in that regard.

Additionally, lands currently used for agriculture, husbandry or market gardens and lands that have a good agronomic potential will be impacted by the Project. This is namely the case for the sectors of the pit, of the Malikoundi waste dump and of the ore processing mill.

In compliance with Senegalese regulatory requirements and World Bank guidelines, measures will need to be taken to mitigation the effects of these involuntary relocations. Thus, a relocation and compensation strategy was developed for the Project by the specialized firm RePlan, which supported several developers with relocation and compensation programs in Senegal.

Should the displacement of revenue-generating activities or of people to other revenue-generating activities be required to complete the Project, AGEM will implement a relocation and compensation program in compliance with the requirements of Senegalese regulations and international standards. For the development and implementation of the provisional relocation and compensation program, a four-phase process will be implemented:

- Phase 1: The provisional program published as part of the current ESIA will be the main output of this phase.
- Phase 2: Comprehensive investigation to document the impacts of relocations and to identify all affected persons.
- Phase 3: Implementation and individual negotiations.
- Phase 4: Monitoring and support, including implementation of the agreed upon livelihood restoration program, the vulnerable person assistance program, and the monitoring-assessment program.

The activities necessary to achieve a comprehensive and operational relocation and compensation program are as follows:

- Development of a stakeholder engagement strategy specific to relocations and compensation, including the creation of a forum for discussions about issues related to land acquisition, relocation, and livelihood restoration.
- Development of a compensation framework, including defining a policy and eligibility criteria, rules, and specific compensation types.
- Selection of relocation sites and action planning to facilitate the relocation.
- Development of a livelihood restoration program.
- Development of a support plan for vulnerable people and groups.
- Implementation of a grievance management system.

20.4.3 Local Community Development Support Program

The Boto sector is characterized by deep poverty, remoteness and isolation during the rainy seasons, and lacking public utilities. The public inquiry conducted in May-June 2016 at the request of the Governor of Kédougou revealed that the local populations are pinning their hopes on the Project, hoping that the project's arrival will promote the development of local communities.

The completion of the Project could represent a structuring project that will allow the State to sustainably improve the inhabitants' living conditions. Therefore, if the Project is implemented, AGEM intends to provide effective support to the authorities and communities to improve living conditions in the zone and help the development of local communities. This support will remain within the financial limits of the company and in accordance with the mandates of the State.

The guiding principles of the local community development support program are as follows:

- Act in partnership with the State, public agencies and specialize non-governmental organizations (NGO's).
- Invest in the sustainable development of local communities.
- Act in partnership with local communities.
- Give preference to one-time investment in projects already identified in priority action plans and other development plans from the authorities.
- Establish an agreement for each investment.

The priorities that the company intends to support as part of the program are as follows:

- Population health.
- Water supply.
- Food security.
- Education and training.
- Population safety.
- Income-generating activities and local economy.
- Energy.
- Collective sanitation of villages and districts.
- Leisure and population retention.
- Income-generating activities.

AGEM will focus as a priority on sectors located near mining operations and indirect activities related to the construction and operation of a mine. To this end, five action zones will be established (map 3.2). These zones are:

Direct impact zone:

- Zone 1 – Médina Baffé, Doumakhia, Touréboung, Babouya, Khérémakhono, Kiribou, Boféto, Bétékhali, Guémedji, Fadougou, Kouliminde, Noumoufoukha, Boto-Boféto, Boto-Guémedji, Diakha-Guémedji, Khouréforé, Guémedji, Niengueya, Fandiandian Sonkhoya.

Indirect impact zones:

- Zone 2 – Rest of the Madina Baffé commune (outside zone 1) and Nafadji village.
- Zone 3 – Saraya Department: nearby villages on the Saraya road and Saraya town (due to the expected presence of the majority of workers and their family).
- Zone 4 – Rest of the Kédougou region.
- Zone 5 – Rest of the Senegalese territory.

AGEM's support of community development will be provided as a priority to zones 1, 2 and 3, but without neglecting zones 4 and 5.

20.5 Environmental and Social Management Program (ESMP)

To effectively reduce environmental and social impacts and adequately monitor activities, Senegal's Environmental Code requires that an Environmental and Social Management Program (ESMP) be developed, implemented and maintained for large-scale projects such as the Project.

An ESMP addresses the main environmental and social issues identified in the ESIA. It must include mitigation measures that will be implemented, monitoring and surveillance measures, key performance indicators, relevant records and emergency measures to be implemented if needed. It also includes a relocation and compensation plan (if required) as well as a local community development support program. The ESMP applies to all project phases: construction, operation and closure.

A preliminary ESMP was presented to the authorities in the ESIA report. Once the environmental permit is obtained for the Project, an official version of the ESMP will be prepared and implemented.

20.6 Reclamation, Restoration and Closure Plan

As required by Senegalese authorities, volume II of the ESIA report details the reclamation, restoration and closure activities planned for the Project. The main activities planned are the following:

- At the end of operations, all infrastructures and service buildings will be dismantled, unless the State formally acquires them, without any civil liability for AGEM.
- Concrete foundations will be crushed and integrated into the soil in place. Lands will return to a state similar to their original state.

- Soil characterization will be carried out at locations that may have been contaminated (e.g. fuel storage zone). Contaminated soil management will be done in compliance with authorities' requirements and/or good practices. Waste dumps, TMF and FWP will be revegetalized using, among other things, the accumulated topsoil. The use of endemic plant species that cattle do not feed on will be favoured. It should be noted that the reclamation strategy will be progressive for waste dumps.

- As soon as water quality is established, a breach will be made in the process water pond of the TMF to allow the free flow of water. A breach will also be made in the FWP to allow the free flow of water.

- The open pits' access ramps will be dismantled. Berms will be made at the edge of the pits to prevent animals and people from accidentally entering.

- Site monitoring will continue for five years after the end of production. The quality of water, air and soils will be monitored to confirm that all parameters studies return to their pre-mining levels. The success of revegetation will be monitored to make sure the new vegetation is self-sufficient.

A cost estimate for reclamation, restoration, and closure is provided in Section 21. This amount does not include the costs to carry out social and community projects. The amount to be invested for that will be determined in the mining agreement that will be signed between AGEM and the Senegal. Wherever practicable, progressive restoration activities for the TMF and the waste dump slopes to reduce the cost of reclamation and restoration activities to be carried out upon closure of the mine.

20.7 Permit

The expiry date of the existing exploration permit is March 4, 2019. Normally, a request for a mining permit or a mining concession permit must be filed at least four months before the expiry date of the exploration permit. The request must be supported by, among other things, a FS, an ESIA, a development and start-up plan and an investment plan.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Summary

The overall capital cost estimate was compiled by Lycopodium and is presented here in summary format. The capital cost estimate reflects the Project scope as described in this report. Mine capital costs (developed by AGP) are included in the estimate tables below. KP provided quantities for the TMF, FWP and Site Water Management Plan.

The capital estimate for the Project is summarized in Table 21.1. The initial capital cost estimate by discipline is provided in Table 21.2.

All costs are expressed in United States Dollars unless otherwise stated and are based on Q2 2018 pricing and deemed to have an overall accuracy of ±15%. The capital cost estimate conforms to AACEI (Association for the Advancement of Cost Engineering International) Class 3 estimate standards as prescribed in recommended practice 47R11.

The capital cost estimate was based on an EPCM implementation approach and typical construction contract packaging. Equipment pricing was based on quotations and actual equipment costs from recent similar Lycopodium projects considered representative of the Project.

Table 21.1 Capital Cost Estimate Summary (Q2 2018, ±15%)

Area	M$ (Excluding Duties and Taxes)
Direct Costs	
Mining	$62.1
Infrastructure	$19.0
Ore Handling & Processing	$57.4
Tailings & Water Management	$16.6
Sub-Total Direct Costs	**$155.1**
Indirect Costs	
Construction In-directs	$49.0
Owner's Costs	$19.2
Contingency	$24.3
Sub-Total Indirect Costs	**$92.5**
Sub-Total Initial Capital Costs	**$247.6**
Additional Indirect Costs	$6.8
Total Initial Capital Cost	**$254.4**
Sustaining Capital Cost	$66.0
Total Project Capital Cost	**$320.4**

Table 21.2 Initial Capital Estimate Summary by Discipline (Q2 2018, ±15%)

Table Title	Supply Cost $	Installation Cost $	Contingency $	Total $
A General	$12,650,627	$647,837	$1,787,148	$15,085,612
B Earthworks	$7,642,408	$2,721,460	$1,750,003	$12,113,872
C Concrete	$10,933,065	$4,559,319	$2,203,404	$17,695,788
D Steelwork	$2,025,286	$1,447,996	$430,687	$3,903,969
E Platework	$3,753,215	$2,147,921	$733,519	$6,634,654
F Mechanical	$34,385,973	$2,176,160	$4,744,109	$41,306,242
G Piping	$5,105,872	$2,274,533	$1,180,909	$8,561,315
H Electrical	$8,496,951	$1,996,766	$1,503,223	$11,996,940
J Instrumentation & Control	$3,367,053	$810,070	$695,533	$4,872,656
M Buildings & Architectural	$2,577,976	$365,601	$295,615	$3,239,192
N Mining	$61,608,163	-	$2,710,759	$64,318,922
O Owners	$25,643,438	$826,946	$2,650,569	$29,120,954
P In-directs	$3,431,499	$14,001,125	$1,849,373	$19,281,996
R Tailings	$14,503,217	-	$1,715,526	$16,218,743
Grand Total	**$196,124,744**	**$33,975,735**	**$24,250,377**	**$254,350,856**

The sustaining costs for the Project are provided in Table 21.3 and include mine equipment purchases and replacement, and stage development of the TMF.

Table 21.3 Summary of Sustaining Capital Costs (Q2 2018, ±15%)

	Total	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13
Mining Capital ($M)	**$42.90**	$8.04	$9.38	$3.71	$1.15	$2.53	$4.82	$3.81	$1.79	$7.45	-	$0.22
Miscellaneous Mine Capital ($M)	**$2.68**	$0.77	$0.35	-	$0.43	$0.13	$0.22	$0.22	$0.13	$0.43	-	-
TMF Capital ($M)	**$20.44**	$2.40	$2.00	$3.68	-	$5.01	-	$3.85	-	-	$3.50	-
Total	**$66.02**	**$11.21**	**$11.73**	**$7.39**	**$1.58**	**$7.67**	**$5.04**	**$7.88**	**$1.92**	**$7.88**	**$3.50**	**$0.22**

The foreign exchange rates in Table 21.4 have been used in the compilation of the estimate.

Table 21.4 Currency Exchange Rates

Currency	$USD
AUD	0.78
USD	1.00
Euro	1.20
Rand	0.07
CAD	0.8
CFA	0.001754

21.2 Mine Capital Cost

The capital costs for the mine are summarized in Table 21.5. These costs exclude contingency, duties and taxes.

Table 21.5 Capital Cost Summary – Mining

Capital Category	Preproduction Capital Year -2, -1 $M	Sustaining Capital $M	Total Capital $M
Mining Equipment	$23.3	$42.9	$66.2
Miscellaneous Mine Capital	$3.3	$2.7	$6.0
Pre-Production Stripping	$35.0	-	$35.0
Total	**$61.6**	**$45.6**	**$107.2**

Initial capital requirements (pre-production) are estimated to be $61.6M and includes pre-production mining which is capitalized. The mining equipment capital reflect full purchase of the equipment. Leasing or financing have not been included for the FS.

The open pit mining equipment capital costs are based on conventional medium scale open pit equipment, common for West Africa and contractor quotes with regional experience

Production drilling is completed with down the hole (DTH) hammer rigs drilling either a 154 mm bit or a 104 mm bit. The drill rigs will do both production drilling and pre-shear drilling as required. The loading fleet is comprised of 15 m^3 hydraulic excavators as the primary loading unit together with 13 m^3 front end loaders (FEL) and 6.7 m^3 hydraulic excavators. The truck fleet is rigid frame trucks with 95 t capacity. The trucks will be outfitted with lightweight boxes to increase their payload. Support equipment includes track dozers, graders, water trucks, support backhoes, stockpile loaders, pumps and additional equipment.

Boto 5 is planned as a contract mining pit which starts in Year 1. The contractor will be using 40 t trucks and the associated support equipment. No capital is required for the Boto 5 pit other than road construction from the plant to the pit which has been included in the Pre-production stripping.

The mining capital detail by period is shown in Table 21.6.

Table 21.6 LOM - Mining Capital by Period

Equipment	Preproduction Year -2, -1 $M	Sustaining Capital $	Total $M
Mining Equipment			
Major Mine Equipment	$23.3	$42.9	$66.2
Subtotal	**$23.3**	**$42.9**	**$66.2**
Miscellaneous Mine Capital			
Engineering Office Equipment	$1.2	-	$1.2
Dispatch System	$1.3	-	$1.3
Dewatering System – pumps/piping	$0.6	$2.0	$2.6
Boto 5 – WMF preparation	$0.2	-	$0.2
Geotechnical Instrumentation – Phase 2	-	$0.2	$0.2
Geotechnical Instrumentation – Phase 3	-	$0.3	$0.3
Geotechnical Instrumentation – Boto 5	-	$0.2	$0.2
Subtotal	**$3.3**	**$2.7**	**$6.0**
Pre-Production Stripping	$35.0	-	$35.0
Total Mine Capital	**$61.6**	**$45.6**	**$107.2**

Equipment pricing was received from the local vendors and Caterpillar, Komatsu, Sandvik and Atlas Copco.

The distribution of the capital costs was completed using the units required within a given period. This may be new units or replacement units as needed for sustaining capital.

The number of units are determined by the annual operating hours required. These were balanced over the periods of time so that if there are fluctuations in the hours from period to period, or year to year, these are distributed over the entire equipment fleet to balance the hours.

Replacement times for the equipment are average values from AGP experience and from vendor estimates of reasonable replacement hours. Options around rebuilds and recertification of equipment like track dozers are not considered nor is used equipment.

The balancing of equipment units was based on operating hours is completed for each major piece of mine equipment. The smaller equipment was based on number of units required based on operational experience.

The most significant quantity of major mine equipment are the haulage trucks. At the peak of mining 20 units are necessary to maintain mine production. This happens from Year 6 onwards and is a result of the stripping requirements of Malikoundi Phase 3. The maximum hours per truck per year are set at 6,000.

The other major mine equipment is determined in the same manner. In some instances, the loaders have a longer period life (same number of hours between replacements) because of this sharing of hours with the other units in the fleet.

The support equipment is usually replaced on a number of year's basis. Pickup trucks are leased on a monthly basis and form part of the operating cost but not the capital.

Major equipment purchase timing is indicated in Table 21.7.

Table 21.7 Mine Major Equipment Purchases

Equipment	Unit Life	Year -2	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
Production Drill	25,000 hrs	2	1	-	1	2	1	1	2	-	-	-	-	-	-	-
Large Excavator(15m^3)	72,000 hrs	1	-	1	1	-	-	-	-	-	-	-	-	-	-	-
FEL (13m^3)	35,000 hrs	1	-	1	-	-	-	-	-	1	-	-	-	-	-	-
Small Excavator	7 years	1	-	-	-	-	-	-	-	-	-	1	-	-	-	-
Haulage Truck	50,000 hrs	7	2	4	4	1	-	-	2	-	-	-	-	-	-	-
Tracked Dozer	35,000 hrs	4	1	-	-	-	-	-	-	-	2	2	-	-	-	-
Grader	20,000 hrs	1	-	2	-	-	-	-	-	2	1	-	-	-	-	-
Support Backhoe	10 years	1	-	-	-	-	-	-	-	-	-	1	-	-	-	-

Miscellaneous Mine Capital

The miscellaneous mine capital includes various separate line items in the costing:

- Engineering Office Equipment.
- Mine Dispatch System.
- Dewatering System - Pumps and Piping.
- Boto 5 – WMF preparation.
- Geotechnical Instrumentation – Phase 2.
- Geotechnical Instrumentation – Phase 3.
- Geotechnical Instrumentation – Boto 5.

The engineering office equipment includes items such as mining software purchase for geology and engineering, survey equipment with associated peripherals including survey drones. This cost is estimated at $200,000 with the majority of the cost being the mining software.

The mine dispatch system will assist in the routing and tracking of material movement during operation.

The dewatering system is a set of pumps in the pit with piping to bring water to the surface ditches which will flow by gravity to the FWP. Two pumps will be purchased initially with 1 km of piping to start the system. Average pumping requirements assumed are expected to be 1,700 m^3/d initially with an initial maximum pumping rate of 3,400 m^3/day during storm events or in the rainy season. As the mine expands in size laterally and depth, the pumping rate will increase and that is considered under sustaining capital for the purchase of additional pumps and piping.

Boto 5 has a different terrain with more vegetation and poorer foundation conditions than the Malikoundi area. One dump facility will be developed for Boto 5, which requires additional preparation of the foundation due to removing more vegetation and topsoil prior to waste placement. Appropriate cost allowances have been made for the Boto 5 waste dump foundation preparation.

The geotechnical instrumentation includes vibrating wire piezometers and other slope monitoring instrumentation as recommended by the geotechnical consultant, Absolute Geotechnics.

Pre-Production Stripping

The mine is scheduled to initiate mining in Year -2. Year -2 is only three months long and Year -1 is 12 months long for a total of 15 months preproduction stripping. The material moved will be used to develop the mine roads, TMF construction, FWP, initial waste management facility development and provide ore for the stockpile. A total of 13.8 Mt of waste will be mined and 2.2 Mt of ore stockpiled.

Waste material will be hauled longer to the TMF areas for construction of the facility prior to plant commissioning. Ore will be stockpiled at the plant for commissioning and surge capacity.

This is expected to cost $35.0 M or $2.19/t material moved. This includes all costs associated with IAMGOLD management, dewatering, drilling, blasting, loading, hauling, support, engineering and geology department labour and ore control.

These construction activities are typically less productive hauls due to narrower working conditions and longer hauls than normally scheduled for the waste material. The TMF construction means the trucks will have to turn around on narrow road widths requiring back and forth movement to negotiate the turns. This plus extended reversing of the loaded trucks to the dumping point results in longer truck cycle times. This has been factored into the haulage times.

Contingency

Significant effort has been undertaken to determine the mine capital costs, and in particular the mine equipment. With the level of confidence from these negotiations, no contingency is applied to the mine capital costs.

21.3 Plant and Infrastructure Capital Costs

The capital costs for the process plant and infrastructure capital are based on the facilities described in Chapters 17 and 18 and prepared by Lycopodium with input from KP on the TMF, FWP and Site Water Management Plan.

The purpose of the capital cost estimate is to provide substantiated costs which can be utilized to assess the preliminary economics of the Project. The capital cost estimate is based on an EPCM implementation approach and horizontal (discipline based) construction contract packaging.

The various elements of the Project estimate have been subject to internal peer review by Lycopodium and have been reviewed for scope and accuracy.

A summary of the estimate methodology is provided in Table 21.8 and estimate methodology is provided in Table 21.9.

Table 21.8 Capital Cost Estimate Basis

Description	Basis
Site	
Geographical Location	Actual site
Maps and Surveys	Available
Geotechnical Data	Preliminary
Process Definition	
Process Selection	Fixed
Design Criteria	Final for FS
Flowsheets/Plant Capacity	Final for FS
P&IDs	Not Required as suitable 'go-by' costs available from Lycopodium's database
Mass Balances	Final for FS
Equipment List	Final for FS
Process Facilities Design	
Equipment Selection	Selection based on duty and budget pricing provided by vendors.
General Arrangement Drawings	Final for FS
3D model	Preliminary to a level of detail suitable for FS
Piping Drawings	No drawings. Plant piping factored. Overland piping material take off.
Electrical Drawings	HV SLD. LV drawings not required
Specifications/Data Sheets	Preliminary for budget quotation requests (BQRs)
Infrastructure Definition	
Existing Services	Not relevant
Design Basis	Fixed
Layout	Fixed

Table 21.9 Capital Cost Estimate Methodology

Description	Basis
Bulk Earthworks	Volume estimated from the layout and available topography for bulk earthworks on all sites. Unit rates discussed in narrative sections below.
Detailed Earthworks	Allowances for under pad excavation and backfill to prepare site for concrete works
Concrete Installation	Estimated from the layout and similar projects of comparable scale. Concrete (wet) supply rates and installation rates applied from project specific BQRs.
Structural Steel	Quantities estimated from the layout and similar projects of comparable scale. Supply and install rates applied from project specific BQRs.
Platework & Small Tanks	Quantities provided in the mechanical equipment list. Large item quantities estimated from reference projects. Smaller items compared to database. Supply and install rates applied from project specific BQRs.
Tankage Field Erect	Quantities provided in the mechanical equipment list. Supply and install rates applied from project specific BQRs.
Mechanical Equipment	Quantities provided in the mechanical equipment list. Costs from responses to BQRs from reputable suppliers for all equipment with a value nominally >$25,000. Costs for low value items taken from the Lycopodium database.
Haul Roads	Refer mining cost estimate.
Mining Fleet	Refer mining cost estimate.
Power Station	Build–Own–Operate project (BOO). BQR to reputable suppliers based on project specific duty.
Conveyors	Concrete & structural estimated from reference projects and layout. Mechanicals supply and install pricing from project specific BQRs.
Plant Piping General	Factored from mechanical costs.
Overland Piping	Size and specification based on engineering selection. Quantity based on site layout. Rates based on project specific BQRs.
Electrical General	Quantities derived from engineering design and site layout. Materials pricing and installation costs from a combination of recent database information and responses to BQRs.
Electrical HV	Quantities derived from engineering design and site layout. Materials pricing and installation costs from a combination of recent database information and responses to BQRs.
Commodity Rates – General	Appropriate rates from a combination of recent database information and responses to project specific BQRs.
Installation Rates – General	Appropriate rates from a combination of recent database information and responses to project specific BQRs based on preliminary contracting strategy.
Heavy Cranes	Requirements estimated based on largest lifts and likely duration.
Freight General	Factors assessed in a line by line basis and validated against similar projects.
Contractor Mobilization / Demobilization	Appropriate rates from a combination of recent database information and responses to project specific BQRs.
Fencing	Costed based on measured length and rate.
EPCM	Scope and deliverables-based estimate based on the EPCM controlled scope.
Vendor Representatives	Assessed by equipment package based on similar projects.
Site Establishment	Requirements estimated using base rates.
Construction Facilities	Allowance based on projects of a similar size.

Description	Basis
Owner's Costs	
Opening Stocks, First Fill Reagents and Consumables	Estimated from consumption rates and costs in operating cost estimate.
Spare Parts	Based on the Capital Spare List and costs derived from a combination of quoted rates and factors of equipment supply.
Owner's Project Team Labour	Client estimate.
Owner's Project Team Expenses	Client estimate.
IT & Communication	Client estimate.
Project HSEC	Client estimate.
Mobile Equipment	Estimated based on projects of a similar size.
Travel Expenses	Client estimate.
Operational Readiness	Client estimate.
Commissioning and Ramp-Up	First principle build-up of costs based on the assessed scope of work
Corporate Administration	Client estimate.
Condemnation Drilling	Client estimate.
Exclusions	
Duties and Taxes	Excluded on the assumption the Project will be exempt.
Escalation	Excluded.

21.3.1 General Estimating Methodology

The process plant was broken down into unit operation areas with quantity take-offs benchmarked against similar facilities from previous projects to provide the additional scope and level of confidence needed to confirm a FS level estimate was achieved.

Overall plant layout and equipment sizing was prepared with sufficient detail to permit an assessment of the engineering quantities for the majority of the facilities for earthworks, concrete, steelwork, and mechanical items. The layouts enabled preliminary estimates of quantities to be taken for all areas and for interconnecting items such as piperacks.

Unit rates for labour and materials were derived from responses to BQRs sent to fabricators and contractors experienced in the scale and type of work in the region.

Budget pricing for equipment was obtained from reputable suppliers with the exception of low value items which were costed from Lycopodium's database of recent project costs.

For the offices, workshops and similar items appropriate budget pricing was obtained from reputable suppliers of similar prefabricated designs with erection/installation costs derived from solicited contractor rates.

For the TMF and sediment control facilities bills of quantities were provided by KP based on their preliminary designs. Appropriate mining and construction rates were then applied to the quantities by Lycopodium. The bulk earthworks rates were based on estimated fleet requirement, wet hire rates for that equipment from local suppliers and appropriate allowances for supervision.

21.3.2 Quantity Development

Overall estimated quantities of key commodities are shown in Table 21.10.

The derivation of quantities within these categories by percentage is provided in the table weighted by value of the direct permanent works (i.e. excluding temporary works, construction services, commissioning assistance, EPCM, escalation and contingency).

Preliminary Engineering refers to quantities taken from data and layouts prepared specifically for the Project. Estimated refers to quantities derived from similar project designs factored and/or modified to suit.

Table 21.10 Derivation of Quantities - Bulks

Classification	Quantity	Unit	Preliminary Engineering %	Estimated %	Factored %
Concrete (excluding blinding)	10,214	m^3	100%	-	-
Structural Steel	793	t	100%	-	-
Chutes/Hoppers/Bins	242	t	100%	-	-
Field Erected Tanks	741	t	100%	-	-
Piping Bulks	-	-	-	-	100%
Overland Pipeline	19.2	km	100%	-	-
Electrical and Instrumentation	-	-	62%	-	38%
Modular Buildings	5,544	m^2	100%	-	-

21.3.3 Pricing Basis

Estimate pricing has been derived from a combination of the following sources:

- Budget Quotation: Budget pricing solicited specifically for the study or project estimate.
- Database: Historical database pricing that is less than one year old.
- Estimated: Historical database pricing older than one year, escalated to the current estimate base date.
- Factored: Factored from costs with a basis.
- Allowances.

Pricing has been identified by the following cost elements, as applicable, for the development of each estimate item.

Table 21.11 summarizes the source of pricing by major commodity, weighted by value of the direct permanent works (excluding temporary works, construction services, commissioning assistance, EPCM costs and contingency), including supply and installation.

Table 21.11 Source of Pricing

Classification	Budget Quotation %	Database %	Estimated %	Factored / Allowance %
Concrete	99%	1%	-	-
Structural Steel	100%	-	-	-
Platework / Tanks	99%	1%		
Mechanical Equipment	91%	8%	1%	-
Piping - Plant	-	-	-	100%
Overland Pipeline	100%	-	-	-
Electrical and Instrumentation	19%	21%	10%	50%
Buildings	46%	29%		25%

Plant Equipment

This component represents prefabricated, pre-assembled, off-the-shelf types of mechanical or electrical equipment, either fixed or mobile. Pricing is inclusive of all costs necessary to purchase the goods ex-works, generally excluding delivery to site (unless otherwise stated) but including operating and maintenance manuals. Vendor representation and commissioning spares have been allowed for separately in the estimate.

Bulk Materials

This component covers all other materials, normally purchased in bulk form, for installation on the Project. Costs include the purchase price ex-works, any off-site fabrication, transport to site (unless otherwise stated), and over-supply for anticipated wastage.

Installation

This component represents the cost to install the plant equipment and bulk materials on site or to perform site activities. Installation costs are further divided between direct labour, equipment and contractors' distributable.

The labour component reflects the cost of the direct workforce required to construct the Project scope. The labour cost is the product of the estimated work hours spent on site multiplied by the cost of labour to the contractor inclusive of overtime premiums, statutory overheads, payroll burden and contractor margin.

The equipment component reflects the cost of the construction equipment and running costs required to construct the Project. The equipment cost also includes cranes, vehicles, small tools, consumables, PPE and the applicable contractor's margin.

Contractors' in-direct costs encompass the remaining cost of installation and include items such as offsite management, onsite staff and supervision above trade level, crane drivers, mobilization and demobilization, R&Rs, meals and accommodation costs, and the applicable contractors' margin.

21.3.4 Temporary Construction Facilities

Facilities will be capable of servicing the Owner’s and EPCM teams.

Included in the estimate for construction facilities are the following:

- Construction offices.
- Computers and computing servers, telephones, printers, etc. and office furniture.
- Provision of services.

21.3.5 Heavy Lift Cranage

A heavy lift crane of 250 t capacity has been allowed for in the estimate for the duration of the installation of the mill.

21.3.6 Contractor Distributable

Mobilization/Demobilization

Costs for mobilization/demobilization of labour and equipment to/from the Project site were, where practical, adopted from budget quotation enquiries to contractors or adjusted from current tenders/contracts to reflect the Project location.

21.3.7 Earthworks

Quantities for plant site bulk earthworks have been estimated from the layout. Quantities for the TMF, FWP and surface water management facilities were provided by KP.

Rates were derived from quotations for the wet hire of the earthworks fleet built-up with allowances for miscellaneous additional labour and equipment and supervision. It is considered that the rates used are appropriate to complete the work as an addition to the mining contract, as a ‘self-perform’ activity by the Company using hired equipment or as a contract let to a local ‘second tier’ contractor.

As place and compact earthworks activities to construct the TMF embankment will continue virtually uninterrupted for the life of mine it is considered likely that the earthworks will be undertaken by the mining contractor.

Quantities for the ROM pad (at the process plant) are limited to the engineered fill and drainage and site earthworks required around the ROM pad retaining wall. The cost of the balance of the ROM pad is included in the mining estimate using material that would otherwise go to the waste dumps.

21.3.8 Concrete

Quantities for concrete works were established using the following:

- Material take-offs from layouts prepared for the FS.
- Benchmarking against detailed drawings for similar sized projects completed by Lycopodium.

Rates for this estimate were based on responses to BQRs from regional subcontractors with experience on this kind of work and capacity to perform the works.

Rates and quantities were prepared on a composite per cubic metre basis. Mobilization, demobilization and indirect costs were separated to reflect contract methodology.

21.3.9 Steelwork

Quantities for structural steel were established using:

- The layout and equipment elevation drawings/sketches prepared for the FS.
- Benchmarking against detailed drawings for similar sized projects completed by Lycopodium.

Rates for this estimate were based on responses to BQRs from fabricators with experience on global supply.

Site installation hours were applied from responses to BQRs from regional subcontractors with experience on this kind of work and capacity to perform the works.

21.3.10 Platework/Tankage

Platework and tankage quantities were determined using the sizing provided in the mechanical equipment list prepared for the FS as the basis. A preliminary design was undertaken for each tank to select appropriate plate thicknesses to develop tank tonnages. Lining materials, where applicable, were quantified separately.

Rates for this estimate were based on responses to BQRs from fabricators with experience on global supply.

Site installation hours were applied from responses to BQRs from regional subcontractors with experience on this kind of work and capacity to perform the works.

21.3.11 Mechanical Equipment

The mechanical equipment list prepared for the FS provided the quantities and sizing for the cost estimate.

Budget pricing was obtained from reputable suppliers for the majority of mechanical equipment, based on equipment data sheets prepared for the FS.

Equipment installation hours were estimated based on responses to BQRs solicited from contractors and installation hours estimated by Lycopodium. For each individual item of equipment due allowances were made for retrieval from the storage location, handling, placing, installing and commissioning the equipment.

21.3.12 Plant Pipework

The supply and installation estimate for in-plant piping was derived using factors derived from previously built projects. These factors are a percentage of the mechanical equipment supply and are calculated per individual plant area. The plant piping costs allow for the supply and installation of pipe, fittings, mountings and manual valves.

21.3.13 Overland Pipework

The overland piping, i.e. raw water supply, tailings discharge line and decant water return line were quantified based on material take-offs.

Supply Rates and Installation Cost for this estimate were based on responses to BQRs from regional subcontractors with experience on this kind of work and capacity to perform the works.

21.3.14 Electrical and Instrumentation

The supply of electrical equipment was estimated in detail and compiled using electrical equipment lists, loads lists, GA drawings and supplier pricing. The Plant Control System was quoted by reputable suppliers and the major instruments and valves were estimated using first principles engineering.

The supply and installation estimate for electrical and instrumentation bulks was estimated using factors derived from previously built projects.

21.3.15 Erection and Installation

Included in the discipline by discipline assessment of erection/installation costs detailed above, allowances were made for major construction cranage and equipment and construction costs such as site establishment, construction personnel meals, accommodation, transportation from/to site, flights and fuel usage, etc.

21.3.16 Architectural/Buildings

Budget pricing for prefabricated and steel frame buildings were sourced from reputable suppliers based on preliminary layout drawings.

21.3.17 Transport

The transport costs included in the estimate are based on factors on supply costs and benchmarked against detailed drawings for similar sized projects completed by Lycopodium.

21.3.18 EPCM

The EPCM estimate was based on a first principle build-up of costs based on the assessed scope of work managed by the EPCM Engineer for the Project and is based on the EPCM controlled scope.

Expenses such as catering and accommodation for the Engineer's site personnel are included in the estimate.

21.3.19 Vendor Representatives

Assessed by equipment package based on similar projects.

21.3.20 Qualifications/Clarifications

The estimate is subject to the following qualifications and clarifications:

- All labour rates, materials and equipment supply costs are current as of Q2 2018. Contingency has been allowed based on the quality of the information, however no allowance for escalation has been included.
- Construction contractor rates include mobile equipment, vehicles, fuel, construction power and consumables for the duration of construction. Potable water and raw water supply will be provided by the Company and available at site for use by contractors.
- Accommodation, meals and mobilization/demobilization/R&R flights of construction contractor personnel are incorporated in the contractor in-direct labour rates on the basis of individual contractors making their own accommodation arrangements.
- Meals and accommodation for the Owner’s and EPCM teams have been allowed in the estimate.
- Project spares are a percentage allowance of the mechanical supply cost based on similar size projects.
- A commissioning assistance crew is allowed for in the EPCM allowance.
- PLC programming for the process plant has been allowed for in the instrumentation and control budget and not the EPCM budget.

- Site supply of power and raw water (for operations and construction), sewage removal and treatment, communications network for construction facilities are included in the infrastructure costs.

21.3.21 Owner's Costs

The following items are included in the Owner's costs:

- Owner's project management team (labour and expenses).
- Owner's Consultants.
- Health Safety Environment and Community.
- Site IT and Communication.
- Mobile Equipment.
- Owner’s travel expenses.
- Operation Readiness.
- Commissioning and Ramp-up.
- Corporate Administration.
- Condemnation Drilling.

21.3.22 Spares

Spares have been estimated based on the Critical Spares List. Estimate pricing has been derived from a combination of quoted rates and factors of equipment supply and benchmarked against the spares expenditure on projects of a comparable scale.

The approach assumed is that a minimal quantity of spares will be purchased at the outset of operations with spares stocks progressively expanded during operations.

21.3.23 First Fill and Opening Stocks of Consumables

Quantities for opening stocks and first fill consumables have been assembled from basic principles and using the Project design criteria. Unit rates are based on budget quotations solicited from suitable suppliers.

21.3.24 Contingency

The purpose of contingency is 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 is an integral part of an estimate and has been applied (after careful analysis) to all parts of the estimate on a line by line basis, i.e. direct costs, indirect costs, services costs, etc.

21.4 Operating Costs

21.4.1 Introduction

The project operating cost estimate is built-up from three components:

- The mine operating costs developed by AGP.
- The process plant operating costs developed by Lycopodium.
- The General and Administration (G&A) operating costs developed by IAMGOLD.

The estimated life of mine operating cost per tonne of ore processed is summarized in Table 21.12.

Table 21.12 Life of Mine Operating Costs (Q2 2018)

	Total Cost ($M) from first gold pour	$/t Processed
Mining	$456	$13.01/t
Processing	$528	$15.04/t
G&A	$148	$4.22/t
Total Cash Cost	**$1,132**	**$32.27/t**

The foreign exchange rates summarized in Table 21.3 were used to develop the operating costs.

21.4.2 Mining Operating Costs

Mining costs were estimated based on the hourly costs of the various equipment, equipment utilization rates and productivity assumptions. The mine operating costs by total material and per tonne of ore are illustrated in Table 21.13 and Table 21.14.

Table 21.13 Open Pit Mine Operating Costs ($/t total material)

Open Pit Operating Category	Unit	Year 1	Year 5	Year 10	Year 1 - 13 Average Cost
General Mine and Engineering	$/t	0.33	0.26	0.67	0.34
Drilling	$/t	0.05	0.17	0.18	0.14
Blasting	$/t	0.14	0.37	0.52	0.33
Loading	$/t	0.21	0.18	0.21	0.19
Hauling	$/t	0.36	0.47	0.76	0.50
Support	$/t	0.24	0.24	0.54	0.29
Grade Control	$/t	0.06	0.07	0.24	0.09
Dewatering	$/t	0.01	0.02	0.06	0.02
Contract Services	$/t	0.68	-	-	0.21
Total	**$/t**	**2.07**	**1.78**	**3.18**	**2.11**

Table 21.14 Open Pit Mine Operating Costs ($/t Ore)

Open Pit Operating Category	Unit	Year 1	Year 5	Year 10	Year 1 - 13 Average Cost
General Mine and Engineering	$/t	3.10	2.63	1.67	2.24
Drilling	$/t	0.43	1.69	0.46	0.91
Blasting	$/t	1.27	3.76	1.31	2.07
Loading	$/t	1.91	1.81	0.52	1.23
Hauling	$/t	2.93	4.65	1.89	3.07
Support	$/t	2.09	2.37	1.36	1.85
Grade Control	$/t	0.57	0.67	0.60	0.59
Dewatering	$/t	0.10	0.17	0.16	0.15
Contract Services	$/t	6.31	-	-	1.32
Total	**$/t**	**18.71**	**17.75**	**7.96**	**13.43**

Mine Operating Costs

Diesel is estimated based on equipment hourly consumption with a price of 365 CFA/L provided by IAMGOLD. There is a separate fuel delivery fee that is carried in the mining General and Mine Engineering cost category. This covers the cost of fuel storage and delivery to the equipment reducing the need for the mine to have fuel trucks or manpower to deliver the fuel.

Labour costs for the various job classifications were obtained from IAMGOLD. These rates were used and include the appropriate burden for each category to cover items such as health care, vacation and federal holidays. The mine labour is based on a 12-hour shift schedule with three crews.

The mine labour includes a mixture of expatriate and local labour. The expatriate workforce is scheduled to train the local workforce and then be reduced over time. The only expats that are expected to remain on site over the mine life is that of the Superintendent of Mine Operations.

Year 6 is the peak in manpower but the staff manpower remains fairly constant through this period. Year 2 manpower levels are shown in Table 21.15.

Table 21.15 Mine Staffing Requirements and Annual Employee Salaries (Year 2)

Staff Position	Employees	Annual Salary $/y
Mine Maintenance		
Superintendent – Maintenance	1	$305,900
Coordinator – HME/Drill/Ancil.	3	$241,400
Coordinator – Maintenance Planning	2	$38,500
Supervisor – HME/Drill/Ancil.	14	$27,700
Leadhand – Trades Trainer	1	$23,600
Subtotal	**21**	
Mine Operations		
Mine Manager	1	$437,900
Superintendent – Mine Operations	1	$305,900
Coordinator – D&B or Mining	2	$241,400
Supervisor – D&B or Mining	6	$27,700
Leadhand – Drill/Mine Trainer	6	$23,600
Leadhand – Mine Operations	6	$17,900
Subtotal	**22**	
Mine Engineering		
Superintendent – Technical Services	1	$305,900
Coordinator – Senior Mine Engineer	1	$241,400
Supervisor – Mine Engineer	3	$27,700
Supervisor – Survey	2	$23,400
Leadhand – Mining Technician	3	$14,000
Leadhand - Surveyor	9	$19,700
Subtotal	**19**	
Geology		
Coordinator – Senior Geologist	1	$38,500
Supervisor – Geologist	3	$27,700

Staff Position	Employees	Annual Salary $/y
Leadhand – Geology Technician	6	$17,900
Subtotal	**10**	
Total Mine Staff	**72**	

Table 21.16 Hourly Personnel Requirements and Annual Salaries (Year 2)

Hourly Position	Employees	Annual Salary $/y
Mine Operations		
Driller	12	$16,500
Blaster	-	-
Blaster's Helper	-	-
Loader Operator	3	$16,500
Excavator Operator	15	$16,500
Haul Truck Driver	58	$16,500
Dozer Operator	12	$16,500
Grader Operator	6	$16,500
Water Truck Driver	4	$16,500
Transfer Loader/Backhoe	7	$16,500
Subtotal	**117**	
Mine Maintenance		
HME Trades	15	$23,100
Drill Maintenance Trades	33	$23,100
Apprentice/Ancillary	8	$18,100
Subtotal	**56**	
Total Hourly	**172**	

Labour costs are based on an owner operated scenario for Malikoundi. Blasting will be handled by the explosives contractor who will also load and shot the holes. Tire maintenance is handled by a tire contractor who is responsible for changing the tires. Fuel is also under contract and delivered to the equipment in the field.

Drilling labour force is based on one operator per drill per crew while operating. This peaks at 21 drillers in Year 5 then drops down over time as the drilling hours are diminished. Maintenance factors are used to determine the number of HME Trades, Drill Maintenance Trades and apprentices required. The number of people required is a factor based on the number of operators. The factors for the various mine cost categories are summarized in Table 21.17.

Table 21.17 Maintenance Labour Factors (Maintenance per Operator)

Maintenance Job Class	Drilling	Loading	Hauling	Mine Operations Support
HME Trades	0.25	0.25	0.25	0.25
Drill Maintenance Trades	0.25	0.25	0.25	0.25
Apprentice/Ancillary	0.05	0.01	-	0.25

The number of excavator, loader, truck and support equipment operators is estimated using the projected equipment operating hours. The maximum number of employees is 3 per unit to match the mine operating crew schedule. This schedule is comprised of 2 each 12-hour shifts per day requiring 3 operating crews.

Repair and maintenance (R&M) costs for each piece of equipment were provided by vendor. The costs for the R&M are expressed in a $/hr form. Fuel consumption rates are based on vendor supplied data and estimated for the conditions expected at the Boto Gold Project. R&M and fuel costs are used in the detail costs for the mine equipment.

AGP solicited purchase costs from local vendors for various tire sizes that will be used during the Project. Estimates of the tire life are based on AGP experience and information from other mine operators. The operating cost of the tires is expressed in a $/h form. The life of the haulage truck tires is estimated at 5,000 hours per tire for the rigid frame trucks with proper rotation from front to back. On the rigid frame trucks (95 tonne) each tire costs $13,100 so the cost/h for tires is $15.67/hr for the truck using the six tires in the calculation.

Ground Engaging Tools (GET) costing is estimated from other projects. This is an area of cost that is expected to be fine-tuned with mine operations.

Drill consumables were estimated as a complete drill string using the parts list and component lives provided by the vendor. The bit life has been estimated at 600 m per bit. Drill productivity for both ore and waste is estimated at 17.5 m/h for the 5 m bench holes and 24.1 m/h for the 10 m bench holes.

Lubricants, oil and filters are estimated as a percentage of the fuel cost per hour. The summary of the equipment operating costs is shown in Table 21.18.

Table 21.18 Equipment Operating Costs – No Labour ($/h)

Equipment	Fuel	Lube/ Oil	Tires	R&M	GET/ Consumables	Total
Production Drill – (154 mm)	$32.70	$3.27	-	$46.36	$46.09	$128.41
Production Drill – (104 mm)	$32.70	$3.27	-	$46.36	$30.39	$112.71
Hydraulic Excavator (15m^3)	$114.78	$11.48	-	$105.11	$15.00	$246.37
Front End Loader (13m^3)	$50.18	$5.02	$23.57	$53.00	$11.00	$142.77
Hydraulic Excavator (6.7m^3)	$40.91	$8.18	-	$58.30	$8.00	$115.39
Haulage Truck – 95 t	$49.24	$4.92	$15.67	$36.20	$5.00	$111.03
Track Dozer	$51.85	$5.19	-	$48.60	$5.00	$110.64
Grader	$19.42	$1.94	$5.18	$24.10	$5.00	$55.65
Support Backhoe	$19.35	$1.94	-	$29.10	$5.00	$55.39

Drilling in the open pit will be performed using down the hole hammer drills with two different bit sizes depending on the bench being drilled. The pattern size is the same for ore and waste but varies by the bit size. The drill pattern specifications are shown in Table 21.19. There is a slight expansion in the patterns for laterite/saprolite and transition but the majority of the material is rock.

Table 21.19 Drill Pattern Specifications - Rock

Specification	Unit	Ore/Waste	Ore/Waste
Bench Height	m	5	10
Sub-Drill	m	0.9	1.3
Blasthole Diameter	mm	104	154
Pattern Spacing – Staggered	m	3.3	5.1
Pattern Burden – Staggered	m	2.9	4.4
Hole Depth	m	5.9	11.3

The sub-drill was included to allow for caving of the holes in the weaker zones, reducing re-drilling of the holes or short holes that would affect bench floor conditions. Proper floor breakage will improve travel times and decrease tire and overall maintenance costs.

Table 21.20 outlines the parameters used for estimating drill productivity. This is based on studies of these parameters at similar operations and vendor supplied information. Due to the bench heights and the specifications of the drill, additional steel will be required from the carousel.

Table 21.20 Drill Productivity Criteria - Rock

Drill Activity	Unit	Ore/Waste	Ore/Waste
Bit Size	mm	104	154
Pure Penetration Rate	m/min	0.40	0.50
Hole Depth	m	5.9	11.3
Drill Time	min	14.75	22.60
Move, Spot, and Collar Blasthole	min	3.00	3.00
Level Drill	min	0.50	0.50
Add Steel	min	0.50	0.50
Pull Drill Rods	min	1.50	1.50
Total Setup/Breakdown Time	min	5.50	5.50
Total Drill Time per Hole	min	20.3	28.1
Drill Productivity	m/h	17.5	24.1

An emulsion product will be used for blasting to provide water protection. With the high rainfall it is expected that a water-resistant explosive will be required. The powder factors used in the explosives calculation are shown in Table 21.21.

Table 21.21 Design Powder Factors - Rock

	Unit	Ore/Waste	Ore/Waste
Drill Bit Size	mm	104	154
Powder Factor	kg/m^3	0.827	0.818
Powder Factor	kg/t	0.301	0.296

Presplit holes will be drilled along the rock walls as the pit depth deepens. The burden of those holes will be 2.2 m with a spacing of 1.2 m. These will have a charge factor of 0.5 kg/m^2 of blast face.

The blasting cost comes from quotations of the local vendors. They are responsible for the explosives plant, magazines, loading and shooting of the holes. The price per kg of emulsion provided was $98.07/100 kg. In addition, there is a monthly fee that covers all other cost components of blasting and their support equipment.

Ore and waste loading costs were estimated using the front-end loaders and excavators as the only loading units. The smaller excavators are primarily for the ore loading to reduce dilution but have been scheduled for some waste material movement as required. The excavators are responsible for the majority of the waste rock movement.

The average percentage of each material type that the various loaders is responsible for is also shown in Table 21.22.

Table 21.22 Loading Parameters – Year 2

	Unit	Hydraulic Excavator	Front-End Loader	Hydraulic Excavator
Bucket Capacity	m^3	15	13	6.7
Truck Capacity Loaded	t	95	95	95
Waste Tonnage Loaded	%	76	2	22
Ore Tonnage Mined	%	73	2	25
Bucket Fill Factor	%	95	95	95
Cycle Time	Sec	38	40	38
Trucks Present at the Loading Unit	%	80	80	80
Loading Time	min	2.60	2.70	5.13

Haulage profiles were developed for each pit phase and destination by end of period plans. These profiles were used in Alastri's Haul Infinity to determine haul cycle times for each of these hauls and then used in the cost model. Various limits were applied to the haulage profiles to consider rolling resistance, up or down hauls and congested area speeds. These are shown in Table 21.23.

Table 21.23 Haul Infinity Design Parameters

	Rolling Resistance %	Distance Applied m	Empty Uphill km/h	Empty Downhill km/h	Flat km/h	Loaded Uphill km/h	Loaded Downhill km/h
Working Face	10	50	-	-	30	-	-
Roads/Ramps	3	full	35	35	50	35	30
Waste Dump	8	300	35	35	30	35	30
Crusher	3	100	-	-	30	-	-

The distance applied refers to the length of that particular segment of the haulage profile. In the case of the working face, it is the first 50 m near the shovel or loader. On the waste dump, it is the last 300 m of the dump as the truck approaches to dump their load.

Support equipment hours and costs are allocated using the percentages as shown in Table 21.24.

Table 21.24 Support Equipment Operating Factors

Mine Equipment	Factor %	Factor Units
Track Dozer	25	Of haulage hours to a maximum of 5 dozers
Grader	10	Of haulage hours to a maximum of 3 graders
Transfer Loader (992/WA900)	14	Hours per day per unit – outside the fence
Crushed Pile Loader (988/WA600)	12	Hours per day per unit – inside the fence
Support Backhoe	7	Of loading hours to a maximum of 2 backhoes
Water Truck	7	Of truck hours
Tire Manipulator	-	Handled by tire contractor
Lube/Fuel Truck	-	Handled by fuel contractor
Mechanic's Truck	10	hours/day
Welding Truck	10	hours/day
Integrated Tool Carrier	3	hours/day
Compactor	8.5	hours/day
Lighting Plants	12	hours/day
Pickup Trucks	4	hours/day
Dump Truck – 20 ton	5	hours/day

These percentages resulted in the need for five track dozers, three graders and one support backhoe. Dozer tasks include clean-up of the excavator faces, construction and maintenance of roads, dumps and blast patterns and ripping of competent rock that is too thin to be blasted. The graders will maintain the ore and waste haul routes. In addition, water trucks have responsibility for patrolling the haul roads and controlling fugitive dust for safety and environmental reasons. The small backhoes will be responsible for cleaning out sedimentation ponds and water ditch repairs.

The transfer loader is full time at the primary crusher tramming material in to maintain full capacity of the crusher. The crushed pile loader is inside the fence and is used to tram material on the crushed pile and load onto the belt as required for maintenance or any other issues.

These hours are applied to the individual operating costs for each piece of equipment. Many of these units are support equipment so no direct labour force is allocated to them due to their function.

Grade Control

Reverse circulation drilling and sampling will be employed as the method of ore definition due to the steep inclination of the mineralized zone. The RC program will work in advance of mining using tight inclined drillhole spacing to accurately define the ore/waste contacts. This ore-waste boundary information is then built into the short- range models and then marked in the field to guide the loading equipment. This practice is widespread in Australia with great success and also in Canada and Africa.

The method involves using a dedicated grade control drill rig and crew in the pit to drill a series of shallow inclined holes drilled in a pattern similar to the blast hole pattern. For this study, that is assumed to be a contract drill rig and team with experience in this type of drilling. The grade control drill will be drilling the saprolite where blast holes are not required. The pattern for drilling will be a 10 m spacing and a 5 m burden with samples taken every 1 m in presumed mineralized zones as outlined by both previous ore control drilling and the exploration drilling. The volume of the sample to be assayed and sample intervals will need be determined in a grade deportation study in subsequent studies. An additional 25% will be drilled along the contacts to ensure that unknown structures are not missed in the saprolite. These “waste” samples will be drilled with a 20 m spacing and 10 m burden and sampled over 6 m.

The amount of reverse circulation drilling peaks in Year 2 at 33,200 m then drops off after that averaging 22,300 m/y from Year 2 until Year 12. No ore control drilling is required after Year 12.

The reverse circulation drills will operate for 24 h/d to minimize disturbance and be in advance of mine operations with the information. A three-man crew per drill is required; one driller and two drill helpers. In addition, geologists will provide guidance throughout the day and be on call if unknown issues arise.

The drill penetration rate is estimated at 10m/h with setups, sampling, etc. Overall, the cost for the RC drill program with contract labour is estimated at $63/m. Sampling and assay costs are estimated from laboratory quotes at $14.25/sample.

The data from the grade control drilling is then interpreted by the geologist and the ore is then remodelled. The production drilling and blasting will then be designed to mine the ore material separately from the waste.

Dewatering

Pit dewatering is a significant work function at the Boto Gold Project. Efficient and cost-effective dewatering will be crucial to the success of the Boto Gold Mine project development and slope stability. Dewatered slopes allow a reduction in the strip ratio by permitting steeper inter-ramp angles that are also be inherently safer.

Initial estimates of the daily pumping requirements were made while waiting for additional testwork. It is estimated that on average in the initial years 3,700 m^3/day will need to be pumped from within the pit to the pit rim with peak of 8,200 m^3/day. As the mine expands its footprint and goes deeper those numbers rise to 5,100 m^3/day on average with a peak of 11,200 m^3/day. This water will be pumped to the pit rim and discharged to the FWP or discharged if the water quality is met.

Storm events are common and have the potential to impact mining operations. Pumping rates as high as 11,200 m^3/day of pumping may be required for several weeks to recover from one of these events. The capital cost has considered this in the estimation for the number of pumps required on site to handle such an event.

The dewatering cost estimate is broken into two components:

- In-pit.
- Ex-pit.

In-pit includes the pumps, sumps, pipelines responsible for moving water from the pit to the pit rim and any additional items. Labour for this is already included in the General and Mine Engineering category of the mine operating cost.

Ex-pit gravity will be employed in the surrounding ditches to transport the water from the pits to the various discharge points around the mine property. In some instances, water in this ditch may require transfer to another discharge point and that has been allowed for in the cost estimate.

The in-pit pumps, like the ex-pit pumps are the same size to allow them to be interchanged as required. All the pumps are diesel powered.

The pumps required changes as the pit deepens with 2 pumps will be required for normal operation in the pit. Storm events would require up to 4 pumps for short periods of time. The operating parameters are shown in Table 21.25.

Table 21.25 Dewatering Parameters – Year 2

Parameter	Units	In-Pit (Normal)	In-Pit (Storm)
Average Pumping			
Days	days per year	365	365
Pumping Rate	m^3/day	5,100	11,200
Pump Capacity	m^3/hr/unit	160	160
Pump Working time	hrs/day	16	20
Pumps required	units	2	4

The dewatering operating cost works out to \$0.01/t moved over the period of Year 1 to 13.

21.4.3 Process Plant Operating Costs

Overview

The process plant operating costs have been developed based on an ore processing rate of 2.75 Mtpa. The plant will normally operate 24 hrs/day for 365 d/y with 75% (6,570 h/y) crushing plant utilization and 92% milling plant utilization (nominal 8,059 h/y).

The operating cost estimates are expressed in United States Dollars in Q2 2018 terms and are deemed to have an overall accuracy of ±15%.

The process operating costs for the Project have been developed according to typical industry standards applicable to gold ore processing plants.

Quantities and cost data were compiled from a variety of sources including:

- Metallurgical testwork.
- Consumables prices from IAMGOLD.
- Advice from IAMGOLD.
- Lycopodium data.
- First principles.

Qualifications and Exclusions

The process operating cost estimate includes all direct costs associated with the project to allow production of doré. Each cost estimate is presented with the following exclusions:

- All taxes and import duties.
- Any impact of foreign exchange rate fluctuations.
- Any business interruption costs.
- Any escalation beyond the date of the estimate.
- First fill and opening stocks costs (included in the capital cost estimate and financial model).
- Tailings storage, rehabilitation or closure costs (included in the capital cost estimate).
- Land lease or other compensation costs.
- Product costs (transportation, refining, marketing and insurance), however, are included in the financial model.
- Licence fees or royalties (included in cash flow model).
- Contingency.

Cost Categories

The operating cost estimate includes the following major categories as defined below:

1. Operating Consumables.
2. Plant Maintenance.
3. Power.
4. Plant Laboratory.
5. Labour (Operation and Maintenance).

A description of each cost category is provided in the following sections.

Operating Consumables

The consumables category covers all wear parts, consumable materials, reagents, fuel used to operate the process plant. Consumption and cost per tonne rates for consumables and reagents are summarized in Table 21.26. The tables are based on the following:

- Comminution consumables (crusher liners, mill liners and grinding media) were evaluated based on comminution testwork performed on various samples representing the lithologies in the Boto ore bodies. Crusher liner, SAG mill liner and steel ball consumption rates are based on calculations and vendor supplied data.

- Laboratory testwork results are used to determine the reagent consumption rates for the most costly reagents (cyanide, lime, lead nitrate and oxygen). In the absence of testwork data, reagent consumption rates are assumed based on first principle calculations, Lycopodium experience and generally accepted practice within the industry.

- Diesel fuel consumption rates for the mobile equipment and plant operations (elution heating, carbon regeneration and gold smelting) are based on first principles calculations and Lycopodium experience.

- Fuel costs of CFA 365.176/L for diesel and CFA 328.329/L for HFO delivered to site was provided by IAMGOLD.

- Consumables and reagents pricing were provided by IAMGOLD.

- Laboratory costs are based on a contract laboratory at site.

Table 21.26 Consumables Cost Summary

Area	Consumption Rate	Consumables Cost $/y	Consumables Cost $/t
Primary crusher liners and related		$212,431	$0.08
SAG Mill liners	0.24 kg/t	$2,174,653	$0.79
SAG Mill balls	1.47 kg/t	$5,117,763	$1.86
Cyclone parts and screen panels		$173,217	$0.06
Sodium cyanide	0.27 kg/t	$1,732,692	$0.63
Quicklime	1.93 kg/t	$2,487,249	$0.90
Lead nitrate	0.20 kg/t	$1,384,900	$0.50
Other reagents		$596,182	$0.22
EW and Gold room		$16,158	$0.01
Diesel	1,017 kL/y	$678,725	$0.25
Water treatment and lubricants		$94,158	$0.03
Total		**$14,668,128**	**$5.33**

Maintenance

Maintenance material costs were estimated by applying factors to the fixed capital investment in each area of the plant. Note that the fixed capital investment constitutes the installed cost of all equipment excluding civil works, foundations and concrete and indirect costs. Crusher and mill wear parts are included in the consumables allowance. The factors applied are based on Lycopodium's database and experience and are average costs over the life of the mine. As such, actual spares costs may be lower during the initial years but rise later. An overall factor of 3.7% is applied to the fixed capital investment for the process plant areas. The estimated annual maintenance cost for process plant and mobile equipment is $3.6 M. Note that the mobile equipment element includes the cost of leasing light vehicles as well as medium/heavy equipment. The maintenance costs are summarized in Table 21.27.

Table 21.27 Process Plant Maintenance Cost

Area	Maintenance Cost $/y	Maintenance Cost $/t
Process Plant spares	$2,217,721	$0.81
Mobile equipment lease and maintenance	$532,906	$0.19
Maintenance systems	$177,000	$0.06
Contracted maintenance	$696,614	$0.25
Total	**$3,624,240**	**$1.32**

Power

Electricity is generated on site. The power unit cost for the first five years of operation of $ 0.181/kWh is based on HFO and diesel fuel consumption rates and BOO costs. From year 6 onwards the fee for the photovoltaic solar plant decreases resulting in a reduced energy price of $0.172/kWh.

Electricity consumption estimate is based on the installed power excluding standby equipment. Electrical load factors and utilization factors are applied to the installed power to arrive at the annual average power draw, which is then multiplied by total hours operated per annum and the electricity price to obtain the annual cost.

Power consumptions and costs per plant area are summarized in Table 21.28. The estimated annual power cost for the site during the initial years of operation, including infrastructure and the permanent camp, is $ 20.69 M or $ 7.52/t of ore.

Table 21.28 Site Power Cost by Area (Years 1 to 5)

Area	Average Power (kW)	Annual Power Consumption (kWh)	Total Annual Power Cost ($)	Cost, $/t
TOTAL AREA 550 - POTABLE WATER	25	214,620	38,768	0.01
TOTAL AREA 551 - FIRE PROTECTION	2	21,199	3,829	0.00
TOTAL AREA 552 - SEWAGE TREATMENT & DISPOSAL	47	410,581	74,167	0.03
TOTAL AREA 604 - CRUSHING	192	1,683,584	304,119	0.11
TOTAL AREA 605 - ORE HANDLING	95	835,879	150,991	0.05
TOTAL AREA 610 - GRINDING	10,171	89,101,464	16,095,096	5.85
TOTAL AREA 620 - PRELEACH THICKENING	114	995,224	179,775	0.07
TOTAL AREA 625 - LEACHING & ADSORPTION	703	6,159,769	1,112,687	0.40
TOTAL AREA 630 - ACID WASH & ELUTION	20	173,273	31,300	0.01
TOTAL AREA 631 - CARBON REGENERATION & RECOVERY	2	17,345	3,133	0.00
TOTAL AREA 632 - ELECTROWINNING & REFINING	40	349,787	63,185	0.02
TOTAL AREA 645 - TAILS PUMPING	104	907,448	163,920	0.06
TOTAL AREA 650 - REAGENT PREPARATION & STORAGE	27	240,112	43,373	0.02
TOTAL AREA 655 - AIR SERVICES	399	3,497,255	631,737	0.23
TOTAL AREA 660 - WATER SERVICES	265	2,319,648	419,016	0.15
TOTAL AREA 805 - TAILINGS MANAGEMENT FACILITY	10	90,929	16,425	0.01
TOTAL AREA 810 - RECLAIM WATER (PROCESS WATER POND)	71	620,646	112,112	0.04
TOTAL AREA 820 - RAW WATER (FRESH WATER)	89	779,728	140,848	0.05
TOTAL AREA 830 - EVENT POND (POLLUTION POND)	1	8,410	1,519	0.00
TOTAL AREA 420 - ON-SITE POWER DISTRIBUTION - LSP	220	1,926,149	347,935	0.13
TOTAL AREA 500 - INFRASTRUCTURES - BUILDINGS	167	1,460,730	263,863	0.10
TOTAL AREA 300 - MINING	159	1,394,329	251,869	0.09
TOTAL AREA 515 - PERMANENT CAMP	154	1,349,040	243,688	0.09
TOTAL	**13,077**	**114,557,148**	**20,693,356**	**7.52**

The overall average power consumption is estimated at 13,077 kW. The installed power, maximum continuous draw and details pertaining to efficiency and utilization factors are provided in the electrical load list.

Plant Laboratory

The operating cost for the plant laboratory is based on the BOO (Build, own and operate) contract with 28% of the total contract cost allocated to the Process Plant. This equates to $ 319,173/year.

Process Plant Labour

The process labour burdened costs and complement numbers are provided by IAMGOLD. The estimated annual process plant labour cost is $3.85 M or $ 1.40/t ore. Table 21.29 summarizes the annual labour costs for each position.

Table 21.29 Process Plant Labour Compensation

Strata	Position	Number	Total $/y	Total $/t
Manager	Process	1	437,850	0.16
Superintendent	Metallurgical	1	305,932	0.11
	Process Maintenance	1	305,932	0.11
	Process Operations	1	305,932	0.11
Coordinator	Sr Production Metallurgist	1	241,385	0.09
	Sr Project Metallurgist	1	36,462	0.01
	Process Mechanical Maintenance	1	241,385	0.09
	Process Electrical Maintenance	1	241,385	0.09
	Power Plant	1	241,385	0.09
	Process Plant Operations	1	241,385	0.09
	Process Maintenance planning	2	72,925	0.03
Supervisor	Metallurgist	2	44,330	0.02
	Process Mechanical Maintenance	2	44,330	0.02
	Process Electrical Maintenance	2	44,330	0.02
	Power Plant	0	0	0.00
	Process Plant Operations	3	78,846	0.03
	Process Power Plant	3	78,846	0.03
Leadhand	Metallurgical Tech	3	50,837	0.02
	Process Trainer	2	44,673	0.02
	Trades Trainer	2	37,321	0.01
	Process Mechanical Maint Trades	9	119,425	0.04
	Process Electrical Maint Trades	9	119,425	0.04
	Power Plant Trades	9	119,425	0.04
	Process Plant Operations	3	50,837	0.02
	Control Room	3	50,837	0.02
Operator	Process	24	290,242	0.11
Total		**88**	**3,845,664**	1.40

Annualized Plant Operating Cost

During the initial years of operation the ROM blend will contain more saprolite and transition material than the LOM average for the ore body. These two ore types are significantly softer to grind and are also less abrasive. Consequently, it is expected that the grinding energy and grinding media consumption rates will be lower than average during the initial payback period. Table 21.30 below summarizes the estimated annual Plant operating cost linked to the mine plan.

Table 21.30 Annualized Plant OPEX by Year

Year	Tonnes Treated	% Saprolite and Transition	Grinding Steel Cost ($/yr)	SAG Power Cost ($/yr)	Total Annual Cost $ /y	Total $/t
1	2,589,584	24.1	6,020,509	13,263,348	37,694,259	14.56
2	2,750,000	30.0	6,025,874	13,402,610	39,175,197	14.25
3	2,750,000	30.0	6,024,752	13,202,398	39,879,923	14.50
4	2,750,000	30.0	6,026,019	13,295,725	39,974,519	14.54
5	2,750,000	10.0	7,273,834	15,162,173	43,088,779	15.67
6	2,750,000	13.6	7,046,476	14,138,881	41,838,131	15.21
7	2,750,000	11.6	7,172,928	14,291,525	42,117,226	15.32
8	2,750,000	10.0	7,273,833	14,292,434	42,219,040	15.35
9	2,750,000	10.0	7,273,834	14,311,089	42,237,697	15.36
10	2,750,000	10.0	7,273,834	14,226,954	42,153,561	15.33
11	2,750,000	6.4	7,510,574	14,666,785	42,830,132	15.57
12	2,750,000	0.0	7,934,906	15,184,081	43,771,761	15.92
13	2,222,333	0.0	6,412,358	12,270,565	32,444,028	14.60

21.4.4 General and Administration

The G&A costs were provided by IAMGOLD and include labour and expenses. The average G&A costs is $4.22/t processed.

22.0 ECONOMIC ANALYSIS

22.1 Introduction

An economic assessment of the Project was completed using a pre and after-tax cash flow model prepared by IAMGOLD. The model was structured using an EXCEL workbook which presents annual cash flows during the expected life of mine of the project. Parameters affecting the project cash flow are: production schedule, revenues, royalties, sustaining and initial capital requirements, operational costs, working capital, financing costs, mine closure costs and Senegalese fiscal regime. Previous costs related to the valuation of the project are estimated at $70.3 M and are considered in the financial analysis in terms of future tax depreciation.

The after-tax financial analysis and results are based on a NPV, the IRR, the payback period which is initiated at project approval (except for payback which refers to an amount of time after commercial production is declared) and the all-in sustaining costs (AISC) which is a gold industry standard in benchmarking costs per ounce of gold.

The costs were evaluated in United States Dollars. All amounts are in constant 2018 dollars, no provision is made for inflation nor increase in gold price. All cash flows are estimated on the project base solely and are excluding debt financing and a discount rate of 6% was used for the calculation of the NPV.

At the award of the ''permis d'exploitation'', a mining company will be created where the Government of Senegal will hold a 10% free carrying interest. IAMGOLD will hold the other 90% remaining interests. Hence the Senegalese government will receive 10% of all declared divided produced by the Boto mining company.

Input data was provided from a variety of sources, including the various consultants' contributions to this report, pricing obtained from external suppliers and contractors, and exchange rates and project specific financial data such as the expected project taxation regime. The assessment was based upon:

- Capital cost estimates and expenditure schedules prepared by Lycopodium and AGP.
- Mine schedule and mining operation cost estimates based on owner operated mining operations, as developed by AGP for the study.
- Process operating cost estimates prepared by Lycopodium, with contributions from IAMGOLD.
- General and administration cost estimates prepared by IAMGOLD.
- Metallurgical performance characterized by testwork completed for the study, managed by Lycopodium.
- Sustaining capital cost estimates for the mine development provided by AGP and TMF stage development provided by KP.
- Owner's capital cost estimates prepared by IAMGOLD.

- Royalty, tax, discount rates and other model inputs provided by IAMGOLD.
- Closure costs estimated by IAMGOLD.

22.2 Assumptions

Key economical parameters consist of: price of gold, price of hydrocarbons and exchange rate EUR/USD which dictate the value of the local currency since it is pegged to EUR (XOF). The financial model was built according to those assumptions and the ones stated below.

22.2.1 Metal Price

The price of gold considered for the revenue base is at $1,250/oz. No price variation was considered for this model.

22.2.2 Fiscal Regime

The corporative fiscal regime is specified by the Code Minier 2003-36 and in the mining convention between AGEM and the Government of the Republic of Senegal, which states the advantages given during the operation phase.

According to the mining convention, during the investment period including production start-up the producing company benefit from an exoneration of all import taxes and duties. Those exonerations starts of the delivery date of ‘’permis d’exploitation’’ or of the mining concession and are valid for a maximum of four years.

Finally, a seven years total tax exoneration is applied at year 1 of the project was applied for the purpose of this study, which is compliant with the mining convention. This exoneration applies on all corporate income tax as well as VAT on good and services procured locally or outside Senegal.

In the financial model, the corporate taxes are at 30% and applied on taxable benefits starting year 8 of the project based on the seven years tax exoneration. The corporate taxes for this project stands at $93.4 M and $856 k for other taxes.

22.2.3 Royalties

Royalties applied are standing at 3% of revenues, which is compliant to the Code Minier 2003-36. The royalties amount to $64.7 M.

22.2.4 Working Capital

This amount represents dues and incomes, inventory, storage fees and bullion inventory before payment. The working capital was estimated at $10.0 M which includes necessary inventories for spare parts, other inventories as well as initial cash requirements.

22.3 Financial Results

The LOM capital cost for the project is estimated at $320.5M, with an initial capital expenditure of $254.4M. Table 22.1 presents a summary of the production information on which the financial model is based, while the summary of the financial results is presented in Table 22.2.

Table 22.1 Production Summary

	Value
Ore milled	35.1 Mt
Total tonnes mined DMT	239.4 Mt
Average head grade	1.71 g/t Au
Contained gold in material	1 926 Moz
Total gold produced	1 724 Moz
Average gold recovery	89.5%
Production life (processing)	12.8 years
Nominal annual processing rate	2.75 Mtpa

Table 22.2 Initial Capital Cost Estimate Summary

Area	M$ (Excluding Duties and Taxes)
Direct Costs	
Mining	$62.1
Infrastructure	$19.0
Ore Handling & Processing	$57.4
Tailings & Water Management	$16.6
Sub-Total Direct Costs	**$155.1**
Indirect Costs	
Construction In-directs	$49.0
Owner's Costs	$19.2
Contingency	$24.3
Sub-Total Indirect Costs	**$92.5**
Sub-Total Initial Capital Costs	**$247.6**
Additional Indirect Costs	$6.8
Total Initial Capital Cost	**$254.4**

General

- Annual mined tonnage and head grade have been based on the mining schedule as presented in Section 16 and process plant throughput and production rates as presented in Section 17.
- The mining, processing and administration costs are based on the operating cost estimates presented in Section 21.
- Gold recovery is based on testwork and interpretation presented in Section 13 and Section 17.
- The initial capital cost at $254.4 M is based on the estimates presented in Section 21.

- Sustaining costs are $66 M.
- Closure costs of $22.6 M have been included.
- A salvaged value of $14.3 M was estimated at the end of mine life.
- No provision has been made for interest on cost of capital with exception of 5% for the purpose of calculation of income taxes.
- No provision has been made for escalation or inflation.

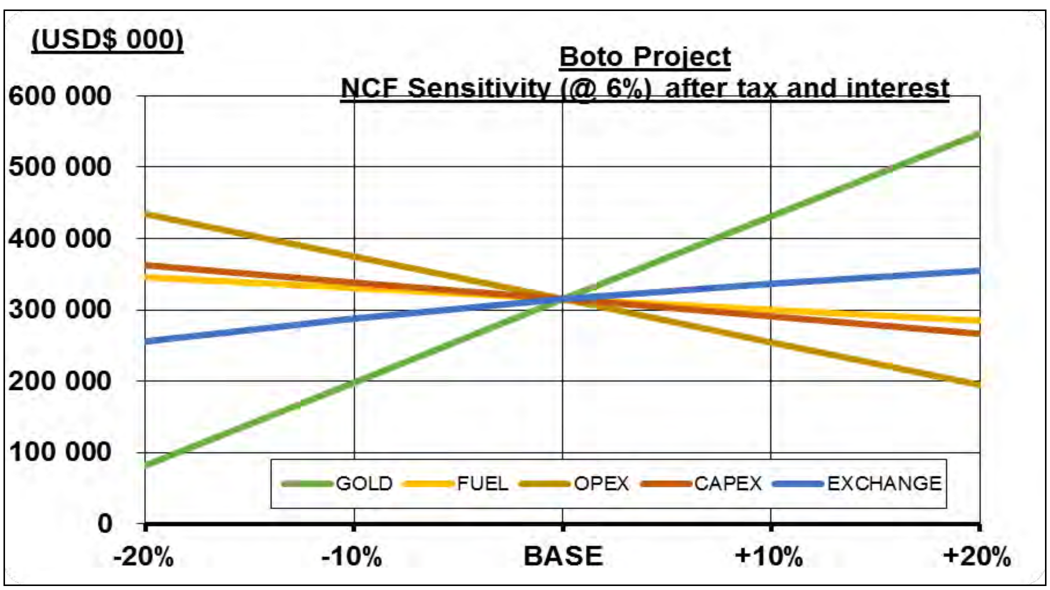
Table 22.3 After-tax Financial Results

AISC	$753/oz Au
IRR	23.0 %
NPV (6%)	$260.9 M
Payback	3.4 years

22.4 Sensitivity Analysis

A sensitivity analysis has been carried out on the base case scenario described above to assess the impact of changes on gold price, fuel price, operating costs and capital costs and currency exchange rates. The project is most sensitive to changes in gold price and then operating costs. The results of all sensitivity analyses are presented Figure 22.1.

Figure 22.1 NPV Sensitivity Analysis (After-tax)



22.5 Summary

The Table 22.4 presents a summary of the key results of the FS inclusive of financial results and Table 22.5 provides the financial model of the Project.

Table 22.4 Summary of the Main Results of the Project Feasibility Study

Gold Price	$1,250/oz
Tonnes mined (Mt)	239.4
Strip Ratio	5.8
Tonnes Processed	35.1
Average grade (g/t)	1.71
Gold contained (koz)	1,926
Gold produced (koz)	1,724
Process capacity (Mtpa)	2.75
Average recovery (%)	89.5
Total Revenue (M$)	2,155
Initial Capital (M$)	254.4
Sustaining Capital (M$)	66.0
Total Capital (M$)	320.5
Total Taxes (M$)	73.4
Royalties (M$)	64.7
Closure Costs (M$)	22.6
Salvage value (M$)	14.3
AISC ($/oz)	753
NPV at 6% after tax (M$)	260.9
IRR after tax (%)	23.0
Payback (years)	3.4

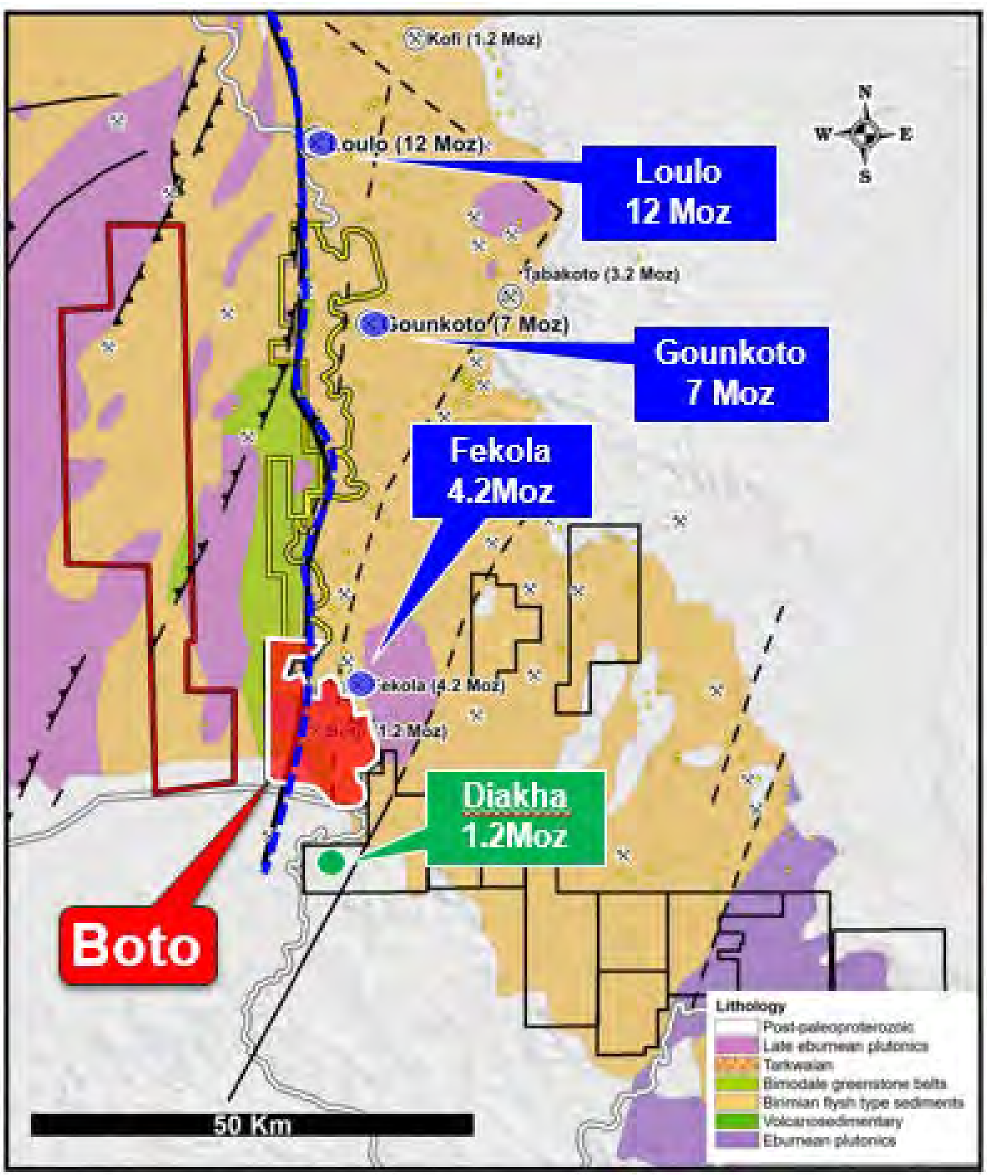
Table 22.5 Financial Model

Boto LOM Plan																	
Total Mined	*(kt)*	239 397	2 296	13 705	24 084	33 426	29 624	27 550	27 580	27 359	18 257	13 311	10 164	6 887	4 137	1 017	0
Strip Ratio		5,83	0,00	5,10	9,91	8,62	8,07	8,62	8,09	5,84	5,71	3,85	2,55	1,52	0,86	0,47	0,00
Total Milled	*(kt)*	35 062	0	0	2 590	2 750	2 750	2 750	2 750	2 750	2 750	2 750	2 750	2 750	2 750	2 750	2 222
Weighted Grade	*(g/t)*	-	0,00	0,00	2,34	2,27	1,67	1,93	1,81	2,14	1,58	1,39	1,79	1,74	1,59	1,08	0,72
Weighted Recovery Rate	*(%)*	-	0%	0%	90,3%	90,6%	90,5%	90,2%	89,4%	89,6%	89,2%	88,7%	89,2%	89,1%	88,9%	88,1%	86,9%
Total Gold Production	*(koz)*	1 724	0	0	176	182	133	154	143,1	169	125	109	141	137	125	84	45
Total Mining Revenue	***(US$'000)***	**$2 155 028**	**$0**	**$0**	**$219 040**	**$226 907**	**$166 719**	**$192 346**	**$178 921**	**$211 785**	**$155 945**	**$135 851**	**$176 855**	**$171 765**	**$156 563**	**$105 283**	**$57 048**
Mining Costs	*(US$'000)*	$456 217	$0	$0	$47 172	$62 986	$50 958	$45 528	$45 816	$50 269	$39 281	$33 463	$28 374	$21 686	$16 532	$10 762	$3 391
Processing Costs	*(US$'000)*	$527 543	$0	$0	$39 668	$41 536	$41 498	$41 077	$41 033	$41 717	$41 327	$41 317	$41 307	$41 307	$41 307	$41 317	$33 131
G&A Costs	*(US$'000)*	$148 117	$0	$0	$12 486	$12 237	$11 827	$11 396	$11 371	$11 486	$11 408	$11 393	$11 392	$11 376	$11 357	$11 296	$9 093
Total Operating Costs	***(US$'000)***	**$1 131 878**	**$0**	**$0**	**$99 326**	**$116 759**	**$104 282**	**$98 001**	**$98 220**	**$103 472**	**$92 015**	**$86 173**	**$81 073**	**$74 369**	**$69 197**	**$63 376**	**$45 615**
Gov't Royalty	*(US$'000)*	$64 651	$0	$0	$6 571	$6 807	$5 002	$5 770	$5 368	$6 354	$4 678	$4 076	$5 306	$5 153	$4 697	$3 158	$1 711
Operator Management Fee	*(US$'000)*	$21 550	$0	$0	$2 190	$2 269	$1 667	$1 923	$1 789	$2 118	$1 559	$1 359	$1 769	$1 718	$1 566	$1 053	$570
Patent, water, eletrical and other taxes	*(US$'000)*	$13 629	$0	$0	$1 025	$1 244	$1 252	$1 274	$1 098	$967	$967	$967	$967	$967	$967	$967	$967
Sustaining Capital	*(US$'000)*	$66 021	$0	$0	$11 211	$11 724	$7 390	$1 586	$7 668	$5 042	$7 874	$1 919	$7 881	$3 504	$221	$0	$0
Salvage Value	*(US$'000)*	($14 300)	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	($14 300)
Reclamation	*(US$'000)*	$22 623	$0	$0	$1 740	$1 740	$1 740	$1 740	$1 740	$1 740	$1 740	$1 740	$1 740	$1 740	$1 740	$1 740	$1 740
Operating Cash Flow	***(US$'000)***	**$1 023 150**	**$0**	**$0**	**$121 279**	**$112 091**	**$63 473**	**$94 833**	**$81 249**	**$110 825**	**$64 031**	**$49 891**	**$96 386**	**$97 708**	**$86 341**	**$37 918**	**$7 126**
Gov't Royalty	*(US$'000)*	($64 651)	$0	$0	($6 571)	($6 807)	($5 002)	($5 770)	($5 368)	($6 354)	($4 678)	($4 076)	($5 306)	($5 153)	($4 697)	($3 158)	($1 711)
Operator Management Fee	*(US$'000)*	($21 550)	$0	$0	($2 190)	($2 269)	($1 667)	($1 923)	($1 789)	($2 118)	($1 559)	($1 359)	($1 769)	($1 718)	($1 566)	($1 053)	($570)
Patent, water, eletrical and other taxes	*(US$'000)*	($13 629)	$0	$0	($1 025)	($1 244)	($1 252)	($1 274)	($1 098)	($967)	($967)	($967)	($967)	($967)	($967)	($967)	($967)
Retrenchment	*(US$'000)*	($1 148)	$0	$0	($88)	($88)	($88)	($88)	($88)	($88)	($88)	($88)	($88)	($88)	($88)	($88)	($88)
Salvage Value	*(US$'000)*	$14 300	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$14 300
Reclamation	*(US$'000)*	($21 475)	$0	$0	($1 652)	($1 652)	($1 652)	($1 652)	($1 652)	($1 652)	($1 652)	($1 652)	($1 652)	($1 652)	($1 652)	($1 652)	($1 652)
EBITDA	***(US$'000)***	**$914 998**	**$0**	**$0**	**$109 753**	**$100 031**	**$53 812**	**$84 125**	**$71 254**	**$99 646**	**$55 087**	**$41 748**	**$86 604**	**$88 129**	**$77 370**	**$31 000**	**$16 438**
Interest Expense	*(US$'000)*	($45 084)	($2 440)	($9 174)	($13 806)	($9 892)	($5 970)	($3 802)	$0	$0	$0	$0	$0	$0	$0	$0	$0
Taxable Income Before Depreciation	***(US$'000)***	**$869 914**	**($2 440)**	**($9 174)**	**$95 947**	**$90 139**	**$47 842**	**$80 323**	**$71 254**	**$99 646**	**$55 087**	**$41 748**	**$86 604**	**$88 129**	**$77 370**	**$31 000**	**$16 438**
Depreciation	*(US$'000)*	($399 819)	$0	$0	($27 817)	($28 836)	($30 008)	($30 829)	($31 028)	($32 123)	($32 963)	($34 538)	($35 018)	($37 645)	($39 397)	($39 618)	$0
Taxable Income Before Losses	***(US$'000)***	**$470 095**	**($2 440)**	**($9 174)**	**$68 130**	**$61 303**	**$17 834**	**$49 494**	**$40 226**	**$67 523**	**$22 124**	**$7 210**	**$51 587**	**$50 485**	**$37 973**	**($8 618)**	**$16 438**
Income Taxes	*(US$'000)*	($73 416)	$0	$0	$0	$0	$0	$0	$0	($20 257)	($6 637)	($2 163)	($15 476)	($15 145)	($11 392)	$0	($2 346)
Net Income	***(US$'000)***	**$396 679**	**($2 440)**	**($9 174)**	**$68 130**	**$61 303**	**$17 834**	**$49 494**	**$40 226**	**$47 266**	**$15 487**	**$5 047**	**$36 111**	**$35 339**	**$26 581**	**($8 618)**	**$14 092**
Free Cash Flow																	
EBITDA	*(US$'000)*	**$914 998**	**$0**	**$0**	**$109 753**	**$100 031**	**$53 812**	**$84 125**	**$71 254**	**$99 646**	**$55 087**	**$41 748**	**$86 604**	**$88 129**	**$77 370**	**$31 000**	**$16 438**
Add: Non-cash stockpile movement	*(US$'000)*	($0)	$0	$0	($1 565)	($1 943)	($1 037)	($487)	($548)	($2 512)	($101)	($213)	($604)	($312)	$1 026	$3 989	$4 307
Less: Interest Expense	*(US$'000)*	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Less: Income Taxes	*(US$'000)*	($73 416)	$0	$0	$0	$0	$0	$0	$0	($20 257)	($6 637)	($2 163)	($15 476)	($15 145)	($11 392)	$0	($2 346)
Less: Development Capital	*(US$'000)*	($254 511)	($97 588)	($156 923)	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Less: Sustaining Capital	*(US$'000)*	($66 021)	$0	$0	($11 211)	($11 724)	($7 390)	($1 586)	($7 668)	($5 042)	($7 874)	($1 919)	($7 881)	($3 504)	($221)	$0	$0
Less: Retrechment & Reclamation cash adjust.	*(US$'000)*	$0	$0	$0	$1 740	$1 468	$1 317	($176)	$1 317	$1 589	$1 589	$1 589	$1 589	$1 589	$1 438	$1 438	($2 165)
Less: Changes in Working Capital	*(US$'000)*	$0	$0	($10 000)	($6 628)	$512	$2 623	($1 744)	$747	($1 517)	$2 440	$785	($2 523)	($84)	$553	$7 495	$6 682
FCF	***(US$'000)***	**$521 049**	**($97 588)**	**($166 923)**	**$92 089**	**$88 344**	**$49 325**	**$80 131**	**$65 103**	**$71 908**	**$44 504**	**$39 828**	**$61 709**	**$70 673**	**$68 774**	**$43 922**	**$22 916**

23.0 ADJACENT PROPERTIES

The host rocks and observed structural setting demonstrated at the Project are also observed at many of the economic gold deposits located along the Senegal-Mali shear zone. Well established gold mines are situated along this trend such as: Fekola (B2Gold), Loulo and Gounkoto (Randgold) and Sadiola and Yatela (IAMGOLD) (Figure 23.1).

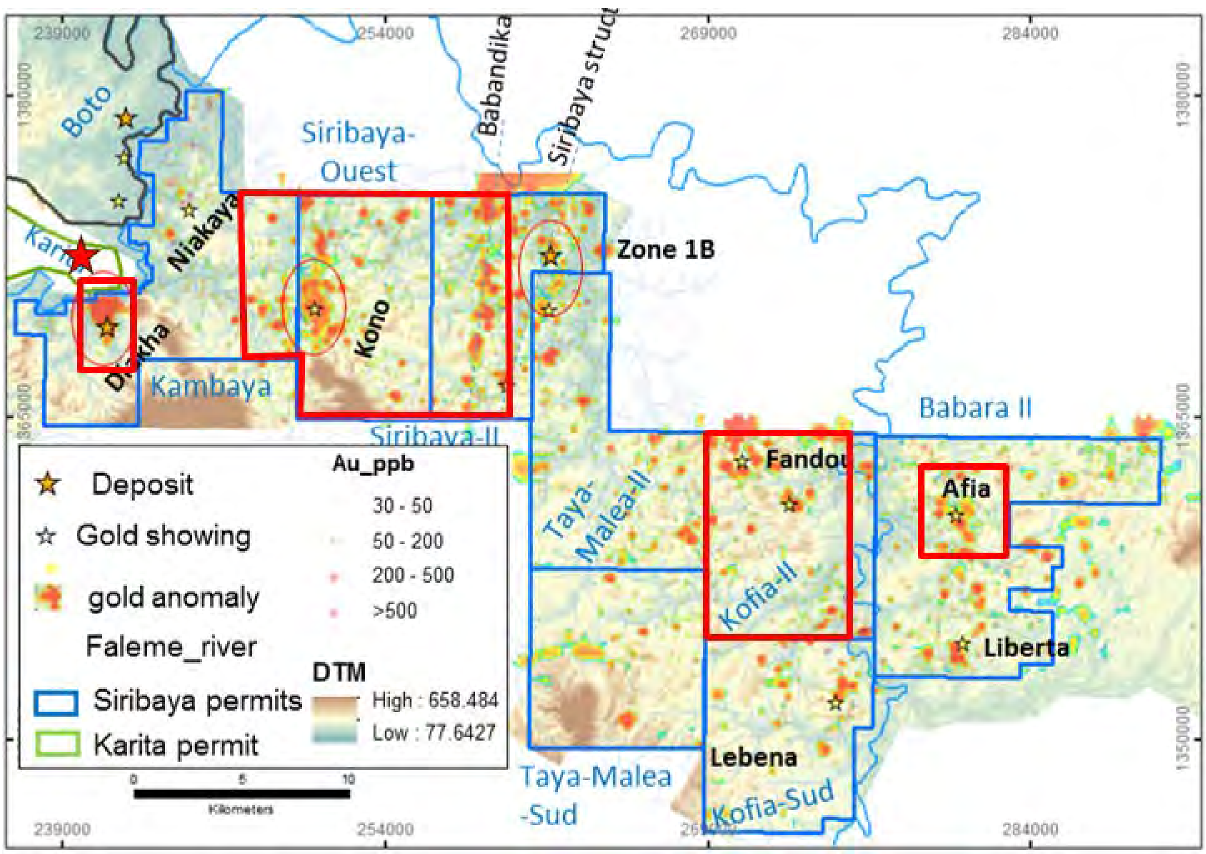
Figure 23.1 Adjacent Properties to the Boto Project



The most proximal and significant property to the Project is the operating Fekola gold mine located in southwest Mali and is owned and operated by B2Gold. The Fekola gold mine is situated approximately 6 km north and along strike with the Malikoundi/Boto 2 deposit. The most recent mineral resource estimate for the Fekola deposit reports, at a 0.6 g/t Au cut-off grade, Indicated Resources of 56.6 Mt at 2.04 g/t Au (inclusive of Reserves), and an Inferred Resource of 4.2 Mt at 1.69 g/t Au (B2Gold AIF, 23 March 2018; www.sedar.com; most recently viewed on 2 May 2018). This information is not necessarily indicative of the mineralization at the Project.

In Mali, IAMGOLD holds eight exploration permits covering 600 km^2 at the triple junction between Mali, Senegal and Guinea. A recent discovery has been made on the Fekola-Malikoundi trend, known as Diakha Project with reported Inferred mineral resources of 14.8 Mt at 1.81 g/t Au (RPA, 2015). Exploration is still ongoing on the project with step out and infill drilling at Diakha as well as some sub-surface sampling through the project area.

Figure 23.2 IAMGOLD Properties in Mali



In Guinea, IAMGOLD holds an exploration license located south of the Project and north of Diakha in Mali. A termite mount sampling has been performed and an outstanding anomaly determined. This anomaly is being RC drilled at the moment.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation and Schedule

The implementation strategy for the Project is based on an engineering, procurement and construction management (EPCM) implementation approach and horizontal discipline-based contract packaging. Horizontal packages are for the earthworks, building works, concrete works, field erected tankage, structural, mechanical and piping installation, electrical and instrumentation supply and installation. An experienced engineering firm will be engaged to provide engineering and procurement services for the development of the process plant and the associated infrastructure, and provide construction management services as part of an integrated team with IAMGOLD for the development of the Project.

The initial strategy for implementation of the Project is to commence early works to enable access to site year-round and to meet the first gold pour end of 2020. Early works includes upgrading the main access road and main bridges, commencing engineering related to long lead items and finalizing contracts for immediate award following board approval.

Lycopodium has developed a preliminary project schedule for the execution of the detailed engineering, procurement and construction of the Project in accordance with IAMGOLD Work Breakdown Structure.

The major preliminary Project Milestones are provided as follows:

- FS Completion 28-Sep-18
- Environmental Permit 15-Oct-18
- Exploitation Permit Application 02-Nov-18
- Approval for Early Engineering 2-Jan-19
- Project Execution Plan Complete 30-Jan-19
- Reception of Exploitation Permit 1-Mar-19
- IAMGOLD Board Approval 1-Apr-19
- EPCM Award/Commence Project 2-Apr-19
- SAG Mill PO Award 9-Apr-19
- Access Road Completion - Two Bridges 2-Jul-19
- Start of Mining 1-Oct-19
- Start of Process Plant Construction 4-Nov-19
- Engineering and Design Complete 27-Jan-20
- SAG Mill Delivered to Site 24-Jun-20
- Plant Concrete Works Completed 30-Jun-20
- Fuel Facility Operational 30-Jun-20
- Power Plant Operational - Power On 23-Jun-20
- Process Plant Practical Completion 19-Oct-20
- TMF Completion 30-Nov-20
- First Gold Pour 9-Dec-20
- EPCM Project Complete 25-Dec-20

The schedule has been set-up in sufficient detail to demonstrate the logical steps required to progress the work scope to conclusion and be organized according to the work packages within the WBS of the project. The schedule also identifies the critical path.

Activities of the schedule are based on the engineering deliverables, procurement and contract quotations (Budgetary and Firm pricing) and construction scope which have been identified during the Feasibility Study.

Following equipment packages are identified as major equipment and/or Long Lead:

Package	Description	Lead Time (weeks)
Mechanical Equipment:		
SEBO-3-MS-101	SAG Mill	53
SEBO-3-MS-102	Jaw Crusher	20
SEBO-3-MS-103	Apron Feeders	30
SEBO-3-MS-104	Belt Conveyors Package	26
SEBO-3-MS-110	Pre-leach Thickener	28
SEBO-3-MS-112	Mill Liner Handler	45
SEBO-3-MS-117	Agitators	28
SEBO-3-MS-123	Slurry Pumps	32
SEBO-3-MS-126	CIP (Pumpcells)	36
SEBO-3-MS-109	Cyclone Clusters	25
Electrical Equipment:		
SEBO-3-ES-200	Transformers	20
SEBO-3-ES-204	HV Switchgear	24
SEBO-3-ES-203	MCCs	19

The critical path for the Project schedule runs through following activities:

- Completion of two bridges for Access Road.
- SAG Mill tender and procurement.
- SAG Mill manufacture and delivery.
- SAG Mill on-site installation
- Grinding area SMP installation.
- Grinding area E&I installation.
- Commissioning.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Geology

The Project is located in the West African Craton (WAC), in the south-eastern part of the Early Proterozoic formation of the Kédougou-Kéniéba inlier, which covers the eastern part of Senegal and western Mali. At Boto, the material near the surface consists of a layer of regolith which is varying in thickness and includes lateritic plateaus. Below the regolith, the geology may be divided into three north trending litho-structural domain: Western flyschoid domain, central deformation Corridor and eastern Siliciclastic Domain. One of the important features of this group is the north-south oriented lineament, known as the Senegal-Mali Shear Zone (SMSZ), located in the eastern part of the inlier.

The western domain is dominated by a sequence of flyschoid turbidites, black shales (or graphitic pelite), carbonate rocks, minor volcanics (mainly basalt with subordinate rhyolite and pyroclastic breccia or agglomerate), and dioritic intrusions. The eastern domain is dominated by a detrital assemblage composed of greywacke and sandstone (± quartzite), called Guémédji sandstone. Between the west and east domains is a highly deformed north-trending domain (020˚ N) that is well defined in magnetic geophysical data. It is likely this highly deformed domain corresponds to a regional scale structural corridor that branches from the SMSZ. Lithologically, it is composed of fine schistose sediments that are carbonaceous in places, locally referred to as the "Pelite Unit", and fine laminated sediments (± carbonates) that subtly grade into an impure marble, locally called the "Cipolin Unit".

The known gold deposits at the Project occur on the margins of the central deformation corridor. The Malikoundi/Boto 2, Boto 4, and Boto 6 deposits lie along the eastern boundary (in contact with the Guémédji sandstone) and where the Boto 5 deposit lies along the western boundary (in contact with carbonaceous turbidites). The mineralization at Malikoundi/Boto 2, Boto 4 and Boto 6 is hosted in the upper part of the Guémédji sandstone, near a structurally modified contact with the overlying, finer grained, sedimentary sequence. The mineralization occurs in quartz-tourmaline-pyrite alteration and veins and in hematite-calcite-pyrite alteration and veins. The mineralization at Boto 5 is mainly hosted within albitite and follows a phase of quartz tourmaline veining as well as pyrite and related alteration/bleaching.

Drilling to date has been primarily focussed on the development of the Malikoundi/Boto 2, Boto 5 and Boto 6 deposits. As of March 2018, the database for the Project contained 784 diamond core and reverse circulation holes totalling 126427 m. AGP is satisfied that the drill hole database is adequate to conduct an estimate of mineral resources. AGP completed a site visit to the Boto Project in December 2017 that included an inspection of drill core logging, sampling and storage facilities, a review of sampling methods, chain of custody; a review of drill hole locations. AGP completed an independent verification of the drill logging and drill hole database. AGP is of the opinion that the sample preparation, security, and analytical procedures followed by IAMGOLD are to industry-standard and the resulting data and database are adequate for use in the estimation of mineral resources.

The majority of drilling to date, on the principal deposits that make up the Project, has been conducted on drill sections oriented to the south-southeast, on a nominal drill hole spacing of between 50 m by 50 m, and up to

100 m, on the known mineralized zones. Most recent drilling in 2017 and 2018 at Malikoundi/Boto 2 has targeted the wider spaced drill fences to in-fill these areas to a 50 m by 50 m spacing to confirm continuity between the north and south areas of the deposit, to increase mineral resources and to upgrade Inferred resources to an Indicated classification.

The mineral resources were estimated using Geovia GEMS 6.8.1 resource estimation software. The current mineral resource estimates for the Project was carried out using inverse distance cubed interpolation method within 3D wireframe models representing interpreted mineralized zones for each of the deposits.

The Mineral resources are classified as Indicated Resources and Inferred Resources in accordance with the CIM (2014) Standards and Definitions of Mineral Resources and Mineral Reserves. The classification is based on the number of drillholes and distance to nearest sample and are reported within an optimized constraining shell. Mineral resources are reported within optimized constraining shells using Hexagon Mining MineSight 3D software using a gold price of $1,500/oz. Cut-off grades vary between 0.37 and 0.51 g/t Au, and densities range between 1.70 g/cm^3 and 2.76 g/cm^3, depending on alteration zone. The effective date of the mineral resources for the Project is 8 May 2018.

Mineral resources are reported inclusive of mineral reserves.

Table 25.1 presents the Mineral Resources for the Project.

Table 25.1 Mineral Resources for the Boto Project; effective date 8 May 2018

Zone	Classification	Tonnes	Grade	Contained Gold
		(,000 t)	g/t Au	(,000 oz)
Malikoundi/Boto 2	Indicated	41,915	1.66	2,240
	Inferred	1,974	2.00	127
Boto 5	Indicated	2,469	1.86	148
	Inferred	34	0.75	1
Boto 6	Indicated	3,661	0.84	99
	Inferred	475	1.06	16
Boto 4	Indicated	-	-	-
	Inferred	-	-	-
Total	**Indicated**	48,045	1.61	2,487
	Inferred	2,483	1.80	144

Notes:

1. The mineral resources are reported within optimized constraining shell using a gold price of $1,500/oz.
2. Summation errors may occur due to rounding.
3. Mineral Resources are reported inclusive of Mineral Reserves.
4. Mineral Resources are classified in accordance with the CIM (2014) Standards and Definitions of mineral resources.
5. Cut-off grades used to report mineral resources vary from 0.37 Au g/t and 0.51 Au g/t depending on alteration zone.
6. Capping of grade outliers varies between 1.71 Au g/t and 42.02 Au g/t depending on interpreted mineralized zone and sub-domain.
7. The density varies between 1.70 g/cm^3 and 2.76 g/cm^3 depending on alteration zone.

The optimized constraining shells for Malikoundi/Boto 2 and Boto 6 mineral resources, had a 500 m offset applied from the Balinko and Falémé Rivers. The Boto 5 deposit is located approximately 1 km west of the Balinko River and is not affected by the 500 m offset. However, the Boto 5 deposit is affected by workings from artisanal miners that have affected the surface of the deposit. These workings were taken into consideration and excluded from the current mineral resources by creating two wireframes to represent material removed from the mineralized zones. The Boto 4 has adequate drill information to complete resource estimate, however, this deposit is currently not classified as having mineral resources due to the proximity of the Balinko River, the border between Senegal and Mali, and is situated within a 500 m exclusion zone of the river and below the village of Guémédji. Should the exclusion zone change or be lifted mineral resources the Boto 4 block model and mineral resources may be re-evaluated.

AGP is not aware of any information not already discussed in this report, which would affect their interpretation or conclusions regarding the subject property. AGP is required to inform the public that the quantity and grade of reported Inferred resources in this estimation must be regarded as conceptual in nature and are based on limited geological evidence and sampling. The geological evidence is sufficient to imply, but not verify, geological grade or quality of continuity. For these reasons, an Inferred resource has a lower level of confidence than an Indicated resource. It is reasonably expected that most of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. The rounding of values, as required by the reporting guidelines, may result in apparent differences between tonnes, grade, and metal content.

It should benoted that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

25.2 Mining

Mining studies have been completed using the resource estimates as of May 8, 2018 for both Malikoundi and Boto 5 and include the following aspects:

- Pit optimization utilized the Lerch-Grossman algorithm to determine the ultimate pit limits. A metal price of $1,044 per ounce gold was used to define the ultimate pit for Malikoundi Pit, $960 per ounce gold for Malikoundi North and $900 per ounce gold for Boto 5 for the feasibility study.

- Final pits were designed for Boto 5, Malikoundi and Malikoundi North. Bench and overall pit slope designs were based on recommendations by the geotechnical consultants (Absolute Geotechnics).

- Mineral reserves have been determined from mineral resources by taking into account geologic, mining, processing, legal and environmental considerations and are therefore classified in accordance with the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves.

- There are no Proven Mineral Reserves. Probable Mineral Reserves amount to 35.06 Mt at an average grade of 1.71 g/t Au. Total estimated Mineral Reserves amounts to 35.06 Mt at an average grade of 1.71 g/t Au. Inferred Mineral Resources have not been converted to reserves and instead are treated as waste for mine planning purposes.

- The mine schedule moves 204.3 Mt of waste and 35.1 Mt of ore for a strip ratio of 5.83:1.

- Waste storage areas were designed near Boto 5, Malikoundi and Malikoundi North. They have been designated as WDNE, WDNW, Boto 5 and FWP.

- Mining at Malikoundi and Malikoundi North will be completed with size appropriate equipment in the form of 95 t haulage trucks matched to 15 m^3 hydraulic excavators, 13 m^3 front-end loaders and 6.7 m^3 hydraulic excavators. Support for dilution control will be the responsibility of the smaller excavators in delineating the ore/waste contacts. Drilling and blasting will be required of all horizons but to a lesser amount in the saprolite/lateritic zones. Support equipment such as dozers, graders and water trucks will assist in the mining operation. Boto 5 will be mine by contractor using 40 t trucks and the appropriate loading and support equipment.

- Grade control will be provided by a separate fleet of contract reverse circulation drills working in advance of the active mine faces in the saprolite, transition and fresh rock horizons.

- Dewatering activities will be of a large scale to accommodate the annual 2+ m of rainfall. Water will be pumped from the pits to the pit rim then pump horizontally (or by gravity) to various discharge points if the water is of suitable quality.

- Estimates of both mine capital and operating costs are made. Capital costs consider owner-operated mining for the Malikoundi pits and contract mining in Boto 5. Pre-production stripping will be completed by IAMGOLD mining equipment. Replacement and additional equipment purchase costs have been included over the life of the Project.

- The mine operating cost estimate is dominated by equipment operating costs (fuel, tires, labour and maintenance). Blasting, mine operations administration, and services have also been included.

25.3 Processing

25.3.1 General Conclusions

The following conclusions can be made from the metallurgical testwork with regards to a leach/CIP process:

- Fresh rock, saprolite, and saprock at Boto are readily amenable to whole ore cyanidation.
- The optimum grind size was determined to be P_{80} 75 µm.
- The ore at Boto is not expected to have preg-robbing properties.
- A pre-oxygenation step with oxygen sparging during leach, combined with lead nitrate addition is critical in achieving the maximum possible recovery.
- Leach extraction rates are essentially completed by 24 hrs to 36 hrs.
- Cyanide consumption rates are expected to be low, averaging about 0.13 kg NaCN/t ore. When accounting for cyanide residue in CIP tailings, an addition rate of 0.27 kg NaCN/t ore is expected.
- Lime consumption rates are expected to be moderate, average at 1.64 kg CaO/t ore at the design ore blend. When accounting for 85% purity of the supplied lime, and an addition rate of 1.93 kg CaO/t ore is expected.

25.3.2 Grind Selection

Cyanidation tests were conducted on both gravity tailings and whole ore samples at the finer grind size (targeted P_{80} of 53 µm). The results for the finer grind, ~P_{80} 53 µm, yielded 2% to 3% higher gold extraction than the results for grind P_{80} 75 µm.

A grind size trade-off study was conducted by Lycopodium to assess the benefit of incorporating a regrind milling circuit into the plant design. A simple financial analysis was completed and based on a set of inputs which included a base case gold price of $1,200/oz. At this base case gold price, the analysis yielded a negative NPV and no payback on the IRR for the addition of a regrind circuit. A sensitivity analysis showed that the NPV and IRR becomes positive with a payback at the end of a 10-year mine life if gold price becomes $1,250/oz. It was concluded that a regrind milling circuit is not justifiable at the base case gold price and is not recommended for inclusion in the design at this stage. However, the plant layout should include provision for future installation.

25.3.3 Gravity Circuit

A review of all the previous gravity separation results showed that out of 56 tests (from 2013 to 2018), only three yielded a GRG close to 35%, one close to 50%, and one Boto 5 sample over 65%. The overall GRG average is at approximately 20%, which is considered to be on the low end of the GRG range.

The E-GRG test results conducted in 2018 indicated that the majority of the GRG amount in MC-2 is classified as very fine. Although the GRG number is considered to be moderate at 39.6, recovery with gravity at the full scale would be difficult due to this fine nature.

The whole ore leaching results for MC-1 and MC-2 produced similar results to the leach tests on gravity tailings.

All the results combined provided no justification for inclusion of a gravity circuit in the flowsheet.

25.3.4 Pre-treatment and Leach Conditions

Extensive leach optimization testwork were conducted to determine the ideal pre-treatment and leach conditions for the Project. The results showed that the use of oxygen in pre-treatment and during leaching, along with lead nitrate addition will yield at least 1.8% higher gold extraction than if these additions were not considered. The leach kinetics will be more rapid, and the cyanide consumption will also be reduced significantly.

A trade-off study was conducted by Lycopodium to justify the inclusion of the oxygen plant and the lead nitrate system. A simple financial analysis yielded a positive NPV of $15.3M and an IRR of 157%, providing a payback period within the first year.

The leach conditions recommended based on the results of the leach optimization tests include:

- Pulp pH at 10.5 to 11 to be maintained with the addition of lime.
- Pre-treatment with oxygen, and leaching with oxygen spargers to maintain a dissolved oxygen level of ~15 mg/L or higher.
- Lead nitrate addition of 200 g/t ore.
- Pre-treatment time of 4 hours minimum, and leach time of approximately 36 hours.
- Cyanide concentration of 0.5 g NaCN/L to be maintained in first leach tank.

26.0 RECOMMENDATIONS

26.1 Geology

In order to continue the development of the known deposits and determine the potential of other exploration targets, the following drill programs and exploration activities are recommended:

- Continue to collect density measurements from the drilling, both within and outside the mineralized zones, and to validate density measurements externally as needed. A 5% check of the current database at an external lab is recommended with an estimated budget of approximately $5,000.

- Continue drilling on Malikoundi/Boto 2 and Boto 5 both along strike and at depth to upgrade inferred resources to an indicated classification. An additional 10,500 m of RC and diamond core drilling is recommended with an estimated budget of approximately: $700,000.

- Drilling on Malikoundi/Boto 2 to test an area of 200 m x 200 m with a grid spacing of 25 m by 25 m to confirm the continuity of grades, geology and mineralized zones. A diamond core drill program of approximately 4,500 m of is recommended with an estimated budget of approximately: $550,000.

- Continue drilling to the east of the known mineralized zones in order to determine the extents of the different alteration zones within the optimized constraining shells. Since the majority of drill holes are drilled from northwest to southeast, there is less geological and geotechnical information to the east and southeast of the various deposits. A combination of RC and diamond core drilling of roughly 2,500 m is recommended with an estimated budget of approximately: $220,000.

Pending positive results from these exploration and drill activities, additional recommendations may be made.

26.2 Geotechnical

Recommendations for the next stage of engineering for the Project (detailed design) are summarized below:

Site Investigations

- Additional site investigations should be completed as part of detailed design to namely confirm the foundation conditions below the FWP embankment, select Waste Dumps, and potential construction materials. Additional undisturbed samples should be collected for laboratory testing based on the encountered conditions to refine the foundation material strength estimates and optimize the designs.

TMF

- Develop a Failure Modes and Effects Analysis (FMEA) for the TMF to identify and characterize potential risks associated with the TMF for the construction, operations and closure phases. The FMEA should include a risk evaluation matrix that provides a rating for the likelihood and consequence for each of the identified risks for the TMF and identify design mitigation measures to reduce the identified risks.

- Complete a breach analysis for the TMF to estimate the potential flood inundation zones that could result if the embankment associated with the facility were to fail. The estimated downstream inundation zone will help identify any residences, infrastructure and/or receptors that are at risk downstream of the TMF. An EPP should be developed to reduce the risk of human life loss and injury, and minimize property damage in the event of the occurrence of an unusual or emergency at the TMF.

- Develop and Operations, Maintenance and Surveillance (OMS) Manual for the TMF as part of the detailed design.

- Development of a full closure plan for the TMF based on the final design configuration.

- Assess the compaction characteristics of the ferricrete/durricrust through the construction of a test pad prior to completion of final design.

FWP

- Develop a FMEA for the FWP to identify and characterize potential risks associated with the FWP for the construction, operations and closure phases. The FMEA should include a risk evaluation matrix that provides a rating for the likelihood and consequence for each of the identified risks for the FWP and identify design mitigation measures to reduce the identified risks.

- Complete a breach analysis for the FWP to estimate the potential flood inundation zones that could result if the embankment associated with the facility were to fail. The estimated downstream inundation zone will help identify any residences, infrastructure and/or receptors that are at risk downstream of the FWP. An EPP should be developed to reduce the risk of human life loss and injury, and minimize property damage in the event of the occurrence of an unusual or emergency at the FWP.

- Perform a trade-off study to investigate the use of geosynthetics (i.e. geonet) as a replacement for the sand and gravel liner bedding cushion layer on the upstream face of the FWP Embankment.

- Perform stability analysis between the FWP and Malikoundi North Pit.

- Develop and OMS Manual for the FWP as part of the detailed design.

Water Balance

- The catchment areas contributing runoff to the Plant Site, open pits and waste dumps, and the amount of groundwater inflow to the open pits with time need to be confirmed based on the ultimate mine plan.

- Collection of site specific meteorological and hydrology data. This data will be used to refine seasonal runoff values and design storms to be used in future detailed design work.

- Calibration of the water balance should be completed, based on actual site data, during early years of operation

Water Management Measures

- Perform further analysis to characterize the geochemical composition of runoff water to provide insight into the acceptable discharge limits.

- Perform bench scale settling testing to characterize the required retention time for suspended solids in the runoff water.

- Develop an operations and management schedule to confirm the design criteria and performance of the water management design, specifically the operational, monitoring and maintenance requirements.

26.3 Open Pit Mining

Significant work has been completed to date on the open pit designs and costing for the Boto 5 and Malikoundi pits. This work demonstrates the potential for economic development of the Project. There are still some areas that can be enhanced to assist in the mine development.

26.3.1 Geotechnical

The Boto 5 area and Malikoundi North areas are lacking geotechnical monitoring. Recommendations by Absolute Geotechnics included the installation of instrumentation for monitoring of pore pressure and slope stabilities. This instrumentation has been included in the cost estimate but will need to be detailed and installed in the appropriate time frames.

Waste management facility foundations for Boto 5 and FWP area were a concern on how fast material could be placed to allow for sufficient pore depressurization. That has been considered in the FS but improvements on these facilities may assist mine operations during construction. Further examination needs to be undertaken to optimize the existing designs. This can be completed as part of basic engineering.

26.3.2 Ore Control Procedures

The FS went into detail with the specific methods of ore control but this area represents a significant area of benefit to the overall mine operation if dilution is reduced. Additional study is required to evaluate exact equipment for use in the grade control program and methodologies regarding appropriate sample size to be employed in mine operations. There are specific experts in the field that should be contacted and it would benefit the project to consider them prior to mine operations commencing.

26.3.3 Dewatering

A more detailed design from a mine operation perspective needs to be completed on the dewatering system. This will need input from the hydrogeological engineers to provide further guidance on flow rates.

26.3.4 Low Grade Material

An opportunity exists dependent on mill capacity and the gold price. A review was completed of material that existed below the marginal cut-off due to the fact the diluting material was carrying grade. If the marginal cut-off was to drop from the existing 0.46 – 0.48 g/t Au to 0.35 g/t Au, an additional 15.6 Mt grading 0.49 g/t Au is currently categorized as waste. This has 246,000 oz contained within that material. Testing of the gold recovery at that low level has not been completed but should be part of any basic engineering. With that information a proper economic evaluation of the below cut-off material can be completed to determine if the opportunity to process this material is valid. It is not currently in the mine plan but could be processed in the final year if it makes economic sense.

26.4 Processing

The process plant is scalable and can be expanded at a later stage to accommodate an increase in throughput.

Material meeting minimum 3-inch rock size from future drilling activities should be set aside for CWi tests since the additional CWi tests planned for in the FS phase were not conducted due to material availability.

During plant operations, the following items are recommended:

- Observations made during the test program indicated that an abundance of saprolite in the plant feed may cause materials handling and sedimentation problems. A blending strategy should be implemented as soon as ore is being mined in order to limit the amount of saprolite in the ROM feed to the plant.
- Particular attention should be paid to the operation and commissioning of the thickener and associated flocculant addition systems as this unit operation is expected to be the bottleneck during initial operations (given the high proportion of saprolite and transition material scheduled for treatment in the mine plan during the first 4 years of operations).
- Natural cyanide attenuation (free and WAD) be monitored in the TMF.
- Site water quality (raw and process) be monitored during the initial wet and dry seasons to document the seasonal impact of water quality.

- Gold adsorption rate and equilibrium loading on carbon be monitored as the plant head grade varies during the life of the operation to ensure that carbon movement and management is optimized.

- Slurry percent solids in the leaching stage be monitored during start-up and operation as this parameter could reduce gold extraction if allowed to deviate significantly from the design value of 50% w/w solids.

- Investigate recovering carbon fines with a view to recover additional gold.

26.5 Overall Recommendation

Following board approval, it is recommended that IAMGOLD commence implementation of the Project in line with the preliminary implementation plan and schedule developed during the FS, and committing to the capital expenditure presented in Section 21.

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