NI 43-101 TECHNICAL REPORT
BOTO OPTIMIZATION 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

Qualified Persons:

Manochehr Oliazadeh, P.Eng. Manager of Process, Lycopodium

Niel Morrison, P.Eng., Principal Process Engineer, Lycopodium

Reagan McIsaac, P.Eng., Senior Engineer, Knight Piésold

Tudorel Ciuculescu, P. Geo., Senior Geologist, RPA

Luc-B Denoncourt, ing., Project Manager, IAMGOLD Corporation

Philippe Chabot, ing., Mining Director, IAMGOLD Corporation

Date of the Report	10 February 2020
Effective Date	31 December 2019

File Location: 16.04
Rev: 0

0	10.02.2020	FINAL	BB	JR	CW
REV NO.	DATE	REVIEW	BY	DESIGN APPROVED	PROJECT APPROVED

DATE & SIGNATURE PAGE

Project Name:	**Boto Gold Project**
Title of Report:	**NI 43-101 Technical Report**
Location:	**Senegal**
Effective Date of Report:	**10 February 2020**

[SIGNED]		**DATE**
Manochehr Oliazadeh, P.Eng. (Lycopodium Minerals Canada Ltd.)		10 February 2020
Niel Morrison, P.Eng. (Lycopodium Minerals Canada Ltd.)		10 February 2020
Reagan McIsaac, P.Eng. (Knight Piésold)		10 February 2020
Tudorel Ciuculescu, P.Geo. (Roscoe Postle Associates)		10 February 2020
Luc-B Denoncourt (IAMGOLD Corporation)	Digitally signed by Luc-Bernard Denoncourt DN: cn=Luc-Bernard Denoncourt, o=Iamgold Corporation, ou=Developpement de projets, email=luc-bernard_denoncourt@iamgold.com, c=CA Date: 2020.02.11 13:20:30 -05'00'	10 February 2020
Philippe Chabot (IAMGOLD Corporation)		10 February 2020

Table of Contents

TABLES

FIGURES

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 2019, IAMGOLD contracted several leading consulting firms to undertake a optimization study 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
Roscoe Postle Associates Inc (RPA)	Geology, mineral resources
Knight Piésold (KP)	Tailings management facility, water storage facility, water management, geotechnical
Absolute Geotechnics (AG)	Geomechanical
IAMGOLD	Mineral reserves 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 had an expiry date of 4 March 2019. An application for the Boto exploitation permit has been submitted in November 2018. In December 2019, the Government of the Republic of Senegal approved the mining permit application and granted an exploitation permit for the Boto Project for an initial period of 20 years principally under the provisions of Senegal's 2003 mining code. An application for renewing the exploration permit for Daorala was also submitted in November 2018. This application is currently under review.

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 Koila 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 on site. 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 was 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 (often called the ''western Pelites'') is dominated by a volcano-sedimentary assemblage containing tuffaceous rythmites and tuffs, black shales (or graphitic pelite), carbonate rocks, hypovolcanics (microdiorite, andesite, pyroclastic and magmatic breccia or agglomerate), and dioritic intrusions. Immediately east, the central Siliciclastic domain is dominated by a detrital assemblage composed of greywacke and sandstone (+/- quartzite), called the ''Guémédji sandstone''. It is unclear whether these sandstones/wackes are part of the Kofi / Dialé or of the Dalema unit. The exact stratigraphical relations with the surrounding units are not very obvious given the often important level of strain. However, the (westward/upward) apparent increase in carbonate content near the contact with the main carbonate layer (also corresponding to the main tectonic break) would suggest that the Guemedji Sandstone part of Daléma Unit. The western and central domains are separated by a North-South trending high strain structure (010˚ N) that is well defined in all geophysical data and very evident in drilling. This highly deformed sinistral-reverse corridor 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 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.

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 May 2019, a total of 152,061 m have been completed from 996 drill holes.

1.6 Metallurgy

Extensive metallurgical testwork has been conducted on the Boto ore deposit from 2013 to 2019. The testwork results were analyzed and used in flowsheet development and inputs into the process design criteria.

The comminution parameters were determined as lithology weighted averages per weathering type and are as follows:

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

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. A grind size of P_{80} of 75 µm was determined to be optimal for the Project.
- A 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. 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, a significant reduction in cyanide consumption and gold extraction benefits.
- Cyanide consumption was low with an addition rate of 0.27 kg/t ore expected at the design ore blend (approximately 80% fresh rock and 20% saprolite/saprock).
- Lime consumption was moderate with a consumption rate of 1.92 kg/t ore expected at the design ore blend.

1.7 Mineral Resources

The resource estimate has been prepared using interpreted mineralized lenses from Malikoundi/Boto 2, Boto 5, and Boto 6 deposits. The Boto 4 deposit has been excluded from the resource estimate. Geovia GEMS™ 6.8.2

was used for drill hole database management, geological interpretation, mineralized wireframe modelling, and to generate the block model supporting the resource estimate. The inverse distance cubed (ID3) interpolation method was used to estimate the block model gold grades.

The Mineral Resources for the Project are reported within optimized constraining shells using Hexagon Mining MineSight 3D software using a gold price of US$1,500/oz. Cut-off grades vary between 0.37 g/t Au and 0.50 g/t Au, and densities vary between 1.65 g/cm³ and 2.75 g/cm³, depending on weathering 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. The Mineral Resources have an effective date of December 31, 2019.

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

Table 1.2 Summary of Mineral Resources for the Boto Project – December 31, 2019

Classification	Tonnes ('000)	Grade (g/t Au)	Contained Metal ('000)
Indicated	40,600	1.56	2,033
Inferred	8,200	1.78	469

***Notes**:*
- *Mineral Resources are reported within an optimized constraining shell using MineSight 3D software.*
- *Mineral Resources are reported inclusive of Mineral Reserves;*
- *Cut-off grades vary between 0.37 g/t Au and 0.50 g/t Au, depending on the deposit and the weathering type of material;*
- *Mineral resources were estimated based on a gold price of $US 1,500/ oz;*
- *Capping of grades varied between 2 g/t Au and 25 g/t Au on raw assays by mineralized zone;*
- *The density varies between 1.65 g/cm^3 and 2.75 g/cm^3 depending on weathering zone.*

The mineral resources for the Boto project include the Malikoundi/Boto 2, Boto 5, and Boto 6 deposits. No resources were reported at Boto 4 as it is located within the 500 m wide exclusion zone along the Balinko river shore (the border of Senegal and Mali) and underneath the village of Guémédji. Should these factors change, the Boto 4 deposit will be re-evaluated.

RPA is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate at the time of this report.

1.8 Mineral Reserves

The mineral reserve estimate is consistent with the CIM Definition Standards for Mineral Resources and Mineral Reserves. The reserves for the Project are based on the conversion of the Indicated resources to Probable reserves within the current Technical Report. No Measured resources are currently part of the model. The mineral reserve estimate for the Boto Gold Project deposits is based on the resource block model estimated by RPA and with effective date December 31, 2019.

The reserves are based on the Malikoundi deposit, including the Malikoundi and Malikoundi North pits, and the Boto 5 deposit. 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**		
Pit Area (Cut-off g/t)	Tonnes (kt)	Grade (g/t)	Gold (oz)	Tonnes (kt)	Grade (g/t)	Au (koz)	Tonnes (kt)	Grade (g/t)	Au (koz)
Boto Gold Project									
Saprolite	-	-	-	2,332	1.82	137	2,332	1.82	137
Transition	-	-	-	1,574	2.02	102	1,574	2.02	102
Fresh Rock	-	-	-	25,134	1.68	1,354	25,134	1.68	1,354
Total Boto Gold Project	-	-	-	29,040	1.71	1,593	29,040	1.71	1,593

1. *Reserves estimated assuming open pit mining methods.*
2. *Mineral Reserves are estimated using a long-term gold price of 1,200 US$/oz.*
3. *Average weighted process recovery of 89.4%.*
4. *Quantity of gold payable is 99%.*
5. *Transportation and refining costs estimated at 3.04 US$/t milled.*
6. *Royalty and other charges of 4.0% are applied to the gold metal value.*
7. *Processing costs estimated at 10.82 $/t milled, 11.28 $/t milled and 15.61$/t milled for saprolite, transition and fresh rock material respectively.*
8. *G&A costs estimated at 4.29 $/t milled for saprolite for the Malikoundi deposit and 4.37 $/t milled for all other material at the Malikoundi deposit and for the Boto 5 deposit.*
9. *The cut-off grades for the Malikoundi deposit are 0.42 g/t for saprolite, 0.43 g/t for transition rock and 0.58 g/t for hard rock.*
10. *The cut-off grades for the Boto 5 deposit are 0.41 g/t for saprolite, 0.43 g/t for transition rock and 0.58 g/t for hard rock.*
11. *The tonnes and grades are diluted.*

Numbers may not add due to rounding.

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.

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 and the detailed mine design.

A series of nested shells were generated for a range of revenues from 600$/oz to 1,500$/oz where a $1,150/oz gold price shell for the Malikoundi deposit and $1,200/oz Au price shell for the Boto 5 deposit were selected.

The geologic block models developed for the FS were whole block fully diluted models. Additional contact dilution was integrated in the mining block model to better reflect expected results with mining practices. Preliminary analyses of contact dilution were estimated to determine average dilution percentages, which were applied. The diluted tonnes and grade were reported in the detailed pit designs.

The Malikoundi pit is designed as four phases within the main pit. Malikoundi North is designed with two phases. Boto 5 is a single-phase pit.

The mine schedule delivers 29.0 Mt of ore grading 1.71 g/t gold to the mill over a mine life of approximately 12 years, including 13 months of pre-production. 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. The pit phasing and ore stockpiling strategy will ensure that sufficient mill feed is available during the rainy season. Phases will be advanced quickly in the dry season to provide temporary water storage after a rainfall event. Dewatering pumps will evacuate the water from the pits during the wet season.

The Malikoundi pit will be mined from the beginning of mining operations until Year 8. The Malikoundi North pit will be mined from Year 1 to 4. The Boto 5 pit will be mined from Year 1 to 2. From years 9 to 11, the mill will be fed exclusively from the ore stockpiles until they are completely depleted. Project activities in the pre-production period include haul road construction, fresh water pond (FWP) construction, TMF material placement, initiation of mining in Malikoundi Phase 1 and development of an ore stockpile near the processing plant.

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

Table 1.4 Mine Plan

Mine Plan	Unit	-2	-1	1	2	3	4	5	6	7	80	9	10	11	Total
Material Mined - Total	(kt)	800	14,175	36,023	37,012	36,995	36,986	31,102	31,001	17,464	5,784	0	0	0	247,341
Waste	(kt)	800	12,931	32,749	33,221	33,695	32,006	28,825	27,581	13,667	2,825	0	0	0	218,300
Ore	(kt)	0	1,244	3,273	3,790	3,300	4,980	2,277	3,420	3,797	2,959	0	0	0	29,040
Average Mined Grade	(g/t)	0.66	2.18	1.88	1.82	1.43	1.69	1.26	1.62	1.67	1.98	0.00	0.00	0.00	1.71
Material Reclaimed	(kt)	0	0	1,318	521	770	240	1,352	186	338	348	2,750	2,700	1,990	12,514
Average Reclaimed Grade	(g/t)	0.00	0.00	2.37	1.80	2.72	2.20	1.99	1.00	1.05	0.88	0.65	0.76	0.88	1.26
Total Material Moved	(kt)	800	14,175	37,341	37,532	37,766	37,226	32,454	31,187	17,802	6,132	2,750	2,700	1,990	259,855

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. 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 (WMF).

The mill's production will ramp-up over a three (3) month period before it achieves its nameplate capacity; 60% for the first month, 80% for the second month, and 90% for the third month. Thereafter, the mill will operate at 100%. Throughout the mine life, a blending strategy will be used to feed the mill optimally with soft and hard rock material. See Table 1.5 below.

Table 1.5 Mill Feed

Material Type	Unit	-1	-2	1	2	3	4	5	6	7	8	9	10	11	Total
Direct Feed	(kt)	0	0	1,232	2,229	1,980	2,510	1,348	2,514	2,362	2,352	0	0	0	16,526
Reclaimed Material	(kt)	0	0	1,318	521	770	240	1 352	186	338	348	2,750	2,700	1,990	12,514
Total Feed	(kt)	0	0	2,550	2,750	2,750	2,750	2,700	2,700	2,700	2,700	2,750	2,700	1,990	29,040
Feed Grade	(g/t)	0.00	0.00	2.42	2.06	2.06	2.01	1.81	1.87	2.06	2.01	0.65	0.76	0.88	1.71
Recovery	(%)	0.0%	0.0%	90.5%	90.3%	90.5%	89.4%	88.8%	88.9%	89.3%	89.0%	88.3%	87.0%	87.7%	89.4%
Recovered Ounces	(koz)	0	0	179	164	164	159	139	144	159	155	51	57	50	1,424

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 33.5-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.

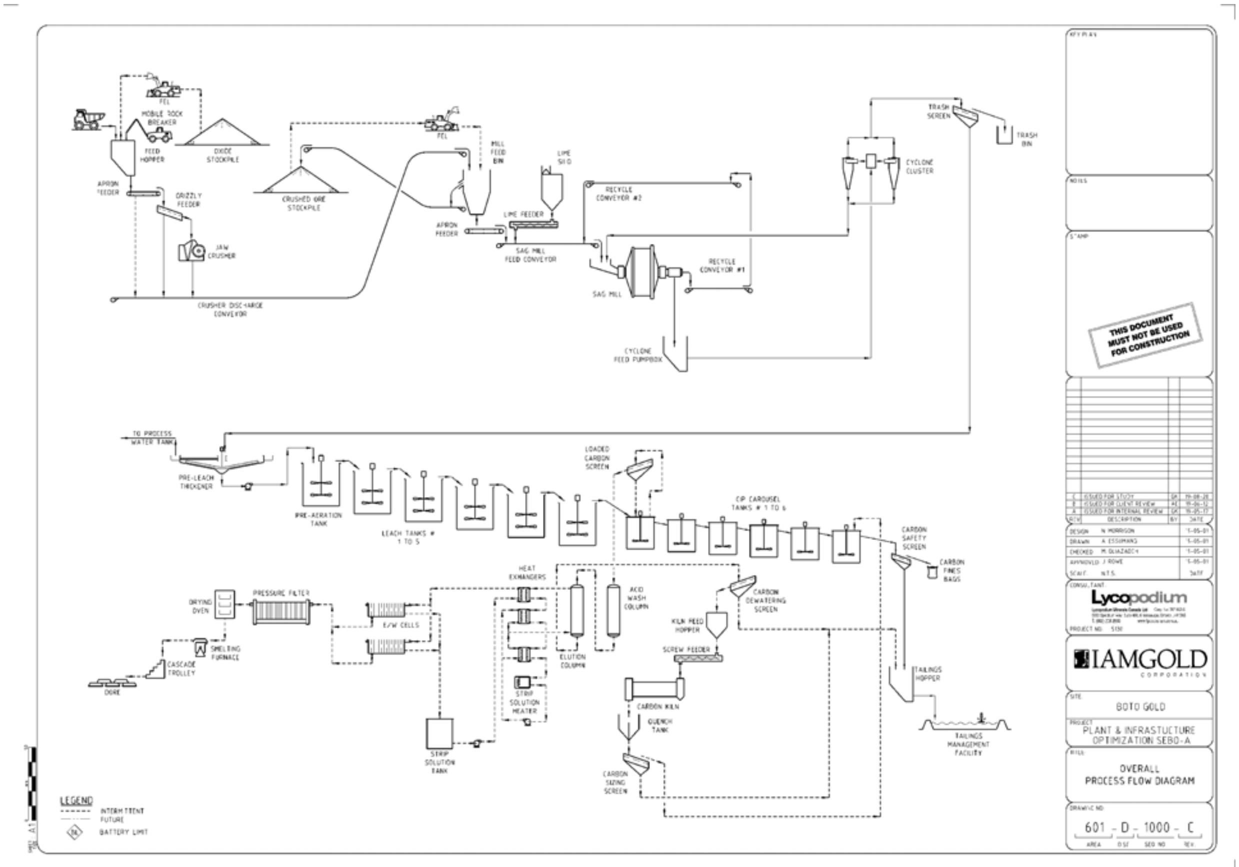
- Carbon-in-pulp (CIP) carousel circuit consisting of six stages for recovery of gold dissolved in the leaching circuit.

- 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

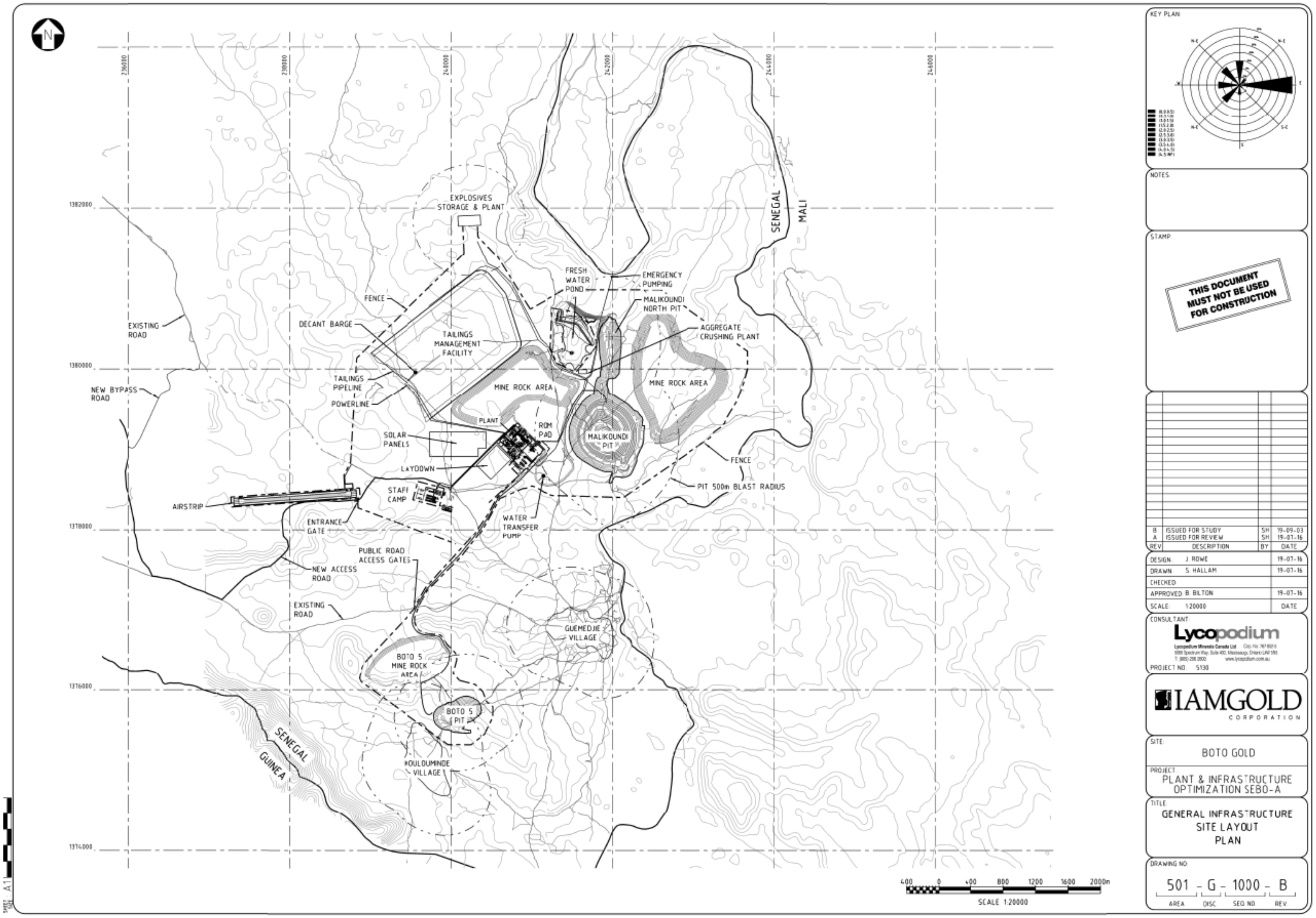


1.11 Infrastructure

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 northeast 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. Additional information was requested by their committee. The amended report taking into consideration this feedback was submitted to the Ministry of Environment in May 2018.

An environmental compliance certificate was issued by the Senegalese Government in October 2018 followed by a decree in November 2018. .

The highlights of the baseline environmental studies and the impact study are provided in Section 20 which 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 2019 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 1.6 Capital Estimate Summary (Q3 2019, ±15%)

Area	M$ (Excluding Duties and Taxes)
Direct Costs	
Site General	29.7
Mining	57.3
Power Supply	2.6
Process Plant	64.7
Tailings & Water Management	14.0
Sub-Total Direct Costs	**168.3**
Indirect Costs	
Construction In-directs	23.2
Owner's Costs	63.1
Contingency	16.7
Sub-Total Indirect Costs	**103.0**
Total Initial Capital Cost	**271.3**
Sustaining Capital Cost	$68.5
Total Project Capital Cost	**339.8**

The estimated life of mine operating cost per tonne of ore processed is summarized in Table 1.7.

Table 1.7 Life of Mine Operating Costs per Tonne

	Total Cost ($M) from first gold pour	$/t Processed
Mining	$487	$16.76/t
Processing	$396	$13.65/t
G&A	$108	$3.70/t
Total Cash Cost	**$991**	**$34.12/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 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 2019 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.

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 $339.8M, with an initial capital expenditure of $271.3M. Table 1.8 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 1.9.

Table 1.8 Production Summary

	Value
Ore milled	29.04 Mt
Total tonnes mined	247.34 Mt
Average head grade	1.71 g/t Au
Contained gold in material	1,593 koz
Total gold produced	1,424 koz
Average gold recovery	89.4%
Production life (processing)	11 years
Nominal annual processing rate	2.75 Mtpa

Table 1.9 After-tax Financial Results

AISC	$842 $/oz
IRR	22.6 %
NPV (6%)	$218.6 M
Payback	3.2 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 optimization study 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 detailed implementation plan and schedule developed during the optimization study, 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 the optimization study of Boto 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, and an earlier feasibility study in 2018.

RPA, KP 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**
Manochehr Oliazadeh	Process Manager	Lycopodium Minerals Canada Ltd	Yes	No Site Visit	P.Eng.	1.6, 1.10, 1.11, 1.13, 18.1, 18.2.1, 18.2.2, 18.3, 18.4.2, 18.4.2-18.4.4, 18.6, 18.7.1, 18.8.1, 18.8.2, 19.9.1-18.9.4, 21.1 and 21.3.
Niel Morrison	Principal Process Engineer	Lycopodium Minerals Canada Ltd	Yes	No Site Visit	P.Eng	13, 17, 21.4.3, 25.3, 26.4.
Reagan McIsaac	Senior Engineer	Knight Piésold	Yes	Did not visit site	P.Eng.	26.2, 18.10-18.12.
Tudorel Ciuculescu	Senior Geologist	Roscoe Postle Associates	Yes	29 May – 02 June 2019	P.Geo.	1.4, 1.5, 1.7, 4-12, 14, 23, 25.1, 26.1.
Luc-B Denoncourt	Project Manager	IAMGOLD Corporation	No	11-16 April 2019	ing.	1.1-1.3, 1.12, 1.14, 1.15, 2.1-2.3, 3, 18.4.1, 18.5, 18.7.2, 19, 20, 21.4.1, 21.4.4, 22, 24, 26.5 and 27.
Philippe Chabot	Director, Mining	IAMGOLD Corporation	No	11-16 April 2019	Ing.	1.8, 1.9, 15, 16, 18.2.3, 18.2.4, 21.2, 21.4.2. 25.2, 26.3.1-26.3.4.

2.3 Visits to the Site

Niel Morrison did not visit the site.

Manochehr Oliazadeh did not visit the site.

Reagan McIsaac did not visit the site.

Tudorel Ciuculescu visited the site from 29 May to 2 June, 2019. The property, deposit geology, geological model, exploration and drilling methods and results, drill core samples and logging, sampling methods and approach, sample and data handling were reviewed during the visit.

Luc-B Denoncourt visited the site from April 11 to April 16, 2019. The property, the current and the future infrastructures location, the road, the surrounding villages, the mine, the tailing and the process plant future location were visited.

Philippe Chabot visited the site from April 11 to April 16, 2019. The property, deposit geology, geological model, drill core and drill hole location, open pit, haul road and waste dump location were visited.

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, 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 East 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 shows the Property location in Senegal and Figure 4.2 shows the outline of permits in the area.

Figure 4.1 Boto Project Location Map

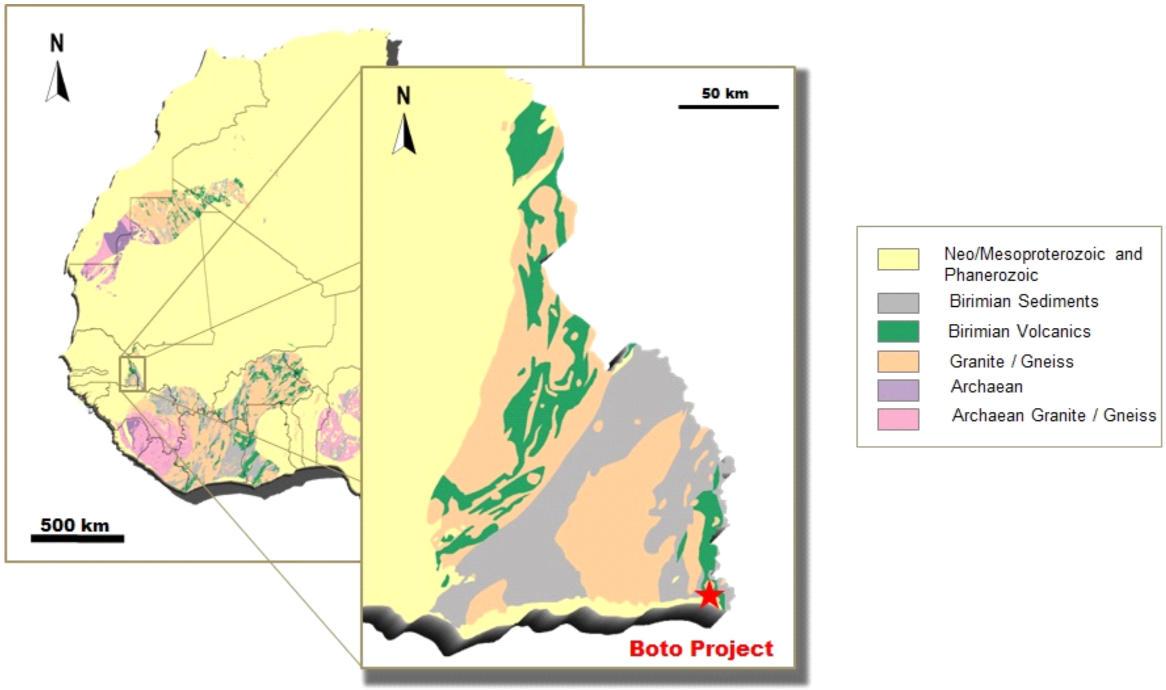
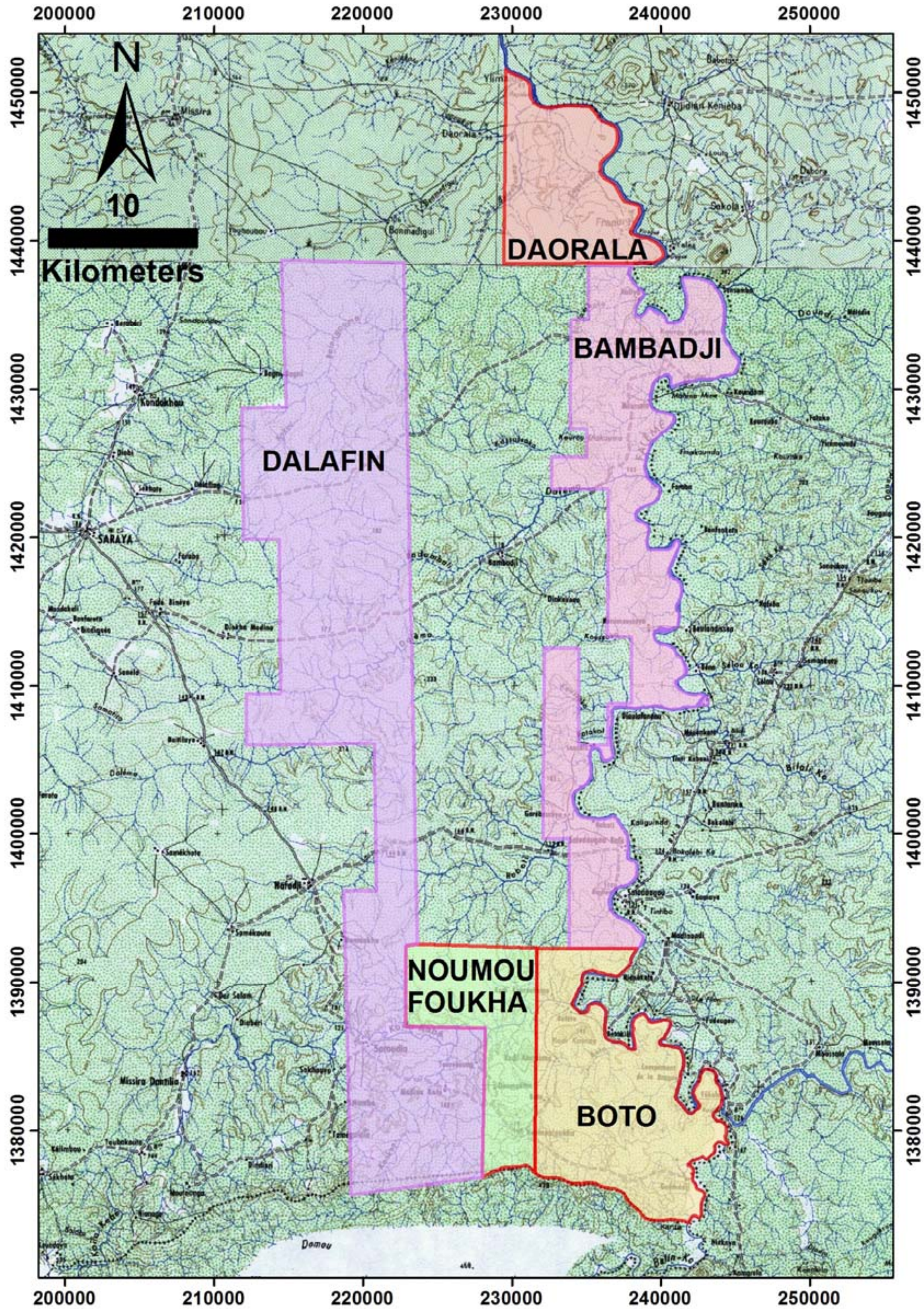


Figure 4.2 Exploration Permits in Daorala-Boto Area



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

Permit Type	Sector	Area (km^2)	Expiry Date	Status
Exploration	Daorala	88	4 March 2019 (Arrêté 13984 MMI/DMGrs)	Permit request in progress since October 2018
Mining Concession	Boto	148	Mining permit granted for a period of 20 years, renewable for every 10 years.	Mining permit granted on December 31st, 2019

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

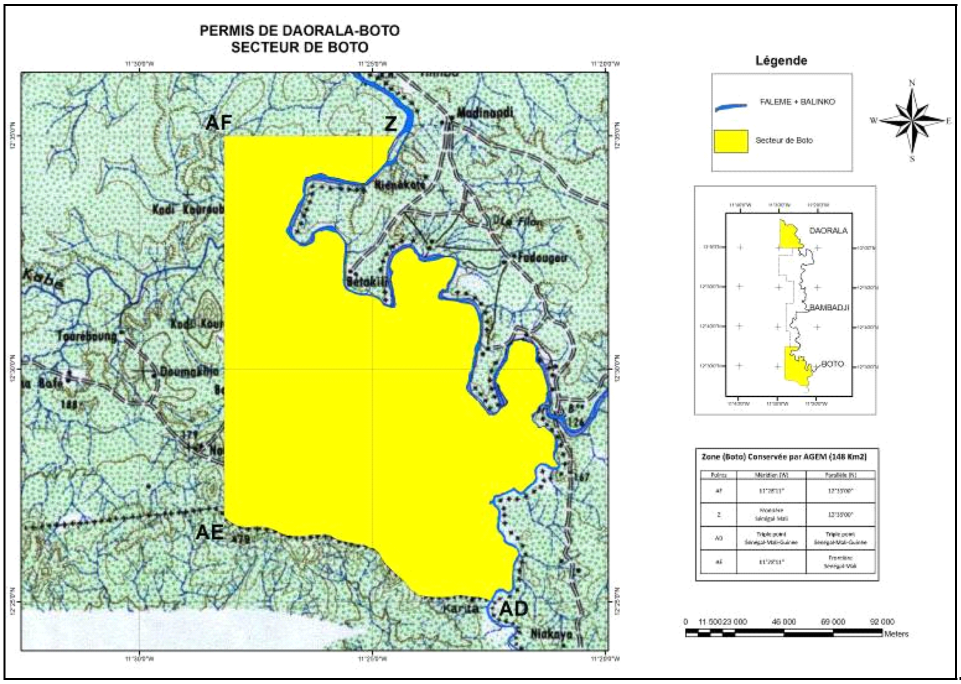
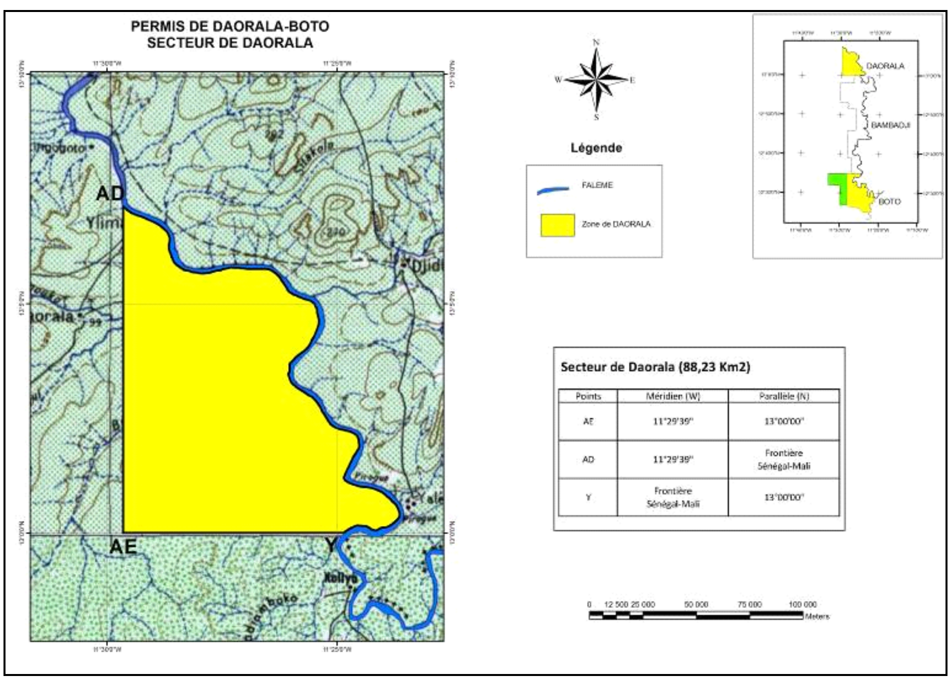


Figure 4.4 Daorala Property Map



An application for the Boto exploitation permit has been submitted in October 22, 2018. At the same time, an application for renewing the exploration permit for Daorala has been submitted. The mining permit has been obtained and received the notification of the decree on December 31, 2019. The request for Daorala research permit is still in progress.

4.3 Bambadji Joint-Venture (IAMGOLD/Barrick)

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

The Bambadji property is a joint venture between AGEM (IAMGOLD) and formerly Randgold Resources Ltd. (Randgold) became Barrick Gold Company (Barrick), a Canadian company headquartered in Toronto. **Randgold merged with Barrick on September 2018.** Barrick holds the mineral rights to the Bambadji property which consists of a single permit, 343 km^2. The Bambadji 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 Barrick. The distribution of the share capital is 65% Barrick and 35% AGEM.

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 Noumoufoukha permit, in the south, and extends approximately 70 km north, east of the town of Saraya.

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%).

- EMC submitted the request to renounce the DALAFIN exploration permit on July 31st, 2019 for the benefit of the joint venture company Stratex-EMC SA.

- On August 1st, 2019, the company Stratex - EMC expressed its interest in taking over the DALAFIN permit under the name of SENELA and submitted an application for an exploration permit which is still in progress.

- The main objective of this resumption of the permit, after the significant investments made, is to allow, within the framework of our agreements with AGEM Senegal, to intensify the works already in progress. The funding commitment is $ 8 million for the next 4 years.

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

4.5 Noumoufoukha Permit

IAMGOLD has applied and obtained the June 5, 2018 by the arrêté 012395/MMG/DMG the Noumoufoukha permit between Boto part of Daorala-Boto permit and Dalafin permit.

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, 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. Certification for a 3 year period was granted on March, 8th 2019.

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. There is some cellular telephone coverage, which is supplied to this area of Senegal, and on the Boto 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 out 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 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.

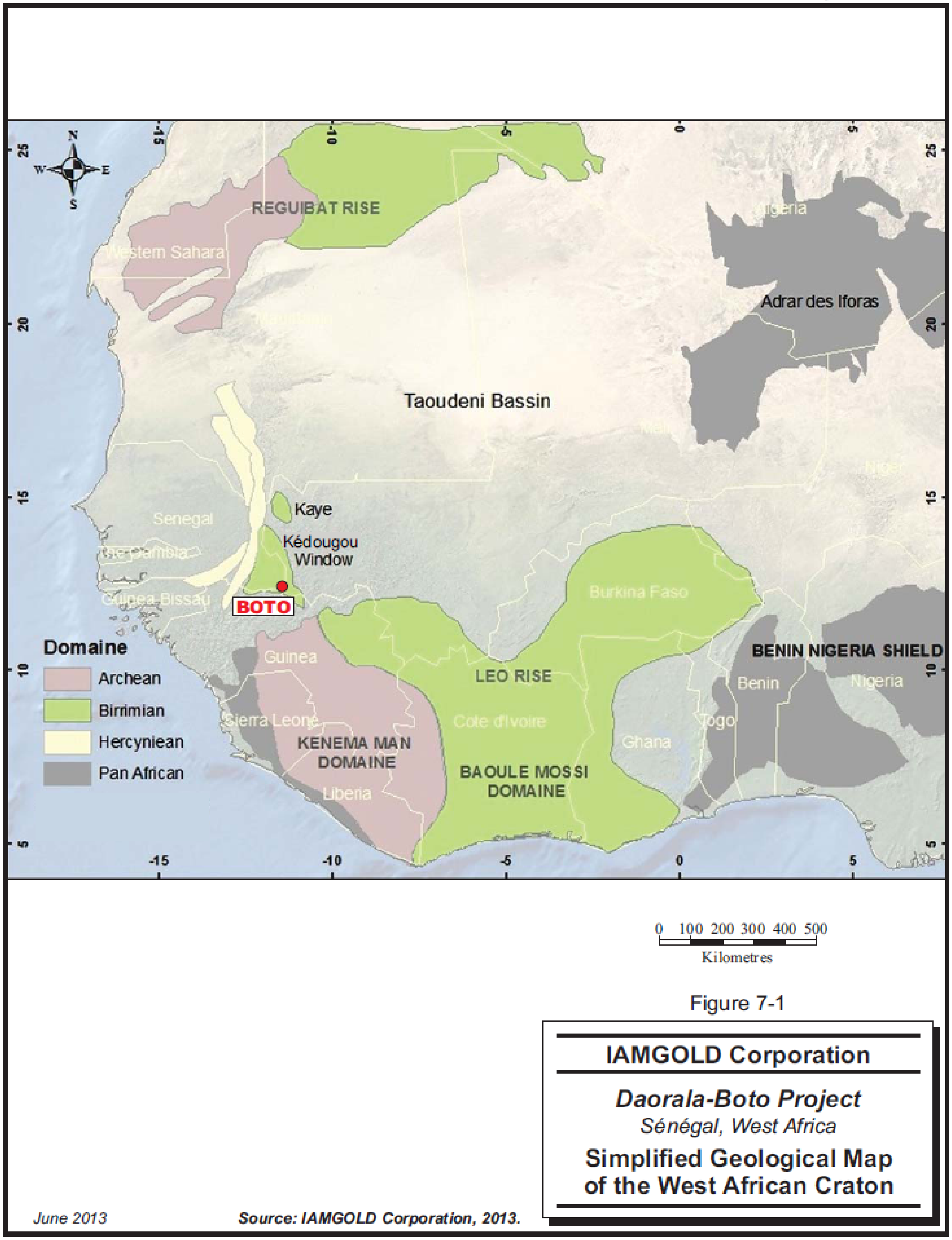
The bedrock of the WAC is exposed inside the Léo-Mann shield, the Reguibat Shield, and the inliers of a Kédougou-Kéniéba and Kaye. It comprises an Archean nucleus (3.0-2.7 Ga, Camil et al. 1983) covered by Lower Proterozoic rock (2.1 Ga, Abouchami et al. 1990; Hirdes et al., 1996) (Figure 7.1).

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 trending 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 Supergroup (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 Supergroup (Kofi and Kénjébandi Formations in Mali

Dialé-Daléma Supergroup is composed mainly of fine grained sedimentary rocks (to the West, in the Dialé Bassin) and of more dominant intrusive, volcanic, pyroclastic and epiclastic rocks (to the East, in the Volcano-Plutonic Arc of the Daléma Group) intruded by coalescent biotite-bearing granitic plutons of the Eburnean Orogeny (i.e., Saraya granite; Pons et al., 1992). Typical lithologies for the Dialé (Kofi in Mali) are folded mudstones, siltstones, and greywackes locally interbedded with calc-alkaline ash-and-lapilli tuffs; the Daléma (Kéniébandi in Mali) being composed of a coarse detrital base evolving to more arkosic facies along with large volumes of intermediate volcanics, pyroclastics and intrusives centered around a volcano-plutonic root (Bassot, 1987; Hirdes and Davis, 2002).

The Dialé serie has a higher proportion of chemical sedimentary rocks; typical lithologies are, from base to top: crystalline limestone and dolomitic marbles, greywacke, arenite sandstone, and schist (Milési et al., 1989). According to Schwartz and Melcher (2004), the Dialé serie has the most extensive occurrences of carbonate in the Birimian. 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.

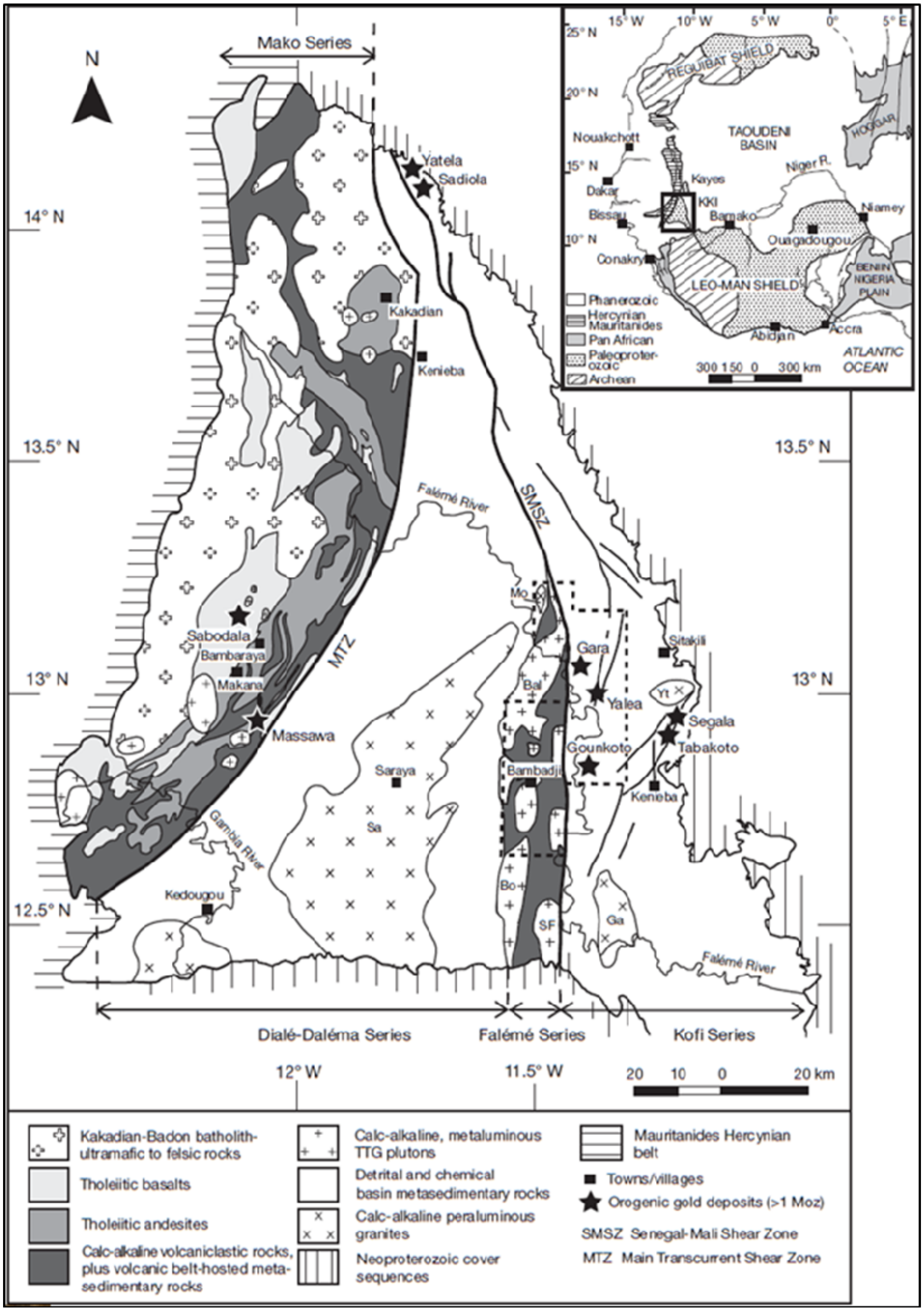
Table 7.1 Senegalese and Malian Terminology for the Birimian Formation

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

One of the important features of the Dialé-Daléma Supergroup 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 Supergroup 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 Supergroup 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 et 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 eastern edge of the Dalema Group (within the Falémé Series formerly known as the Dalema volcano-plutonic complex), a volcanic-plutonic belt that is wedged between the Dialé Group and the Kofi Series, and separated from the latter by the SMSZ. It could be chronologically correlated with the Mako Series but could also be likely slightly later. The easternmost part of the Boto 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 relatively 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 (often called the ''western Pelites'') is dominated by a volcano-sedimentary assemblage containing tuffaceous rythmites and tuffs, black shales (or graphitic pelite), carbonate rocks, hypovolcanics (microdiorite, andesite, pyroclastic and magmatic breccia or agglomerate), and dioritic intrusions. The Boto 5 deposit and the Western sterile part of Malikoundi are located inside this domain. The base of the sequence is composed of a relatively massive and continuous layer of impure bedded limestone (locally called cipolin).

Immediately east, the central Siliciclastic domain is dominated by a detrital assemblage composed of greywacke and sandstone (+/- quartzite), called the ''Guémédji sandstone''. It is unclear whether these sandstones/wackes are part of the Kofi / Dialé or of the Dalema unit. The exact stratigraphical relations with the surrounding units are not very obvious given the often important level of strain. However, the (westward/upward) apparent increase in carbonate content near the contact with the main carbonate layer (also corresponding to the main tectonic break) would suggest that the Guemedji Sandstone to be part of Daléma Unit.

The western and central domains are separated by a North-South trending high strain structure (010˚ N) that is well defined in all geophysical data and very evident in drilling. This highly deformed sinistral-reverse corridor 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 display local orientations averaging 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. In the vicinity of the SMSZ, the whole stratigraphy is affected by a sinistral-reverse mass movement causing a counter clock-wise rotation and N-S reorientation of the units (mainly obvious in plan view) as well a verticalization (best seen in E-W cross section). This causes all three units to trend approximately N-S (000°) along the SMSZ corridor. 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. **Error! Reference source not found.** demonstrates examples of some of the characteristic lithologies at Boto.

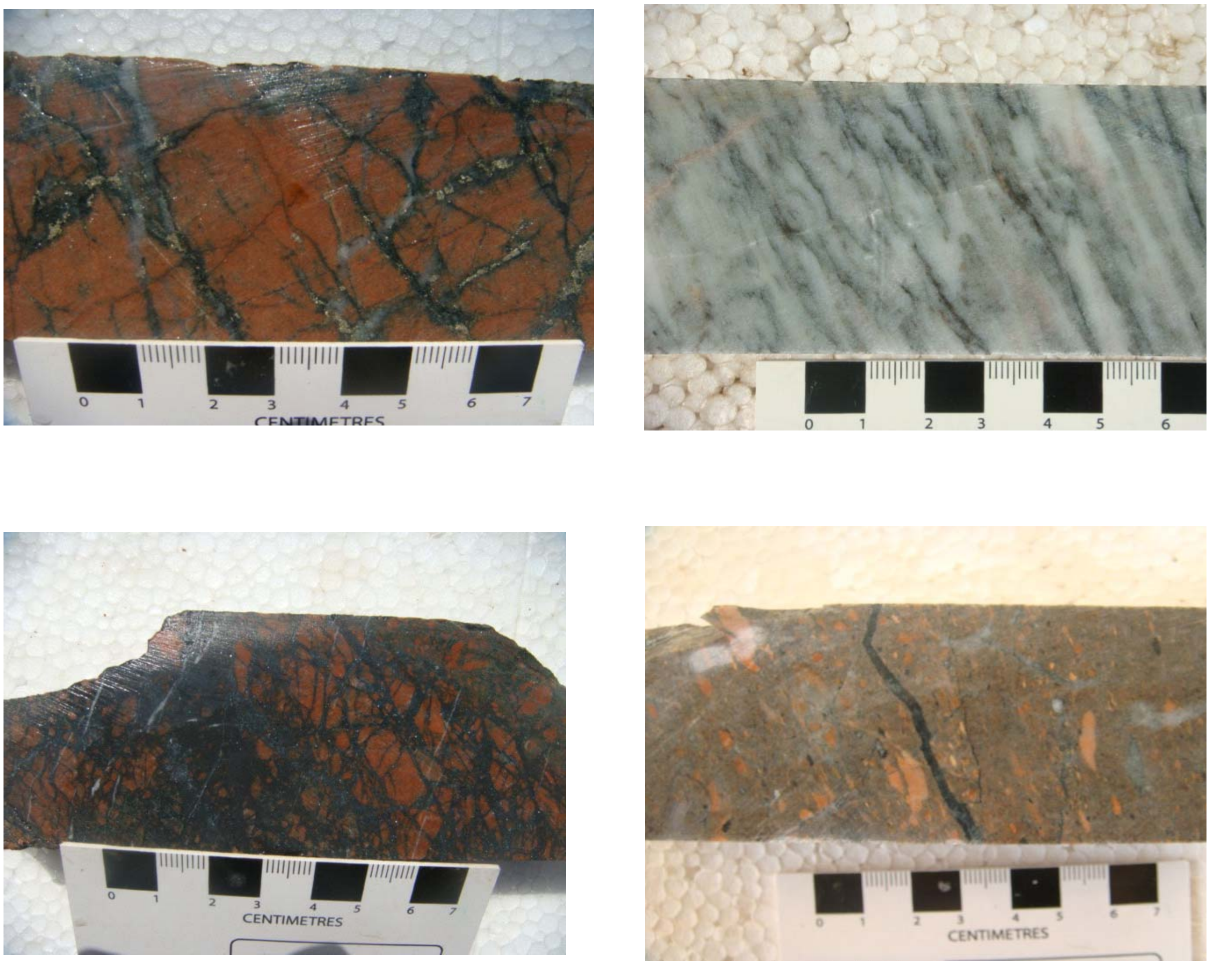
The known gold deposits at the Project occur on the margins of the central deformation corridor (Figure 7.4). Boto 5 lies along the western boundary (contact with carbonaceous turbidites) while Malikoundi/Boto 2, Boto 4, and Boto 6 lie along the eastern boundary (contact with the Guémédji sandstone).

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 $\pm$ magnetite; B) Unclean striated Marble; D) Agglomerate with stretched fragments.

Figure 7.4 Litho-structural Map of the Boto Project

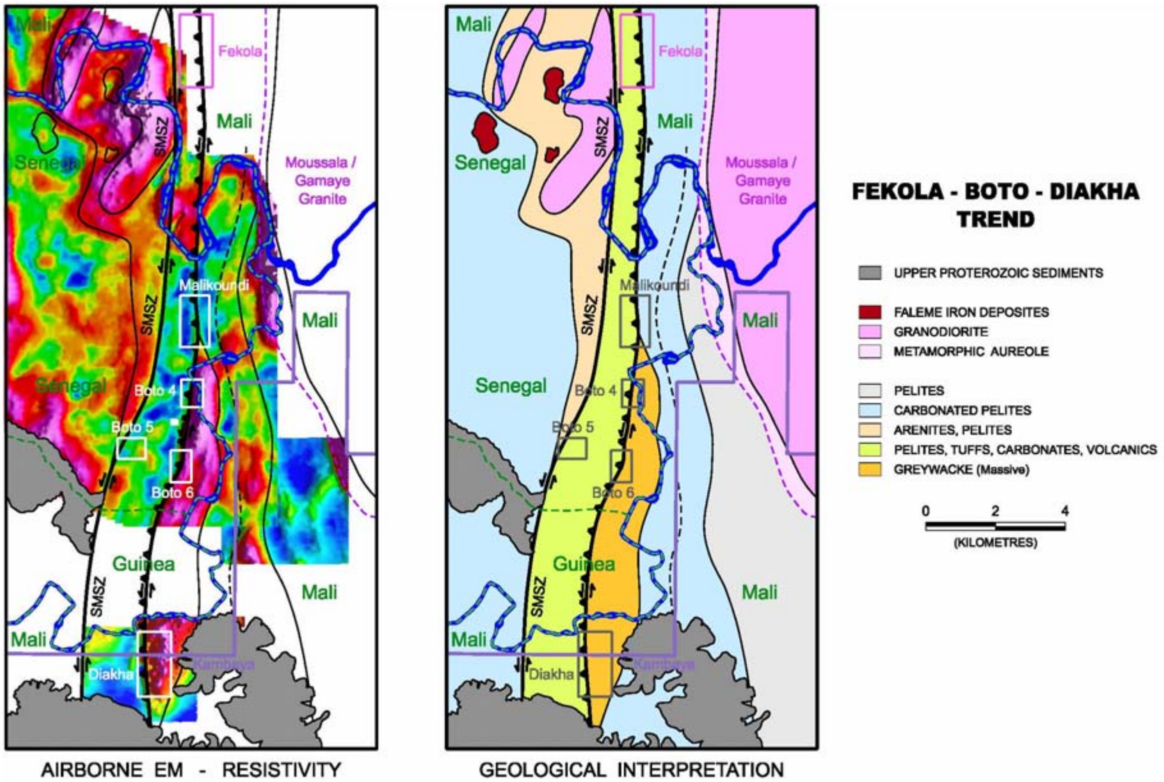
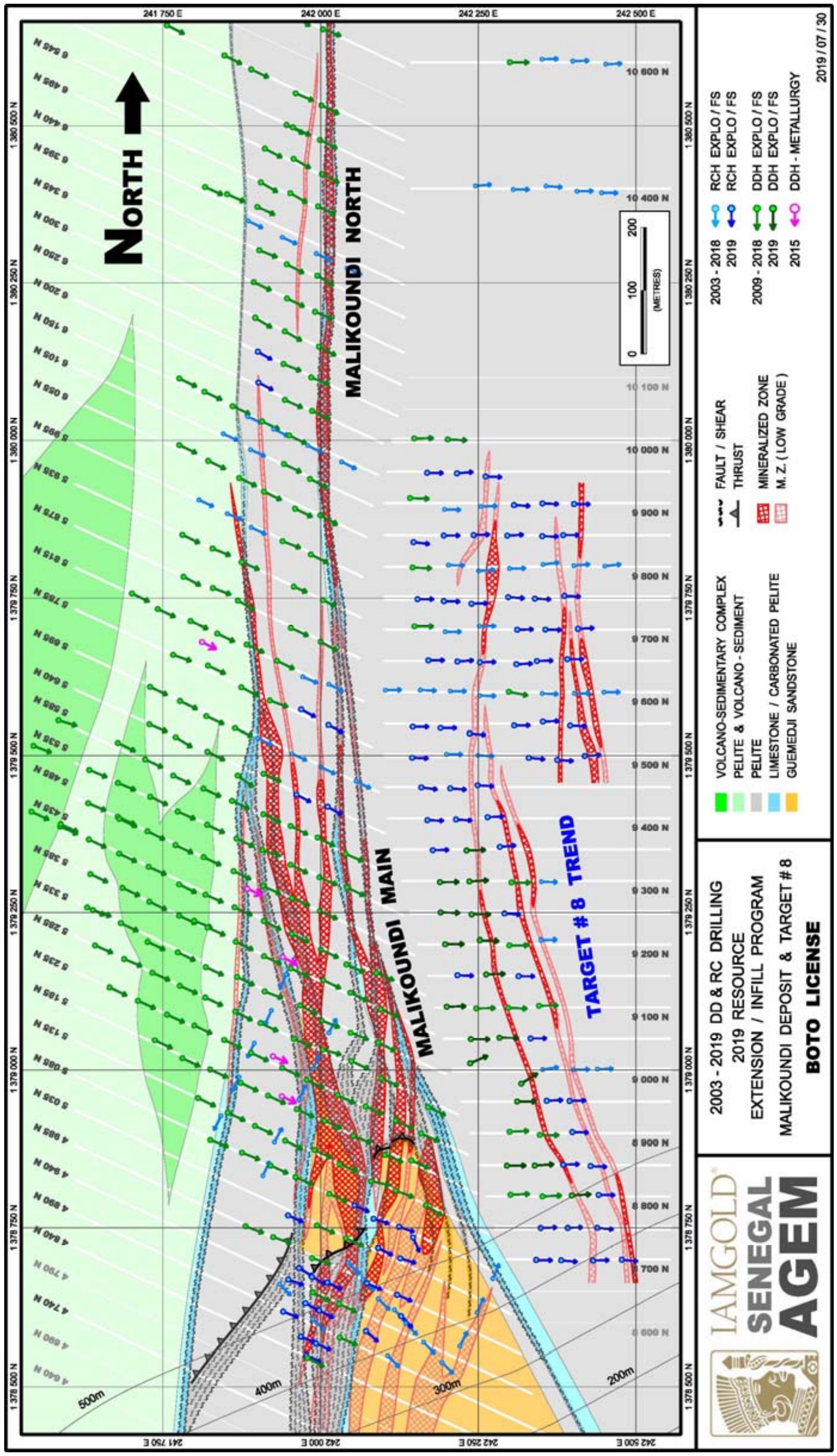


Figure 7.5 Geological Map of Malikoundi



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. 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 (**Error! Reference source not found.**) 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.

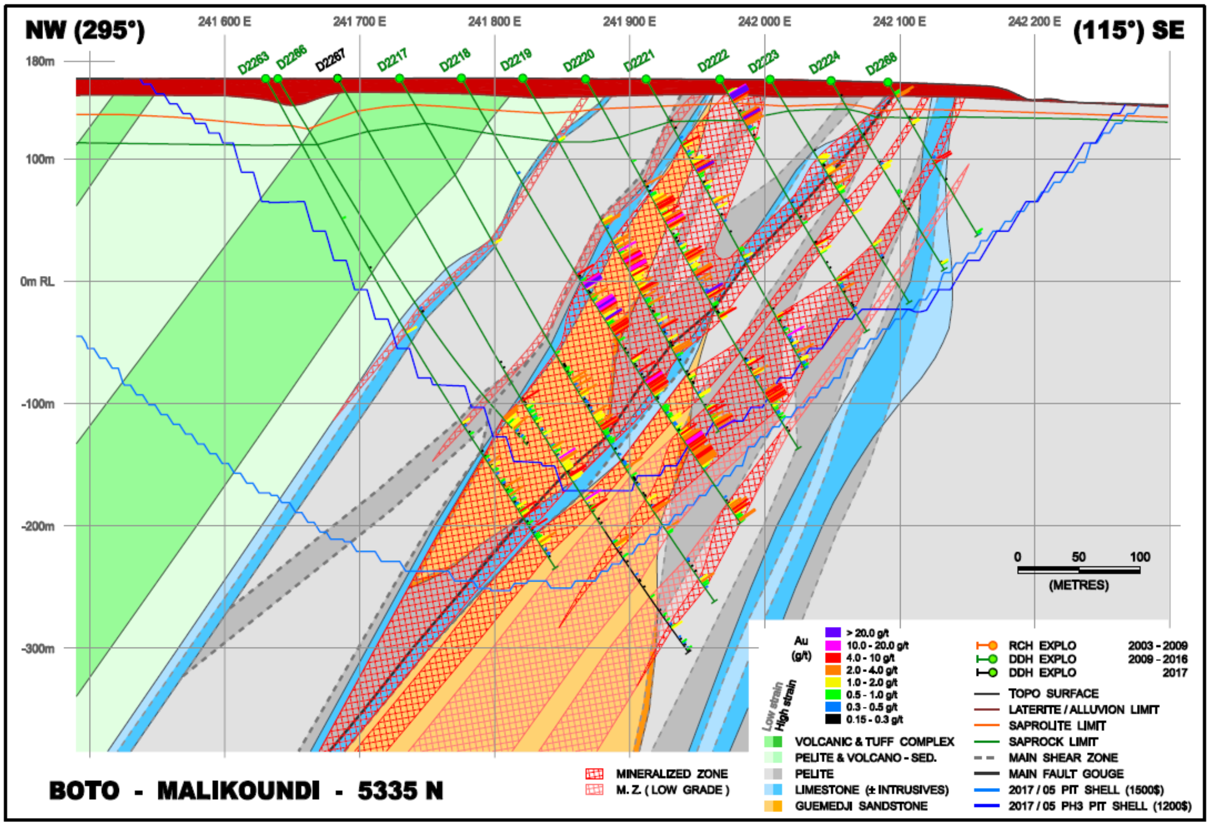
At Malikoundi/Boto 2, a 30° to 60° westward-dipping thrust fault has been observed in drill core at the contact between the Guémédji sandstone and the sequence of marble/laminated sediments. Of particular interest is a large lens of Guémédji sandstone that lies above the fault (Figure 7.6). This over-riding 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.

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

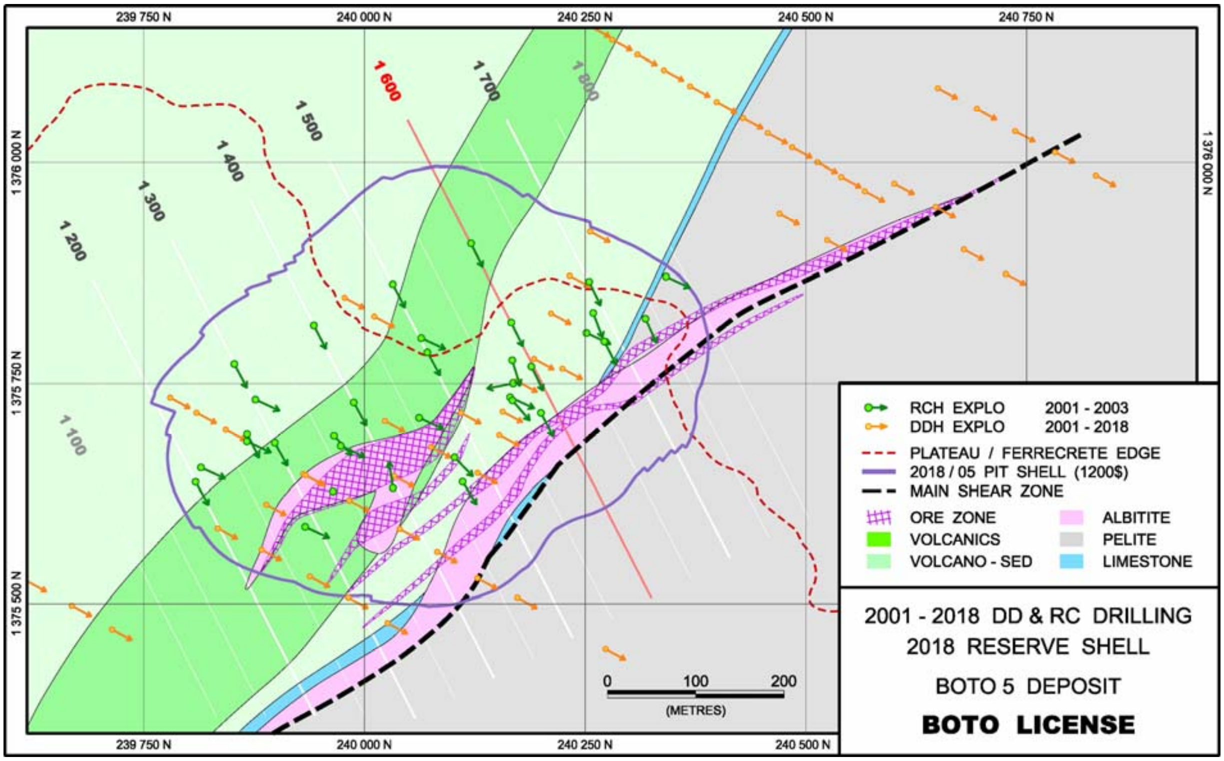


Boto 5

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.

From west to east, the lithological units at Boto 5 strike 015°-020° and are composed of a layer of shale (often possibly graphitic), followed by a complex sequence of volcano-sedimentary material including tuffaceous sediments, tuffs, magmatic breccias and hypovolcanic intrusives that appears to be overlying a relatively thick and continuous layer of bedded limestone (in a configuration identical to the material present in the western sterile of Malikoundi). Two separate bodies of intrusive albitite host the mineralization at Boto 5 and cross-cuts the stratigraphy, striking 045° N dipping between 45° W and 60° W towards the west (**Error! Reference source not found.**).

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.

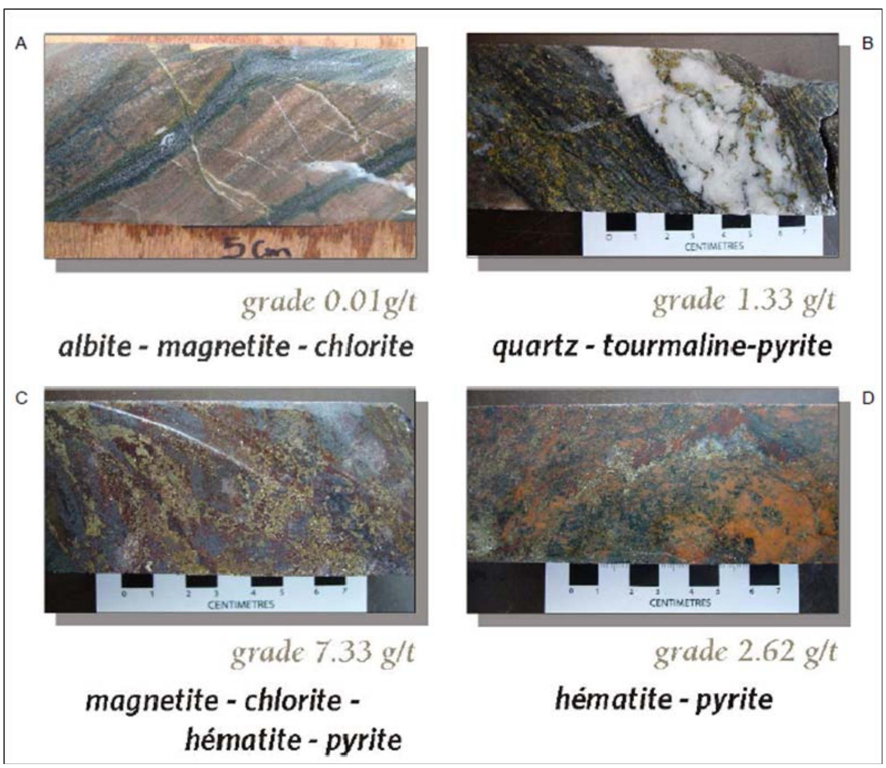
The third phase of alteration is characterized by a wallrock alteration associated with the emplacement of quartz-tourmaline-pyrite veins. Finally, the initial phase of chlorite-magnetite alteration is locally overprinted by pyrite-hematite-calcite alteration.

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 albitite intrusion that penetrates and covers a north-northeast striking sequence of volcano-sediments overlying a layer of 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 (either in sediments or pyroclastic units) is only very limited and insignificant in volume. 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 or in the highly albitised pelite located in immediate continuity, 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:

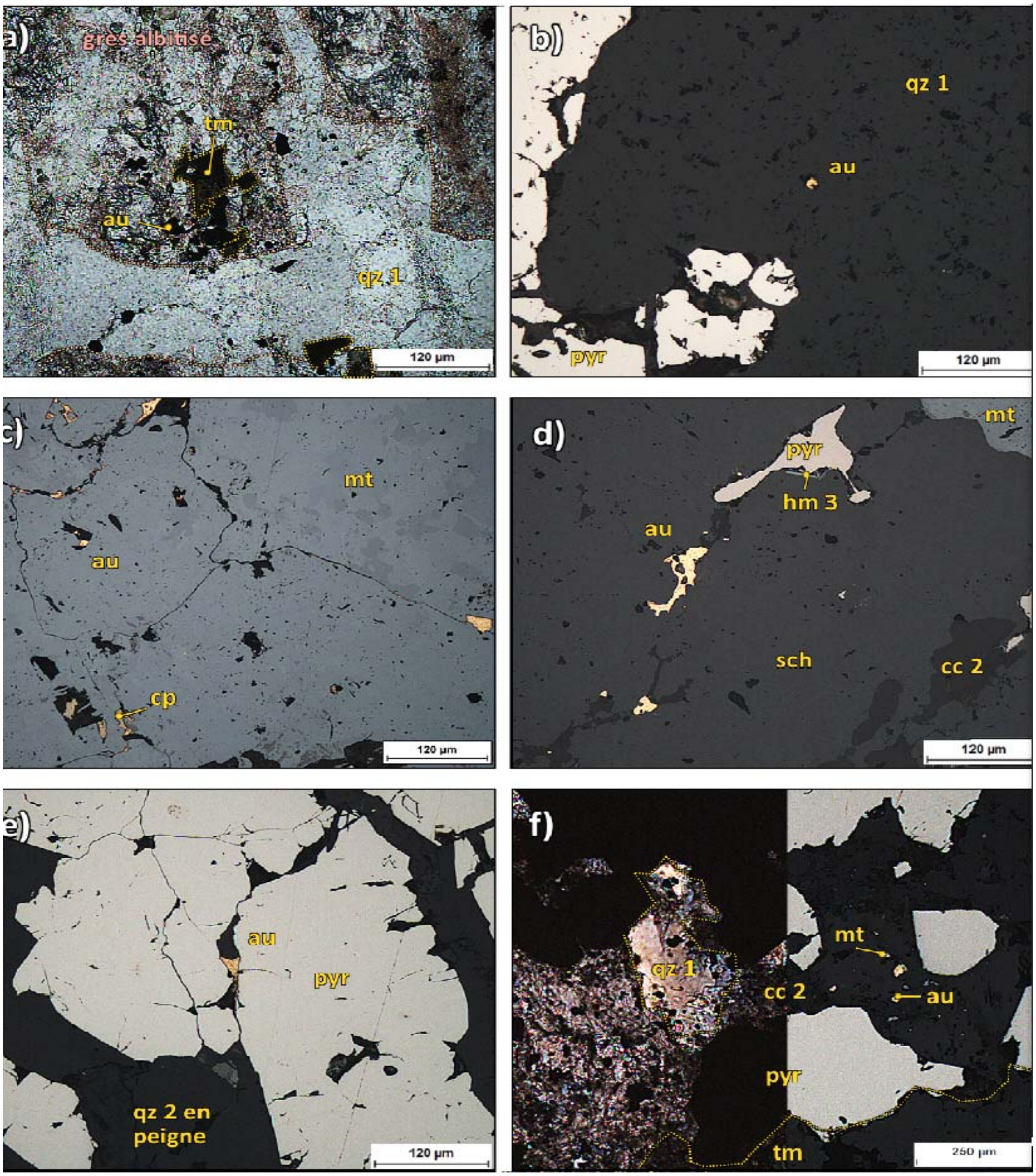
- Chlorite-albite alteration and magnetite-hematite-chlorite veining; calcite-tremolite alteration in distal settings.
- Quartz-tourmaline-pyrite alteration and veins.
- Hematite-calcite-pyrite alteration and veins.

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 Boto 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 Helicopter Electromagnetics (HeliTEM) 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)	13,098 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 totalling 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)	23,414 m in 132 holes
RC Drilling	11,808 m in 119 holes

9.5 Exploration by AGEM, April 2018 – May 2019

A 3D inversion of previous airborne EM has been done to define some new targets.

From April 2018 to end of May 2019, 5,727 m of DD and 19,905 m of RC have been drilled:

- To test some of previous 2017 targets.
- To define the eastern trend of Malikoundi at 50m by 50 m spacing.
- To define Boto 2 at 50 m by 50 m spacing with locally infill at 25 m by 25 m.

- Condemnation drilling for future infrastructure.

Table 9.4 Summary of Exploration Activity by AGEM, April 2018-May 2019

Type of Work	Details
Geophysics	Inversion 3D of previous EM
Diamond Drilling (DD)	5,727 m in 34 holes
RC Drilling	19,905 m in 178 holes

9.6 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, an exploration targeting workshop was organized where various dataset were re-assessed and some fifteen new 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.

Since March 2018, some of the 2017 targets have been tested and the eastern trend of Malikoundi and Boto 2 have been defined at 50 m X 50 m. The eastern trend has been followed on 1.2 km and is still open to the north and partially to the east.

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 May 2019.

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 up to and including May 2019.

Table 10.1 Summary of Drilling for the Project, 2000 – March 2019

Year	DD		RC		Total	
	Metres	Number	Metres	Number	Metres	Number
2000	1,117	8	177	2	1,294	10
2001	2,057	13	2,080	23	4,137	36
2002			1,593	24	1,593	24
2003			3,292	52	3,292	52
2007	2,639	11	10,687	107	13,326	118
2008	3,721	18			3,721	18
2009	3,880	17	7,618	73	11,498	90
2011	284	1			284	1
2012	13,322	50			13,322	50
2013	13,130	52			13,130	52
2014	16,223	60			16,223	60
2015	14,856	58			14,856	58
2016	6,139	38			6,139	38
2017	11,853	69	3,997	41	15,850	110
2018	2,452	14	5,312	53	7,764	67
2019	5,727	34	19,905	178	25,632	212
Total	**97,400**	**443**	**54,661**	**553**	**152,061**	**996**

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

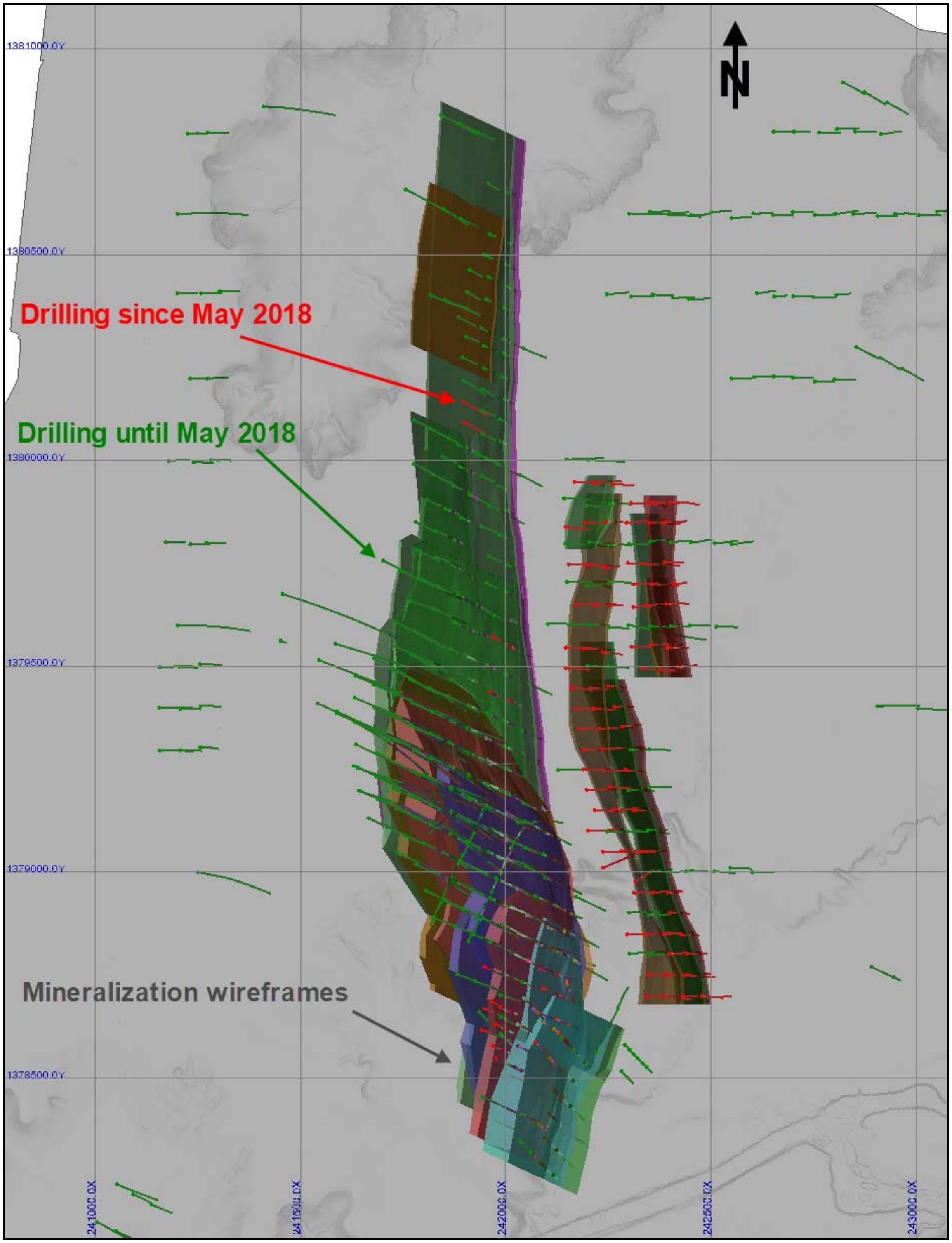


Figure 10.2 Drill Hole Location Map; Boto 5

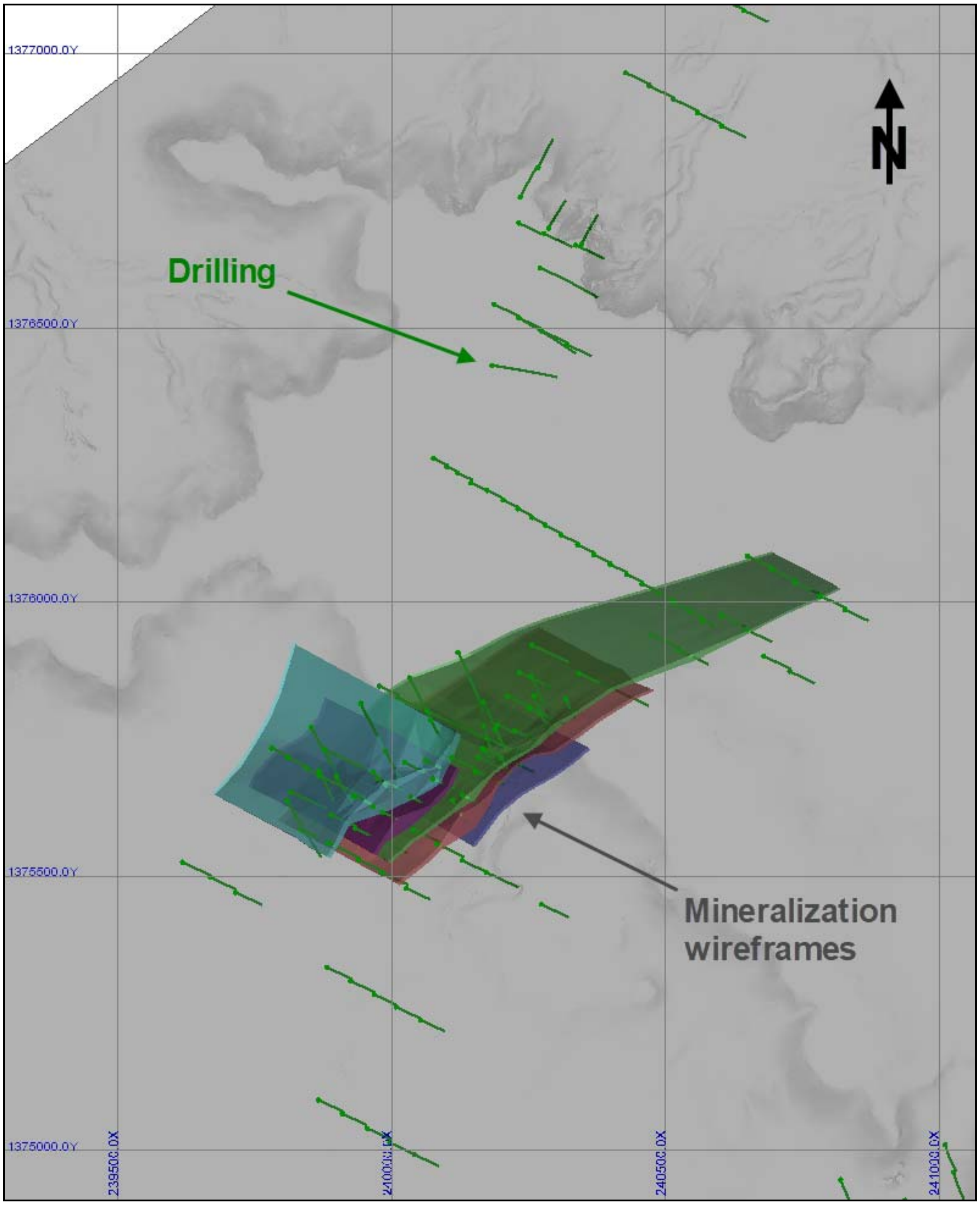


Figure 10.3 Drill Hole Location Map; Boto 6

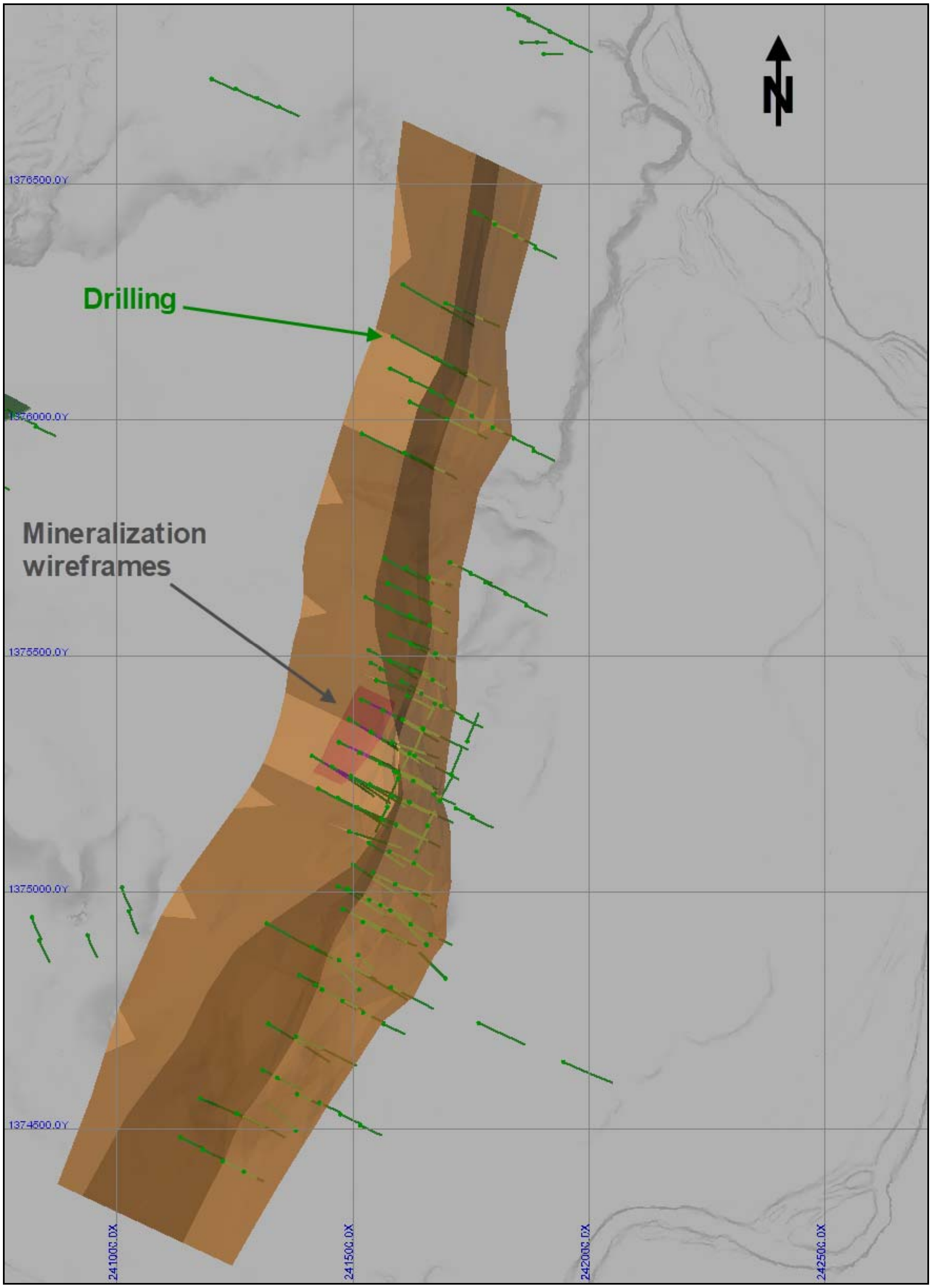
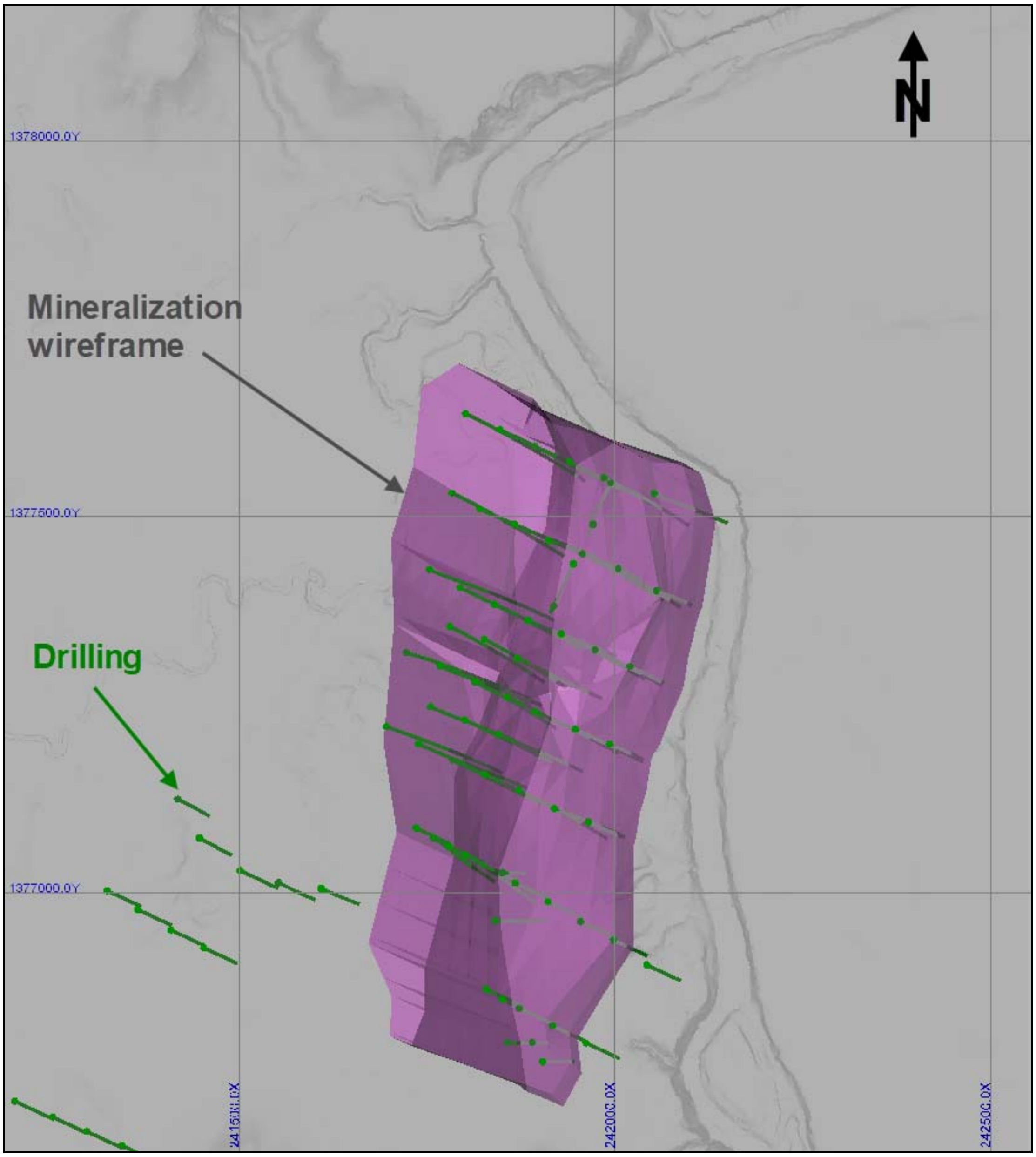


Figure 10.4 Drill Hole Location Map; Boto 4



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 (DGPS). 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 10 m in laterite, saprolite and after saprock and every 25 m in fresh rock 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, 2 m or 3 m 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 1 m 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 2 kg to 3 kg 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 one metre 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 was 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, all one metre and 20 kg mean weight, were completely dried before being divided into two kilogram samples using a riffle splitter. The 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. A portion of crushed sample was then pulverized to have P_{80} of passing 120-mesh. A 30 g aliquot was fire assayed.

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	FA30 g	Re-analysis of Anglo soil samples of 1995
2000	Chimitec in Val-d'or	FA30 g	
2001	Chimitec in Val-d'or	FA30 g	
2002	Chimitec in Val-d'or and Abilab Bamako	FA30 g	
2003	Abilab Bamako	FA50 g	
2004	Abilab Bamako	FA50 g	
2005	Abilab Bamako	FA50 g	
2006	Abilab Bamako	FA50 g	Abilab acquired by ALS Chemex
2007	ALS Chemex Bamako (ex-Abilab)	FA50 g	
2008	ALS Chemex Bamako (ex-Abilab)	FA50 g	

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 included certified standards to comply with the new procedures. From that point on the validation procedures were 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.

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, with certified 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 is 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 were removed from the sample bags and checked against the chain of custody form. Each sample was weighed and assigned a bar code number and a unique file number. The information of the sample was 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 P_{75} of 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 P_{80} 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 aliquots, with a lower detection limit of 5 ppb and an upper detection limit of 10 ppm. 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.

Since 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 P_{70} of 2 mm, with the size of the particles checked on a regular basis. The sample is divided and homogenized to obtain a representative sub-sample before pulverization to P_{85} of 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.

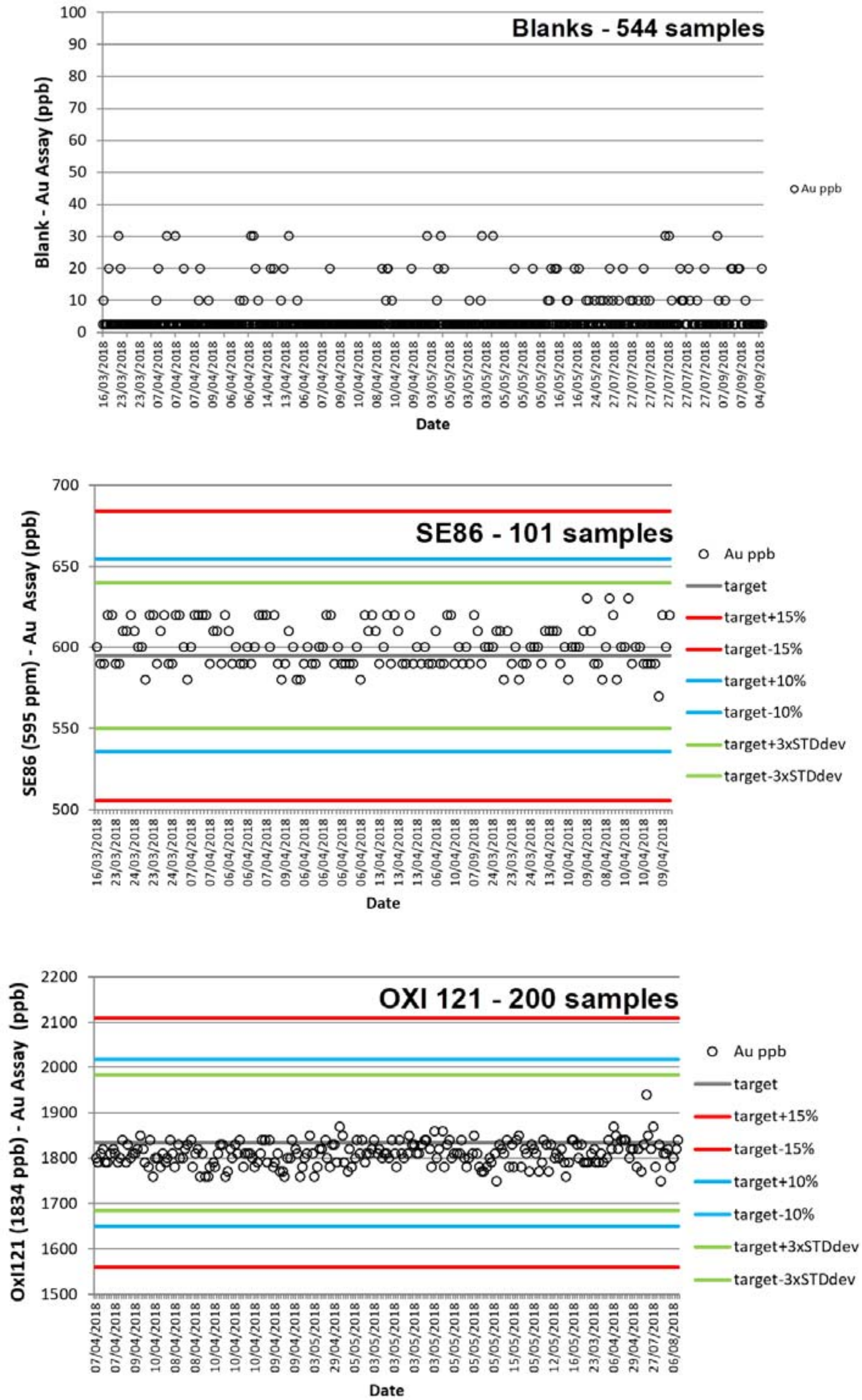
The DD and RC samples are assayed using fire assay with atomic absorption finish on 50 g of pulp (Véritas Code FA450), with a lower detection limit of 10 ppb. Samples of core that are analyzed using fire assay with atomic absorption finish and return a result greater than 10 ppm are re-assayed using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

An example of QA/QC samples inserted in the RC sample stream in 2018 is shown in **Figure 11.1**. The blank material, a low grade certified material SE86 (nominal certified value of 595 ppb Au) close to cut-off grade, and medium grade certified material OXI121 (nominal certified value of 1,834 ppb Au) close to the deposit average grade show good performance.

Figure 11.1 Performance of Blanks, SE86 CRM and OxI121 CRM from 2018 RC Sample Stream



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 was placed in each bag. 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. RPA 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

RPA conducted a site inspection of the Boto project from May 29 to June 1, 2019. There was no drilling activity at the time of the visit. During the site visit, RPA personnel reviewed the deposit geology with the site geologists, visited the Boto deposits, took GPS readings of collar positions, reviewed core from drilling with relevant intercepts, reviewed logging and sampling procedures and visited the site logging, sampling and storage facilities. During the visit, discussions were held with:

- Mr. Benoit Michel, IAMGOLD, Boto Project Manager, Exploration.
- Mr. Philippe Biron, IAMGOLD, Senior Resource Geologist, Exploration.
- Mr. Guillaume Bredillat, IAMGOLD, Project Geologist, Exploration.

Drill core for the Boto Project is logged, sampled, and stored in two locations: (1) at the Boto Exploration Camp, situated approximately 12 km due west of Malikoundi and (2) at the New Camp, situated approximately 1.5 km west of the Malikoundi deposit.

The logging and storage facilities are appropriate. Logging was completed by geologists. The sampling is done generally for the entire length of the hole. The logging and sampling are conducted to industry standards. Core samples are stored in metallic core boxes, while RC samples are stored in rice bags.

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, however, 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. RPA took readings of several collar position from holes in each of the Boto deposits. There were three collars checked for Boto 5, three for Boto 6, and nine for Malikoundi – Boto 2. The RPA handheld GPS readings returned position values within 5 m of the collar positions recorded in the drill hole database.

Core from six representative drill holes and chip boxes for three typical RC holes were reviewed. The logs presented sampling intervals and lithology description consistent with the core and RC chips.

12.1 Boto Database Verification

RPA performed the database validation routine specific for GEOVIA GEMS on relevant tables in the drilling database and no errors were identified. Additionally, RPA checked for zero/extreme values in the collar table, missing or extremely long intervals, extreme high values, overlapping or out of sequence intervals, and visually inspected drill hole traces for unusual azimuths, dips and deviations.

IAMGOLD provided RPA with original assay certificates in digital format for Boto samples. RPA focused on the certificates from 2010 to 2019 drilling campaigns (approximately 750 certificates), from which RPA randomly selected and compiled approximately 300 certificates. The compiled certificates matched approximately 20,000 samples from the drill hole database (15% of the database samples). RPA did not identify any differences between the independently compiled assays and the content of the resource database.

Typically, for deposits in the early stage of exploration independent check assay samples are collected during the site visit from relevant intercepts to confirm presence of mineralization. Given the advanced stage of the Boto project, RPA did not collect check samples from the Project.

12.2 QP Opinion

The QP is of the opinion that the database is acceptable for the purposes of resource estimation. In addition, the logging, sampling, and database management procedures follow industry standards.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

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

The testwork programs investigated the following main topics:

- Sample head assays, chemical analyses and mineralogy.
- Comminution properties.
- Response to pre-concentration (gravity separation, flotation).
- Cyanidation (whole ore CIL, gravity concentrate intensive cyanidation, gravity tailings CIL).
- Sedimentation (thickening) and rheology.
- 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.
- Keckes, Thomas "Thickening Test Report", Project 311872TQA, June 2018. Outotec.

- MacDonald, James. "An investigation into Recovery of Gold from Boto Project Samples", Project 15080-003, July 2018. SGS Mineral Services.

- MacDonald, James. "Rheology Results Summary", Project 15080-03, July 26, 2019. SGS Mineral Services.

13.1 Review of Previous Metallurgical Tests

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

13.1.1 SGS 2013 Testwork Program

The testwork conducted in 2013 was a scoping level metallurgical test program supervised by Pierre Pelletier of IMGOLD. 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 generation 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 g/t	Met #1	Met #2	Met #3	Met #4	Met #5	Met #6	Met #7	Met #8	Met #9
Au	10.7	2.43	2.85	2.68	5.81	2.15	1.48	1.01	4.16
Ag	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Al	55100	69200	47500	56700	42000	47700	153000	167000	64100
Ca	45600	51700	25600	23600	22700	16000	303	332	5420
Cu	22.1	4.1	12.2	8.3	12.2	7.7	6.1	406	141
Fe	85600	36900	54600	56700	62400	33800	52400	9130	27400
K	10900	9290	7830	3370	3210	5220	175	818	529
Mg	23800	36900	14800	19600	11600	8900	252	1900	13600
Mn	729	489	348	300	328	323	5.5	31.1	230
Na	25200	30200	23700	36400	28500	29300	625	810	43500
Ni	41	65	32	35	43	22	100	188	146
P	513	666	268	532	230	240	97	106	612
Sr	60.3	163	43.3	66.5	77.4	68.1	11.6	28.4	82.1
Ti	2140	2880	1690	2490	1310	1680	4010	6410	2110
V	66	126	48	82	48	47	15	101	31
Y	10.5	15.6	7.6	9.2	7.3	7.3	13.2	10.8	12.1
Zn	19	12	10	12	57	7	33	14	18

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

Met #7 and 8 are samples of weathered saprock and saprolite and 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.

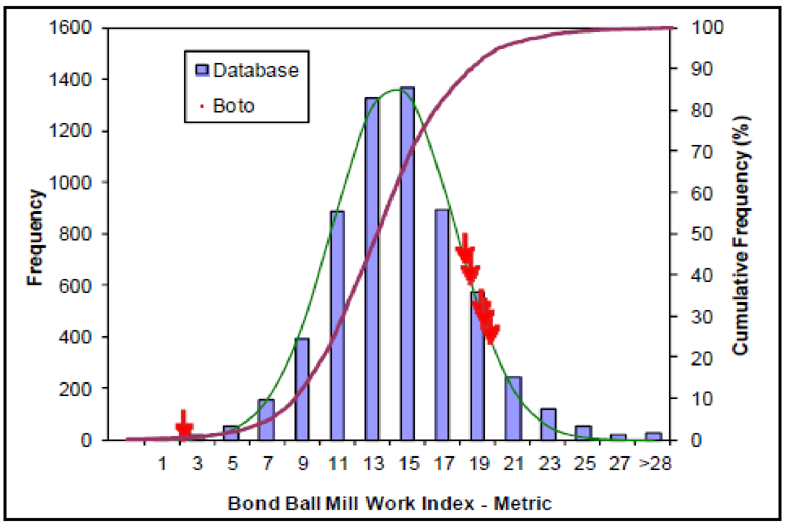
Grindability Testing

Met #1, 3, 4, 5, 6 and 8 were subjected to Bond ball mill grindability testing at a grind size of 106 µm (150 Mesh). A summary of the results is shown in Table 13.3. 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 µm	Feed Size, F_{80} µm	Product Size, P_{80} µm	Work Index, BWi kWh/t	SGS Hardness Percentile
Met #1	106	2,499	82	19.7	91
Met #3	106	2,457	84	19.2	90
Met #4	106	2,464	83	18.6	86
Met #5	106	2,541	86	18.3	85
Met #6	106	2,422	86	19.4	90
Met #8	106	2,211	62	2.2	0

Figure 13.1 SGS 2013 BWi Results Compared to BWi Database



At the time of this testwork, 3,232 Bond ball mill work indices were available in the database. Five of the samples tested were ranked 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, g/t Au*			
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 a pulp density of 40% solids, leaching for 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 g/t Au	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 targeted P_{80}'s of 90 µm, 75 µm 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 fineness of the grind and the extraction achieved.

Figure 13.2 Gold Extraction % vs. Grind Size

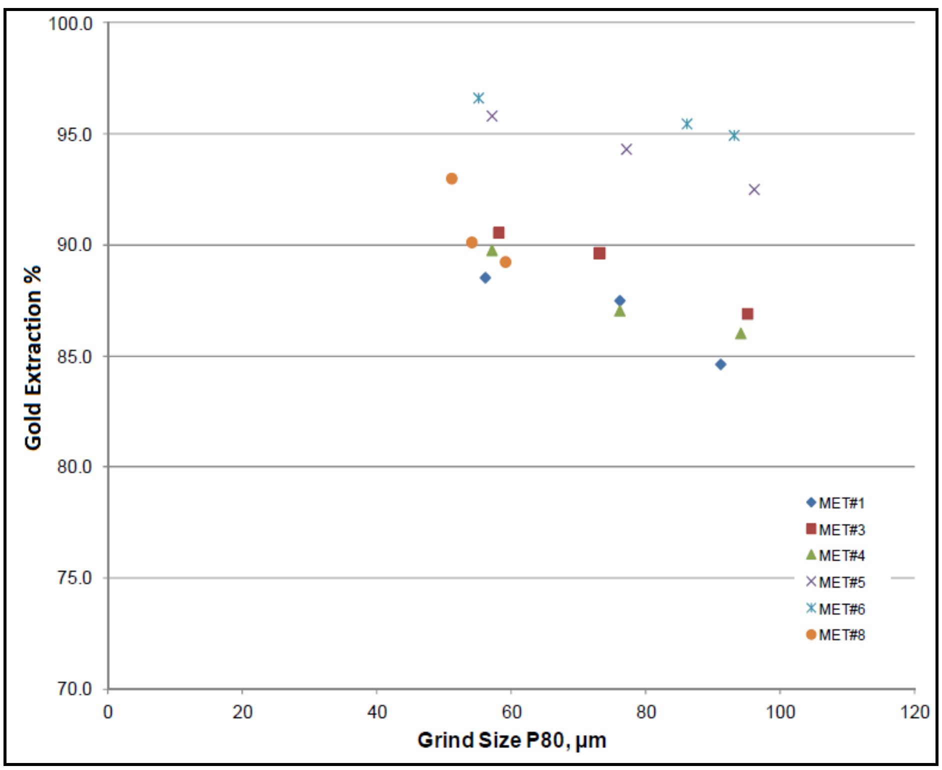


Table 13.6 SGS 2013 Gravity Tailings CIL Test Results

Sample	Feed from Test No.	CN Test No.	CN Feed Size P_{80} (µm)	Reagent Cons (kg/t) NaCN	Reagent Cons (kg/t) CaO	Au Extraction % CN 48 h	Au Extraction % Normalised[1] (to avg calc CN feed grade) CN	Au Extraction % Overall Grav	Au Extraction % Overall CN	Au Extraction % Overall Grav + CN	Au Extraction % Normalised[1] (to avg calc CN feed grade) Grav + CN	Au Extraction % Normalised[2] (to direct feed grade) Grav + CN	Residue Au, g/t	Head Au, g/t CN Calc	Head Au, g/t Avg Calc CN Feed Grade	Head Au, g/t Grav + CN	Head Au, g/t Direct (SM)
Met #1	G-1	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	5.34	8.20	10.74
		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			
		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			
Met #3	G-2	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	3.16	3.28	2.85
		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			
		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			
Met #4	G-3	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	2.47	2.93	2.68
		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			
		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			
Met #5	G-4	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	5.01	6.34	5.81
		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			
		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			
Met #6	G-5	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	3.72	3.86	2.15
		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			
		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			
Met #8*	G-6	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	0.83	1.09	1.01
		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		1.05	
		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	

**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 pulping with synthetic gold stock solution, the pregnant solution of all five samples increased in gold tenor, indicating that there is no apparent 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 with 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 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 g/t	Comp A	Comp B	Comp C
Au	1.81 (SM)	1.70 (SM)	1.01 (SM)
Ag	<2	<2	<2
Al	63800	72300	36600
Ba	224	106	124
Be	0.68	0.89	0.5
Ca	26800	43600	137000
Co	18	14	26
Cr	95	100	34
Cu	32.7	25.3	141
Fe	57100	40500	52400
K	4870	8680	7290
Li	22	18	8
Mg	20800	24300	59900
Mn	444	476	1350
Mo	<5	<5	10
Na	37900	41400	17700
Ni	31	42	48
P	447	651	328
Sr	61.3	80.8	102
Ti	2440	2910	1430
V	64	79	51
Y	10.3	13.9	11.9
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 which 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

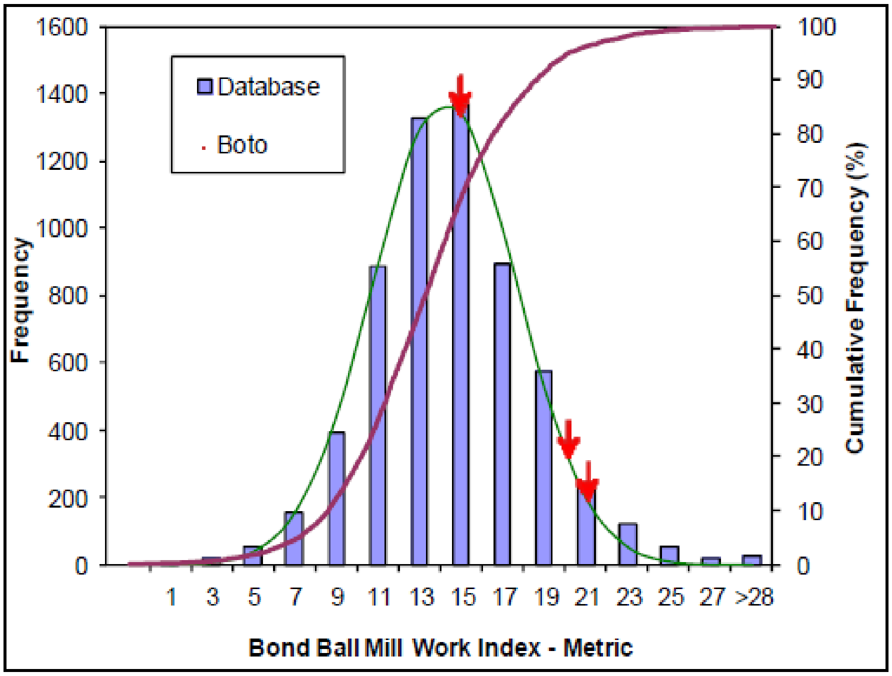
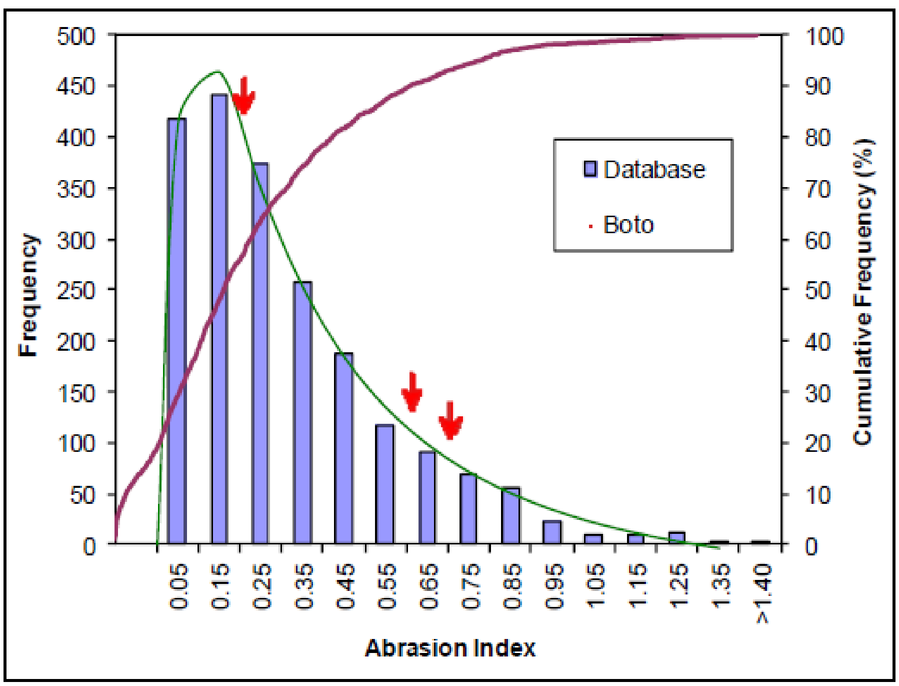


Figure 13.4 SGS 2014 Bond Abrasion Results Compared to Abrasion Database



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 g/t Au	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 results indicated gravity recoverable gold of 15% to 30%. The recommendation at the time of this testwork was that gravity separation could be included in a scoping-level flowsheet to decrease the complexity and size of the downstream cyanidation circuit.

Flotation Testwork

Whole ore bulk sulphide 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

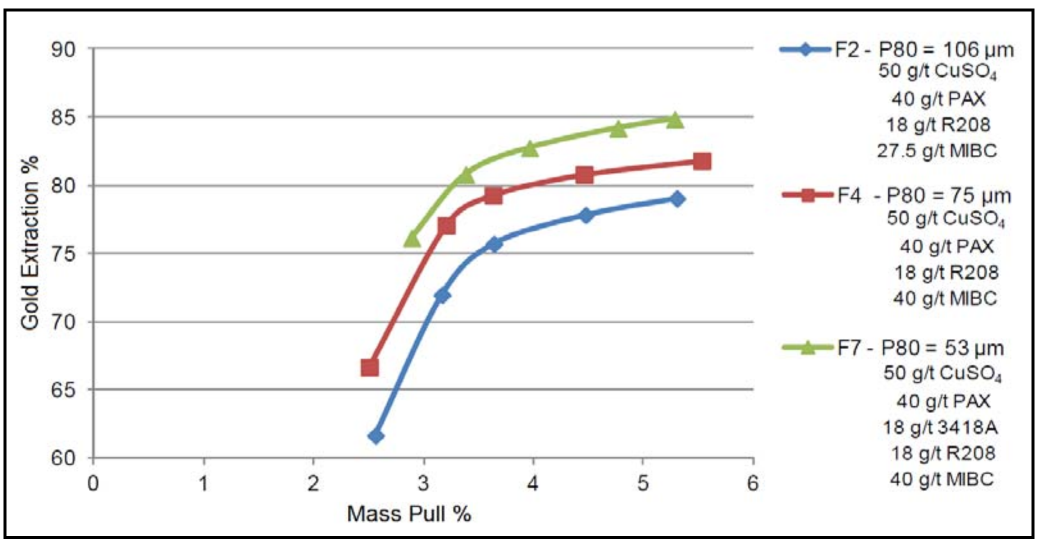


Figure 13.6 SGS 2014 Comp B Flotation Gold Extraction vs. Mass Pull

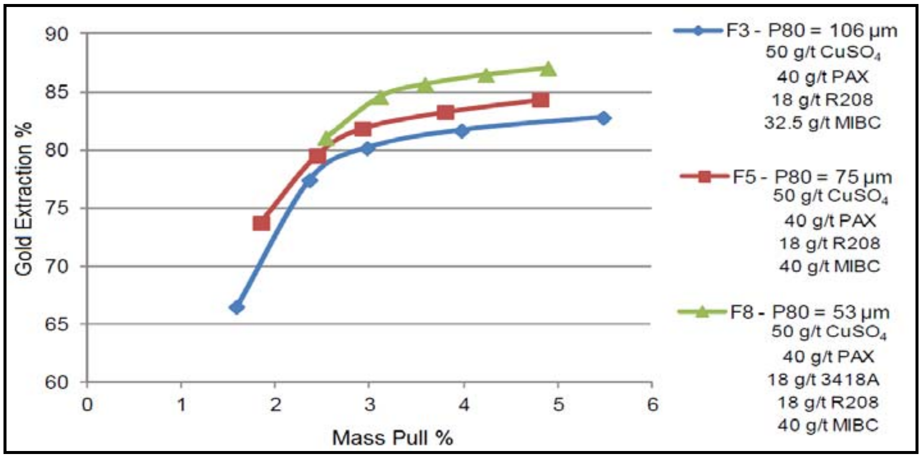
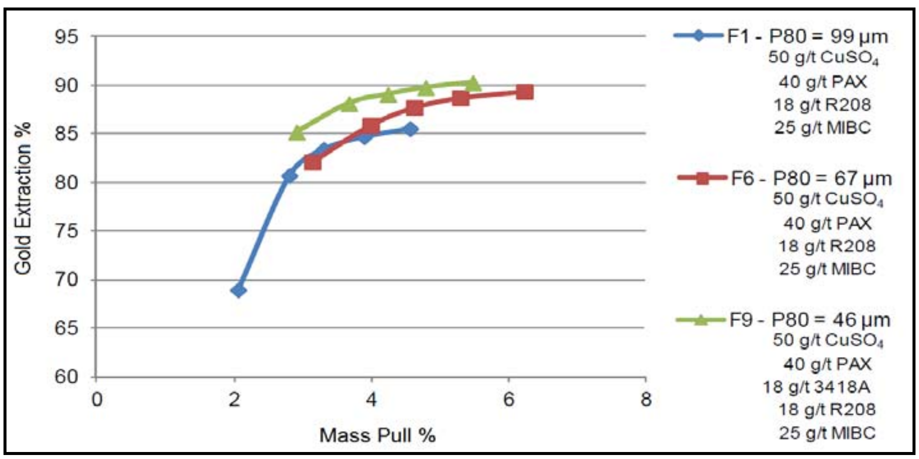


Figure 13.7 SGS 2014 Comp C Flotation Gold Extraction vs. Mass Pull



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.

The 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 47 µm.

Gravity tailings 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 tailings) 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 day 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/t Au	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

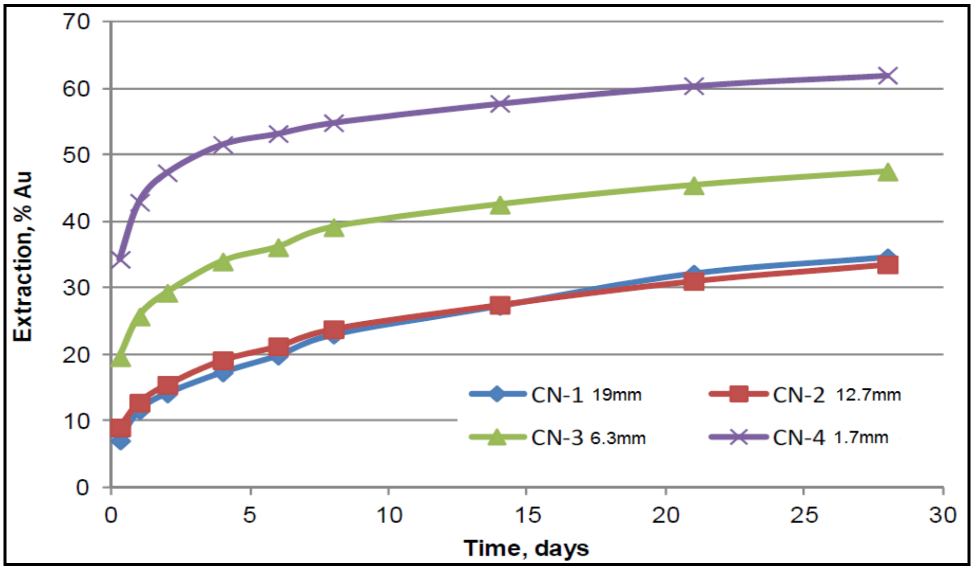


Figure 13.9 SGS 2014 Comp B Coarse Bottle Roll Gold Extraction vs. Time

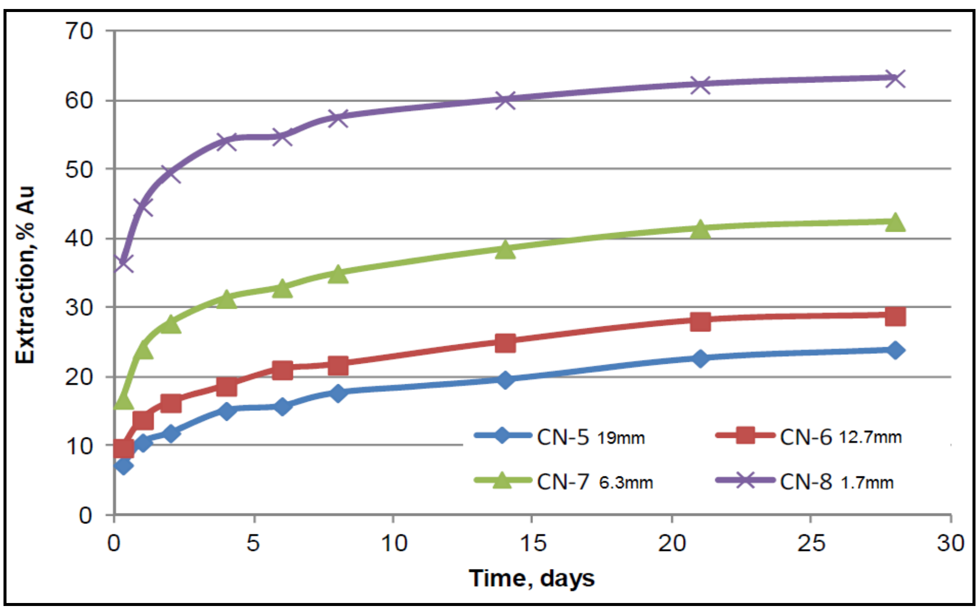
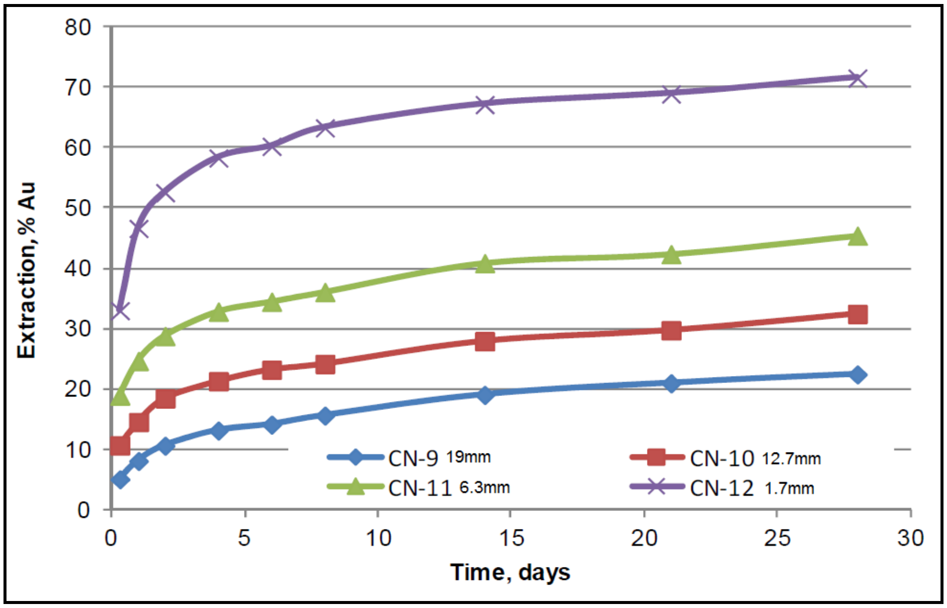


Figure 13.10 SGS 2014 Comp C Coarse Bottle Roll Gold Extraction vs. Time



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-day 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 /t NaCN. Lime consumption ranged from 0.7 kg to 1 kg CaO/t for all the composites.

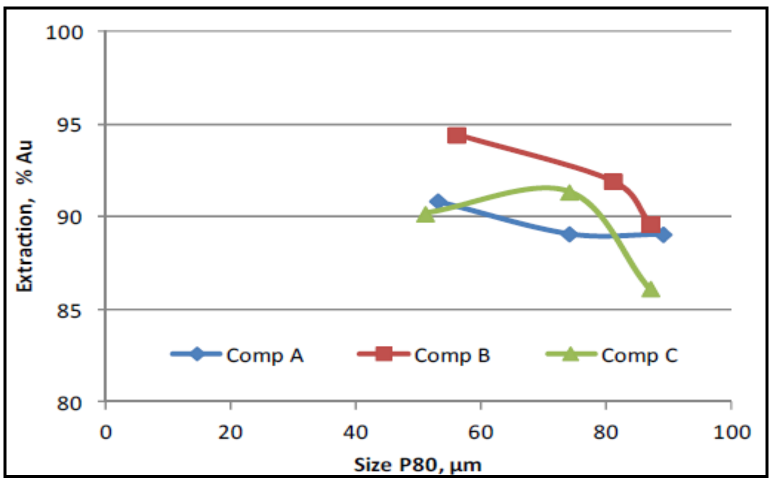
Whole Ore Cyanidation

CIL bottle roll tests were conducted on whole ore at grind P_{80}'s of 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, 10 g/L activated carbon addition, and 48 hour 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 sizes 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/t NaCN and lime consumption ranged from 0.5 kg to 1.0 kg/t CaO.

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

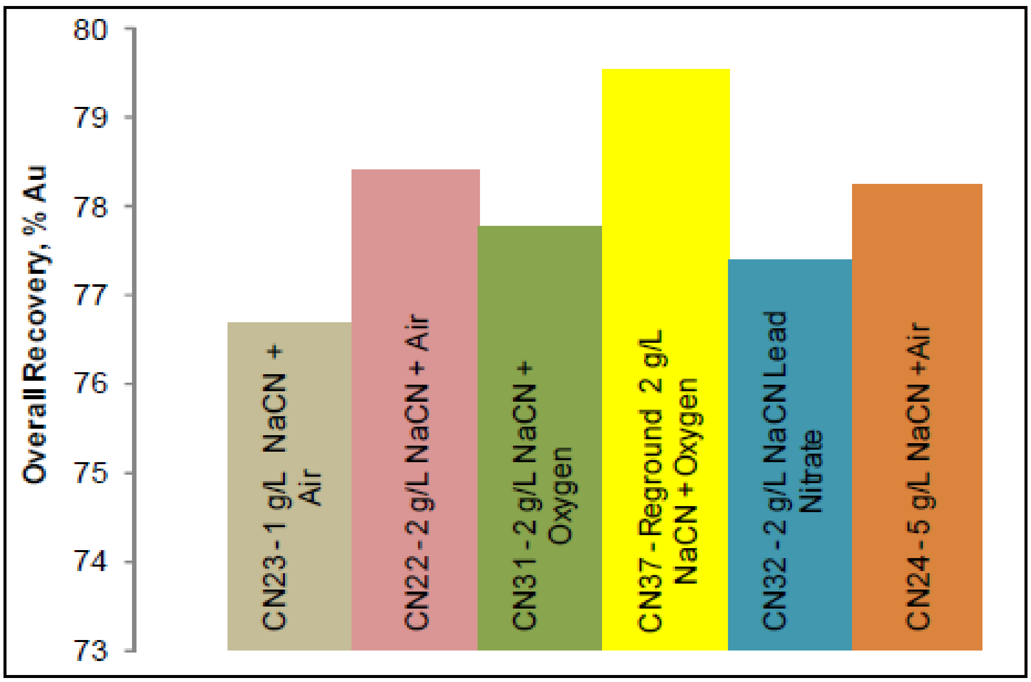


Figure 13.13 SGS 2014 Comp B Flotation Concentrate CIL Overall Gold Recovery

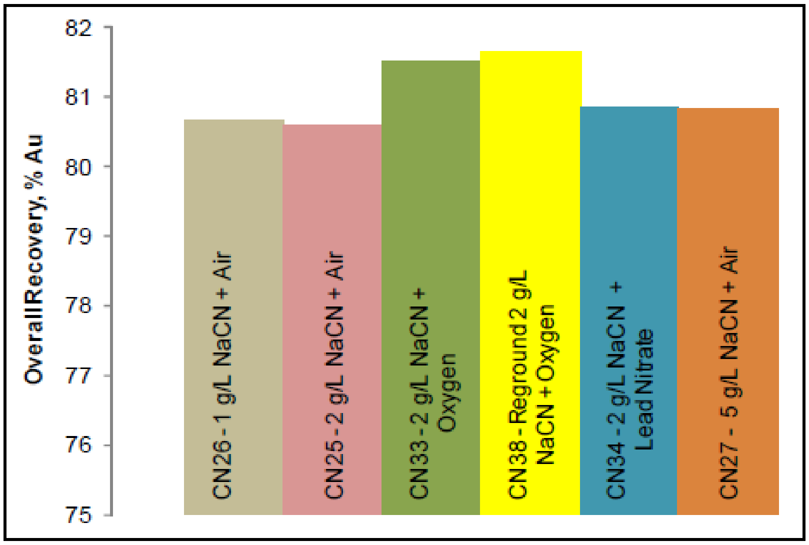
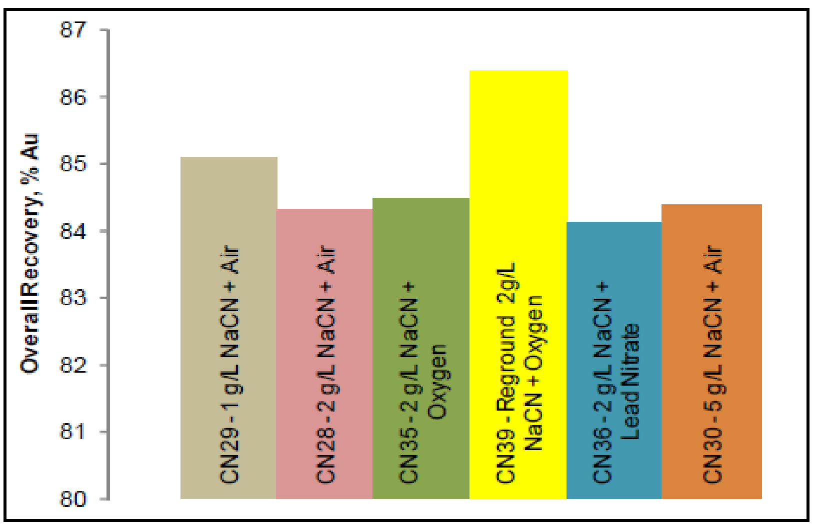


Figure 13.14 SGS 2014 Comp C Flotation Concentrate CIL Overall Gold Recovery



It was concluded by SGS that the optimum conditions for leaching flotation concentrate were maintaining 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	53 µm	1	0.6	90.9	0.17	1.86	
	Flotation	F7	53 µm	---	---	84.9	0.31	1.99	
	Gravity + flotation	F10	53 µm	---	---	81.2	0.36	1.83	
	Flotation concentrate CIL	CN-37	23 µm	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	56 µm	0.6	0.6	94.4	0.08	1.44	
	Flotation	F8	53 µm	---	---	87.1	0.21	1.54	
	Gravity + flotation	F11	53 µm	---	---	80.5	0.28	1.38	
	Flotation concentrate CIL	CN-38	16 µm	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	51 µm	1.4	1	90.2	0.12	1.17	
	Flotation	F9	46 µm	---	---	90.3	0.11	0.97	
	Gravity + flotation	F12	53 µm	---	---	87.2	0.15	1.09	
	Flotation concentrate CIL	CN-39	19 µm	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	21.6	32.5	64.7
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	-	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	units	1.5	1.5	1.5
Vol H_2O_2	g	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	ml	0	0	0
NAG (pH 4.5)	kg H_2SO_4/t	0	0	0
NAG (pH 7.0)	kg H_2SO_4/t	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 IMGOLD supervised additional metallurgical testwork with SGS Lakefield. A sample selection exercise was also conducted by Mr. Guy Desharnais 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 in Table 13.19.

Table 13.19 SGS 2015 Analysis on Gold 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-rythm, sandstone, cipolin and sandstone-rythm. 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 Geo-metallurgical database (from IMGOLD) broken down by count, grade and lithology.

Table 13.20 SGS 2015 Analysis of Geo-metallurgical 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 Geo-metallurgical 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

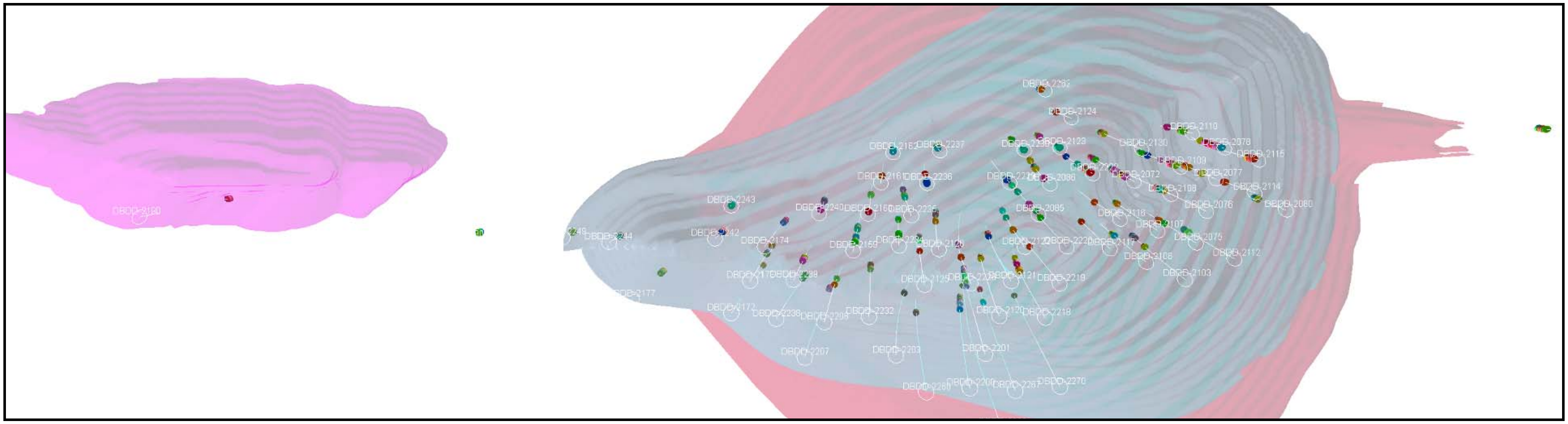


Figure 13.16 SGS 2015 Selected Drill Core Samples for Boto 2/Malikoundi – Side View

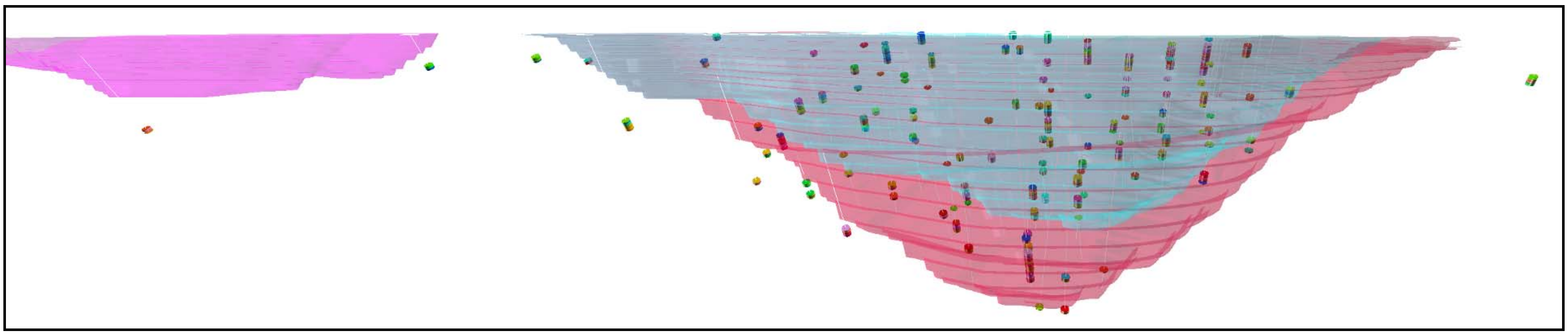


Figure 13.17 SGS 2015 Selected Drill Core Samples for Boto 5 – Top View

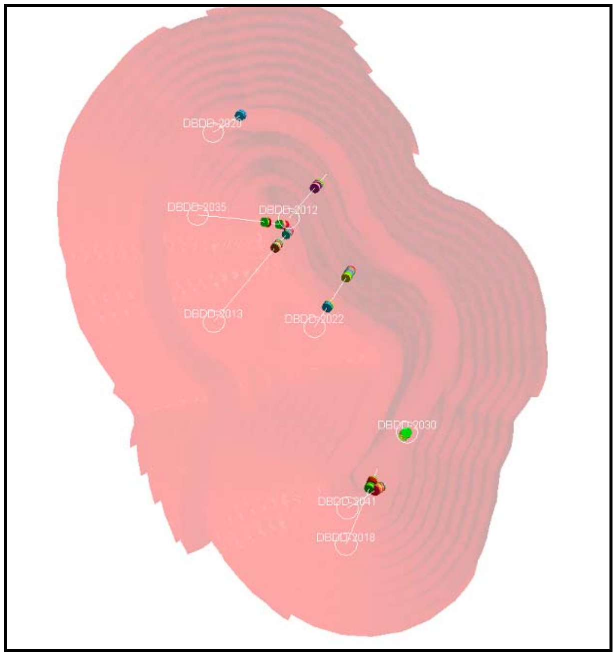
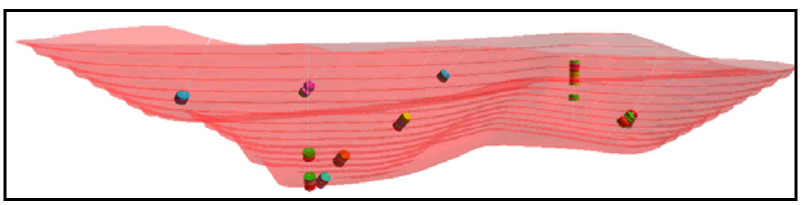


Figure 13.18 SGS 2015 Selected Drill Core Samples for Boto 5 – Side View



Grindability Testwork (2015)

Grindability testwork was conducted on three (3) domain composites and 39 variability samples. The testwork consisted of tests for Bond ball mill grindability, Bond abrasion index, JK drop-weight, SMC, and Bond low-energy impact. A summary of the grindability results is 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	-	0.000
G35	Mottled-zone	-	-	-	-	-	-	10.8	-	0.000
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 types) 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 and 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 native gold mineral abundance for MC-1, MC-2, and MC-3 were 98%, 64%, and 97%, respectively. Calaverite (AuTe) was found in MC-2 with ~35% abundance, 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

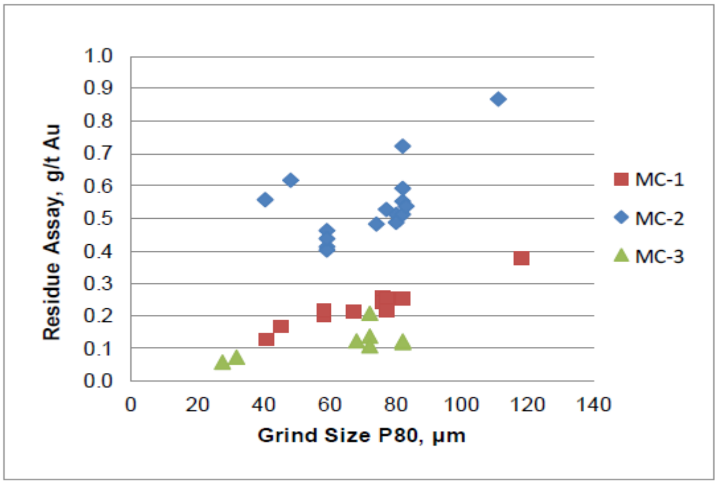
** Note: Boto 2/Malikoundi is denoted with only "Boto 2".*

For MC-3, the bulk cyanidation test (CN-89) had an issue with sample viscosity, 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 to 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 investigated and a plot of the grind size versus the leach residue grade was prepared as 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 exhibited 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 a noticeable improvement in leach kinetics or gold extraction for MC-1 or MC-3. Increasing the cyanide concentration to 0.75 g/L 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)

Forty samples were submitted for gold extraction variability testwork as part of the 2016 SGS metallurgical program. 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		% Au Extraction					Residue g/t Au	Calc. Head g/t Au
			NaCN	CaO	6-h CN	24-h CN	48-h CN	Grav Sep	Grav + CN		
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

2015/2016 Testwork Program Gap Analysis

In 2017, a review of the 2015/2016 metallurgical testwork was completed by Lycopodium to identify potential gaps prior to commencing the metallurgical testwork program for the Boto feasibility study (FS) phase.

The review identified several issues including, but 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 the 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 on gold extraction.

Lycopodium also consulted 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 Most Recent Metallurgical Testwork Programs

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 IMGOLD 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%) to be fed to the plant will come from Boto 5 deposit, therefore MC-1 was formed to better understand the metallurgical behaviour of this material. MC-2 was formed to represent the expected LOM blend and ore grade. The saprolite (SAP) sample was formed without grade consideration, and will only be used in solids-liquid separation 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 to form 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

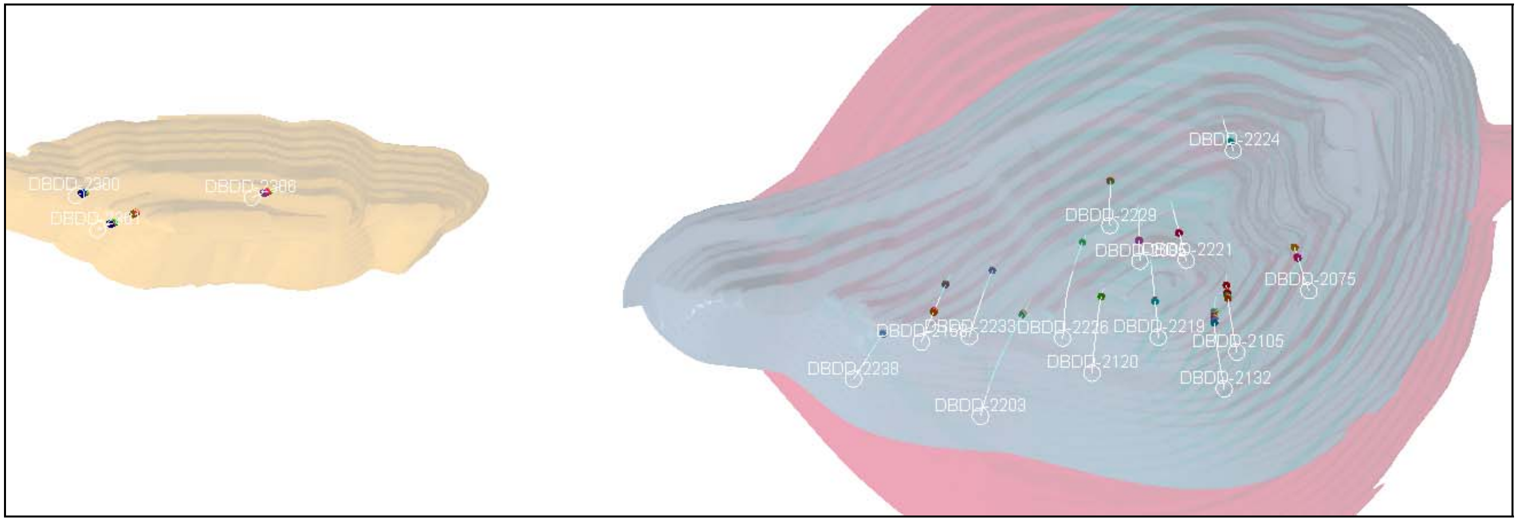


Figure 13.21 SGS 2017 – Boto 2/Malikoundi Samples Used in Master Composites – Side View

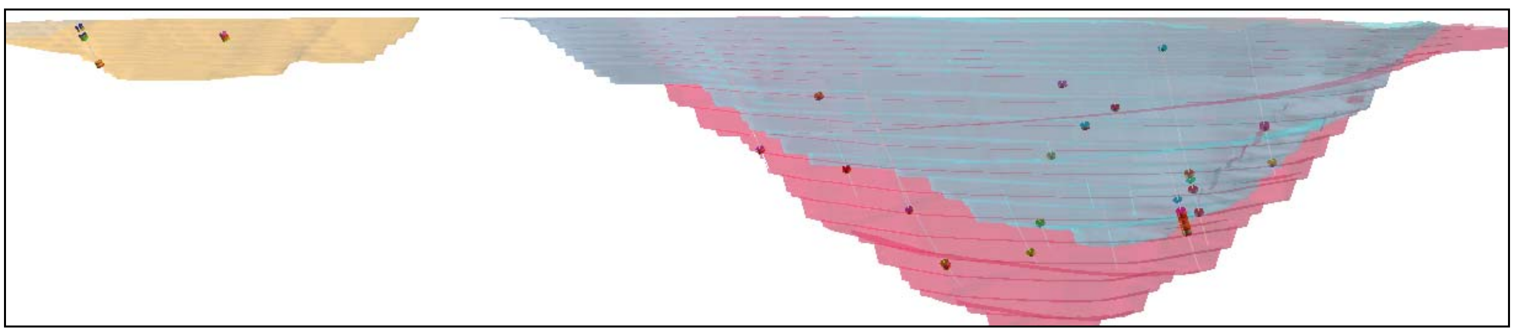


Figure 13.22 SGS 2017 – Boto 5 Samples Used in Master Composites – Top View

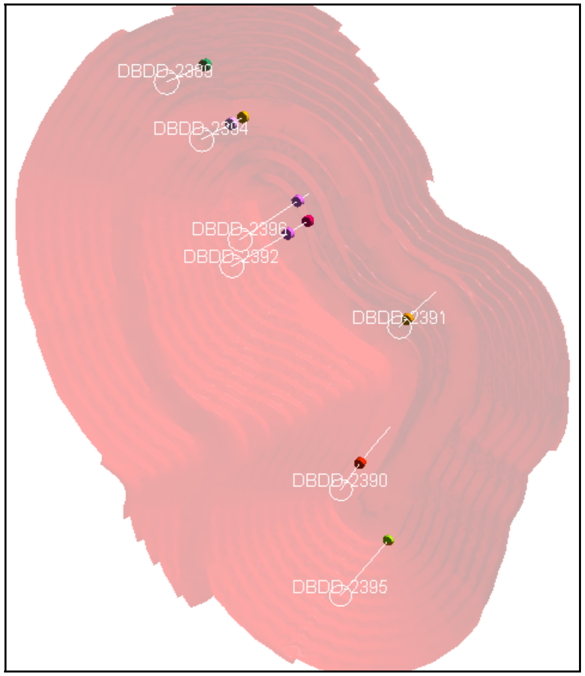


Figure 13.23 SGS 2017 – Boto 5 Samples Used in Master Composites – Side View

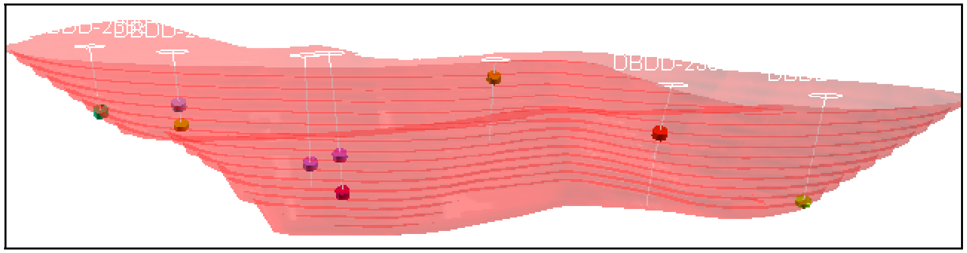


Figure 13.24 SGS 2017 – Boto 2/Malikoundi Samples Used in Variability Composites – Top View

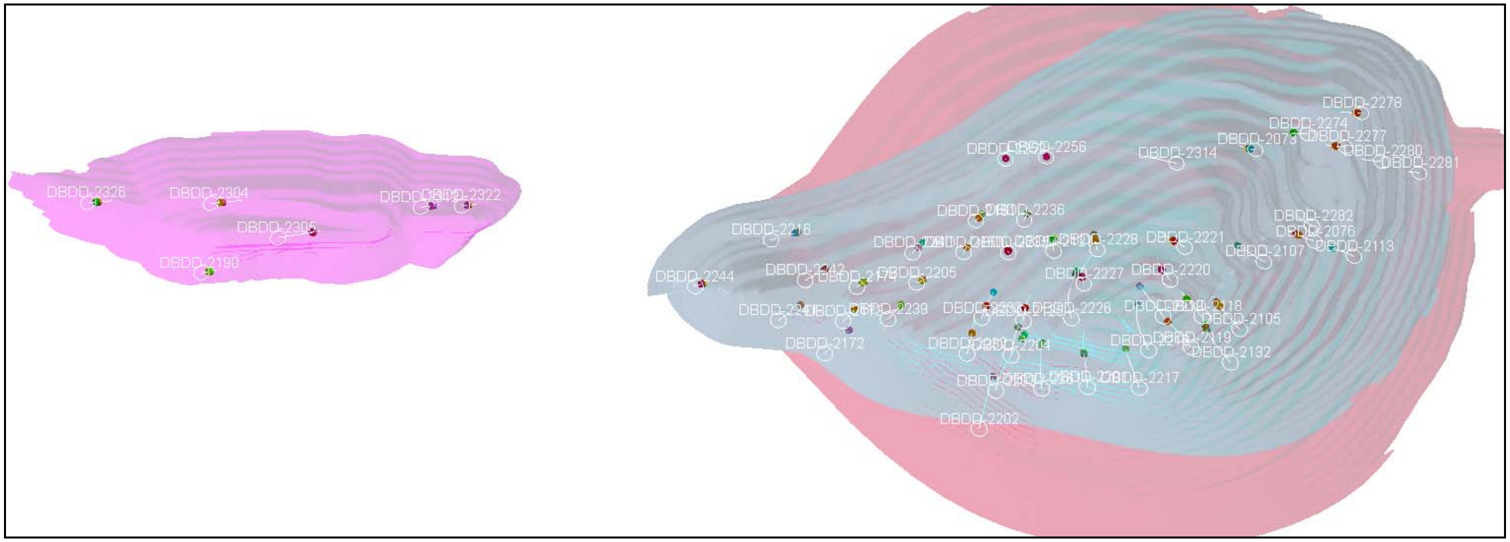
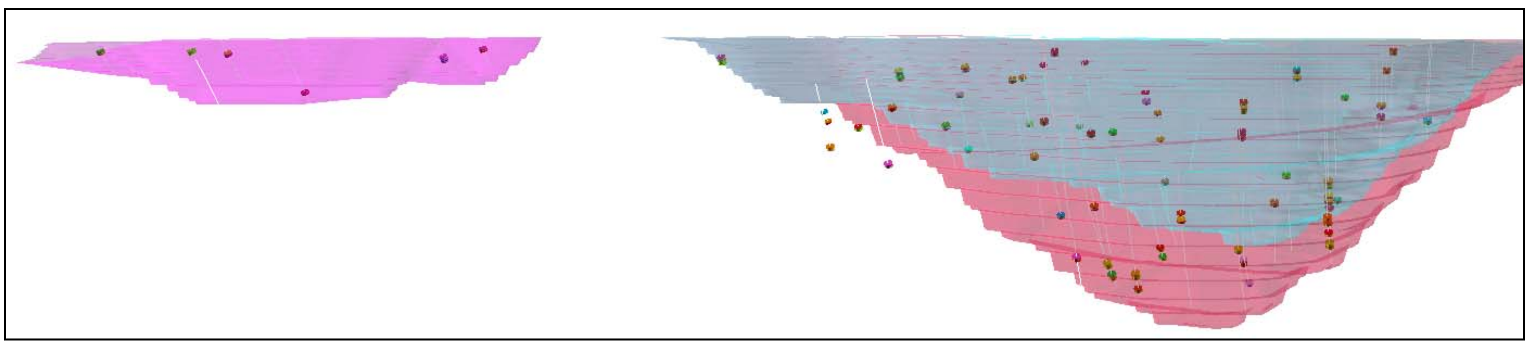


Figure 13.25 SGS 2017 – Boto2/Malikoundi Samples Used in Variability Composites – Side View



Grindability Testwork (2017/2018)

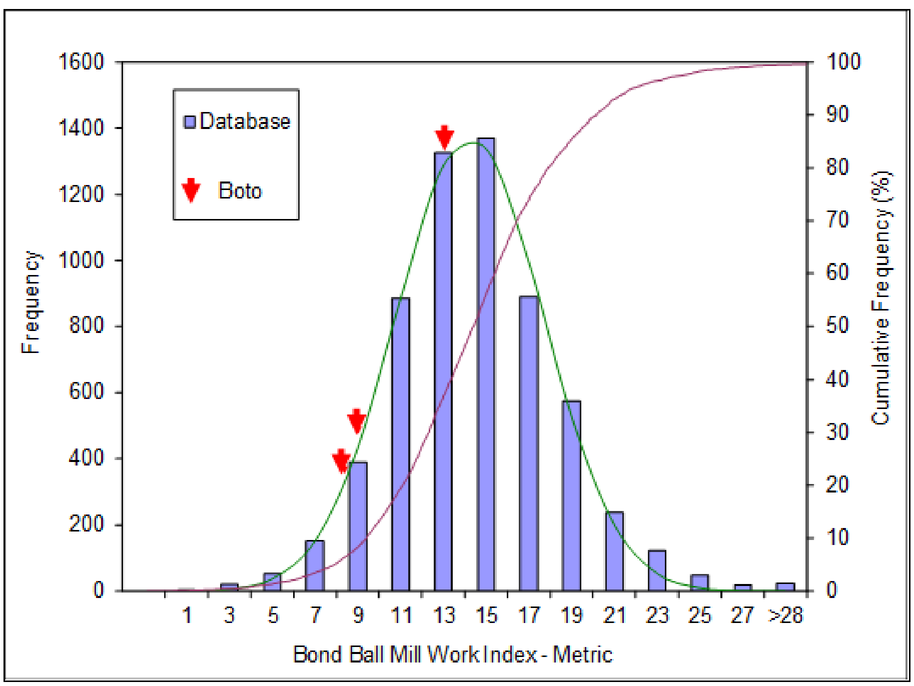
Following the recommendations from OMC's review of the previous comminution data, two saprolite and two saprock samples from Boto 2/Malikoundi were selected for additional BWi tests. Unfortunately, one of the saprolite samples was very fine and disintegrated so the testwork could not be performed. The 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 availability of material that met the testwork procedure rock size requirement.

Head Analysis and Mineralogy Analysis on Master Composites

The specific gravity, determined by pycnometer, was 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 investigate the distribution of tellurides in the ore body and to study the gold deportment and the gold host 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.

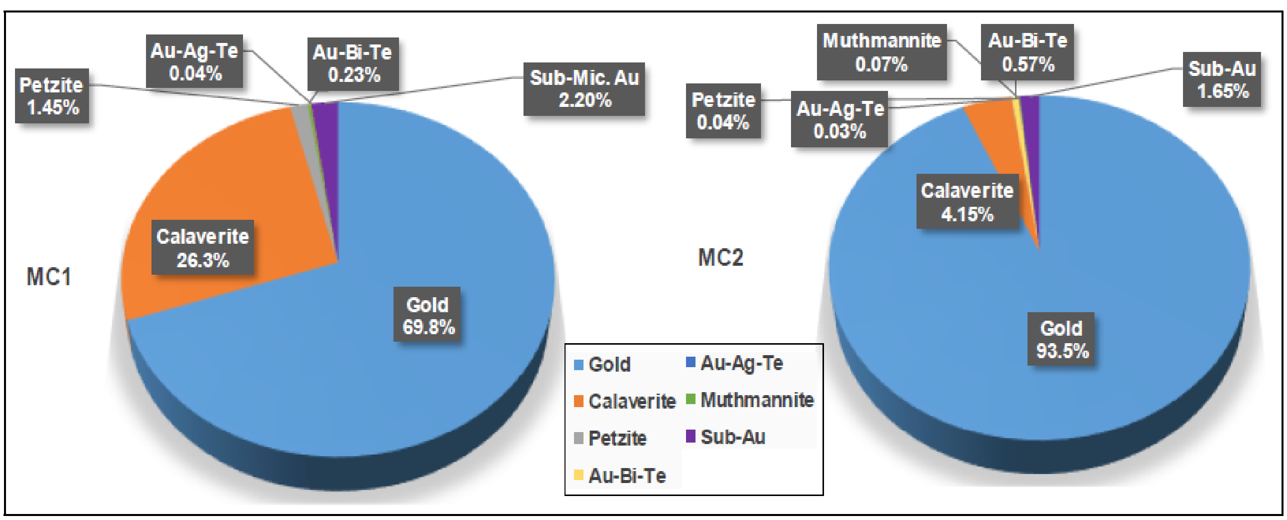
Table 13.30 SGS 2017/2018 Characteristic of Microscopic Gold in MC-1 and MC-2

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



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

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. 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

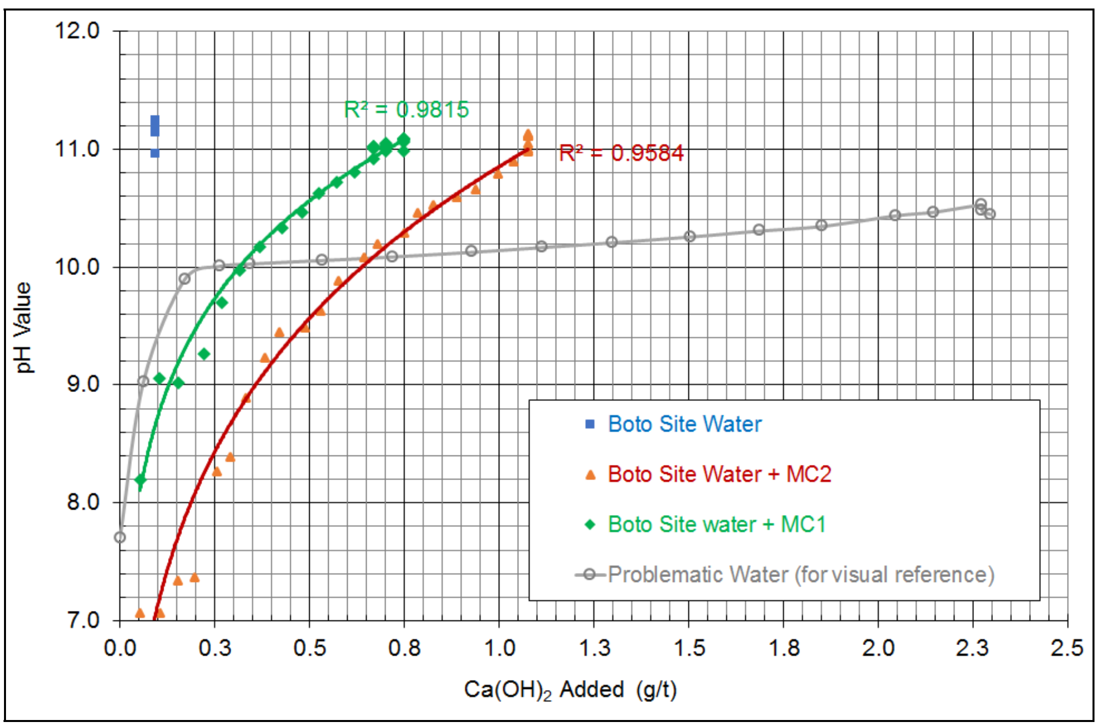
Gravity separation testwork was conducted on MC-1 and MC-2 using a Knelson concentrator coupled with a Mozley laboratory separator to generate material for grind calibration and for downstream testwork such as pH neutralization, rheology and cyanidation tests. Initially, one test was conducted on MC-1 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 a benchmark result for a problematic site water in the grey line for comparison. Excess lime consumption is not predicted by 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 was to determine the maximum solids density for each slurry from an operability perspective. 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)

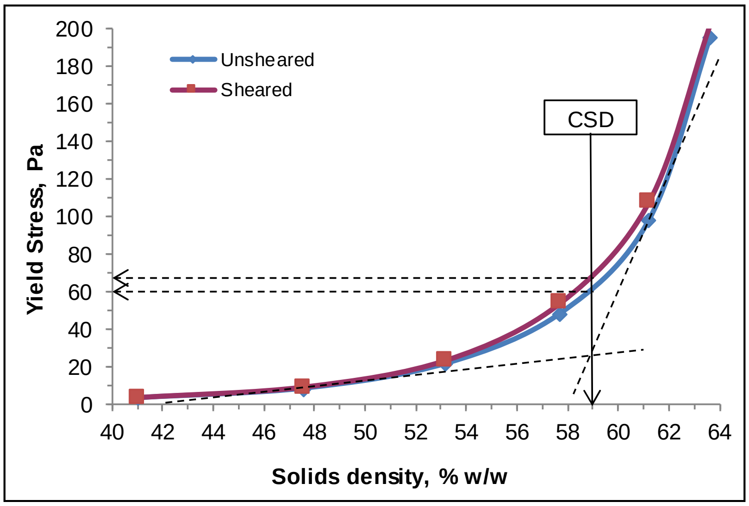
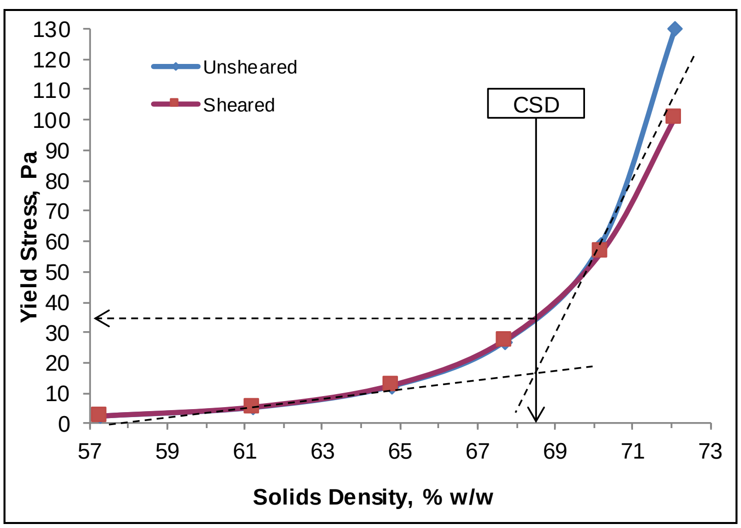


Figure 13.30 SGS 2017/2018 Solids Density vs. Yield Stress for MC-2 Gravity Tailings (G1)

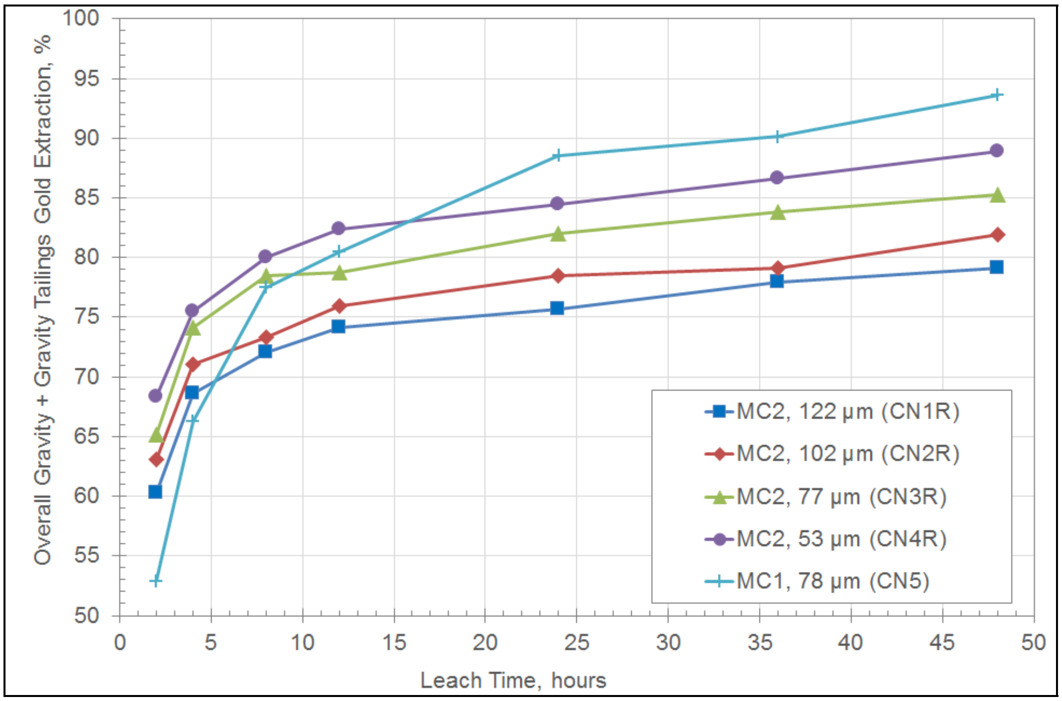


Based on the rheology results, the critical operating density where yield stress is under 10 Pa is ~50% solids for MC-1 and ~64% solids for MC-2.

Grind Optimization Testwork

The grind optimization testwork series included four tests on G3 (MC-2) at targeted grind P_{80}'s of 125 µm, 106 µm, 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 shows 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, a general improvement in leach kinetics and gold extraction is observed with decreasing grind sizes for MC-2. A trade-off study conducted by Lycopodium recommended 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.					Pulp Density (% Solids)	NaCN, g/L, (start/maint.)	Reagents (kg/t of CN Feed)				Au Extraction / Recovery (%)									1 O'all Norm Rec.,%	Leach Res. Au, g/t	Head Grade, Au, g/t		
			Lead Nitrate	Air or O_2		Time, h	DO_2, mg/L (Avg)			Added		Consumed		2 h	4 h	8 h	12 h	24 h	36 h	48 h	Grav.	Grav + CN			CN (calc)	Grav + CN (calc)	Direct
				Pre.	Leach					NaCN	CaO	NaCN	CaO														
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. Although it was expected that cyanide consumption would decrease, this 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. All tests conducted later 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

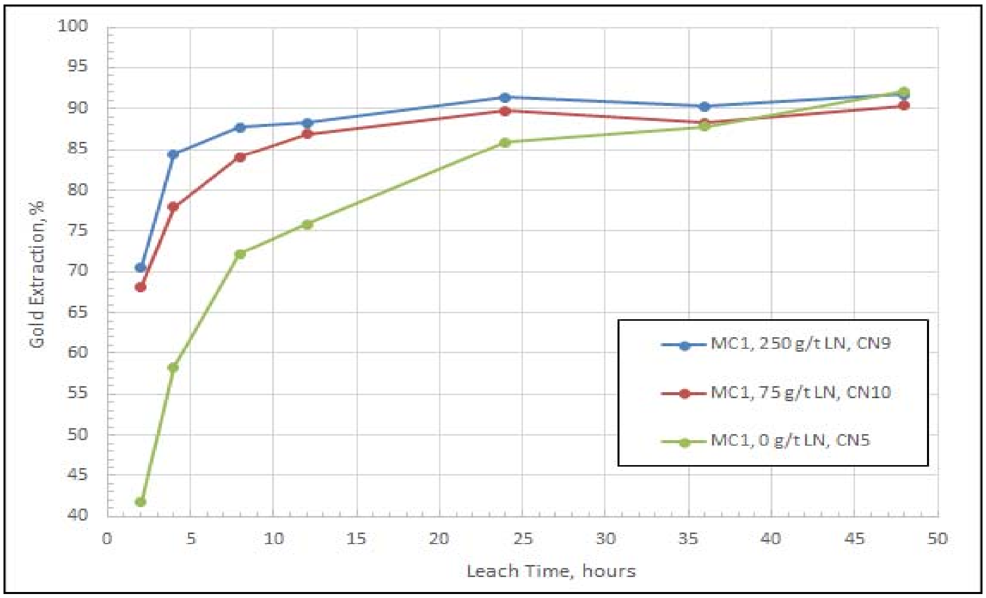
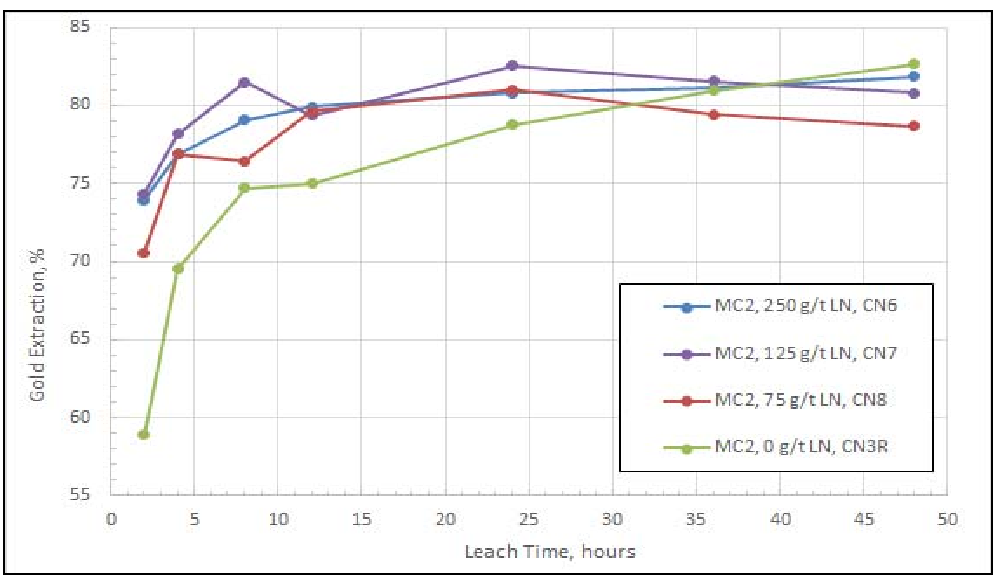


Figure 13.33 SGS 2017/2018 Effect of Lead Nitrate on MC-2 Gravity Tailings Cyanidation



Additional leach optimization tests were conducted using the standard sample mass size of 1,000 g to repeat a number of the lead nitrate tests and 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-aeration, Pre-oxy and O_2 Addition on MC-1

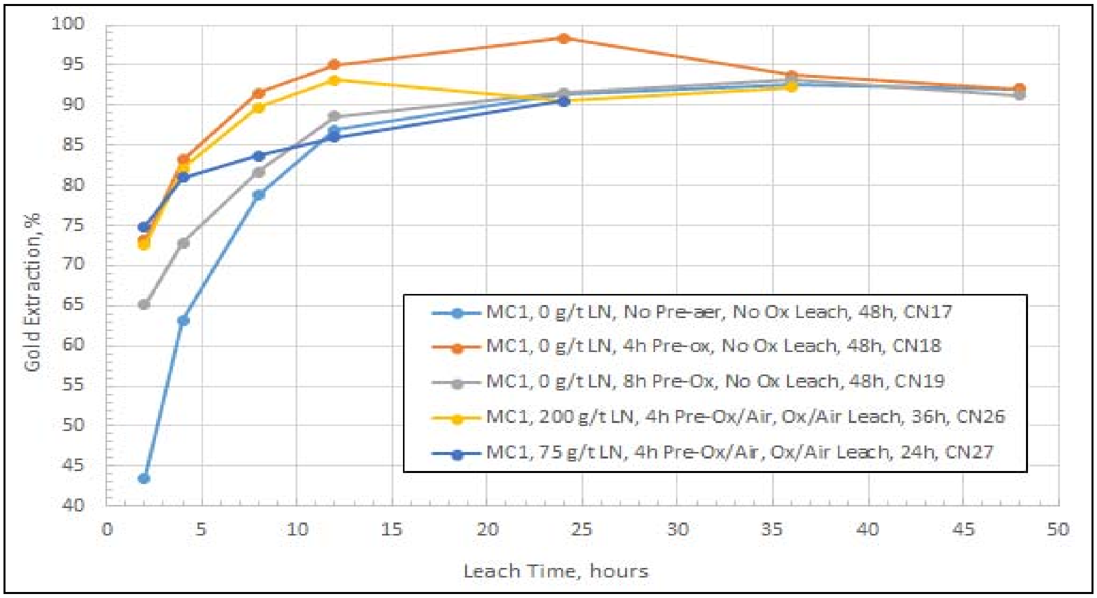
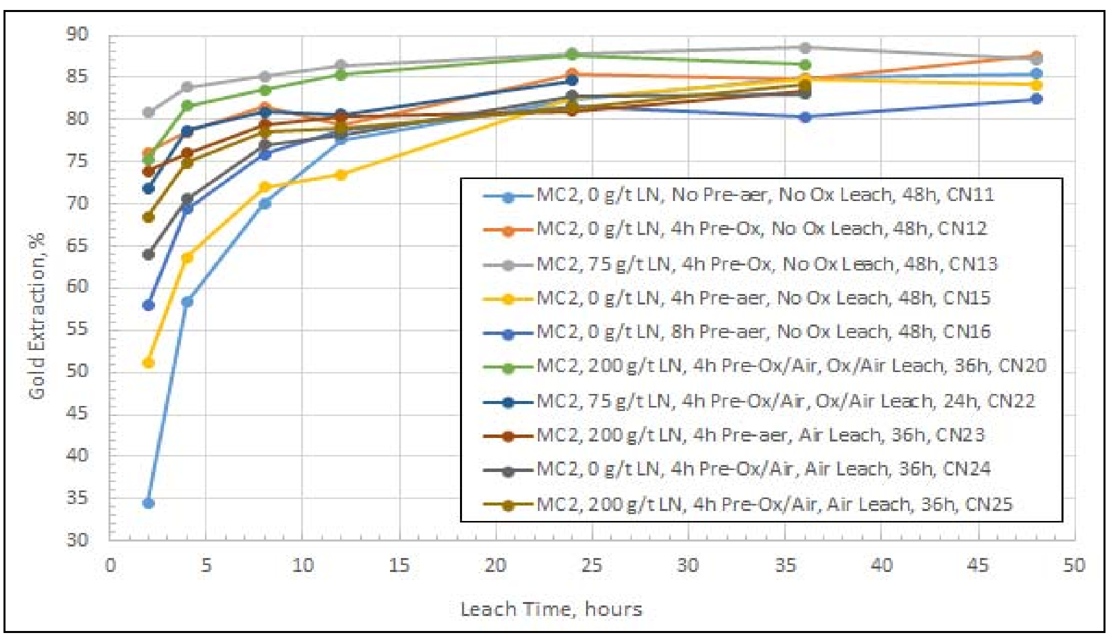


Figure 13.35 SGS 2017/2018 Effect of Lead Nitrate, Pre-aer., Pre-oxy and O_2 Addition on MC-2

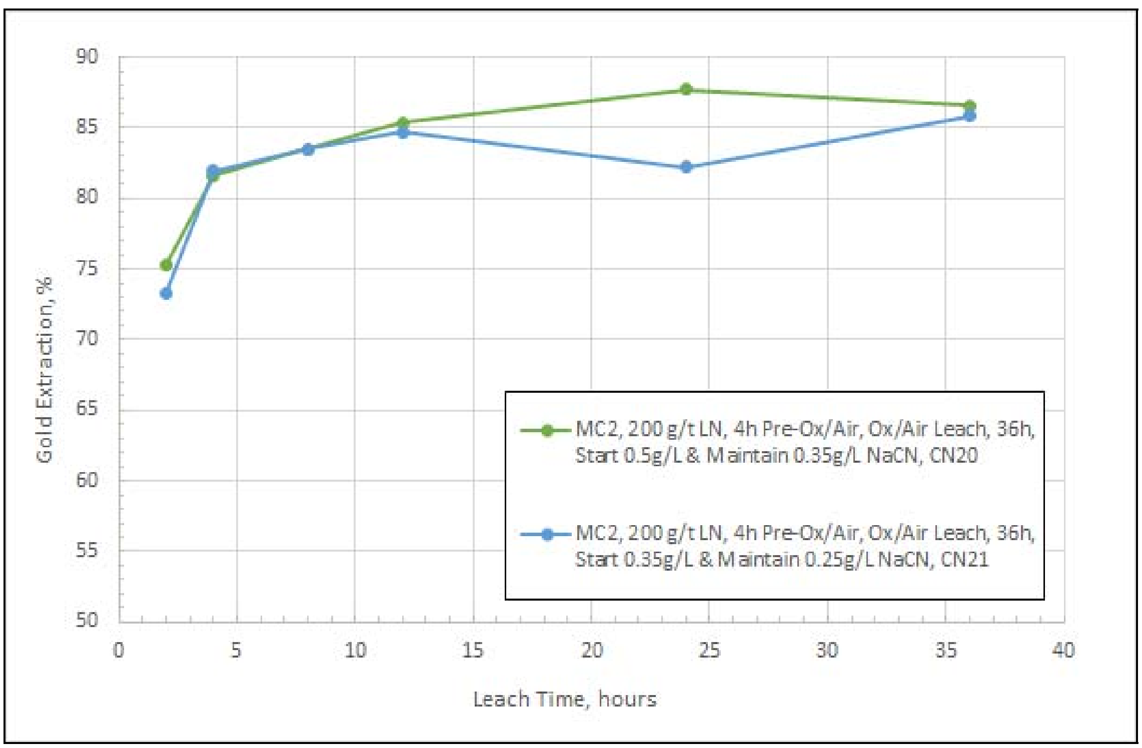


MC-1 gold extractions are consistently high, ranging from 90% to 92%. The addition of lead nitrate and oxygen in the pre-treatment step and during leaching provided insignificant improvement to the gold extraction for MC-1. However, improvement is observed in the leach kinetics during the initial leach period. There was 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 be decreased from 48 hours to 36 hours, and possibly even as low as 24-hours. Lead nitrate addition of 200 g/t provided better leach performance than the test with lead nitrate added at 75 g/t (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 found 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 was allowed to drop below 0.35 g/L during the test. Hence, it was agreed that for future tests cyanide concentration would be maintained above 0.4 g/L to avoid going below the 0.35 g/L level.

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



Leach conditions, as follows, were agreed upon prior to starting the pulp density optimization leach tests (CN28 to CN31):

- Pulp pH at 10.5 to 11 and temperature maintained at 35°C.
- 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.
- 0.5 g/L NaCN at start of test and maintain above 0.4 g/L.

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

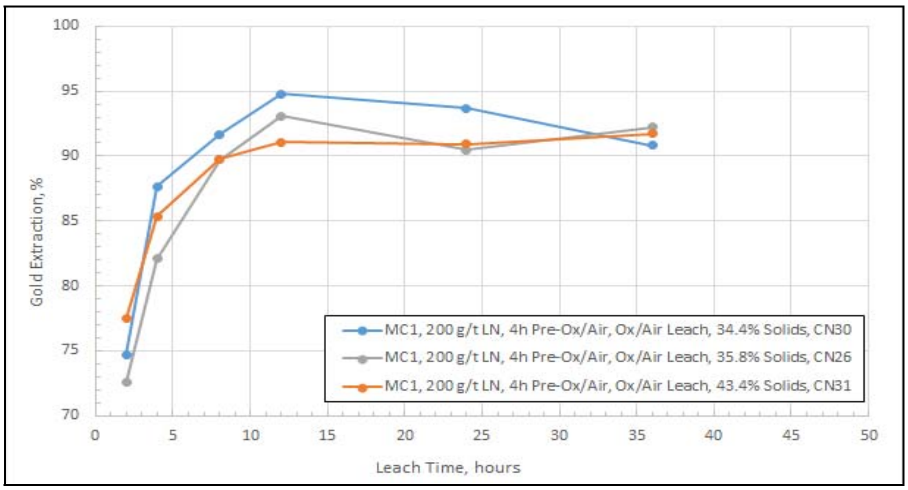
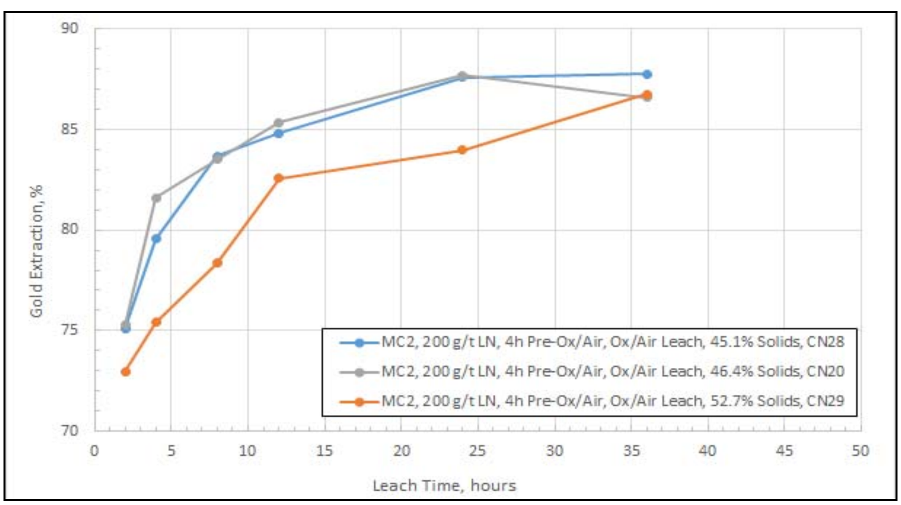


Figure 13.38 SGS 2017/2018 Leach Tests at Varying Pulp Densities for MC-2



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 conducted and the results are presented as part of the whole ore cyanidation testwork section.

The pulp density selected for use in subsequent tests was 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 mainly occurs as 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

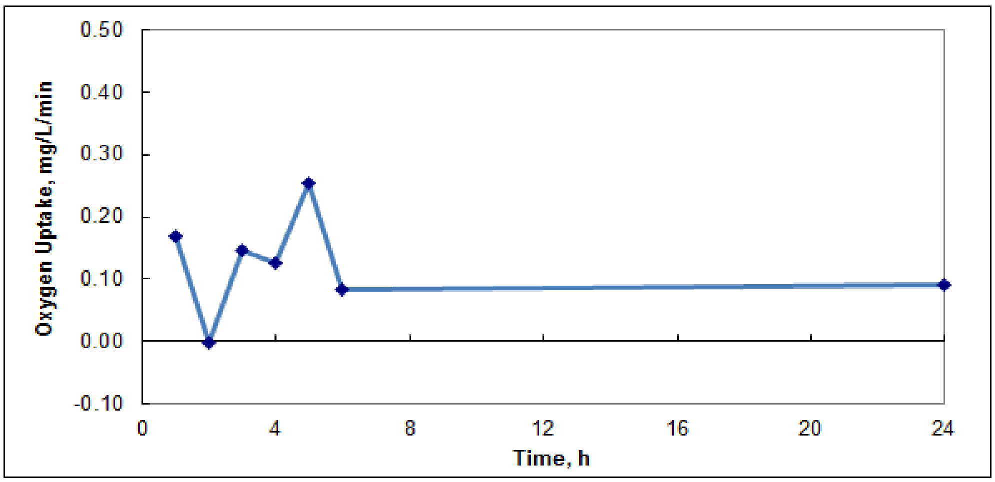
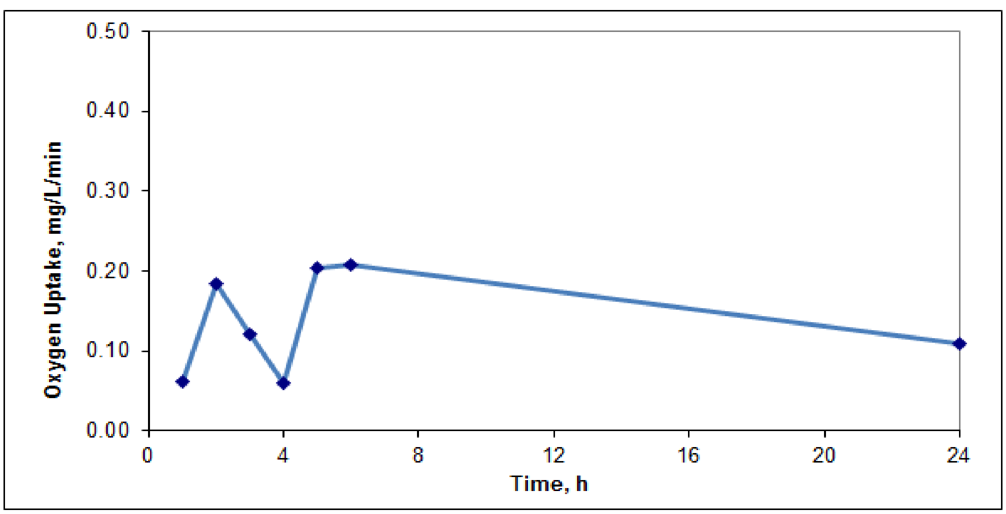


Figure 13.40 SGS 2017/2018 Oxygen Uptake Results for MC-2



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/min for MC-1 and 0.135 mg/L/min for MC-2, which is considered high, therefore oxygen sparging 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 by either MC-1 or MC-2 gravity tailings as the results of the tests with carbon added are very similar to the tests conducted with no carbon.

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

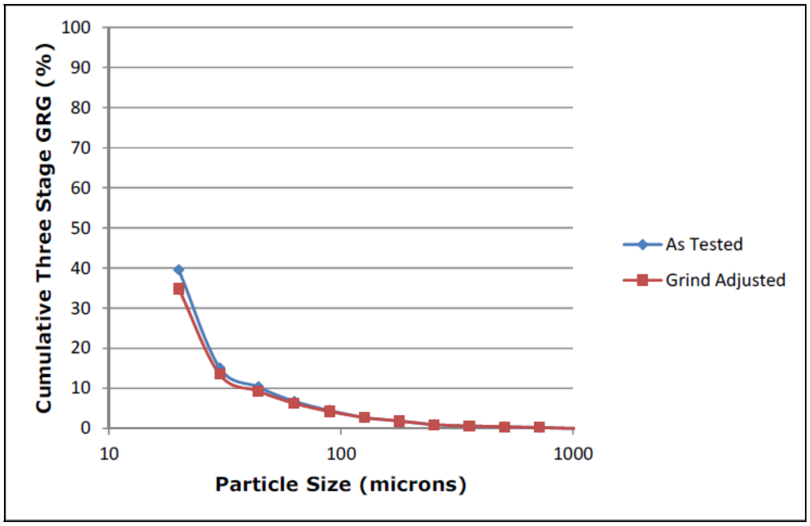
E-GRG test was conducted on MC-2 composite to determine the gravity-recoverable gold (GRG) number. The results from the E-GRG test is presented in Table 13.37 and the GRG number was determined to be 39.6.

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 was carried out on a sample ground 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 this assessment, 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	1.68
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		
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} of 51 µm) 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

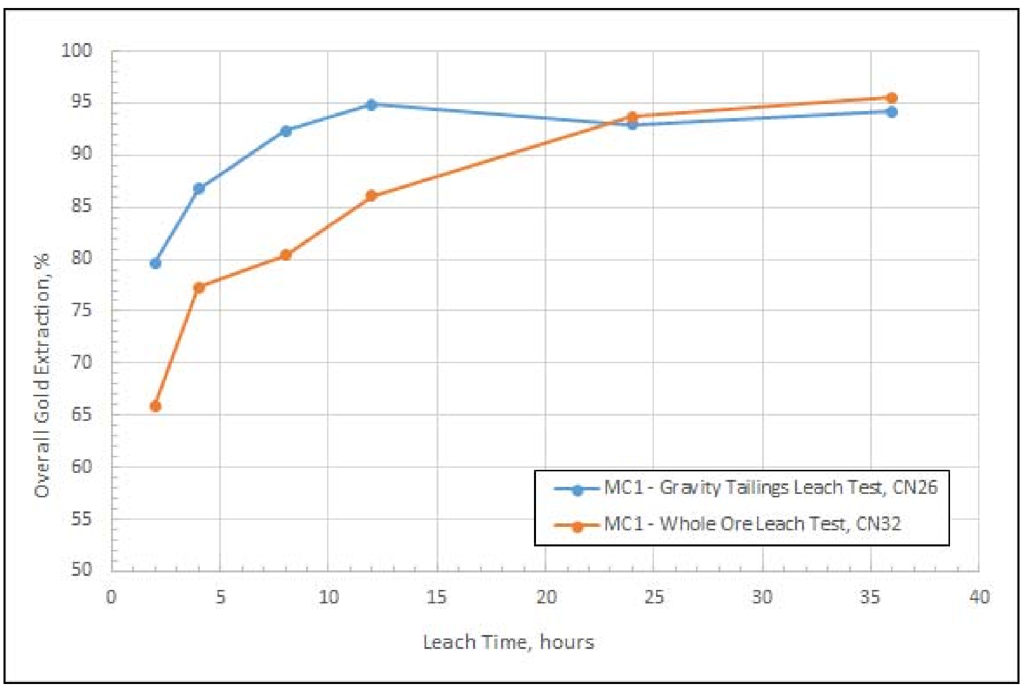


Figure 13.43 SGS 2017/2018 Whole Ore vs. Gravity Tailings Cyanidation for MC-2

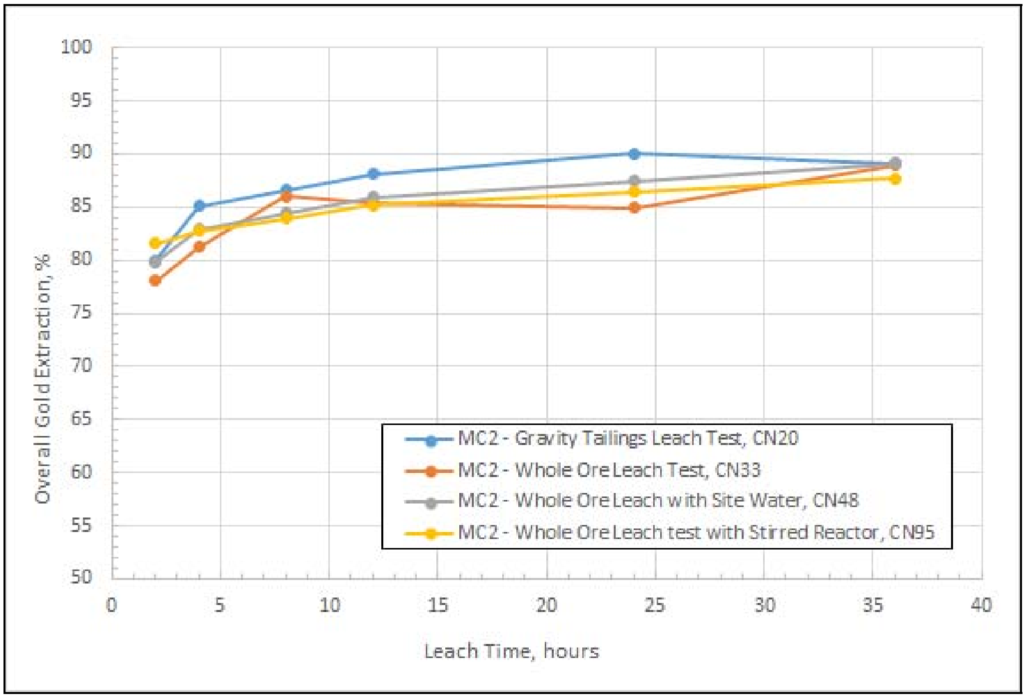


Figure 13.44 SGS 2017/2018 Effect of Whole Ore at Finer Grind for MC-2

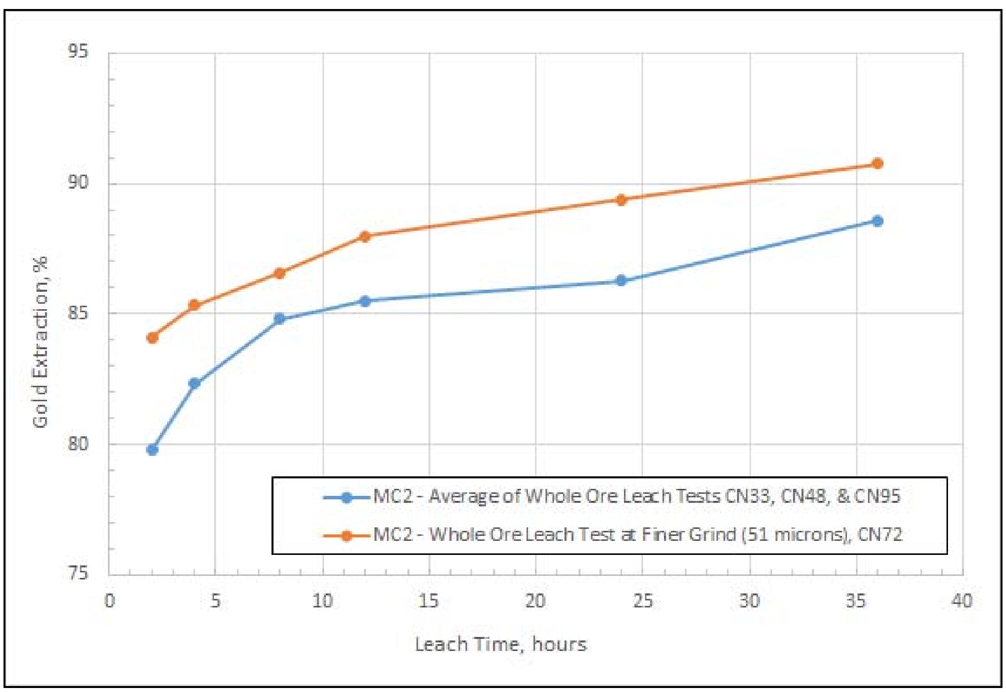
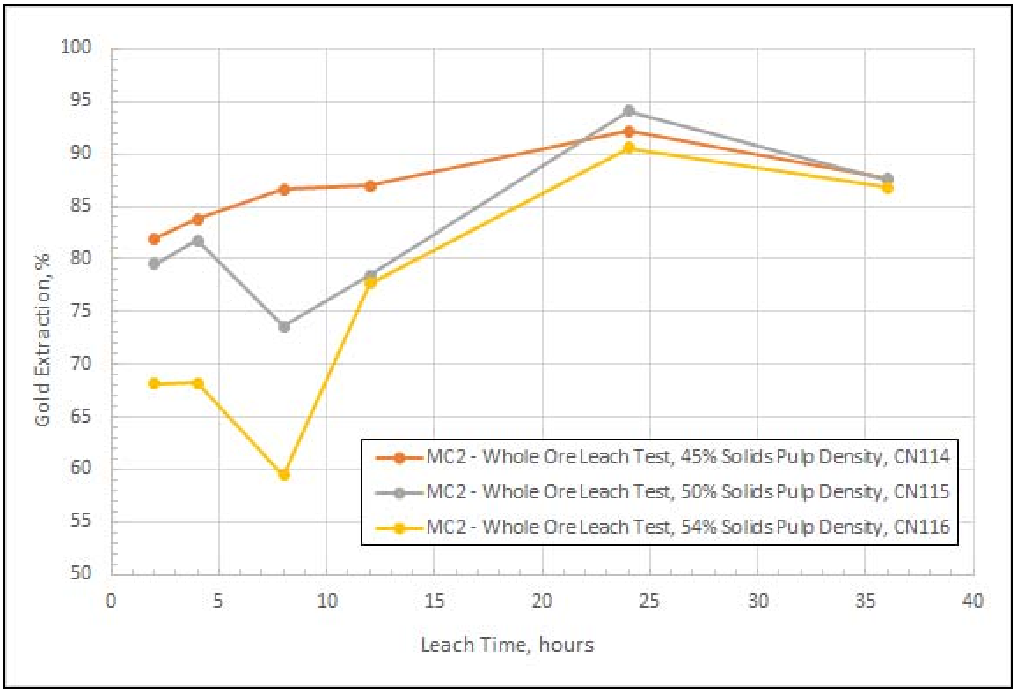


Figure 13.45 SGS 2017/2018 Effect of Pulp Densities on Whole Ore Leaching for MC-2



Carbon-in-Pulp (CIP) Modelling

In order to perform CIP modelling, 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

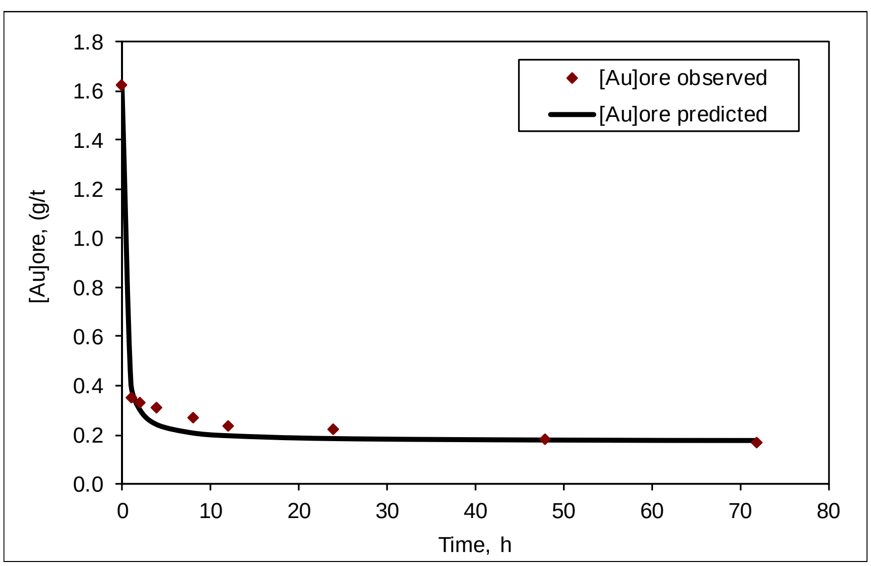


Figure 13.47 SGS 2017/2018 Carbon Absorption Kinetics for CIP Modelling

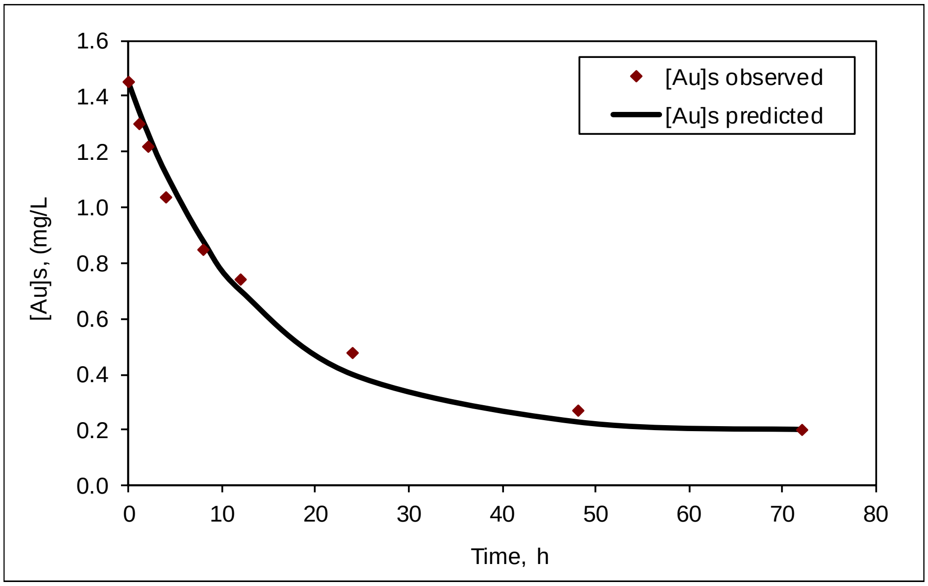
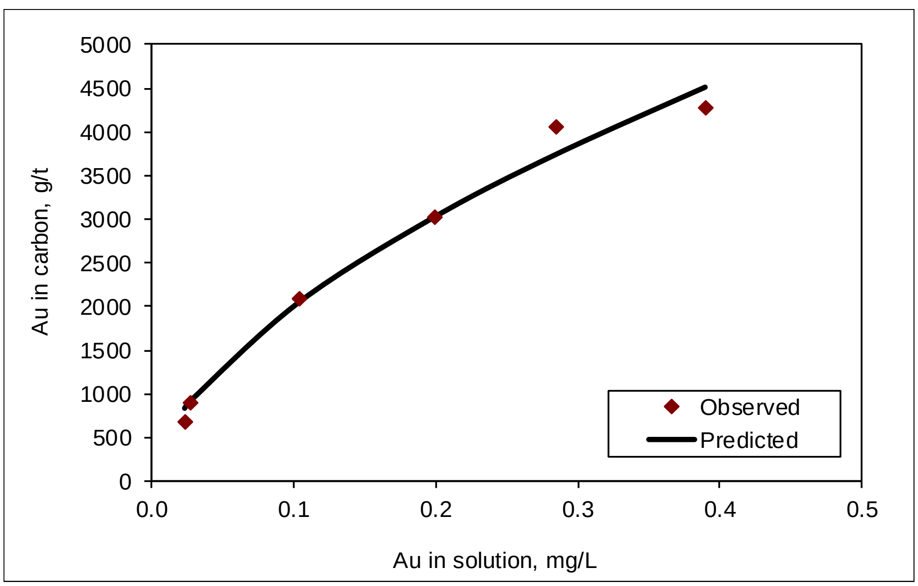


Figure 13.48 SGS 2017/2018 Carbon Absorption Isotherms for CIP Modelling



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 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.

Additional simulations were performed in 2019 to evaluate the effect of the increase in throughput from 2.75 Mtpa to short-term peaks equivalent to 3.0 Mtpa, combined with an increase in head grade from 1.68 to a design grade of 3.0 g/t, on the performance of the adsorption circuit. The simulation results are summarized in Table 13.41 which predicts a modest increase in dissolved gold loss from 0.004 mg/L to 0.005 mg/L when retaining the previously specified 100 m^3 adsorption tanks. Despite this positive outcome, a decision was made to upgrade the tanks to 120 m^3 as this provides more operational flexibility.

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*

Table 13.41 SGS Summary of CIP Modelling 2019 Results

Parameter	Simulation number		
	11	12	13
ROM head grade (g/t)	1.68	2.3	3.0
Tonnage throughput (t/h)	372	372	372
Adsorption tank size (m^3)	100	100	100
Eluted carbon gold grade (g/t)	120	120	120
Kinetic constant (k)	0.010	0.010	0.010
Equilibrium constant (K)	29361	29361	29361
Product of model constants (kK)	303	303	303
Carbon concentration (g/L)	50	50	50
Carbon movement (tpd)	5.0	5.0	5.0
Total carbon inventory (tonnes)	30	30	30
Loaded carbon grade (g/t)	2786	3893	5143
Upgrade ratio	1860	1838	1825
Dissolved gold loss (mg/L)	0.004	0.005	0.005

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 13.42.

Table 13.42 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 13.43 shows the type of lithology that the samples were associated with, along with the sample count and ranges of gold head grades.

Table 13.43 SGS 2017/2018 Variability Samples Head Grade Range by Lithology Type

Lithology Association	Count	Head Grade, g/t Au		
		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 13.44.

Table 13.44 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) Added		Consumed		% Au Extraction 24 h	36 h		Leach Residue g/t Au	Head Grade Au		Te	Ore Weathering Type
					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		Consumed		% Au Extraction 24 h	36 h		Leach Residue g/t Au	Head Grade Au		Te	Ore Weathering Type
					NaCN	CaO	NaCN	CaO	Calc'd Head	Calc'd Head	Direct Head		CN (calc)	Direct	Direct	
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) Added NaCN	Reagents (kg/t of CN Feed) Added CaO	Reagents (kg/t of CN Feed) Consumed NaCN	Reagents (kg/t of CN Feed) Consumed CaO	% Au Extraction 24 h Calc'd Head	% Au Extraction 36 h Calc'd Head	% Au Extraction 36 h Direct Head	Leach Residue g/t Au	Head Grade Au CN (calc)	Head Grade Au Direct	Head Grade Te Direct	Ore Weathering Type
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 13.45 summarizes the variation in leach performance based on the type of lithology.

Table 13.45 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 was to investigate flocculant screening, 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 a targeted grind P_{80} of 75 µm. 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

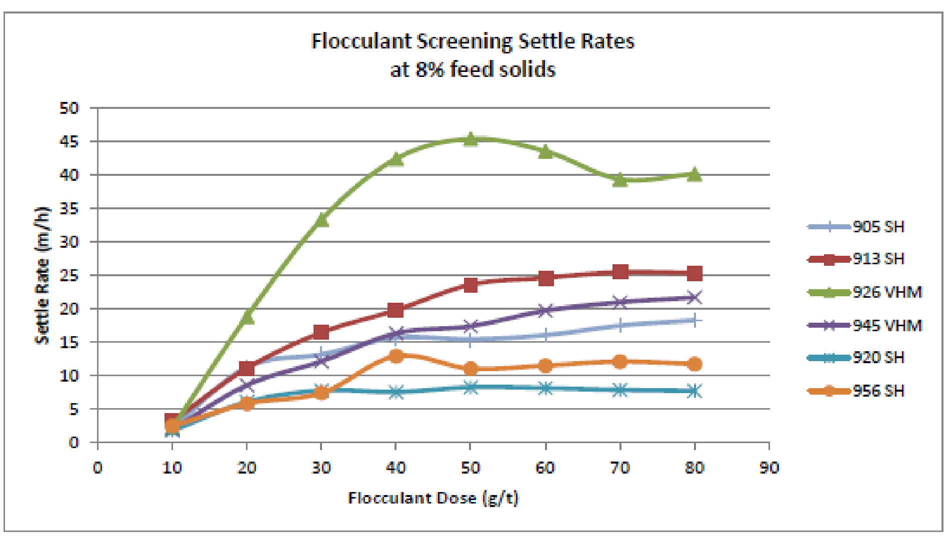
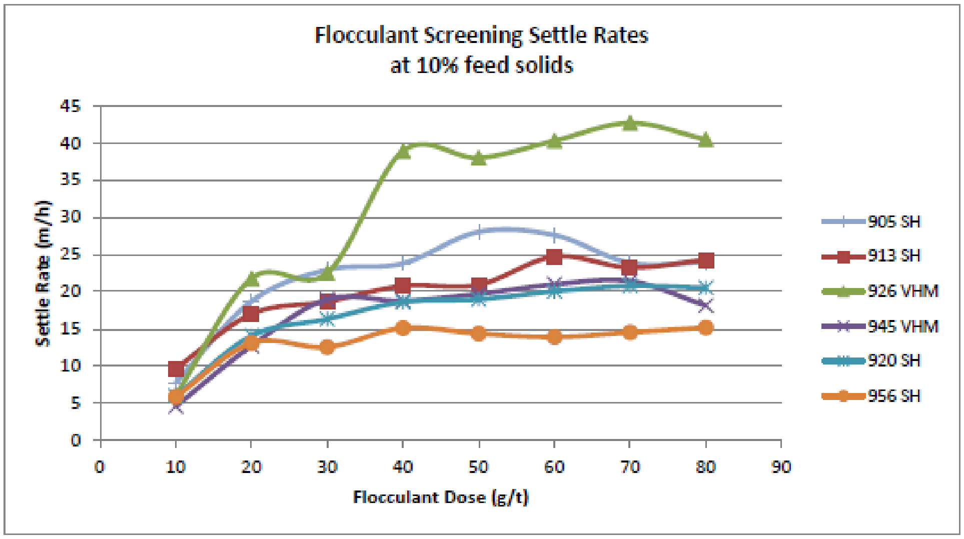


Figure 13.50 Outotec 2018 Flocculant Screening Results for MC-2 Composite



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 13.46 and Table 13.47.

Table 13.46 Outotec 2018 Dynamic Thickening Test Results for SAP Composite

Run No.	Feed Flux (tph/m^2)	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 13.47 Outotec 2018 Dynamic Thickening Testwork Results for MC-2 Composite

Run No.	Feed Flux (tph/m^2)	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 the size of both thickeners both pre-leach and post-leach thickener in the case that a tailings thickening will be required in the future. The thickener sizes in Table 13.48 and Table 13.49 are based on MC-2 composite with only limited feed from the SAP composite.

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

Pre-Leach 17m Thickener					
Feed : MC2 Blend					
Feed t/h dry solids	**Flux $t/m^2/h$**	**Underflow % solids**	**Overflow ppm**	**Flocculant g/t**	**Yield Stress Pa**
310	1.40	61 - 63	<200	15	35
Feed : 10% Sap					
Feed t/h dry solids	**Flux $t/m^2/h$**	**Underflow % solids**	**Overflow ppm**	**Flocculant g/t**	**Yield Stress Pa**
68	0.30	48 - 50	<200	15	40
136	0.60	45 - 47	<200	15	22

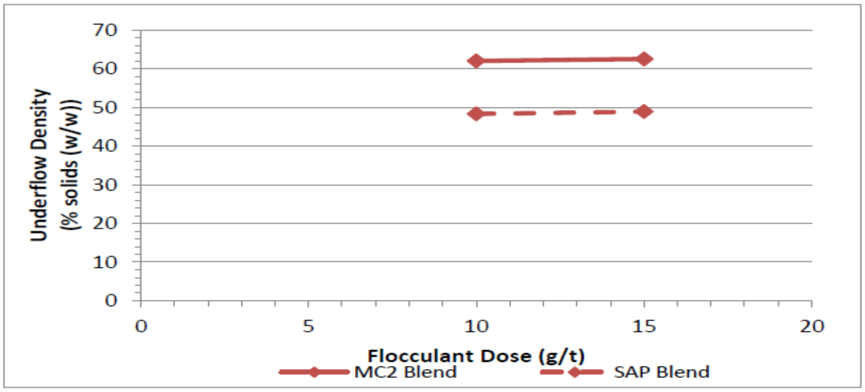
Table 13.49 Outotec 2018 Recommended Post-leach Thickener Size and Flocculant Dosage

Pre-Leach 17m Thickener					
Feed : MC2 Blend					
Feed t/h dry solids	**Flux $t/m^2/h$**	**Underflow % solids**	**Overflow ppm**	**Flocculant g/t**	**Yield Stress Pa**
310	0.60	63 - 65	<200	15	43
Feed : 10% Sap					
Feed t/h dry solids	**Flux $t/m^2/h$**	**Underflow % solids**	**Overflow ppm**	**Flocculant g/t**	**Yield Stress Pa**
68	0.13	49 - 51	<200	15	49
136	0.27	48 - 50	<200	15	40

During the FS and subsequent optimization phases, the plant throughput 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 (referred to as a post-leach thickener above) is currently not included in the design.

Note that a lower flocculant dosage of 10 g/t can also be used as the tests conducted at this lower dosage showed similar results to tests 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 13.50. Based on Pocock's observation, many types of flocculants responded well, however, SNF AN 905 SH was shown to produce a slightly more robust floc structure than the other types tested.

Table 13.50 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 13.51 and Table 13.52.

Table 13.51 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 13.52 Pocock 2018 Recommended High Rate Thickener Operating Parameter Ranges

Material Tested	Tested Feed Solids(1) (%)	Flocculant Type(2)	Flocculant Dose(3) (g/MT)	Flocculant Conc. (4) (g/l)	Design Basis Net Feed Loading (m^3/m^2hr)(5)	Predicted Overflow TSS Conc. Range (mg/l)(6)	Predicted Underflow Density(7)
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/t).
(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 0.45 m septum.
(7) Maximum underflow solids concentration recommended based on viscosity considerations and experience.

Outotec's results were used directly for the process design. The Pocock testing was performed for confirmatory purposes only. These tests yielded similar underflow densities and area requirements but a slightly higher flocculant dosage requirement compared to the Outotec results.

13.2.3 2019 Rheology Testwork at an Increased Saprolite Content

A sample composed of 20% saprolite/saprock and 80% hard rock was subjected to rheology testing in late 2019 during the optimization study phase. The objectives of this additional testing were to obtain data for tailings pipeline design purposes and to evaluate whether the increased saprolite content poses a risk due to the expected increase in viscosity.

The results, presented in Table 13.53, indicate a critical settling density of 68% w/w solids at a yield stress of approximately 40 Pa. This is similar to previous results for a blend containing 10% saprolite. At a density of 58.1% w/w solids (lowest density tested), the yield stress decreased to less than 4 Pa and some minor settling was observed. Both of these densities exceed the expected operating point of below 50% w/w solids. The interpretation is that the increase in saprolite content from the original FS blend of 10% to 20% will not cause any viscosity related problems.

Table 13.53 SGS 2019 Rheology Testing

Sample I.D.	Test Code	Solids Conc. (%wt.)	Coefficient of Rigidity (Pa·s)	Yield Value (Pa)	Apparent Viscosity (Pa·s) at the following Shear Rates:						
					5 sec^{-1}	25 sec^{-1}	50 sec^{-1}	100 sec^{-1}	200 sec^{-1}	400 sec^{-1}	600 sec^{-1}
Rheology Comp July 2019 (as received)	T1	70.8	0.116	101	3.010	1.440	1.143	0.890	0.589	0.370	0.283
	T2	69.2	0.086	55	3.708	1.163	0.852	0.588	0.356	0.221	0.172
	T3	67.2	0.049	29	4.017	0.768	0.531	0.320	0.191	0.120	0.096
	T4	64.7	0.029	14	5.129	0.518	0.288	0.166	0.100	0.064	0.053
	T5	61.7	0.018	7.1	1.845	0.277	0.150	0.087	0.053	0.036	0.031
	T6	58.3	0.012	3.3	0.663	0.138	0.078	0.046	0.029	0.021	0.021
Rheology Comp July 2019 (pH 10.5)	T7	70.7	0.105	116	3.344	1.320	1.091	0.928	0.638	0.397	0.297
	T8	69.2	0.092	61	7.200	1.142	0.874	0.645	0.391	0.242	0.187
	T9	67.3	0.054	32	1.890	0.816	0.560	0.355	0.213	0.133	0.105
	T10	64.7	0.030	15	2.037	0.530	0.297	0.172	0.103	0.067	0.055
	T11	61.7	0.019	7.1	1.364	0.278	0.155	0.089	0.054	0.036	0.032
	T12	58.1	0.013	3.1	1.124	0.140	0.077	0.045	0.029	0.021	0.021

13.3 Results Interpretation

The following sections describe the main testwork results that contributed towards the development of the process flowsheets 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 13.54.

Table 13.54 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

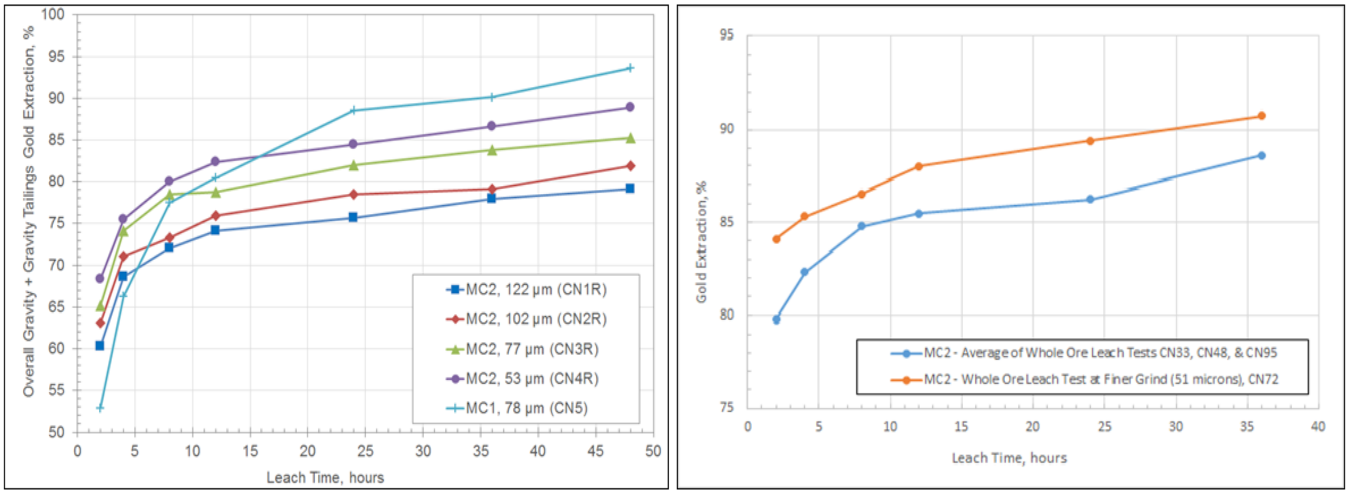
Note: Italicized values are assumed by OMC.

The comminution circuit design is based on a blend of 80% Boto 2/Malikoundi primary ore (fresh rock) and 20% saprolite/saprock (Memorandum No. 5130-MEM-002 Rev 2). With respect to the BWi, A x b and CWi parameters the 85^{th} percentile was used for equipment sizing. The 50^{th} percentile was used for the Ai parameter as this is used for OPEX estimation purposes only.

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} of 53 µm) 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} of 53 µm, yielded 2% to 3% higher gold extraction than the results for a grind P_{80} of 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.

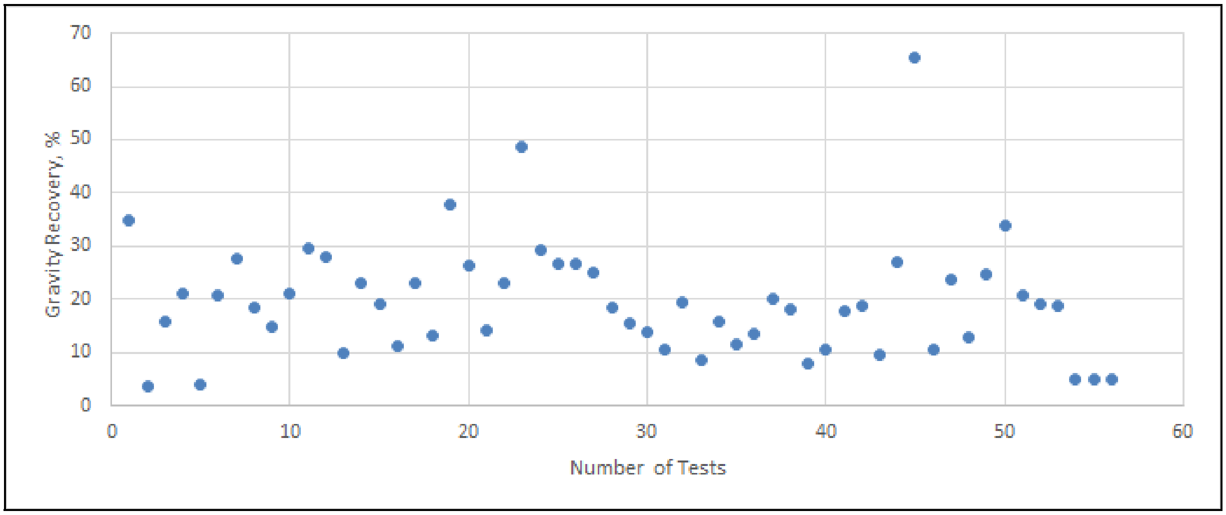
The financial analysis was based on a set of inputs, which included a base case gold price of $1,200/oz. A sensitivity analysis showed that the NPV and IRR becomes positive with a payback at the end of a 10-year mine life if gold price becomes $1,250/oz.

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

13.3.3 Gravity Circuit

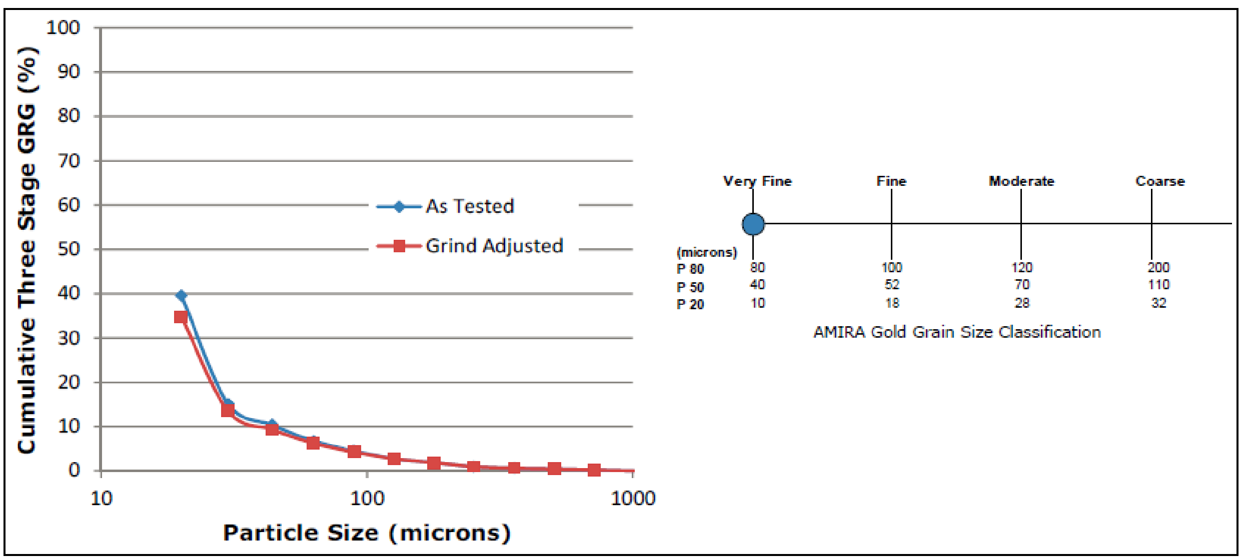
A review of all the previous gravity separation results showed that out of 56 tests (from 2013 to 2018) only three yielded a GRG close to 35%, one close to 50%, and one Boto 5 sample over 65%. The overall GRG average is at approximately 20%. This is considered to be on the low end of the GRG range and inclusion of a gravity circuit cannot be justified. Refer to Figure 13.53 for details.

Figure 13.53 Gravity Separation Results from 2013 to 2018



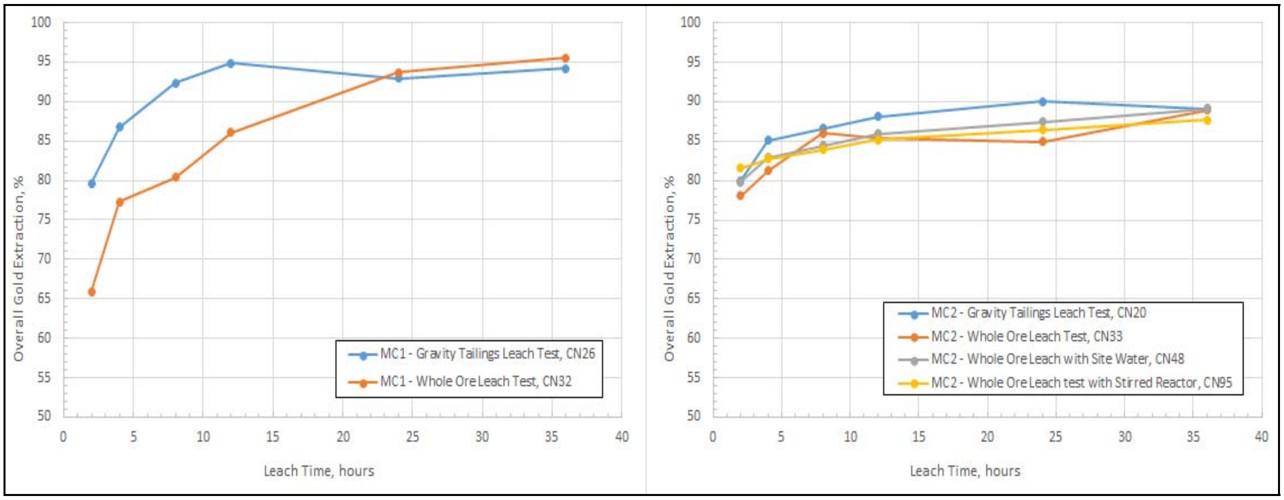
The E-GRG test results combined with the results from whole ore leaching during the SGS 2017/2018 program also did not provide justification for including a gravity circuit in the plant design.

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 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 made to omit a gravity circuit from the recommended flowsheet.

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 be reduced significantly (see Table 13.55). 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 13.55 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 followed by leaching with continuous oxygen sparging 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 be processed. 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 material of this grade and below will not be treated.

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

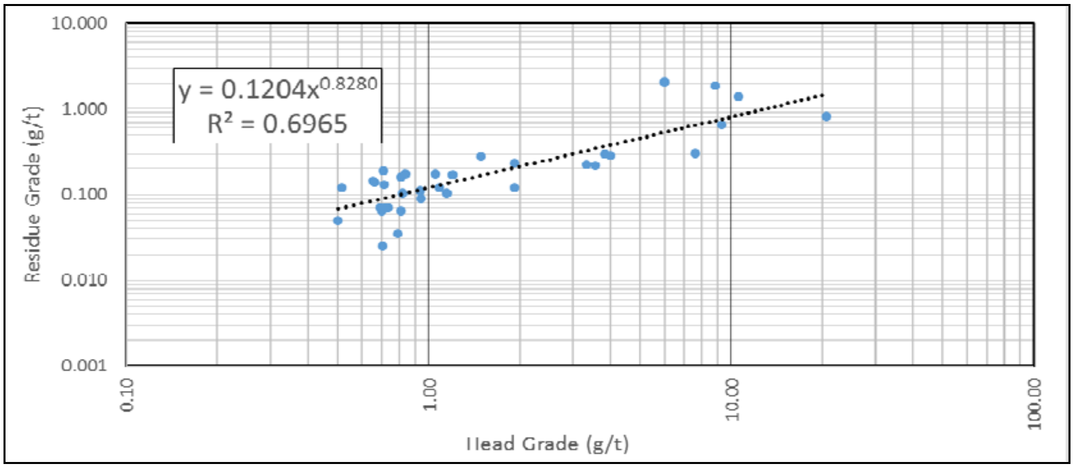
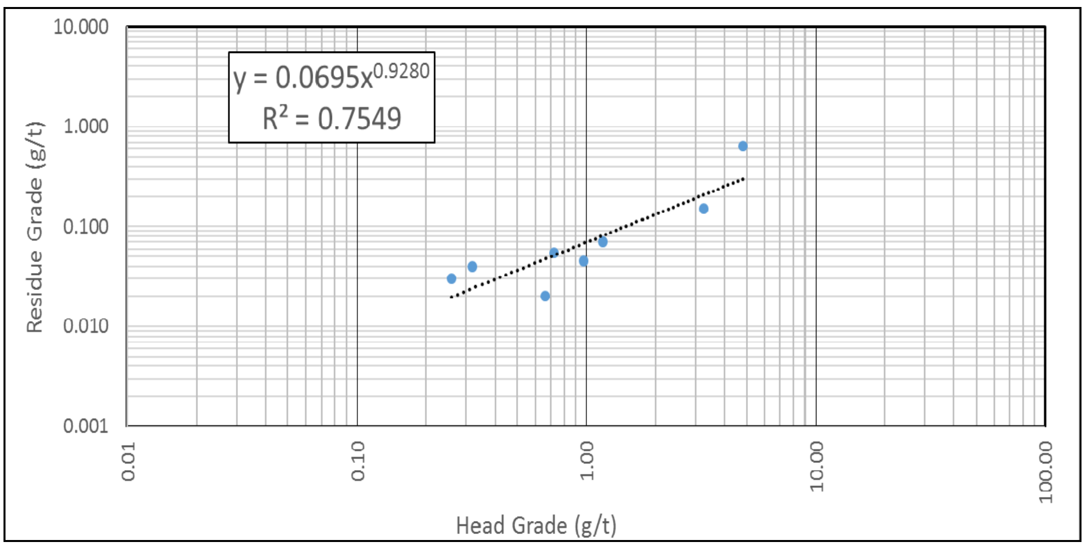


Figure 13.57 Gold Extraction Model for Boto 2/Malikoundi Saprolite/Saprock



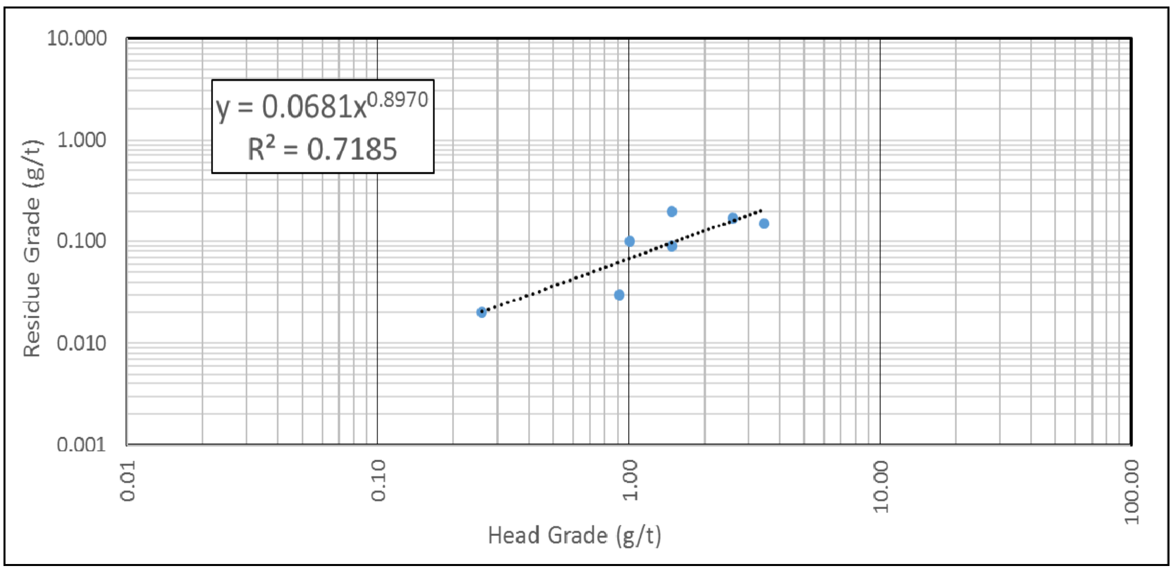
Since MC-1 (Boto 5 Master Composite) showed little gold extraction benefit from lead nitrate and oxygen addition, a combination of old and new test results as seen in

Table 13.56 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 be treated.

Table 13.56 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 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

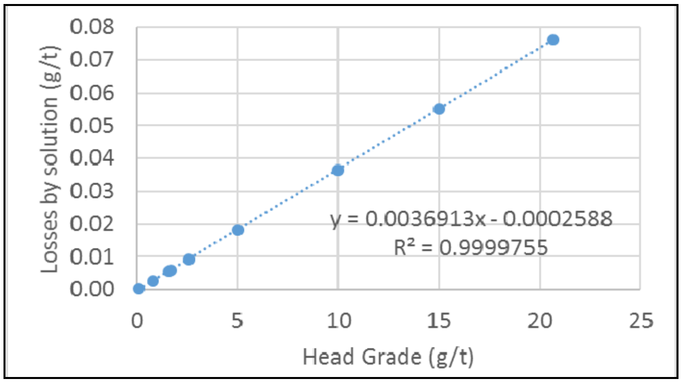


Figure 13.60 Gold Losses Model for Boto 2/Malikoundi Saprolite/Saprock

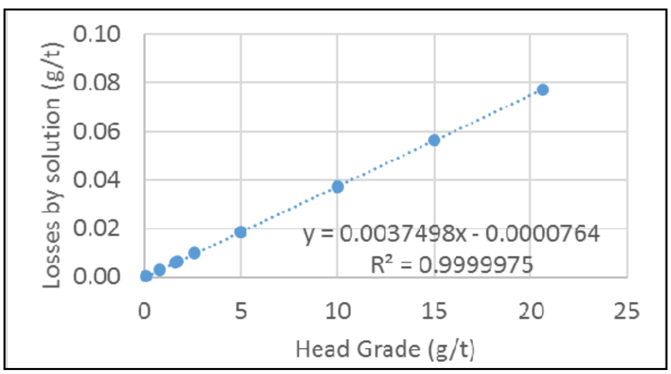
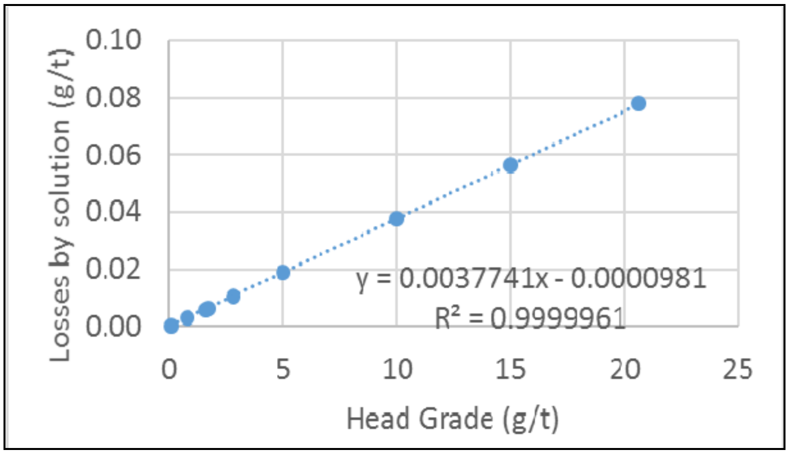


Figure 13.61 Gold Losses Model for Boto 5 Saprolite/Saprock



Gold Recovery Models

Combining the gold extraction and gold losses models, a gold recovery models are 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

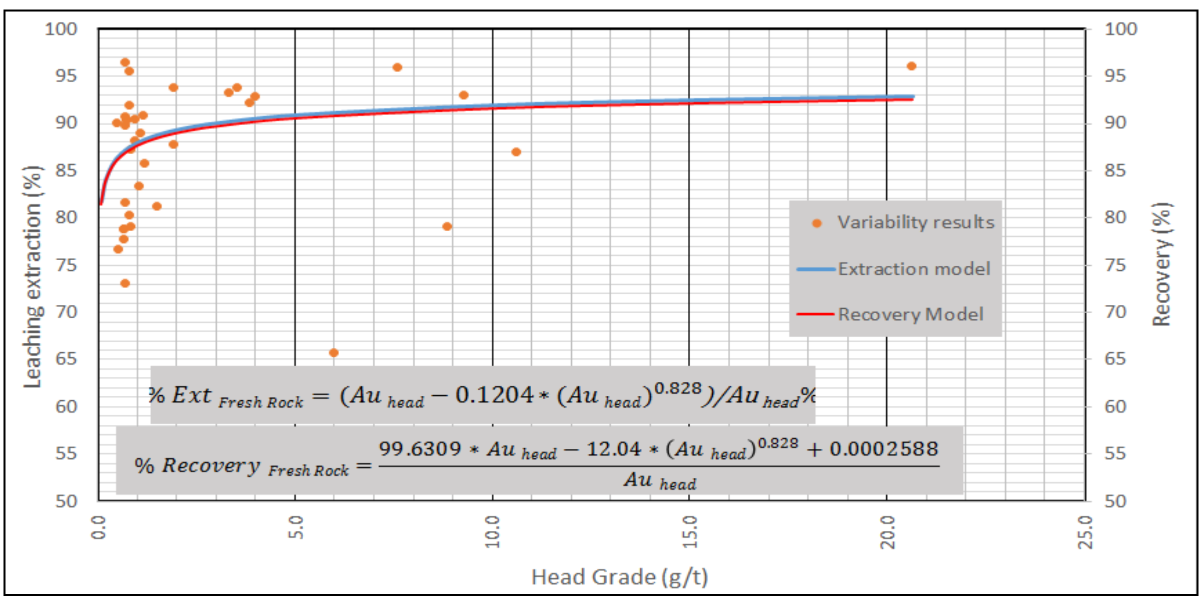


Figure 13.63 Gold Extraction and Recovery Models for Boto 2/Malikoundi Saprolite/Saprock

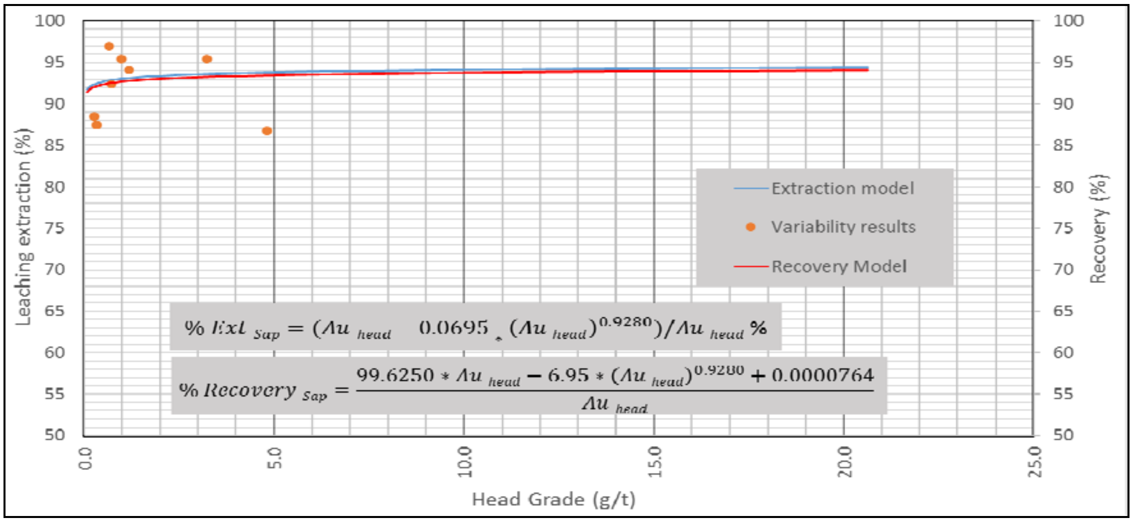
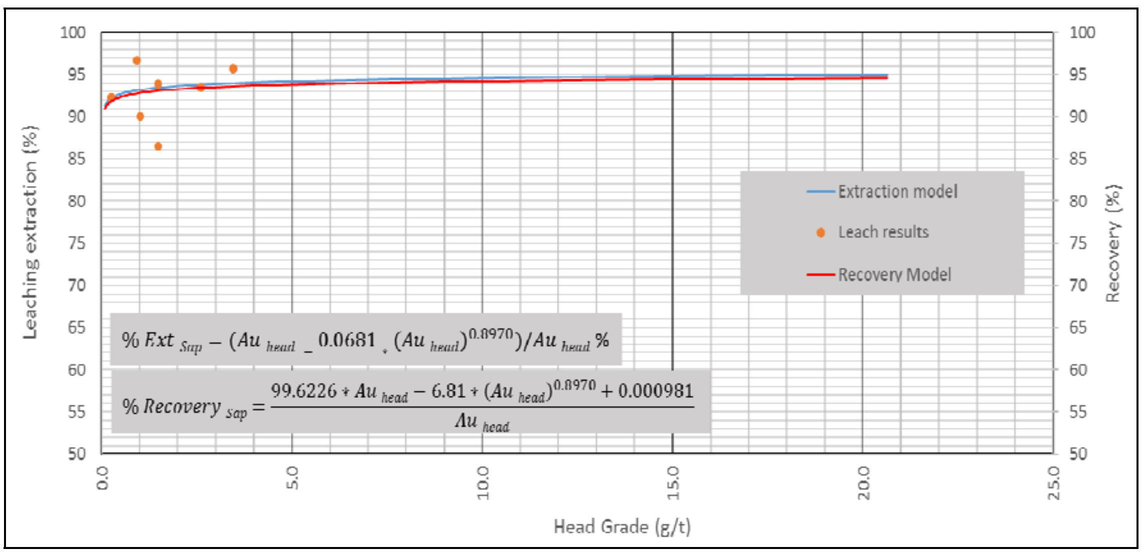


Figure 13.64 Gold Extraction and Recovery Models for Boto 5 Saprolite/Saprock



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 Metallurgical Design Criteria

A summary of the metallurgical inputs in the process design criteria is presented in Table 13.57.

Table 13.57 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
Design Ore Blend - Fresh Rock - Saprolite/Saprock	% %	80 20	Lycopodium
Ore Blend for Comminution Design - Fresh rock - Saprolite/Saprock	 % %	 80 20	Lycopodium
Head Grade - Gold (Ave) - Gold (Design)	g/t Au g/t Au	1.71 2.83	LOM average head grade Annual peak +20%
- Silver	g/t Ag	negligible	Testwork
Ore Specific Density Ore Bulk Density Crushing Work Index (CWi, design) Bond Ball Mill Work Index (BWi, design) Bond Abrasion Index (Ai, average)	t/m^3 t/m^3 kWh/t kWh/t 	2.81 1.53 17.7 18.7 0.390	Testwork Lycopodium Lycopodium Lycopodium Lycopodium
Targeted Grind Size P_{80} Leach & CIP Circuit Residence Time Targeted Pulp Slurry Density	µm hrs % solids	75 35 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 0.125 0.145	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 2.60 1.73	 Testwork Testwork Calculation
Daily Loaded Carbon Advance Rate Maximum Loaded Carbon Grade Targeted Loaded Carbon Grade	t/d g/t Au g/t Au	5.0 Up to 5,000 4,297	Lycopodium Lycopodium Lycopodium

13.4 Conclusions and Recommendations

13.4.1 Conclusions

The following conclusions can be drawn from the metallurgical testwork:

- Fresh rock, saprolite, and saprock at Boto are readily amenable to whole ore cyanidation.
- The optimum grind size was determined to be a P_{80} of 75 µm.

- Gold recovery is predicted to be 89.6% for Boto 2/Malikoundi fresh rock, 93.5% for Boto 2/Malikoundi saprolite/saprock, and 93.2% for Boto 5 saprolite/saprock, at the design head grade of 2.83 g/t Au. These recovery numbers will be marginally lower at the average head grade of approximately 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 leaching, combined with lead nitrate addition, is beneficial to optimise the leach kinetics and gold recovery.

- Leach extraction rates are essentially completed by 24-hours to 36-hours.

- Cyanide consumption rates are expected to be low, averaging about 0.125 kg NaCN/t ore. When accounting for residual cyanide in the CIP tailings solution, an addition rate of 0.27 kg NaCN/t ore is expected.

- Lime consumption rates are expected to be moderate, averaging 1.73 kg CaO/t ore at the design ore blend. When accounting for 90% purity of the supplied lime an addition rate of 1.92 kg lime/t ore is expected

- CIP modelling indicated that the specified configuration represents a robust adsorption circuit, capable of accommodating both an increased throughput (10% above design rate) as well as short-term head grade peaks of 2.83 g/t Au with virtually no change to the dissolved gold loss.

- Rheology testing indicated that a slurry consisting of 20% saprolite is not expected to pose significant viscosity related problems.

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 Boto Gold 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. Tudorel Ciuculescu, P.Geo., Senior Geologist with RPA. The effective date of this mineral resource is December 31, 2019.

The resource estimate has been prepared using interpreted mineralized lenses for three deposits that are part of the Boto Project. The deposits are Malikoundi/Boto 2, Boto 5, and Boto 6, as shown in Figure 14.1. The Boto 4 deposit has been excluded from the resource estimate. Geovia GEMS™ 6.8 was used for drill hole database management, geological interpretation, mineralized wireframe modelling and to generate the block model supporting the resource estimate. The inverse distance cubed (ID^3) interpolation method was used to estimate the block model gold grades.

Figure 14.1 Plan View of the Boto Project

The mineral resources are reported at variable cut-off grades, based on weathering horizons that vary between 0.37 g/t Au and 0.50 g/t Au. Gold grades were capped prior to compositing, with capping levels varying by mineralized lens between 2 g/t Au and 25 g/t. Several mineralized lenses did not require capping.

The mineral resources are reported within optimized constraining shells. The optimized constraining shells were developed for each deposit by IAMGOLD using Hexagon Mining MineSight 3D and incorporates metal recovery, geotechnical parameters, and assumed costs for each weathering 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 Boto Project.

Table 14.1 Summary of Mineral Resources for the Boto Project – December 31, 2019

Classification	Tonnes ('000t)	Grade (g/t Au)	Contained Metal ('000 oz Au)
Indicated	40,600	1.56	2,033
Inferred	8,200	1.78	469

Notes:
- *Mineral Resources are reported within an optimized constraining shell using MineSight 3D software.*
- *Mineral Resources are reported inclusive of Mineral Reserves;*
- *Cut-off grades vary between 0.37 g/t Au and 0.50 g/t Au, depending on the deposit and the* weathering *type of material;*
- *Mineral resources were estimated based on a gold price of $US 1,500/ oz;*
- *Capping of grades varied between 2 g/t Au and 25 g/t Au on raw assays by mineralized zone;*
- *The density varies between 1.65 g/cm^3 and 2.75 g/cm^3 depending on* weathering *zone.*

The mineral resources for the Boto project include the Malikoundi/Boto 2, Boto 5, and Boto 6 deposits. No resources were reported at Boto 4 as it is located within the 500 m wide exclusion zone along the Balinko river shore (the border of Senegal and Mali) and underneath the village of Guémédji. Should these factors change, the Boto 4 deposit will be re-evaluated.

RPA is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate at the time of this report.

14.1.2 Resource Database

The drill hole database for the Boto deposits was provided by IAMGOLD. The database contains records of core drilling and reverse circulation (RC) drilling completed until the end of April 2019. Collar position, downhole deviation survey, gold assay, lithology, weathering profile, density, structural, alteration, mineralization, chemical composition (XRF) and recovery information are stored in separate tables.

The database contains information from 951 drill holes with a total length of 146,195.7 m. Table 14.2presents the drilling available for the Boto deposits. The holes used for mineralized wireframe definition are presented in Table 14.3. Drill hole database verification steps are described in Section 12 of this report.

Table 14.2 Resource Database drilling

Type	Length (m)	Count
Total Core	95,813.7	430
Total RC	50,382.0	521
Total Drilling	**146,195.7**	**951**

Table 14.3 Drilling Defining Mineralized Wireframes

Area/Type	Type	Length (m)	Count
Boto2-MK-T8	Core	57,084.8	241
	RC	15,685.0	149
	Total	72,769.8	390
Boto4	Core	10,596.4	44
	RC	1,374.0	16
	Total	11,970.4	60
Boto5	Core	5,712.0	31
	RC	2,458.0	28
	Total	8,170.0	59
Boto6	Core	15,411.5	76
	RC	3,915.0	39
	Total	19,326.5	115

The database was provided by IAMGOLD to RPA as part of a Geovia GEMS 6.8 project. In addition to data tables, the GEMS project included interpreted mineralized wireframes, geology solids, topography, and weathering surfaces.

14.2 MALIKOUNDI/BOTO 2

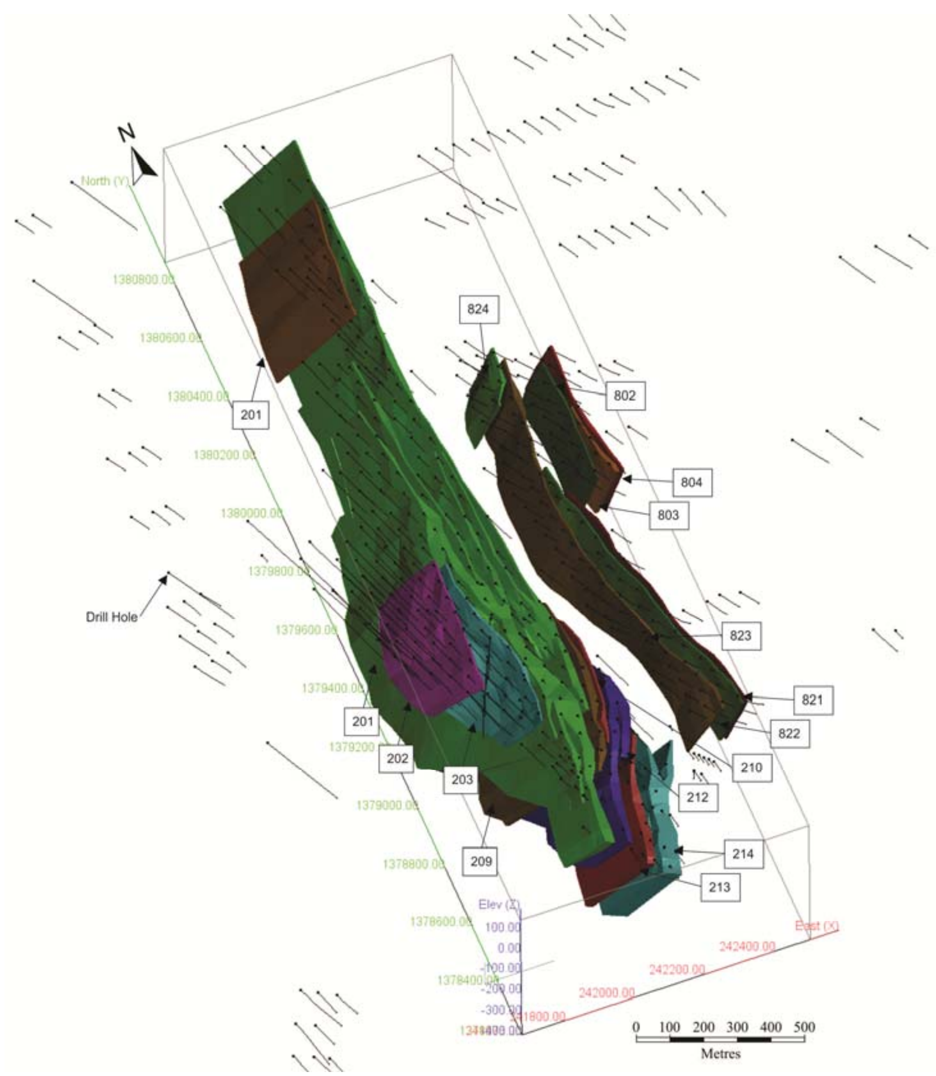
14.2.1 Geological Interpretation

Mineralization wireframes for the Malikoundi/Boto 2 deposit were modelled by IAMGOLD geologists. Drilling completed until the end of May 2019 was used to update and refine the current resource solids. Recent drilling east of the main Malikoundi trend resulted in the definition of the T8 mineralized area, consisting of a set of veins parallel to the main trend. The wireframes were built from 3D rings interpreted on vertical sections spaced at 50 m intervals. The 3D rings, snapped to the beginning and end of sampled intervals down the drill hole, were then connected to create the mineralization solids.

The mineralization solids were defined based on a combination of 0.15 g/t Au nominal cut-of grade, presence of favourable lithology and higher intensity alteration, presence of sulphides, and intensity of fracturing. A minimum nominal thickness of 4 m was used throughout the modelling exercise. The average core length of the mineralized intercepts is approximately 19 m, while the average true thickness is approximately 15 m.

RPA reviewed the modelled mineralized solids, lithology and alteration wireframes, and weathering surfaces. RPA considers the wireframes provided by IAMGOLD a good representation of the mineralization present at Malikoundi/Boto 2 deposit and found them to be appropriate for resource estimation. RPA adopted the wireframes provided by IAMGOLD and used them to constrain the block model supporting the Malikoundi Mineral Resource estimate. The mineralized wireframes were used to select the resource samples and constrain the resource estimate. The weathering surfaces were used to define contacts between different oxidation state material and density flagging in the block model. Figure 14.2shows the Malikoundi/Boto 2 mineralized wireframes.

Figure 14.2 Mineralization Wireframes Malikoundi/Boto 2



14.2.2 Weathering Profile

The weathering profile at Boto has been divided into four major units: laterite, saprolite, transition and fresh rock. The upper unit, laterite, is considered to include transported and reworked material, hence the laterite cover and samples selected from this unit were not considered for resource estimation.

For the current estimate, the contact surfaces between weathering domains have been reviewed and adjusted by IAMGOLD. Visual, hardness, and geochemical information were used to define the contact surfaces. Table 14.4presents the block model flagging codes for the weathering profile.

Table 14.4 Weathering Profile Codes

Horizon	Code
Laterite	40
Saprolite	50
Transition	60
Rock	70

14.2.3 Statistical Analysis

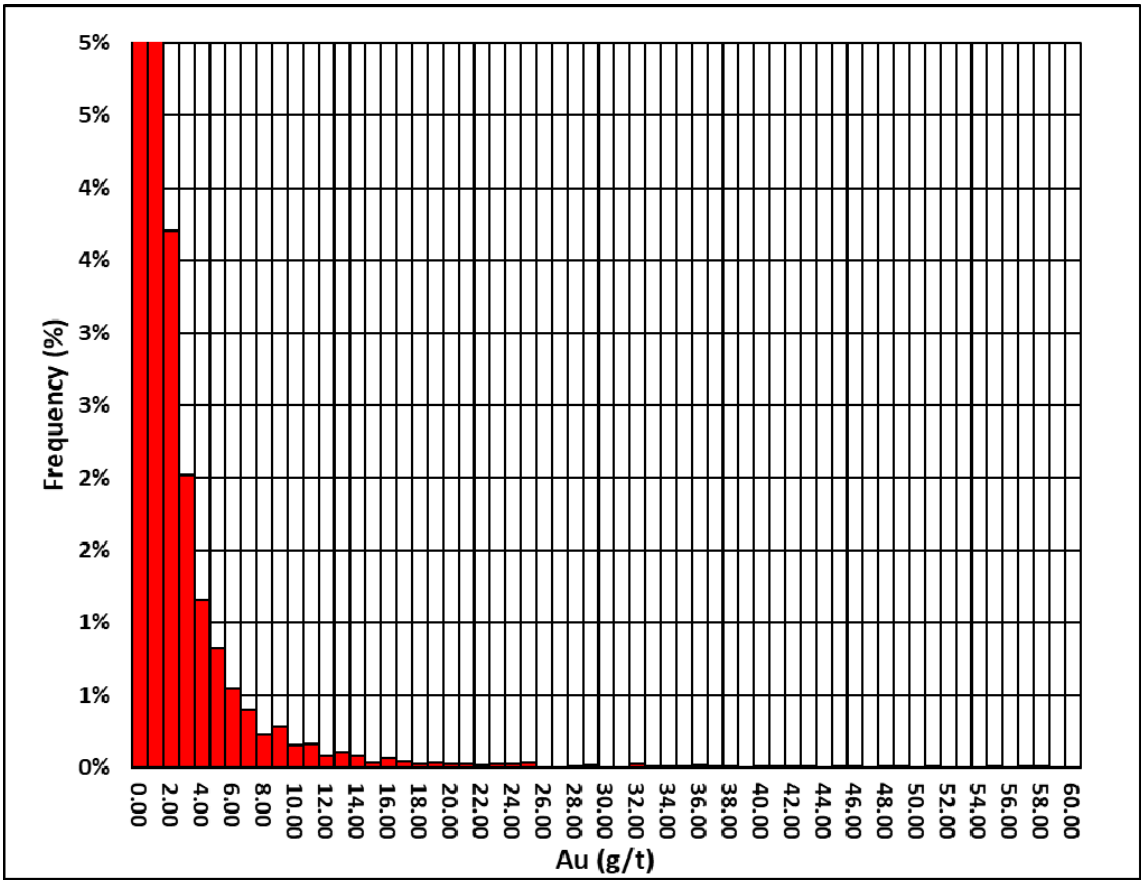
Assays

The Malikoundi/Boto 2 mineralization solids were used to flag the resource samples. The samples retained inside the wireframes were the basis of the resource estimate, consisting of 18,674 samples, with a total sampled length of 19,426.5 m. The resource samples form a positively skewed population, characteristic for gold mineralization, with a relatively large number of low-grade samples and long trail of higher-grade samples. Sample length weighted descriptive statistics of the assays by mineralized lens are presented in Table 14.5. Figure 14.3shows the resource assays histogram for all lenses.

Table 14.5 Descriptive Statistics for Malikoundi/Boto 2 Deposit by Mineralized Lens

Lens	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Stdev (g/t Au)	Variance	CV
202	231	0.00	3.02	0.13	0.38	0.14	2.85
203	435	0.00	13.18	0.38	1.19	1.42	3.12
207	89	0.00	4.86	0.23	0.64	0.41	2.78
209	1,360	0.00	21.77	0.58	1.37	1.88	2.37
210	2,071	0.00	190.00	1.22	4.97	24.74	4.08
212	1,436	0.00	51.64	0.56	2.21	4.87	3.93
213	418	0.00	6.19	0.41	0.69	0.48	1.70
214	606	0.00	9.47	0.42	0.75	0.56	1.80
246	10,393	0.00	129.13	1.09	3.57	12.71	3.27
802	123	0.00	5.93	0.48	0.86	0.74	1.79
803	121	0.00	23.23	0.84	2.64	6.99	3.16
804	273	0.00	10.30	0.47	0.95	0.91	2.05
821	307	0.00	43.58	0.87	3.12	9.76	3.61
822	308	0.00	11.99	0.56	1.51	2.27	2.69
823	468	0.00	11.83	0.43	1.05	1.10	2.42
824	79	0.00	5.93	0.35	0.85	0.72	2.43
All Lenses	**18,671**	**0.00**	**190.00**	**0.90**	**3.25**	**10.57**	**3.60**
Sample Length	**18,673**	**0.50**	**5.00**	**1.04**	**0.20**	**0.04**	**0.20**

Figure 14.3 Assay Histogram – All Lenses Malikoundi/Boto 2



Capping Analysis

Capping of high grade assays prior to compositing is a practice aimed at limiting the influence of erratic high-grade assays, which otherwise have the potential to overpower surrounding lower grade samples. In the absence of production data that would allow the determination of appropriate capping levels, a number of statistical procedures were used. RPA applied statistical methods to establish the capping levels for the Malikoundi/Boto 2 estimation domains. A combination of histograms, decile analysis, probability plots, disintegration and visual inspection of the spatial location of higher-grade assays were used to determine the capping levels for each mineralized lens. RPA capped high-grade assays prior to compositing. Table 14.6shows the capping levels and number of values affected for each of the mineralized lenses.

Table 14.6 Capping Levels by Mineralized Lens - Malikoundi/Boto 2

Lens	Capping Value (g/t Au)	Capped Assays	Capping Grade Percentile	Total Metal (g Au)	Metal Loss %	Average Grade (g/t Au)	CV
202	no capping	0	1.000	34	0	0.13	3.13
203	5	4	0.991	154	13	0.33	2.42
246	25	36	0.997	10,844	6	1.02	2.46
207	2.5	1	0.990	20	11	0.21	2.52
209	8	5	0.996	755	5	0.55	2.01
210	25	4	0.998	2,306	8	1.13	2.27
212	10	5	0.997	733	12	0.50	2.41
213	4	4	0.991	188	3	0.40	1.59
214	4	5	0.994	362	4	0.40	1.54
802	2.5	3	0.976	53	11	0.43	1.51
803	5	4	0.968	69	32	0.57	2.05
804	4	3	0.990	117	8	0.43	1.63
821	10	6	0.983	221	17	0.72	2.55
822	6	6	0.982	152	12	0.49	2.29
823	5	3	0.994	190	7	0.40	2.10
824	4	1	0.992	26	7	0.32	2.35

Samples with assay values higher than the lens-capping threshold were capped at corresponding levels. Table 14.7 shows the descriptive statistics of the capped assays

Table 14.7 Descriptive Statistics of Capped Assays - Malikoundi/Boto 2

Lens	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Stdev (g/t Au)	Variance	CV
202	231	0.00	3.02	0.13	0.38	0.14	2.85
203	435	0.00	5.00	0.33	0.78	0.61	2.35
207	89	0.00	2.50	0.21	0.49	0.24	2.38
209	1,360	0.00	8.00	0.55	1.10	1.20	1.99
210	2,071	0.00	25.00	1.12	2.53	6.42	2.26
212	1,436	0.00	10.00	0.50	1.18	1.39	2.38
213	418	0.00	4.00	0.40	0.62	0.39	1.58
214	606	0.00	4.00	0.40	0.61	0.37	1.53
246	10,393	0.00	25.00	1.02	2.49	6.20	2.43
802	123	0.00	2.50	0.43	0.62	0.39	1.45
803	121	0.00	5.00	0.57	1.14	1.30	2.01
804	273	0.00	4.00	0.43	0.69	0.48	1.61
821	307	0.00	10.00	0.72	1.80	3.25	2.50
822	308	0.00	6.00	0.49	1.10	1.22	2.23
823	468	0.00	5.00	0.40	0.82	0.67	2.03
824	79	0.00	4.00	0.33	0.70	0.49	2.16

Composites

In preparation for grade estimation, samples were composited to intervals of equal length. RPA selected a compositing length of 2 m fixed intervals. Compositing was done from collar to toe within each mineralize lens, starting at the wireframe pierce-point and continuing to the point at which the hole exited the lens. Composites at least 50% of compositing length were considered valid. No composites were discarded for Malikoundi/Boto 2. Capped composites were used for resource estimation.

Table 14.8 and Table 14.9 show the descriptive statistics of the capped and uncapped 2 m composite values by mineralized lens.

Table 14.8 Descriptive Statistics of the Capped 2 m Composites – Malikoundi/Boto 2

Lens	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Stdev (g/t Au)	Variance	CV
202	143	0.00	1.73	0.14	0.32	0.10	2.26
203	226	0.00	4.57	0.33	0.65	0.42	1.95
207	56	0.00	1.28	0.20	0.33	0.11	1.66
209	708	0.00	7.16	0.55	0.92	0.85	1.68
210	1,059	0.00	23.05	1.10	2.13	4.52	1.92
212	745	0.00	7.10	0.50	1.01	1.03	2.01
213	238	0.00	4.00	0.40	0.54	0.29	1.36
214	442	0.00	4.00	0.41	0.57	0.32	1.38
246	5,296	0.00	25.00	1.04	2.10	4.42	2.02
802	57	0.00	2.35	0.45	0.50	0.25	1.11
803	60	0.00	4.79	0.58	0.99	0.98	1.72
804	137	0.00	2.94	0.44	0.56	0.31	1.28
821	152	0.00	10.00	0.73	1.63	2.67	2.22
822	157	0.00	6.00	0.49	0.90	0.81	1.82
823	226	0.00	2.98	0.44	0.65	0.42	1.48
824	35	0.00	2.27	0.38	0.59	0.35	1.55
Boto2MK	9,737	0.00	25.00	0.85	1.80	3.23	2.12

Table 14.9 Descriptive Statistics of the Uncapped 2 m Composites – Malikoundi/Boto 2

Lens	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Stdev (g/t Au)	Variance	CV
202	143	0.00	1.73	0.14	0.32	0.10	2.26
203	226	0.00	8.02	0.38	0.95	0.90	2.46
207	56	0.00	2.44	0.22	0.43	0.18	1.91
209	708	0.00	12.89	0.58	1.10	1.22	1.91
210	1,059	0.00	95.86	1.20	3.68	13.57	3.06
212	745	0.00	25.93	0.57	1.68	2.81	2.95
213	238	0.00	4.97	0.41	0.59	0.35	1.46
214	442	0.00	7.14	0.43	0.68	0.46	1.60
246	5,296	0.00	108.64	1.11	2.99	8.96	2.71
802	57	0.00	3.60	0.51	0.68	0.47	1.35
803	60	0.00	13.90	0.85	2.13	4.55	2.52
804	137	0.00	6.09	0.47	0.75	0.56	1.58
821	152	0.00	22.36	0.88	2.50	6.26	2.84
822	157	0.00	9.28	0.56	1.22	1.49	2.18
823	226	0.00	6.00	0.47	0.80	0.63	1.70
824	35	0.00	3.23	0.41	0.70	0.48	1.70
Boto2MK	9,737	0.00	108.64	0.91	2.64	6.97	2.90

14.2.4 Block Model

A block model was setup in GEOVIA GEMS 6.8 software to support the resource estimate. The block model for the Malikoundi/Boto 2 deposit has a block size of 5 m wide by 10 m deep by 5 m high. The block model is not rotated, with the elongated side of the blocks aligned parallel to the north-south strike of the deposit. The block size is appropriate for the intended open pit operation planning and adequate for the 50 m by 50 m drill hole spacing available at Malikoundi/Boto 2. Table 14.10 summarizes the block model parameters.

Table 14.10 Block Model Parameters for the Malikoundi/Boto 2 Deposit

	Parameters
Easting	241,000 mE
Northing	1,378,000 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

Variography

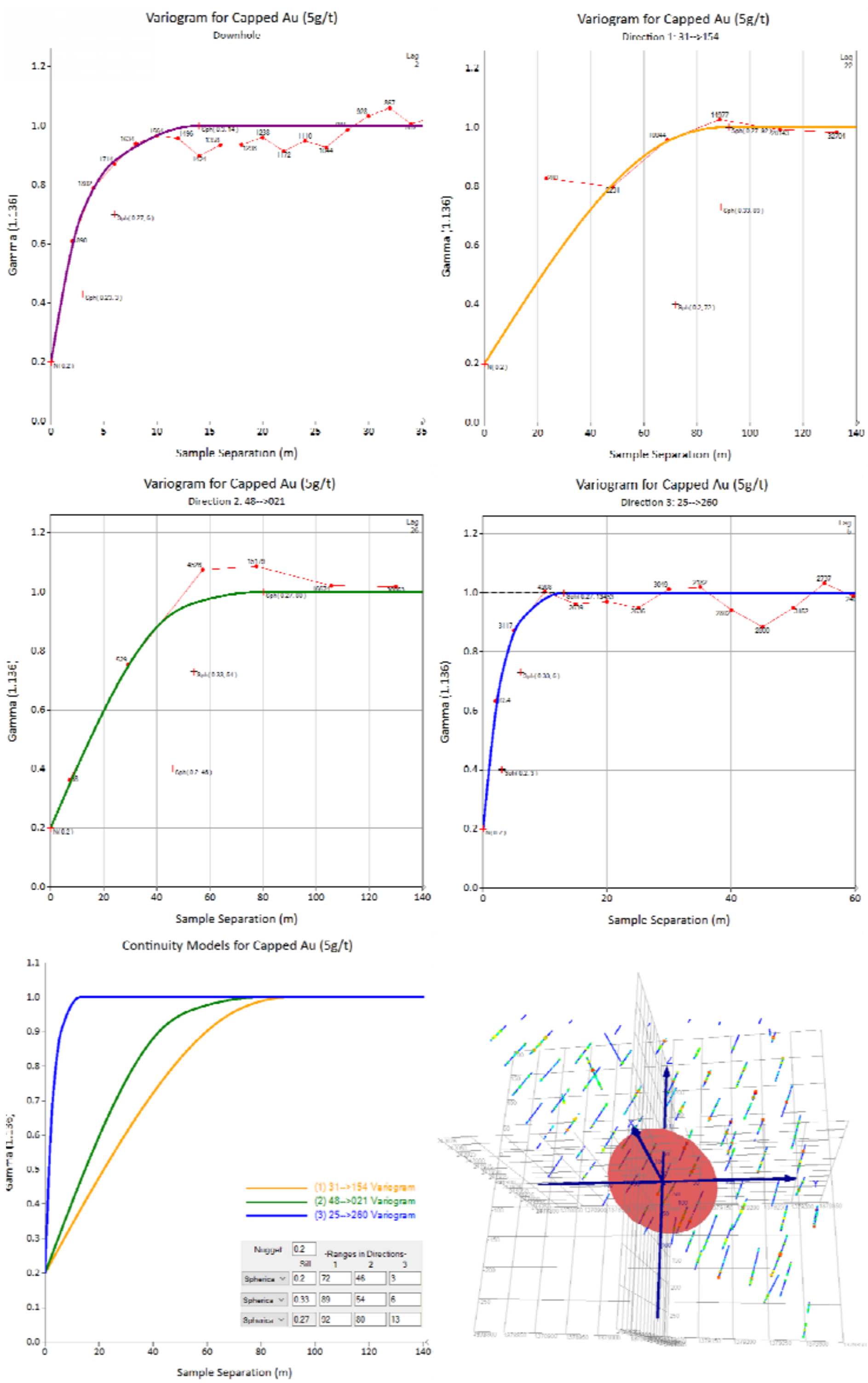
The modelled mineralization wireframes for Malikoundi/Boto 2 deposit capture the favourable lithology, alteration, and intense fracturing, at a nominal cut-off grade of 0.15 g/t Au. The wireframes include a large proportion of low-grade mineralization in order to avoid fragmentation of the modelled lenses. Mineralized zones of higher grade (1 g/t Au and higher) have a relatively short continuity, difficult to extend beyond 2-3 sections spaced 50 m. The mix of low and high-grade samples in the same domain renders the variographic analysis difficult.

RPA attempted variographic analysis for a segment of the vein 246, on 2 m capped composites. An arbitrary section of approximately 400 m along strike, with higher-grade continuity, was selected. Composites associated with branching parts of the mineralized lens were also removed. The oriented variograms were unstable and very sensitive to the angles of tolerance for sample selection, resulting in a range of anisotropy ratios from 2:1 dipping approximately 50° north to almost 1:1. Applying a lower capping to the composites resulted in the reduction of anisotropy. The ranges observed were generally between 80 m and 120 m for the major and 60 m to 100 m for the semi-major directions.

The ranges and orientations observed in the test variographic analysis support a search radius of 100 m and a 1:1 ratio between the major and semi-major ranges.

A set of variograms with a lower capping level of 5 g/t Au is shown in Figure 14.4. The graphs include downhole and oriented variograms.

Figure 14.4 Test Variography Malikoundi/Boto 2



Grade Interpolation

The block model was interpolated in two passes. The gold grades were estimated using the 2 m composites with the ID^3 interpolation method (anisotropic). The ID^3 method was favoured in order to preserve local grades in the context of using mineralized wireframes with occasional internal dilution and with lower grade intercepts. Table 14.11 shows estimation parameters for each pass used to estimate gold grades.

Table 14.11 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	1	12	3	1

Search Ellipses

The search ellipses used for the Malikoundi/Boto 2 block model interpolation were similar to those used in previous resource estimates. The ranges used are appropriate for the 50 m by 50 m drill spacing and supported by the test variographic analysis observations. Search ellipses were oriented along the interpreted mineralized lenses. Where necessary, lenses were subdivided to allow a better local fit of the search ellipse. Occasionally the search ellipses were widened to accommodate local lens geometry. Table 14.12 lists the search ellipse parameters used to estimate the Malikoundi/Boto 2 Deposit.

Table 14.12 Search Ellipse Parameters for Pass 1 and Pass 2 for Malikoundi/Boto 2

Profile Name	Search Rotation	Azimuth (°)	Dip (°)	Azimuth (°)	X Range (m)	Y Range (m)	Z Range (m)	Search Type
Pass 1								
M1_02	Az,Dip,Az	270	-50	1	60.0	60.0	20.0	Ellipsoidal
M1_03N	Az,Dip,Az	330	-50	10	60.0	60.0	25.0	Ellipsoidal
M1_10N	Az,Dip,Az	290	-60	0	60.0	60.0	20.0	Ellipsoidal
M1_10S	Az,Dip,Az	260	-50	1	60.0	60.0	20.0	Ellipsoidal
M1_12_42	Az,Dip,Az	260	-50	1	60.0	60.0	20.0	Ellipsoidal
M1_13_14	Az,Dip,Az	330	-47	35	60.0	60.0	20.0	Ellipsoidal
M1_43	Az,Dip,Az	275	-60	0	60.0	60.0	20.0	Ellipsoidal
M1_MKN	Az,Dip,Az	260	-60	0	60.0	60.0	20.0	Ellipsoidal
M1_WIDEN	Az,Dip,Az	315	-55	0	60.0	60.0	35.0	Ellipsoidal
M1_WIDES	Az,Dip,Az	275	-65	0	60.0	60.0	35.0	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	30.0	Ellipsoidal
M2_10N	Az,Dip,Az	290	-60	0	100.0	100.0	30.0	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
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
M2_MKN	Az,Dip,Az	260	-60	0	100.0	100.0	20.0	Ellipsoidal
M1_WIDEN	Az,Dip,Az	315	-55	0	100.0	100.0	50.0	Ellipsoidal
M1_WIDES	Az,Dip,Az	275	-65	0	100.0	100.0	50.0	Ellipsoidal

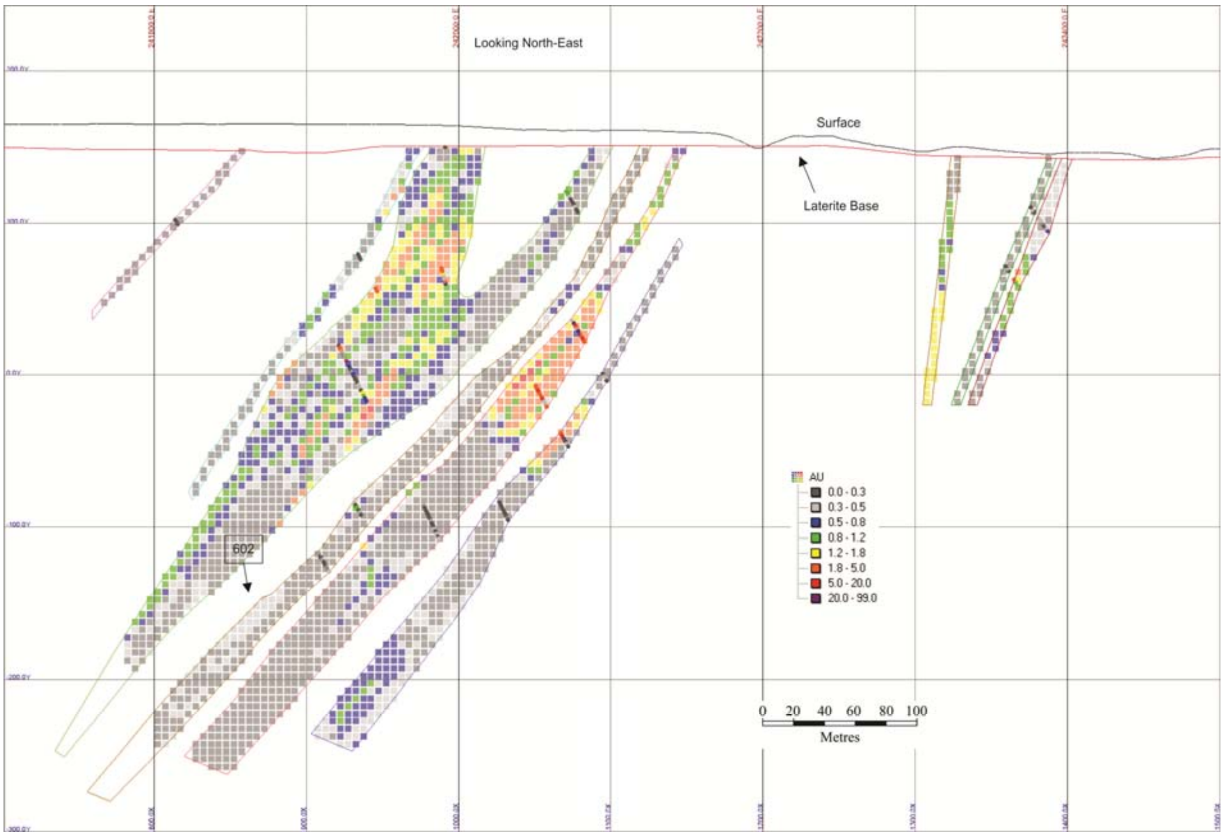
14.2.5 Block Model Validation

RPA used visual and statistical methods to validate the block model attributes flagging and interpolated block grades at Malikoundi/Boto 2. The checks performed included:

- Comparison of mineralized lenses with the flagged blocks.
- Spot checks for search ellipse alignment along mineralized lenses.
- Spot checks for composite and block subdomain flagging.
- Visual check for grade banding, smearing of high grades, and high-grade plumes.
- Visual comparison of composite and block grades on section and plan views.
- Comparison of composite and block grades on swath plots.
- Comparison of interpolated block grades obtained by alternate interpolation methods.

Figure 14.5 presents a vertical section (1,379,005N) showing interpolated block gold grades and the gold grades of the 2 m composites. The grades of the blocks were in good agreement with the composite data used in the interpolation.

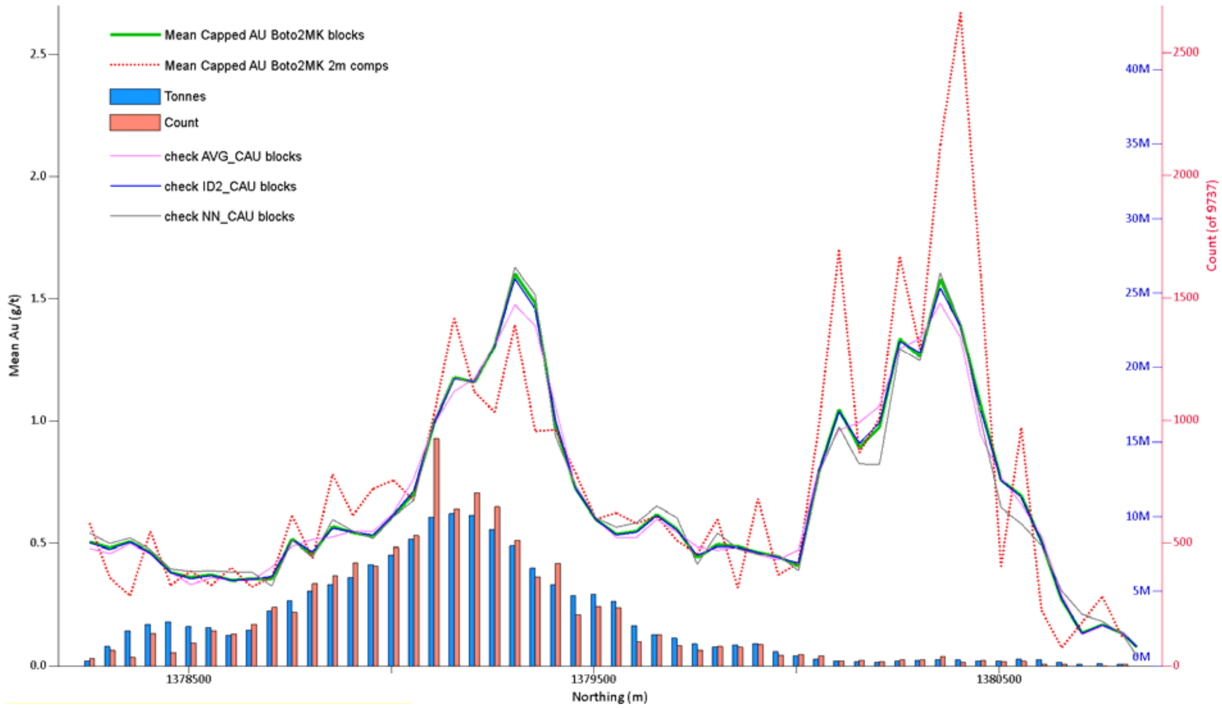
Figure 14.5 Section 1,379,005N – Block Grades and 2 m Composites – Malikoundi/Boto 2



(RPA July 2019)

The alternate grade interpolation method check involves comparison with estimates obtained using different estimators. Along with the ID^3 estimator, RPA ran inverse distance squared (ID^2) and average grade of selected composites. Additionally, RPA generated alternate 6 m composites and ran a nearest neighbour (NN) estimate for Malikoundi/Boto 2. The data from these alternate runs was compared to the ID^3 estimate and the composite data. Figure 14.6 shows the swath plot by northing for Malokoundi/Boto 2. The interpolated grade blocks follow the trends of the composites, in an expected smoothing manner. Departures of the interpolated grades from composite values can be observed for the location of the deep higher-grade holes (1,380,400 mN), and at the north end, where the number of composites is low and a high or a low grade composite has large influence on the average composite value.

Figure 14.6 Swath Plots of Interpolated Gold Grades by Northing – Malikoundi/Boto 2



14.3 BOTO 5

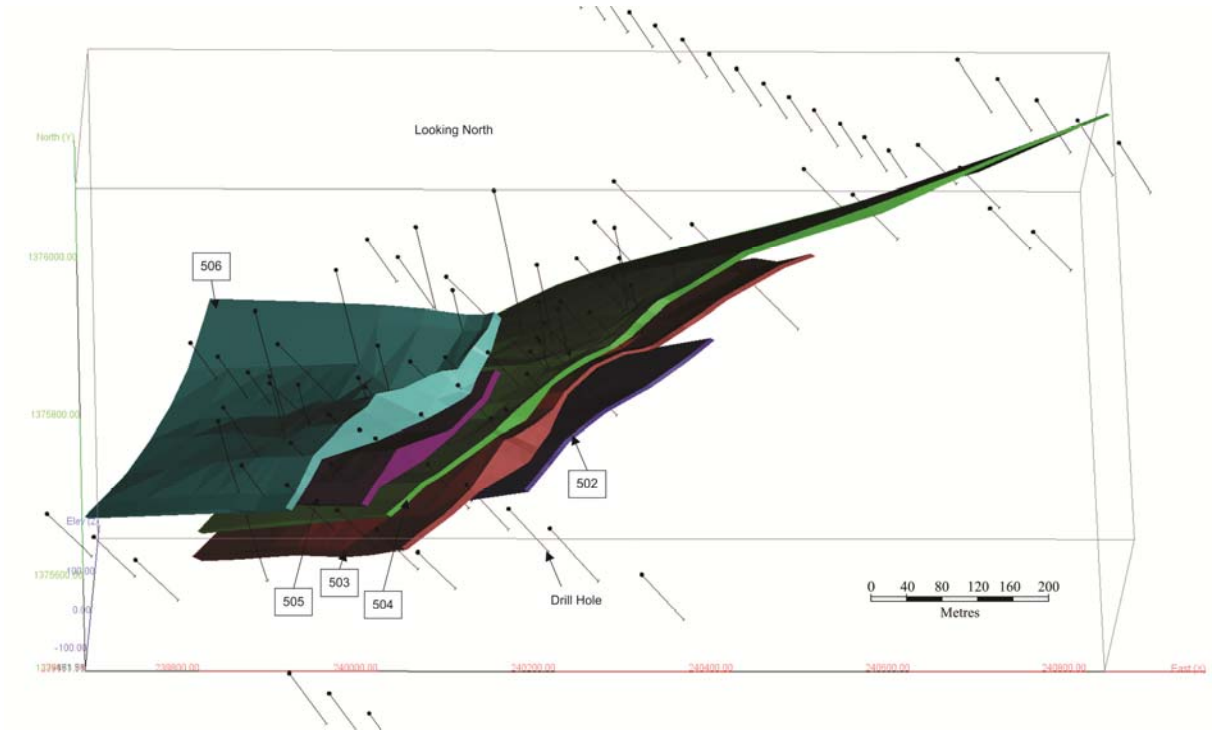
14.3.1 Geological Interpretation

Mineralization wireframes for the Boto 5 deposit were modelled by IAMGOLD geologists. The wireframes were built from 3D rings interpreted on vertical sections spaced at 50 m intervals. The 3D rings, snapped to the beginning and end of sampled intervals down the drill hole, were then connected to create the mineralized 3D solids.

The mineralization solids were defined based on a combination of a 0.15 g/t Au nominal cut-off grade, the presence of favourable lithology and higher intensity alteration, presence of sulphides, and intensity of fracturing. A minimum nominal thickness of 4 m was used throughout the modelling exercise. The average core length of the mineralized intercepts is approximately 11 m, while the average true thickness is approximately 9 m.

RPA reviewed the modelled mineralized solids, lithology and alteration wireframes, and weathering surfaces. RPA considers the wireframes provided by IAMGOLD a good representation of the mineralization present at Boto 5 deposit and found them to be appropriate for resource estimation. RPA adopted the wireframes provided by IAMGOLD and used them to constrain the block model supporting the Boto 5 Mineral Resource estimate. The mineralized wireframes were used to select the resource samples and constrain the resource estimate. The weathering surfaces were used to define contacts between different oxidation state material and density flagging in the block model. Figure 14.7 shows the Boto 5 mineralized wireframes.

Figure 14.7 Mineralization Wireframes Boto 5



(RPA July 2019)

Artisanal small-scale mining is an on-going activity at the Boto 5 deposit and surrounding areas. The artisanal mining affects mainly the laterite cover, which is not considered for the resource estimate. In order to account for possible mined out material, RPA sterilized blocks in the proximity of the artisanal mining outline. The topography surface shows the lower elevation in the areas affected by mining. Blocks situated within 10 m below the topographical surface in the proximity of the artisanal mining were sterilized.

Weathering Profile

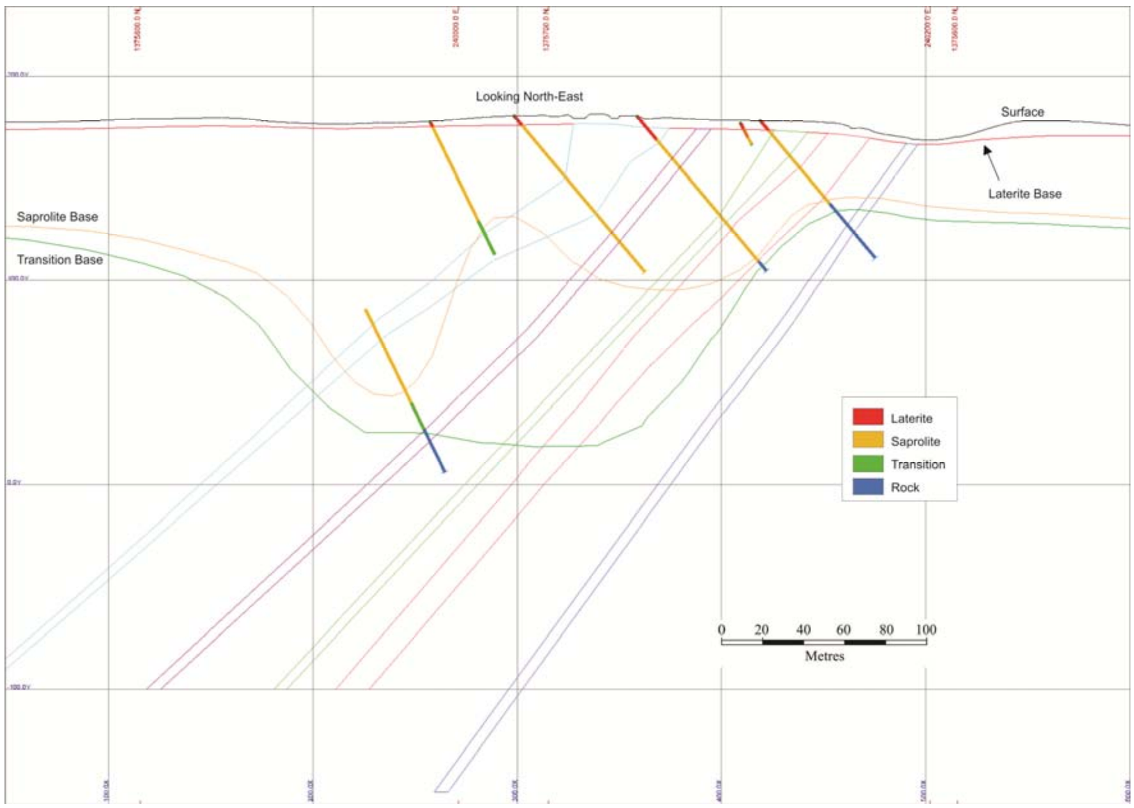
The weathering profile at Boto has been divided into four major units: laterite, saprolite, transition, and fresh rock. The upper unit, laterite, is considered to include transported and reworked material, hence the laterite cover and samples selected from this unit were not considered for resource estimation.

For the current estimate, the contact surfaces between weathering domains have been reviewed and adjusted by IAMGOLD. Visual, hardness, and geochemical information were used to define the contact surfaces. Table 14.13 presents the block model flagging codes for the weathering profile. Figure 14.8 shows the contact surfaces between weathering horizons.

Table 14.13 Weathering Profile Codes

Horizon	Code
Laterite	40
Saprolite	50
Transition	60
Rock	70

Figure 14.8 Section B5_1000N – Weathering Surfaces – Boto 5



(RPA July 2019)

14.3.2 Statistical Analysis

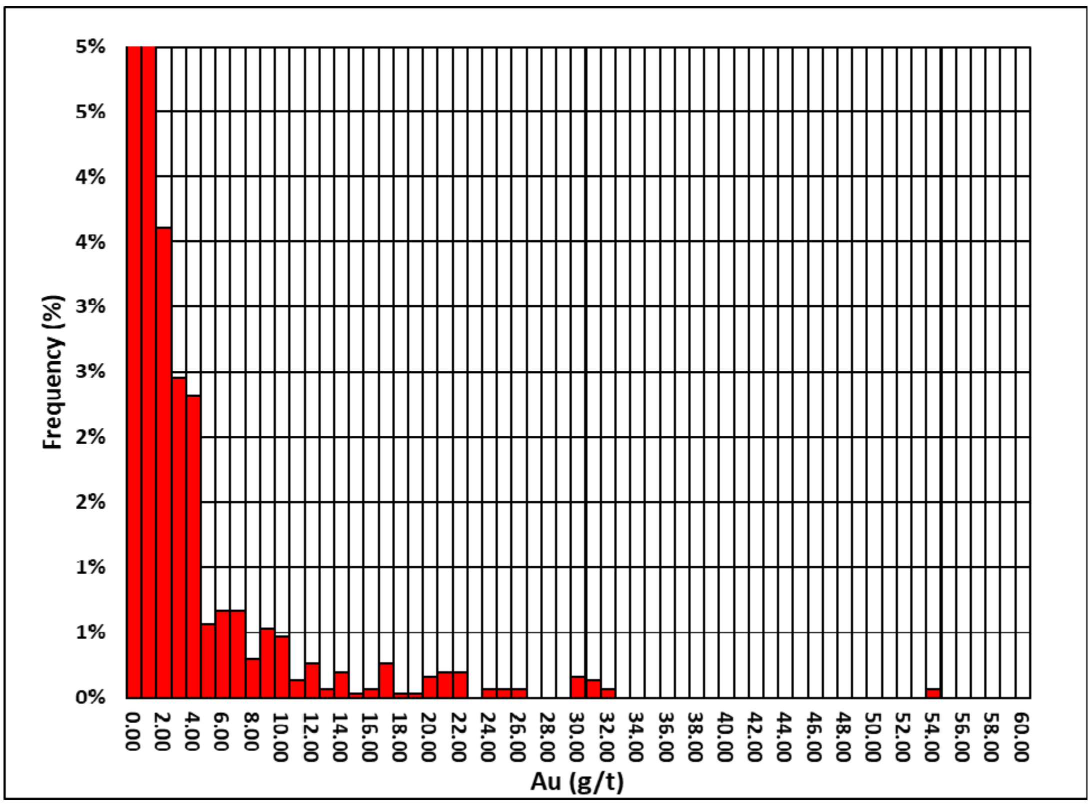
Assays

The Boto 5 mineralization solids were used to flag the resource samples. The samples retained inside the wireframes were the basis of the resource estimate, consisting of 1,484 samples, with a total sampled length of 1,509.0 m. Similar to the Malikoundi/Boto 2 deposit, the Boto 5 resource samples form a positively skewed population, characteristic for gold mineralization, with a relatively large number of low-grade samples and long tail of higher-grade samples. Sample length weighted descriptive statistics of the assays by mineralized lens are presented in Table 14.14. Figure 14.9 shows the resource assays histogram for all lenses.

Table 14.14 Descriptive Statistics for Boto 5 Deposit by Mineralized Lens

Lens	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Stdev (g/t Au)	Variance	CV
502	79	0.00	4.70	0.19	0.61	0.37	3.25
503	616	0.00	149.59	2.37	9.86	97.22	4.16
504	267	0.00	21.64	0.90	2.31	5.32	2.55
505	152	0.00	5.25	0.38	0.96	0.92	2.53
506	370	0.00	106.69	1.57	6.66	44.32	4.25
All Lenses	**1,484**	**0.00**	**149.59**	**1.57**	**7.22**	**52.07**	**4.59**
Sample Length	**1,484**	**0.50**	**2.00**	**1.02**	**0.17**	**0.03**	**0.17**

Figure 14.9 Assay Histogram – All Lenses – Boto 5



Capping Analysis

Capping of high grade assays prior to compositing is a practice aimed at limiting the influence of erratic high-grade assays, which otherwise have the potential to overpower surrounding lower grade samples. In the absence of production data that would allow the determination of appropriate capping levels, a number of statistical methods are used. RPA applied statistical methods to establish the capping levels for the Boto 5 estimation domains. A combination of histograms, decile analysis, probability plots, disintegration, and visual inspection of the spatial location of higher-grade assays was used to determine the capping levels for each mineralized lens. RPA capped high-grade assays prior to compositing. Table 14.15 shows the capping levels and number of values affected for each of the mineralized lenses.

Table 14.15 Capping Levels by Mineralized Lens – Boto 5

Lens	Capping Value (g/t Au)	Number of Caps	Capping Grade Percentile	Total Metal (g Au)	Metal Loss %	Average Grade (g/t Au)	CV
502	2	2	0.986	12	19	0.15	2.74
503	25	9	0.987	1,101	24	1.80	2.32
504	11	2	0.993	247	6	0.85	2.33
505	no capping	0	1.000	58	0	0.38	2.60
506	25	4	0.991	501	15	1.33	2.97

Samples with assay values higher than the lens capping threshold were capped at corresponding levels. Table 14.16 shows the descriptive statistics of the capped assays.

Table 14.16 Descriptive Statistics of Capped Assays – Boto 5

Lens	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Stdev (g/t Au)	Variance	CV
502	79	0.00	2.00	0.15	0.38	0.15	2.51
503	616	0.00	25.00	1.80	4.16	17.26	2.31
504	267	0.00	11.00	0.85	1.93	3.74	2.28
505	152	0.00	5.25	0.38	0.96	0.92	2.53
506	370	0.00	25.00	1.33	3.91	15.33	2.94

Composites

Samples were composited to intervals of equal length. RPA selected a compositing length of 2 m fixed intervals. Compositing was done from collar to toe within each mineralized lens, starting at the wireframe pierce-point and continuing to the point at which the hole exited the lens. Composites with at least 50% of compositing length were considered valid. No composites were discarded for Boto 5. Capped composites were used for resource estimation.

Table 14.17 and Table 14.18 show the descriptive statistics of the capped and uncapped 2 m composite values by mineralized lens.

Table 14.17 Descriptive Statistics of the Capped 2 m Composites – Boto 5

Lens	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Stdev (g/t Au)	Variance	CV
502	47	0.00	1.35	0.13	0.29	0.08	2.23
503	316	0.00	22.67	1.75	3.31	10.96	1.89
504	171	0.00	10.87	0.75	1.65	2.71	2.19
505	89	0.00	4.38	0.34	0.82	0.68	2.43
506	203	0.00	25.00	1.22	3.37	11.37	2.77
Boto5	826	0.00	25.00	1.17	2.81	7.91	2.41

Table 14.18 Descriptive Statistics of the Uncapped 2 m Composites– Boto 5

Lens	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Stdev (g/t Au)	Variance	CV
502	47	0.00	2.37	0.16	0.42	0.18	2.63
503	316	0.00	84.96	2.30	7.31	53.48	3.18
504	171	0.00	16.19	0.80	1.93	3.73	2.42
505	89	0.00	4.38	0.34	0.82	0.68	2.43
506	203	0.00	66.35	1.43	5.49	30.18	3.83
Boto5	826	0.00	84.96	1.44	5.41	29.25	3.75

14.3.3 Block Model

A block model was setup in GEOVIA GEMS 6.8 software to support the resource estimate. The block model for the Boto 5 deposit has a block size of 5 m wide by 5 m deep by 5 m high. The block model is rotated -28.5° (GEMS rotation convention). The block size is appropriate for the intended open pit operation planning and adequate for the 50 m by 50 m drill hole spacing available at Boto 5. Table 14.19 summarizes the block model parameters.

Table 14.19 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*

Grade Interpolation

The Boto 5 block model was interpolated in two passes. The gold grades were estimated using the 2 m composites with the ID^3 interpolation method (anisotropic). The ID^3 method was favoured in order to preserve local grades in the context of using mineralized wireframes with occasional internal dilution and with lower grade intercepts. Table 14.20 shows estimation parameters for each pass used to estimate gold grades.

Table 14.20 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	1	12	3	1

Search Ellipses

The search ellipses used at Boto 5 allow access to closes drilling in the first pass, reaching further for blocks interpolated in the second pass. The ranges were appropriate for the 50 m by 50 m drill spacing. Search ellipses were oriented along the interpreted mineralized lenses. Where necessary, lenses were subdivided to allow a better local fit of the search ellipse. Occasionally, the search ellipses were widened to accommodate local lens geometry. Table 14.21 lists the search ellipse parameters used to estimate the Boto 5 deposit.

Table 14.21 Search Ellipse Parameters for Pass 1 and Pass 2 for Boto 5

Profile Name	Search Rotation	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	20.0	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	30.0	Ellipsoidal

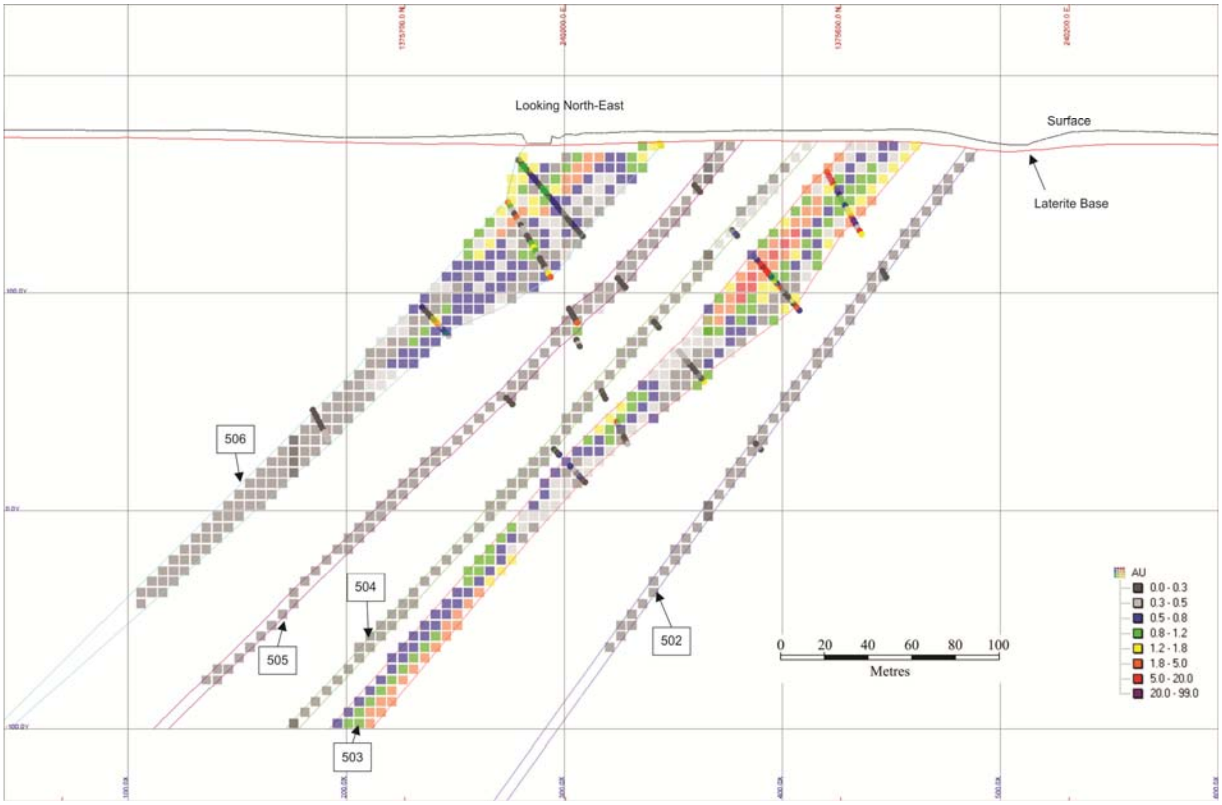
14.3.4 Block Model Validation

RPA used visual and statistical methods to validate the block model attributes flagging and interpolated block grades at Boto 5. The checks performed included:

- Comparison of mineralized lenses with the flagged blocks.
- Spot checks for search ellipse alignment along mineralized lenses.
- Spot checks for composite and block subdomain flagging.
- Visual check for grade banding, smearing of high grades, and high-grade plumes.
- Visual comparison of composite and block grade on section and plan view.
- Comparison of composite and block grades in swath plots.
- Comparison of interpolated block grades obtained by alternate interpolation methods.

Figure 14.10 presents a vertical section (B5_0950E) showing interpolated block gold grades and the gold grades of the 2 m composites. The grades of the blocks were in good agreement with the composite data used in the interpolation.

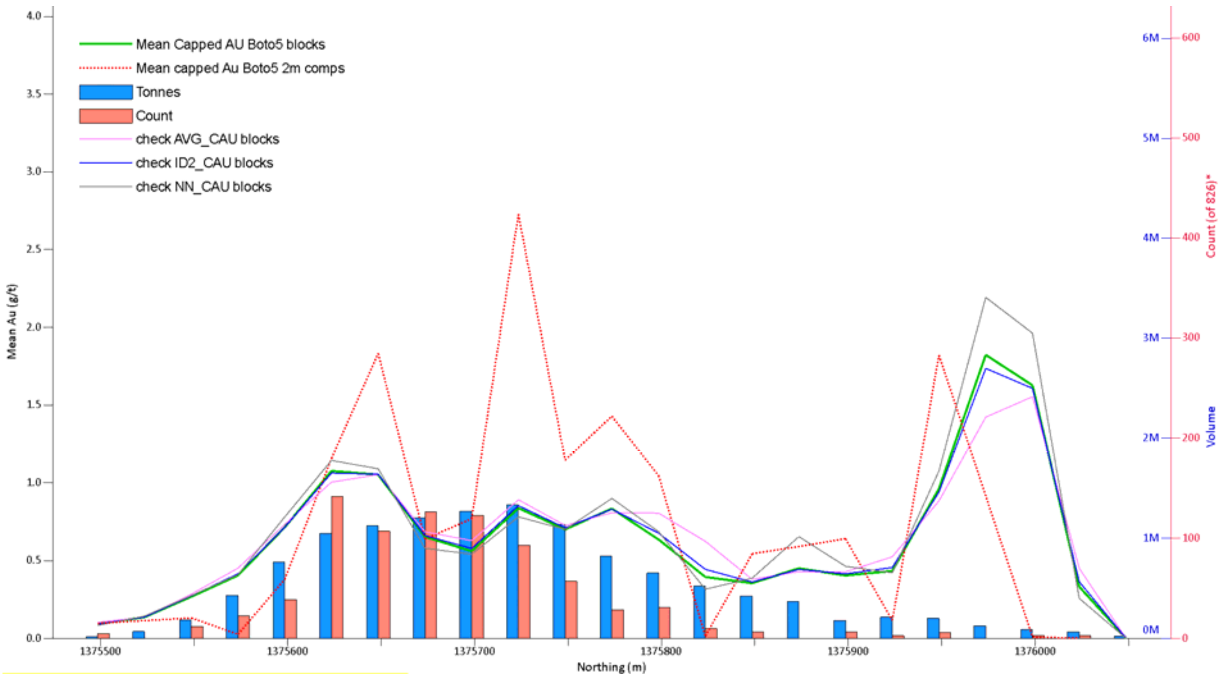
Figure 14.10 Section B5_0950E - Block Grades and 2 m Composites - Boto 5 Deposit



(RPA July 2019)

The alternate grade interpolation method check involves comparison with estimates obtained using different estimators. Along with the ID^3 estimator, RPA ran ID^2 and average grade of selected composites. Additionally, RPA generated alternate 6 m composites and ran a NN estimate for Boto 5. The data from these alternate runs was compared to the ID^3 estimate and the composite data. Figure 14.11 shows the swath plot by northing for Boto 5.

Figure 14.11 Swath Plots for Gold Grades by Northing - Boto 5



14.4 BOTO 6

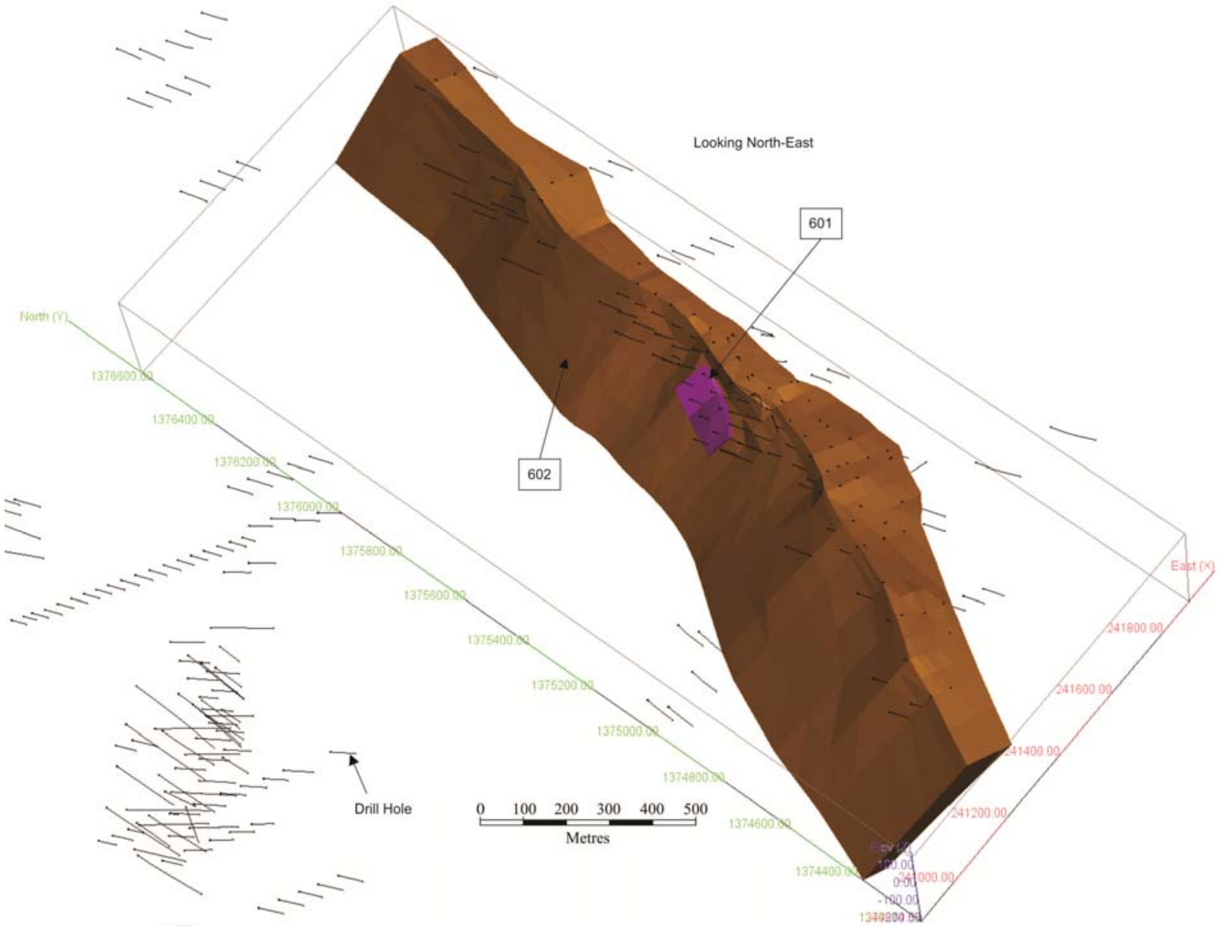
14.4.1 Geological Interpretation

Mineralization wireframes for the Boto 6 deposit were modelled by IAMGOLD geologists. Drilling completed until the end of May 2019 was used to update and refine the current resource solids. The wireframes were built from 3D rings interpreted on vertical sections spaced at 50 m intervals. The 3D rings, snapped to the beginning and end of sampled intervals down the drill hole, were then connected to create the mineralization solids.

The mineralization solids were defined based on a combination of a 0.15 g/t Au nominal cut-off grade, the presence of favourable lithology and higher intensity alteration, presence of sulphides, and intensity of fracturing. A minimum nominal thickness of 4 m was used throughout the modelling work. The average core length of the mineralized intercepts is approximately 92 m, while the average true thickness is approximately 72 m.

RPA reviewed the modelled mineralization solids, lithology and alteration wireframes, and weathering surfaces. RPA considered the wireframes provided by IAMGOLD a good representation of the mineralization present at Boto 6 deposit and found them to be appropriate for resource estimation. RPA adopted the wireframes provided by IAMGOLD and used them to constrain the block model supporting the Boto 6 Mineral Resource estimate. The mineralized wireframes were used to select the resource samples and constrain the resource estimate. The weathering surfaces were used to define contacts between different oxidation state material and density flagging in the block model. Figure 14.12 shows the Boto 6 mineralization wireframes.

Figure 14.12 Mineralization Wireframes - Boto 6



Weathering Profile

The weathering profile at Boto has been divided into four major units: laterite, saprolite, transition and fresh rock. The upper unit, laterite, is considered to include transported and reworked material, hence the laterite cover and samples selected from this unit were not considered for resource estimation.

For the current estimate, the contact surfaces between weathering domains have been reviewed and adjusted by IAMGOLD. Visual, hardness and geochemical information were used to define the contact surfaces. Table 14.22 presents the block model flagging codes for the weathering profile.

Table 14.22 Weathering Profile Codes

Horizon	Code
Laterite	40
Saprolite	50
Transition	60
Rock	70

14.4.2 Statistical Analysis

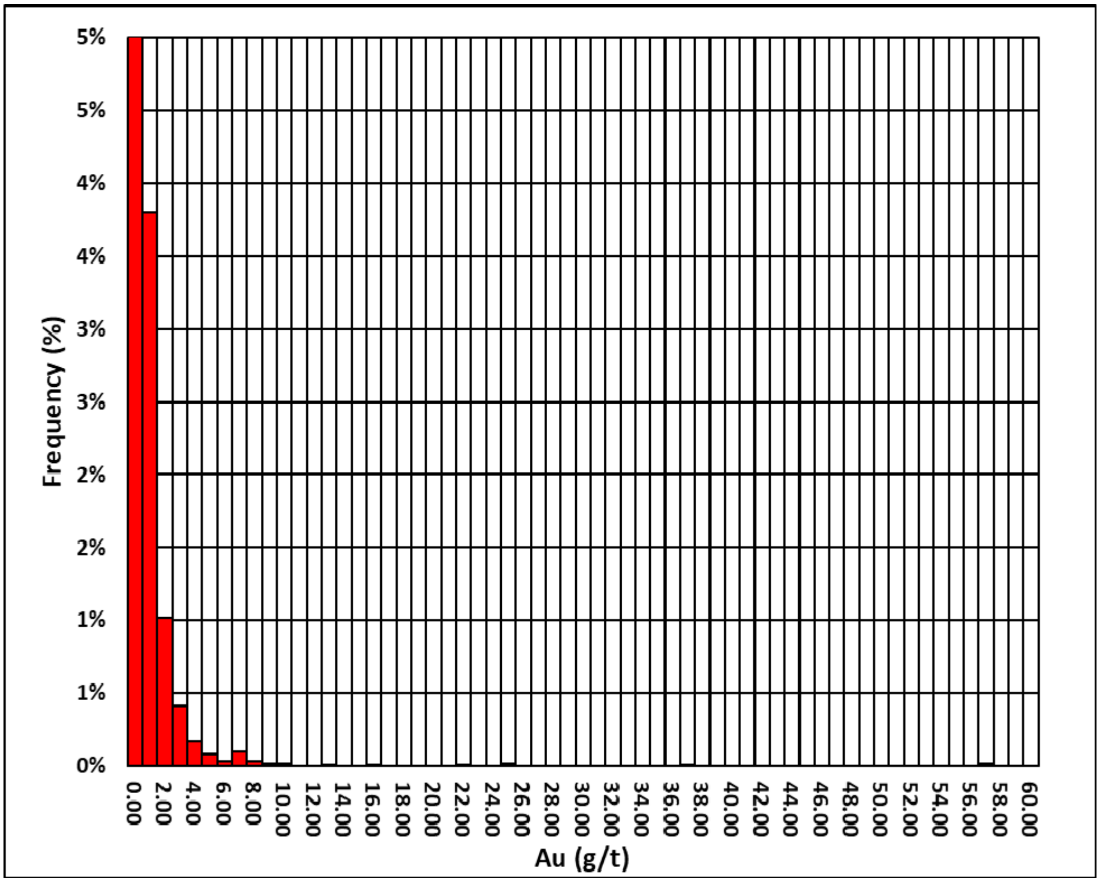
Assays

The Boto 6 mineralization solids were used to flag the resource samples. The samples retained inside the wireframes were the basis of the resource estimate, consisting of 10,307 samples, with a total sampled length of 11,687.0 m. Similar to the Malikoundi/Boto 2 and Boto 5 deposits, the Boto 6 resource samples form a positively skewed population, characteristic for gold mineralization, with a relatively large number of low-grade samples and long tail of higher-grade samples. Sample length weighted descriptive statistics of the assays by mineralized lens are presented in Table 14.23. Figure 14.12 shows the resource assays histogram for all lenses.

Table 14.23 Descriptive Statistics for Boto 6 Deposit by Mineralized Lens

Lens	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Stdev (g/t Au)	Variance	CV
601	44	0.00	2.97	0.47	0.78	0.60	1.64
602	10,263	0.00	62.38	0.31	1.24	1.54	3.95
All Lenses	**10,307**	**0.00**	**62.38**	**0.31**	**1.24**	**1.54**	**3.94**
Sample Length	**10,307**	**1.00**	**3.00**	**1.13**	**0.34**	**0.12**	**0.30**

Figure 14.13 Assay Histogram – All Lenses Boto 6



Capping Analysis

Capping of high grade assays prior to compositing is a practice aimed at limiting the influence of erratic high-grade assays, which otherwise have the potential to overpower surrounding lower grade samples. In the absence of production data that would allow the determination of appropriate capping levels, a number of statistical methods are used. RPA applied statistical methods to establish the capping levels for the Boto 6 estimation domains. A combination of histograms, decile analysis, probability plots, disintegration, and visual inspection of the spatial location of higher-grade assays was used to determine the capping levels for each mineralized lens. RPA capped high-grade assays prior to compositing. Table 14.24 shows the capping levels and number of values affected for each of the mineralized lenses.

Table 14.24 Capping Levels by Mineralized Lens – Boto 6

Lens	Capping Value (g/t Au)	Capped Assays	Capping Grade Percentile	Total Metal (g Au)	Metal Loss %	Average Grade (g/t Au)	CV
601	no capping	0	1.000	22	0%	0.47	1.67
602	11	7	0.999	3,435	6%	0.30	2.27

Samples with assay values higher than the lens capping threshold were capped at corresponding levels. Table 14.25 shows the descriptive statistics of the capped assays.

Table 14.25 Descriptive Statistics of Capped Assays - Boto 6

Lens	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Stdev (g/t Au)	Variance	CV
601	44	0.00	2.97	0.47	0.78	0.60	1.64
602	10,263	0.00	11.00	0.30	0.66	0.44	2.24

Composites

Samples were composited to intervals of equal length. RPA selected a compositing length of 2 m fixed intervals. Compositing was done from collar to toe within each mineralize lens, starting at the wireframe pierce-point and continuing to the point at which the hole exited the lens. Composites with at least 50% of compositing length were considered valid. Only two composites were discarded at Boto 6. Capped composites were used for resource estimation.

Table 14.26 and Table 14.27 show the descriptive statistics of the capped and uncapped 2 m composite values by mineralized lens.

Table 14.26 Descriptive Statistics of the Capped 2 m Composites – Boto 6

Lens	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Stdev (g/t Au)	Variance	CV
601	24	0.00	2.72	0.45	0.71	0.50	1.58
602	5,663	0.00	11.00	0.30	0.55	0.31	1.86
Boto6	5,687	0.00	11.00	0.30	0.55	0.31	1.86

Table 14.27 Descriptive Statistics of the Uncapped 2 m Composites – Boto 6

Lens	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	Stdev (g/t Au)	Variance	CV
601	24	0.00	2.72	0.45	0.71	0.50	1.58
602	5,663	0.00	57.00	0.31	1.05	1.11	3.35
Boto6	5,687	0.00	57.00	0.31	1.05	1.11	3.34

14.4.3 Block Model

A block model was setup in GEOVIA GEMS 6.8 software to support the resource estimate. The block model for the Boto 6 deposit has a block size of 5 m wide by 5 m deep by 5 m high. The block model is rotated 25° (GEMS rotation convention). The block size is appropriate for the intended open pit operation planning and adequate for the 50 m by 50 m drill hole spacing available at Boto 6. Table 14.28 summarizes the block model parameters.

Table 14.28 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*

Grade Interpolation

The Boto 6 block model was interpolated in two passes. The gold grades were estimated using the 2 m composites with the ID^3 interpolation method (anisotropic). The ID^3 method was favoured in order to preserve local grades in the context of using mineralized wireframes with occasional internal dilution and with lower grade intercepts. Table 14.29 shows estimation parameters for each pass used to estimate gold grades.

Table 14.29 Estimation Parameters for the Boto 6 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	1	12	3	1

Search Ellipses

The search ellipses used at Boto 6 allow access to closes drilling in the first pass, reaching further for blocks interpolated in the second pass. The ranges were appropriate for the 50 m by 50 m overall drill spacing. Search ellipses were oriented along the interpreted mineralized lenses. Where necessary, lenses were subdivided to allow a better local fit of the search ellipse. Occasionally the search ellipses were widened to accommodate local lens geometry. Table 14.30 lists the search ellipse parameters used to estimate the Boto 6 deposit.

Table 14.30 Search Ellipse Parameters for Pass 1 and Pass 2 for Boto 6

Profile Name	Search Rotation	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	107.5	-55	0	60.0	60.0	10.0	Ellipsoidal
B602S_P1	Az,Dip,Az	320	-45	38	60.0	60.0	10.0	Ellipsoidal
Pass 2								
B601_P2	Az,Dip,Az	295	-35	28	120.0	120.0	20.0	Ellipsoidal
B602N_P2	Az,Dip,Az	107.5	-55	0	120.0	120.0	20.0	Ellipsoidal
B602S_P2	Az,Dip,Az	320	-45	38	120.0	120.0	20.0	Ellipsoidal

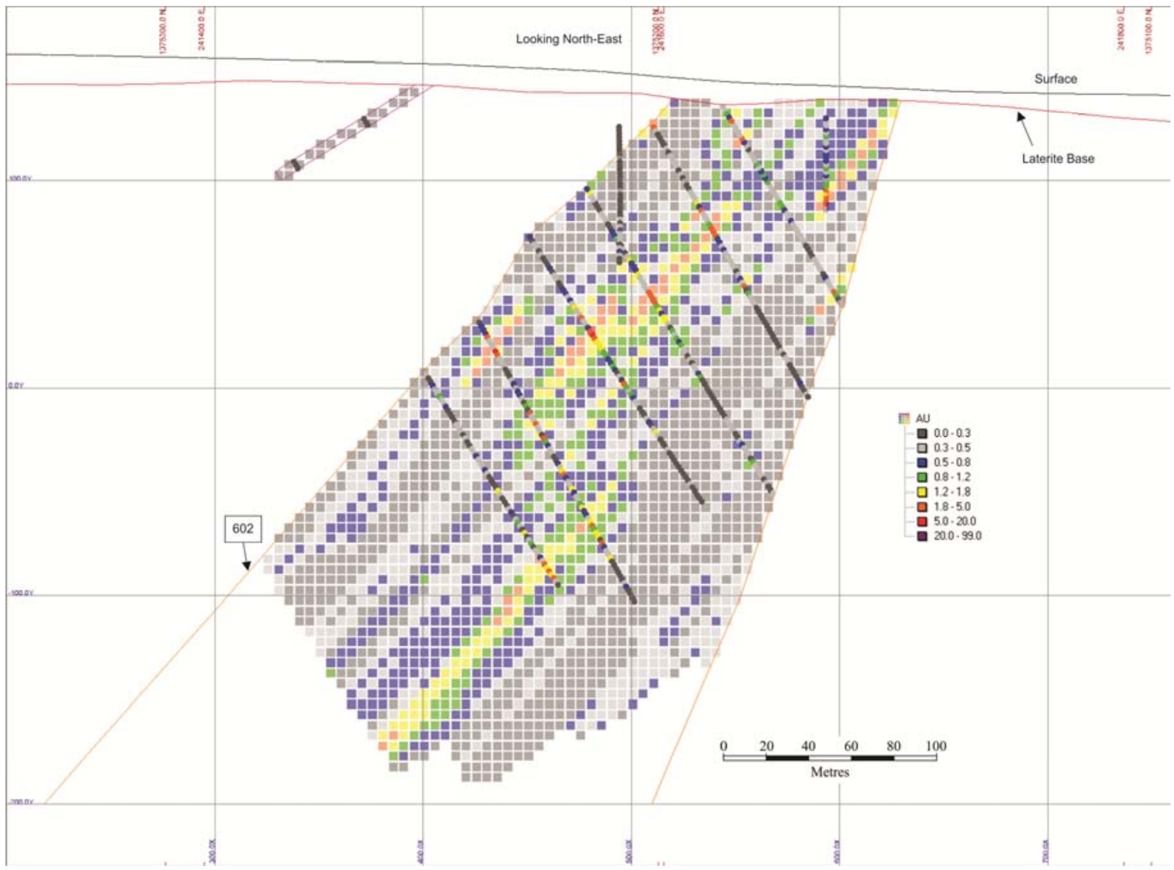
14.4.4 Block Model Validation

RPA used visual and statistical methods to validate the block model attributes flagging and interpolated block grades at Boto 6. The checks performed included:

- Comparison of mineralized lenses with the flagged blocks.
- Spot checks for search ellipse alignment along mineralized lenses.
- Spot checks for composite and block subdomain flagging.
- Visual check for grade banding, smearing of high grades, and high-grade plumes.
- Visual comparison of composite and block grade on section and plan view.
- Comparison of composite and block grades in swath plots.
- Comparison of interpolated block grades obtained by alternate interpolation methods.

Figure 14.14 presents a vertical section (E1645N) showing interpolated block gold grades and the gold grades of the 2 m composites. The grades of the blocks were in good agreement with the composite data used in the interpolation.

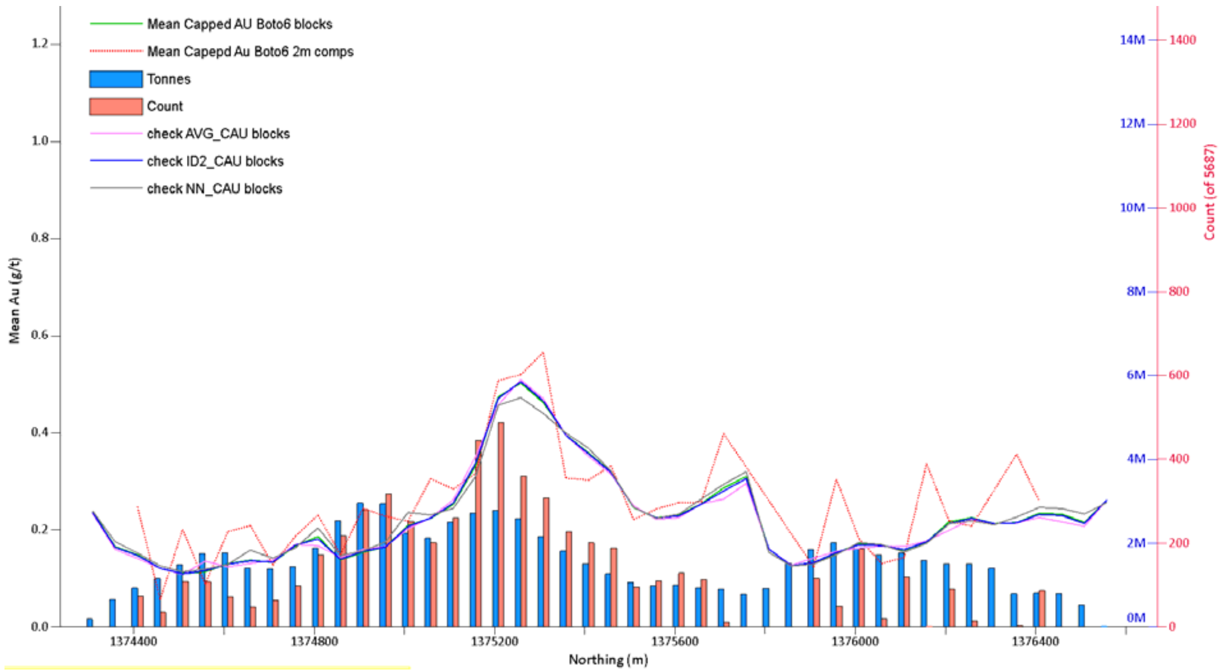
Figure 14.14 Section E1645N - Block Grades and 2 m Composites - Boto 6



(RPA July 2019)

The alternate grade interpolation method check involves comparison with estimates obtained using different estimators. Along with the ID^3 estimator, RPA ran ID^2 and average grade of selected composites. Additionally, RPA generated alternate 6 m composites and ran a NN estimate for Boto 6. The data from these alternate runs was compared to the ID^3 estimate and the composite data. Figure 14.15 shows the swath plot by northing for Boto 6.

Figure 14.15 Swath Plots for Gold Grades by Northing - Boto 6



14.5 MINERAL RESOURCES

14.5.1 Density

The updated drilling database had 4,512 density measurements with available weathering flagging. The weathering flagging was based on position of the measurement in relation to the weathering surfaces. As small pockets of any of the horizons can potentially be included in the domains above or below, occasional larger or lower values can be found in each of the horizons, without affecting the average value of the modelled domain. Approximately 75% of the density measurement data was collected from the Malikoundi/Boto2 deposit. Average density values vary slightly for each of the deposits. RPA decided to use the average density value by weathering horizon using all the data available for the Boto project. The average density values were assigned to blocks in the block model flagged used the weathering surfaces. Table 14.31 presents the descriptive statistics for density values by weathering horizon. Figure 14.16 presents the box plot for the density values by weathering horizon. A stacked histogram of density values for different horizons is shown in Figure 14.17.

Table 14.31 Density Values by Weathering Horizon

Domain	Count	Minimum (g/cm^3)	Maximum (g/cm^3)	Mean (g/cm^3)	StDev (g/cm^3)	Variance	CV
Fresh Rock	2,836	1	4.32	2.75	0.195	0.04	0.07
Laterite	489	1.22	2.57	2.00	0.249	0.06	0.12
Saprolite	657	1.09	2.81	1.65	0.216	0.05	0.13
Transition	530	1	2.95	2.14	0.345	0.12	0.16

Figure 14.16 Box Plots for Density Values by Weathering Horizon

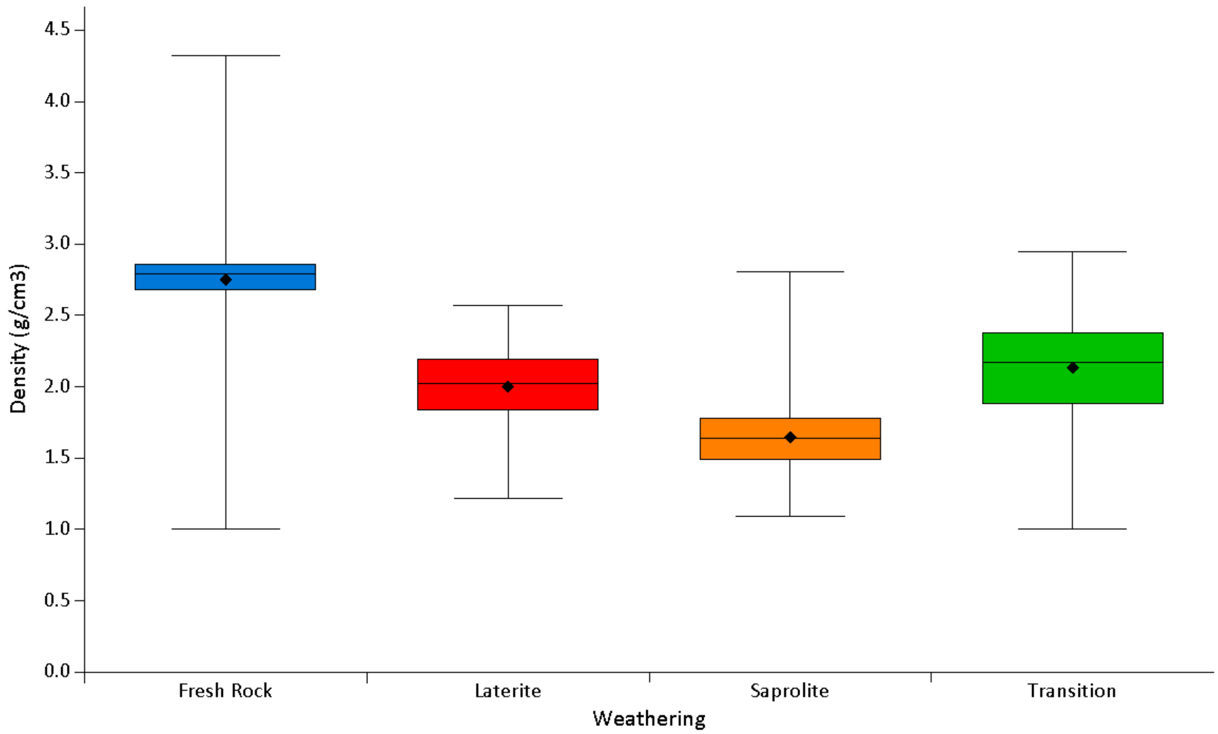
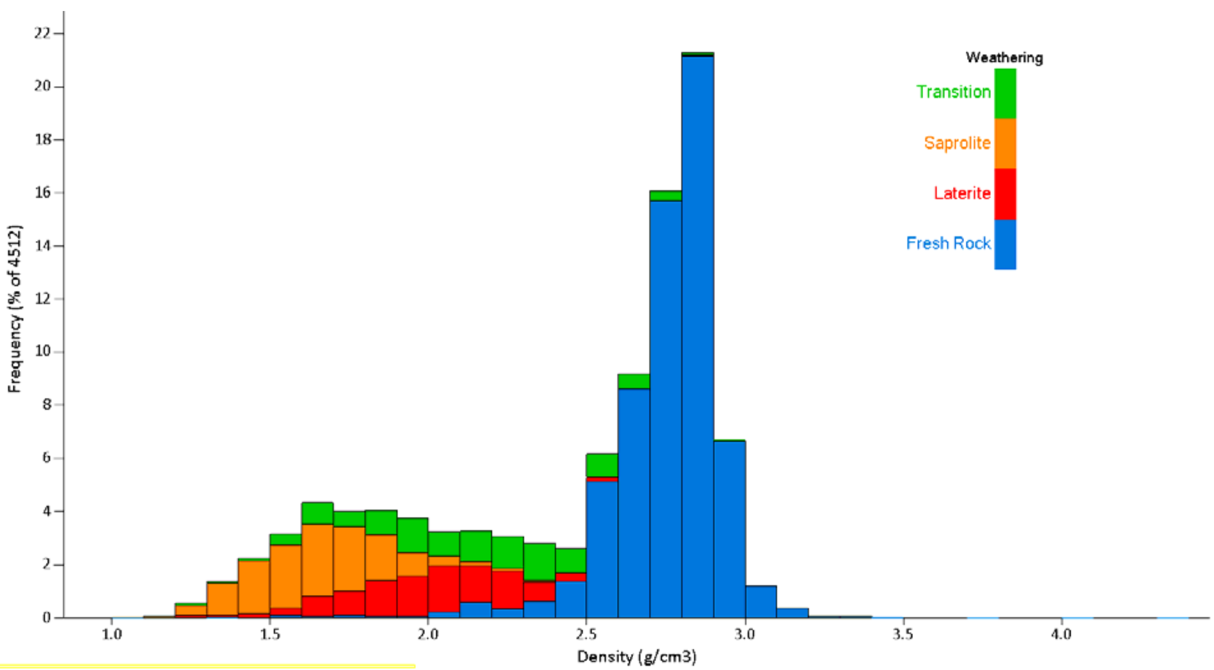


Figure 14.17 Stacked Histogram of Density Values by Weathering Horizon



14.5.2 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 constrained by resource open pit 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. For a 50 m by 50 m drilling area, Indicated blocks are expected to be within a maximum nominal distance of approximately 35 m away from the closest drill hole. Inferred Resources are classified as blocks estimated with a minimum of two drill holes, with a maximum nominal distance to the closest composite of 70 m.

The Malikoundi/Boto2 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 250 m exclusion zone was applied from the edges of these rivers as a protected zone. There are no Mineral Resources declared within the 250 m exclusion zone.

The classification process began with the identification of the blocks satisfying the minimum drill hole count and distance criteria. Blocks were then reclassified using manually drawn contours in order to clean isolated blocks or block clusters of different classification.

For Malikoundi/Boto 2, an additional step was used to refine the classification, using open pit shells as a guide. Isolated clusters of Inferred blocks situated within a preliminary reserve shell were reclassified as Indicated. Similarly, Indicated blocks situated below a preliminary resource shell based on more restrictive grade capping were reclassified as Inferred. Figure 14.18 and Figure 14.19 show the distribution of classified blocks by distance from closest hole and by northing.

Figure 14.18 Distance from Block to Closest Sample - Malikoundi/Boto 2

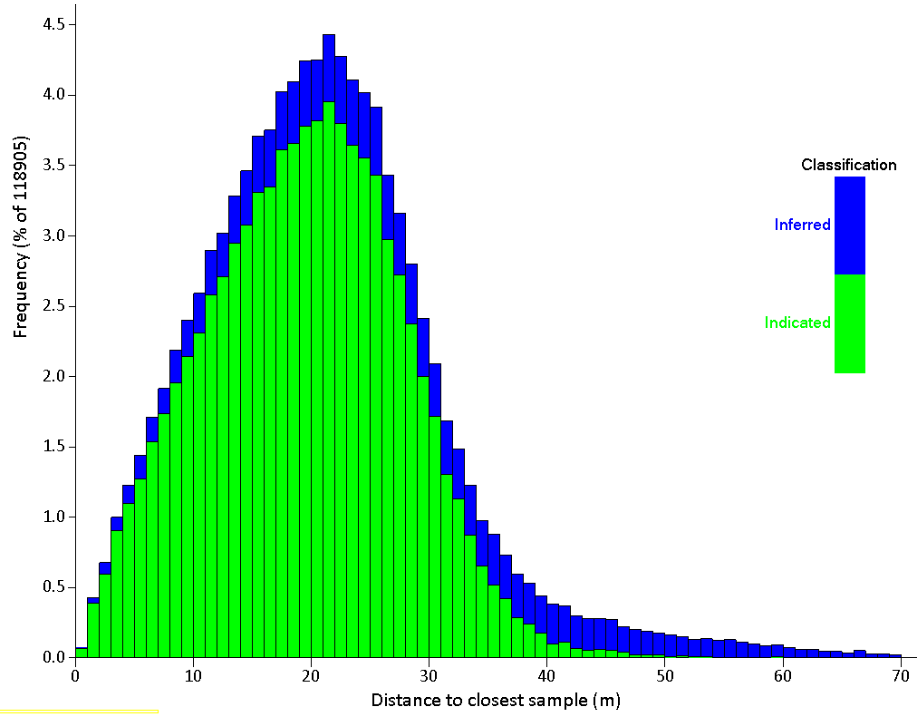


Figure 14.19 Distribution of Classified blocks - Malikoundi/Boto 2

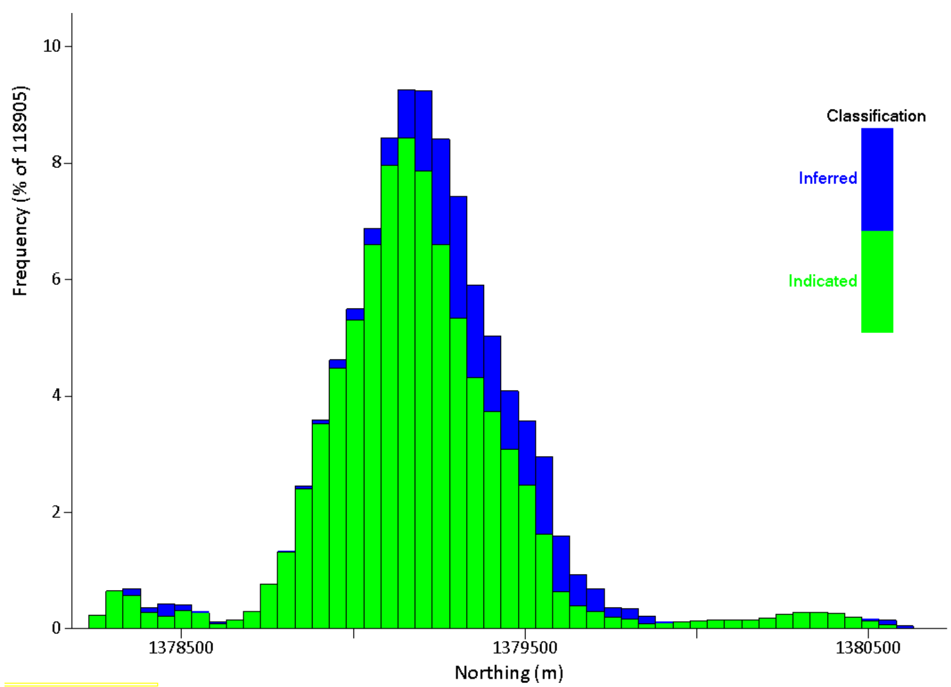


Figure 14.20 to Figure 14.22 show the classified blocks for Malikoundi/Boto 2, Boto 5, and Boto 6.

Figure 14.20 Distribution of Classified blocks – Malikoundi/Boto 2

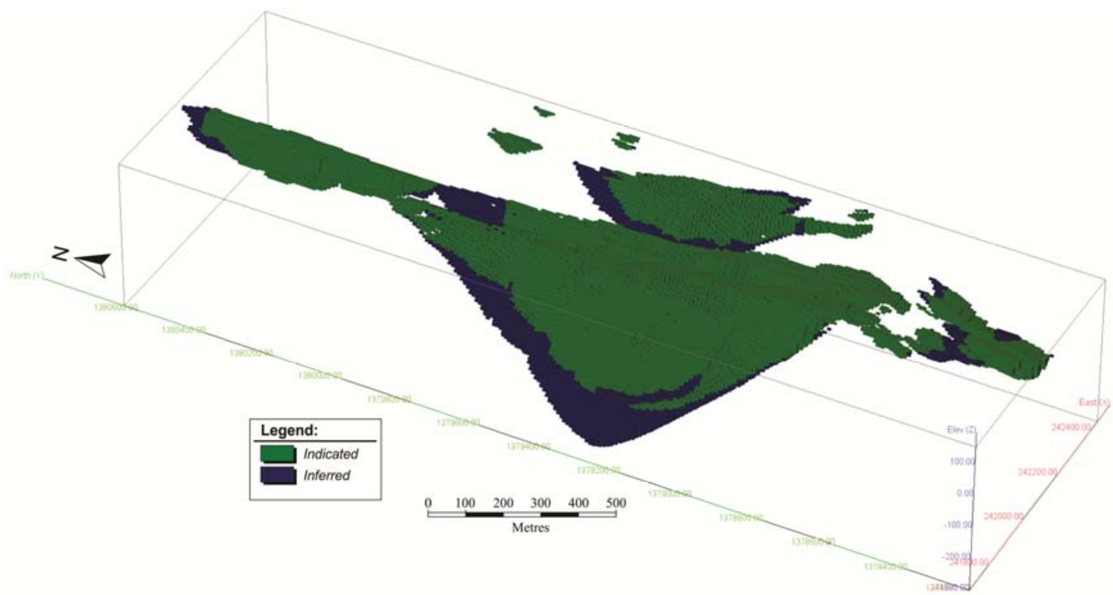


Figure 14.21 Distribution of Classified blocks Boto 5

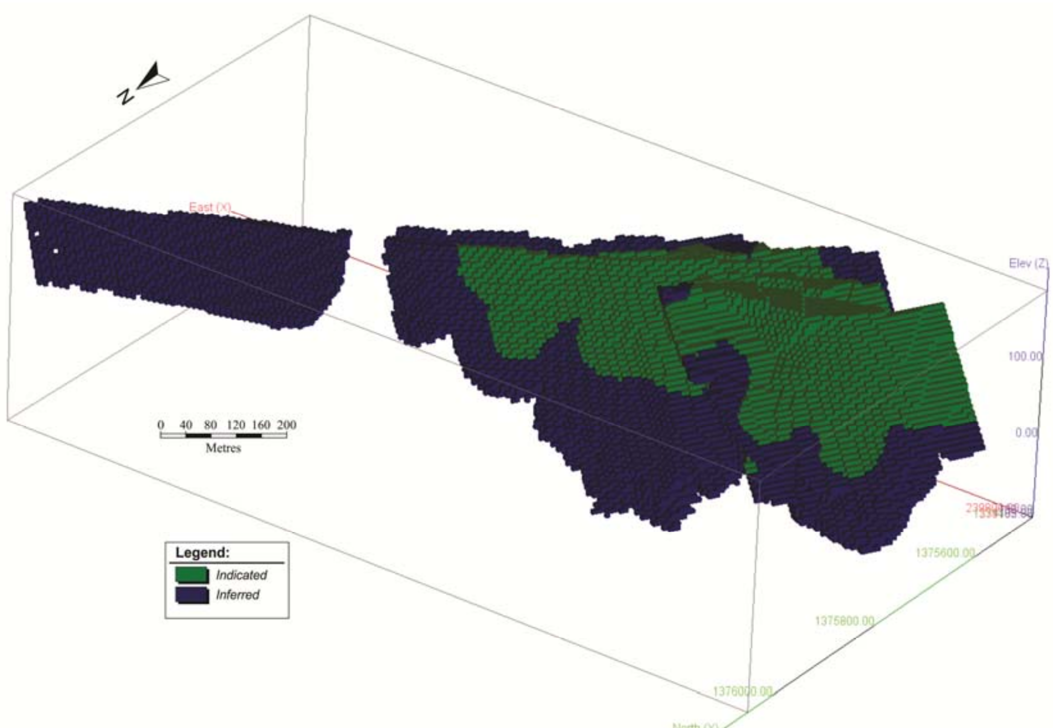
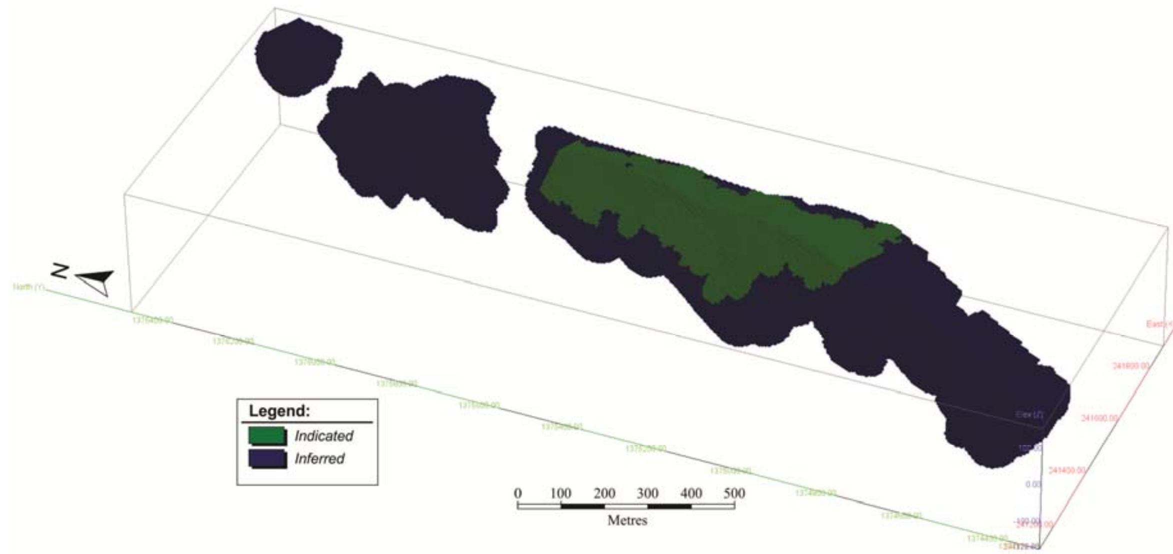


Figure 14.22 Distribution of Classified blocks - Boto 6



14.5.3 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 14.32 lists the parameters used for the optimized constraining shell by weathering zone.

Table 14.32 Parameters for the Optimized Constraining Shell for the Malikoundi/Boto2 Deposit

		Malikoundi/Boto2 Parameters			
Description	Units	Laterite [40]	Saprolite [50]	Transition [60]	Rock [70]
Gold Price	$US/oz	n/a	1,500	1,500	1,500
Mining Cost (Ore)	US$/ t milled	n/a	1.50	2.05	1.87
Mining Cost (Waste)	US$/ t milled	1.1	1.1	1.65	1.72
Incremental Mining Cost (Ore)	US$/ t milled	n/a	0.022	0.022	0.022
Incremental Mining Cost (Waste)	US$/ t milled	0.028	0.028	0.028	0.028
Mining Throughput	Mt/a		2.75	2.75	2.7
Processing Cost	US$/ t milled	n/a	10.82	11.28	15.61
Metallurgical Recovery	%		93	93	88.8
G&A	US$/ t milled	4.29	4.29	4.37	4.37
Slope Angle	°	31 - 33	32 - 33	36.5-39.2	44.9-55.3
Cut-off Grade	g/t Au	n/a	0.37	0.38	0.50

Note: n/a = not applicable

The cut-off grades established for the Malikoundi/Boto2 deposit by weathering zone are:

- Saprolite 0.37 g/t Au
- Transition 0.38 g/t Au
- Rock 0.50 g/t Au

Mineral Resources

The Mineral Resources are reported inclusive of Mineral Reserves. The mineral resources for the Malikoundi/Boto 2 deposit include an Indicated Resource of 35.5 Mt at 1.60 g/t Au and an Inferred Resource of 7.2 Mt at 1.88 g/t Au. Table 14.33 presents the Mineral Resources for the Malikoundi/Boto 2 deposit.

Table 14.33 Mineral Resources for the Malikoundi/Boto 2 Deposit – December 31, 2019

Classification	Tonnes ('000 t)	Grade (g/t Au)	Contained Gold ('000 oz Au)
Indicated	35,546	1.60	1,830
Inferred	7,192	1.88	435

Notes:

1. *The Mineral Resources are reported within an optimized constraining shell using a gold price of US$1,500/oz.*
2. *Mineral Resources are reported inclusive of Mineral Reserves.*
3. *Mineral Resources are classified in accordance with the CIM (2014) Standards and Definitions of Mineral Resources.*
4. *Cut-off grades vary from 0.37 g/t Au and 0.50 g/t Au depending on weathering zone.*
5. *Capping of grade outliers varies between 2 g/t Au and 25 g/t Au depending on interpreted mineralized zone and sub-domain.*
6. *The density varies between 1.65 g/cm^3 and 2.75 g/cm^3 depending on weathering zone.*

RPA is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate at the time of this report.

14.5.4 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 14.34 lists the parameters used for the optimized constraining shell by weathering zone.

Table 14.34 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	$US/oz	n/a	1,500	1,500	1,500
Mining Cost (Ore)	US$/ t milled	n/a	1.50	2.05	1.87
Mining Cost (Waste)	US$/ t milled	1.1	1.1	1.65	1.72
Incremental Mining Cost (Ore)	US$/ t milled	n/a	0.022	0.022	0.022
Incremental Mining Cost (Waste)	US$/ t milled	0.028	0.028	0.028	0.028
Slope Angle	°	31 - 33	32 - 33	36.5-39.2	44.9-55.3
Mining Throughput	Mt/a	n/a	2.75	2.75	2.7
Processing Cost	US$/ t milled	n/a	10.82	11.28	15.61
Metallurgical Recovery	%		93.4	93.4	89.8
G&A	US$/ t milled	4.37	4.37	4.37	4.37
Cut-off Grade	g/t Au	n/a	0.37	0.38	0.50

Note: n/a = not applicable

The cut-off grades established for the Boto 5 deposit by weathering zone are:

- Saprolite 0.37 g/t Au
- Transition 0.38 g/t Au
- Rock 0.50 g/t Au

Mineral Resources

The Mineral Resources are reported inclusive of Mineral Reserves. The Mineral Resources for the Boto 5 deposit include an Indicated Resource of 1.7 Mt at 2.00 g/t Au; and an Inferred Resource of 0.2 Mt at 1.27 g/t Au. Table 14.35 presents the Mineral Resources for the Boto 5 deposit.

Table 14.35 Mineral Resources for the Boto 5 Deposit – December 31, 2019

Classification	Tonnes ('000 t)	Grade (g/t Au)	Contained Gold ('000 oz Au)
Indicated	1,731	2.00	111
Inferred	235	1.27	10

Notes:

1. *The Mineral Resources are reported within an optimized constraining shell using a gold price of US$1,500/oz.*
2. *Mineral Resources are reported inclusive of Mineral Reserves.*
3. *Mineral Resources are classified in accordance with the CIM (2014) Standards and Definitions of mineral resources.*
4. *Cut-off grades used to report mineral resources vary from 0.37 g/t Au and 0.50 g/t Au depending on weathering zone.*
5. *Capping of grade outliers varies between 2 g/t Au and 25 g/t Au depending on interpreted mineralized zone and sub-domain.*
6. *The density varies between 1.65 g/cm^3 and 2.75 g/cm^3 depending on weathering zone.*

RPA is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate at the time of this report.

14.5.5 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 14.36 lists the parameters used for the optimized constraining shell by weathering zone.

Table 14.36 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	$US/oz	n/a	1,500	1,500	1,500
Mining Cost (Ore)	US$/ t milled	n/a	1.50	2.05	1.87
Mining Cost (Waste)	US$/ t milled	1.1	1.1	1.65	1.72
Incremental Mining Cost (Ore)	US$/ t milled	n/a	0.022	0.022	0.022
Incremental Mining Cost (Waste)	US$/ t milled	0.028	0.028	0.028	0.028
Slope Angle	°	31 - 33	32 - 33	36.5-39.2	44.9-55.3
Mining Throughput	Mt/a	n/a	2.75	2.75	2.7
Processing Cost	US$/ t milled	n/a	10.82	11.28	15.61
Metallurgical Recovery	%		93.4	93.4	89.8
G&A	US$/ t milled	4.37	4.37	4.37	4.37
Cut-off Grade	g/t Au	n/a	0.37	0.38	0.50

Note: n/a = not applicable

The cut-off grades established for the Boto 6 Deposit by weathering zone are:

- Saprolite 0.37 g/t Au
- Transition 0.38 g/t Au
- Rock 0.50 g/t Au

Mineral Resources

The Mineral Resources are reported inclusive of Mineral Reserves. The Mineral Resources for the Boto 6 deposit include an Indicated Resource of 3.3 Mt at 0.87 g/t Au; and an Inferred Resource of 0.8 Mt at 1.00 g/t Au. Table 14.37 presents the Mineral Resources for the Boto 6 deposit.

Table 14.37 Mineral Resources for the Boto 6 Deposit – December 31, 2019

Classification	Tonnes ('000 t)	Grade (g/t Au)	Contained Gold ('000 oz Au)
Indicated	3,290	0.87	92
Inferred	770	1.00	25

Notes:

1. *The Mineral Resources are reported within an optimized constraining shell using a gold price of US$1,500/oz.*
2. *Mineral Resources are reported inclusive of Mineral Reserves.*
3. *Mineral Resources are classified in accordance with the CIM (2014) Standards and Definitions of mineral resources*
4. *Cut-off grades vary from 0.37 g/t Au to 0.50 g/t Au depending on weathering zone.*
5. *Capping of grade outliers at 11 g/t Au.*
6. *The density varies between 1.65 g/cm^3 and 2.75 g/cm^3 depending on weathering zone*

RPA is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate at the time of this report.

14.5.6 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 250 m exclusion zone from the river and the situation of the village of Guémédji above the deposit. Should the 250 m offset limit change or be lifted, the block model and mineral resources for the Boto 4 deposit will be re-evaluated.

14.5.7 Mineral Resources – Summary

The following is a summary of the Mineral Resources for the Boto project with an effective date of December 31, 2019. The resources were estimated using Geovia GEMS 6.8 resource estimation software. Mineral resources are reported within optimized constraining shells using Hexagon Mining MineSight 3D software using a gold price of US$1,500/oz. Mineralized zones were modelled using 3D wireframes and gold grades were estimated within these zones using the ID^3 interpolation method. Cut-off grades vary between 0.37 g/t Au and 0.50 g/t Au, and densities vary between 1.65 g/cm³ and 2.75 g/cm³, depending on weathering 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 14.38 presents the Mineral Resources for the Boto Project.

Table 14.38 Mineral Resources for the Boto Project – December 31, 2019

Zone	Classification	Tonnes ('000 t)	Grade (g/t Au)	Contained Gold ('000 oz)
Malikoundi/Boto 2	Indicated	35,546	1.60	1,830
	Inferred	7,192	1.88	435
Boto 5	Indicated	1,731	2.00	111
	Inferred	235	1.27	10
Boto 6	Indicated	3,290	0.87	92
	Inferred	770	1.00	25
Boto 4	Indicated	-	-	-
	Inferred	-	-	-
Total	**Indicated**	40,567	1.56	2,033
	Inferred	8,196	1.78	469

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 0.50 g/t Au depending on weathering zone.*
6. *Capping of grade outliers varies between 2 g/t Au and 25 g/t Au depending on interpreted mineralized zone and sub-domain.*
7. *The density varies between 1.65 g/cm^3 and 2.75 g/cm^3 depending on weathering zone.*

RPA is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate at the time of this report.

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. These grade-tonnage curves are not to be taken as the reported Mineral Resources and serve only to show the sensitivity of the block model to selected cut-off grades.

Figure 14.23 and Figure 14.24 show the grade tonnage curve for the Indicated Resources and the Inferred Resources within constraining shells, respectively, at various cut-off grades for the Boto project.

Figure 14.23 Boto Project Grade-Tonnage Plot - Indicated Resources within Constraining Shells

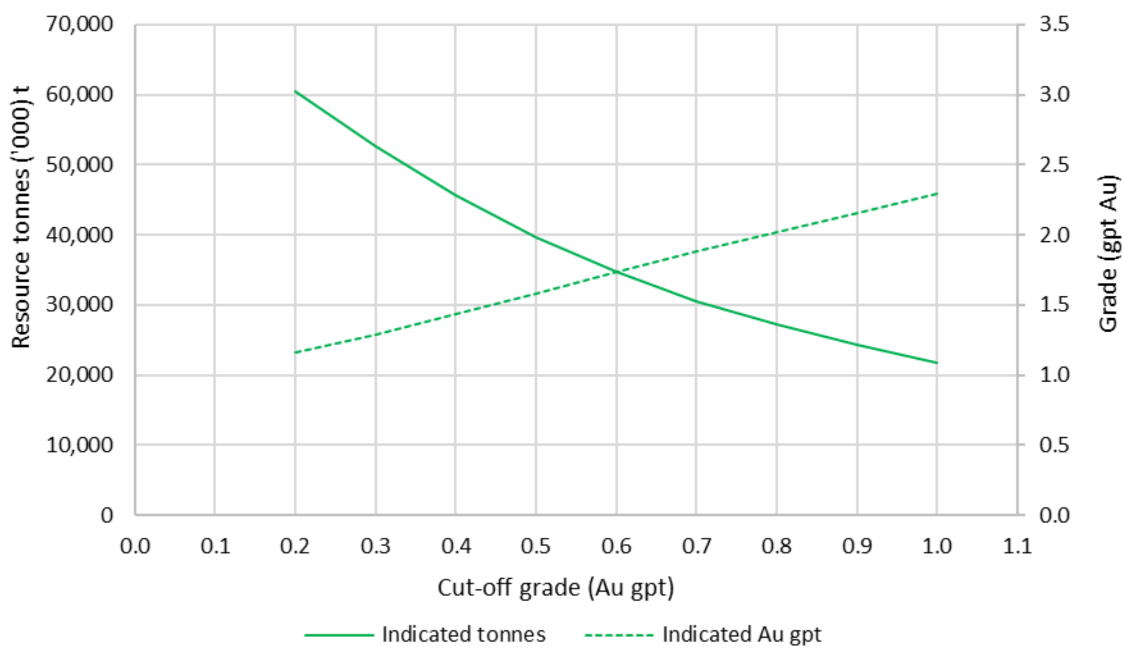
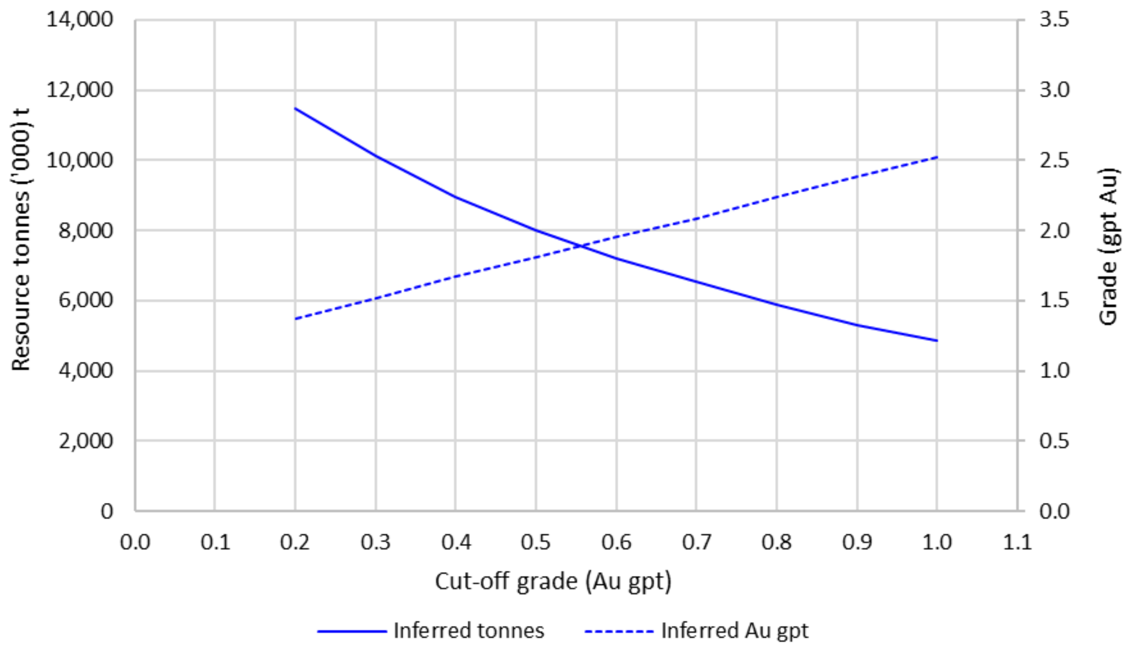


Figure 14.24 Boto Project Grade-Tonnage Plot - Inferred Resources within Constraining Shells



14.6 PREVIOUS MINERAL RESOURCES

14.6.1 Previous Mineral Resources

Since the initial Mineral Resources were reported on the Boto project, there has been an overall increase in resource tonnes and contained metal. The current mineral resource estimate represents a refinement of the previous estimate, benefiting from additional drilling, updated and revised mineralization wireframe interpretations, updated weathering surfaces, slight change of density values, revised classification, updated open pit parameters, and adjusted reporting cut-off grades. There has been a minor decrease in both overall tonnes and contained metal. For each of the deposits the ratio of Inferred to Indicated resources increased, with Inferred resources representing now 17% at Malikoundi/Boto 2, 12% at Boto 5 and 19% at Boto 6 of the total resources. For Malikoundi/Boto 2, classification in deeper parts of the deposit was downgraded due to the presence of a deep high-grade intercept that was observed to have a large influence over the depth of the resource shell. Additional drilling around the area will help in upgrading the classification for future estimates.

Table 14.39 summarizes the mineral resources from 2013 to present.

Table 14.39 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
December 2019	40.6	1.56	2.03	8.2	1.78	0.47

14.6.2 Malikoundi/Boto 2

Compared to the previous May 2018 Mineral Resource Estimate, the Malikoundi/Boto2 Mineral Resources show an overall modest decrease in resource tonnages and contained metal. For the current estimate, there were more drill holes available in both previously modelled areas and for a new target to the east leading to adjusted and new mineralized zones, adjusted weathering surfaces and classification approach, updated pit optimization parameters, and minor changes to the cut-off grades. Most notable, resources from deeper parts of the deposit related to a high-grade intercept were downgraded to Inferred category.

Table 14.40 presents the comparison of the May 2018 Mineral Resources to the current Mineral Resources for the Malikoundi/Boto 2 deposit.

Table 14.40 Comparison of Mineral Resources for Malikoundi/Boto 2 - May 2018 vs December 2019

Classification	Malikoundi/Boto 2 (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)			Malikoundi/Boto 2 (December 2019) in updated 2019 Resource Shell COG > 0.37 g/t Au (Saprolite) and > 0.38 g/t Au (Transition) and > 0.50 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	41,915	1.66	2,240	35,546	1.60	1,830	-15%	-4%	-18%
Inferred	1,974	2.00	127	7,192	1.88	435	264%	-6%	243%

14.6.3 Boto 5

Compared to the previous May 2018 Mineral Resource Estimate, the Boto 5 Mineral Resources show a decrease in resource tonnages and contained metal. This is due to minor adjustments to the mineralized zones, one additional infill drill hole, adjusted weathering surfaces and classification approach, updated pit optimization parameters and minor changes to the cut-off grades.

Table 14.41 presents the comparison of the May 2018 Mineral Resources to the current Mineral Resources for Boto 5 deposit.

Table 14.41 Comparison of Mineral Resources for Boto 5 - May 2018 vs December 2019

Classification	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)			Boto 5 (December 2019) in updated 2019 Resource Shell COG > 0.37 g/t Au (Saprolite) and > 0.38 g/t Au (Transition) and > 0.50 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	2,469	1.86	148	1,731	2.00	111	-30%	7%	-25%
Inferred	34	0.75	1	235	1.27	10	585%	69%	1058%

14.6.4 Boto 6

Compared to the previous May 2018 Mineral Resource Estimate, the Boto 6 Mineral Resources show a minimal reduction overall, with a minimal increase in contained ounces. Indicated tonnage and ounces decreased by 10% and 7%, respectively, with the same number of tonnes and ounces now being added to the Inferred category. This is due to additional infill drill holes, adjusted weathering surfaces and classification approach, updated pit optimization parameters, and minor changes to the cut-off grades.

Table 14.42 presents the comparison of the May 2018 Mineral Resources to the current Mineral Resources for Boto 6 deposit.

Table 14.42 Comparison of Mineral Resources for Boto 6 - May 2018 vs December 2019

Classification	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)			Boto 6 (December 2019) in updated 2019 Resource Shell COG > 0.37 g/t Au (Saprolite) and > 0.38 g/t Au (Transition) and > 0.50 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	3,661	0.84	99	3,290	0.87	92	-10%	3%	-7%
Inferred	475	1.06	16	770	1.00	25	62%	-5%	53%

15.0 MINERAL RESERVE ESTIMATES

15.1 Introduction

The mineral reserve estimate is consistent with the CIM Definition Standards for Mineral Resources and Mineral Reserves. The reserves for the Project are based on the conversion of the Indicated resources to Probable reserves within the current Technical Report. No Measured resources are currently part of the model. The mineral reserve estimate for the Boto Gold Project deposits is based on the resource block model estimated by RPA and with effective date December 31, 2019.

The reserves are based on the Malikoundi deposit, including the Malikoundi and Malikoundi North pits, and the Boto 5 deposit.

15.2 Pit Optimization

A series of nested shells were generated for a range of revenues from 600$/oz to 1,500$/oz. In addition to these pit optimizations, COMET Strategy's Optimal Scheduler software was used to generate an optimal mine plan. This optimal mine plan was considered in order to select the pit optimization shells. Finally, the $1,150/oz gold price shell for the Malikoundi deposit and $1,200/oz Au price shell for the Boto 5 deposit were selected.

Optimized pit shells were generated using the pseudoflow algorithm in Geovia's Whittle strategic mine planning software. The pseudoflow algorithm generates the same results as the Lerchs-Grossman algorithm, however produces the results much faster. The parameters for the pit optimizations are shown in Table 15.1 and Table 15.2.

Table 15.1 Pit Optimization Parameters

Parameter	Units	Malikoundi Deposit	Boto 5 Deposit
Metal Prices			
Gold Price	$/oz	1200	1200
Payable	%	99%	99%
Participation (on profits)	%	100%	100%
Transportation & Refining	$/oz	3.04	3.04
Royalty & Other Charges	%	4.0%	4.0%
General			
Mill throughput	Mtpa	2.75	2.75
Resources blocks used		M+I	M+I
General & Administration Cost – Saprolite	$/t milled	4.29	4.37
General & Administration Cost – Transition	$/t milled	4.37	4.37
General & Administration Cost – Fresh Rock	$/t milled	4.37	4.37
Process Recovery			
Laterite	%	0.0%	0.0%
Saprolite	%	93.0%	93.4%
Transition	%	93.0%	93.4%
Fresh Rock	%	88.8%	89.8%
Process Costs			
Laterite Process Cost	$/t milled	-	-
Saprolite Process Cost	$/t milled	10.82	10.82
Transition Process Cost	$/t milled	11.28	11.28
Fresh Rock Process Costs	$/t milled	15.61	15.61
Mining Costs	Ref. elev. = 170		
Incremental haul cost - waste	$/5m bench	0.028	0.028
Incremental haul cost - ore	$/5m bench	0.022	0.022
Waste			
Laterite	$/t mined	1.10	1.10
Saprolite	$/t mined	1.10	1.10
Transition	$/t mined	1.65	1.65
Fresh Rock	$/t mined	1.72	1.72
Ore			
Laterite	$/t mined	1.11	1.11
Saprolite	$/t mined	1.50	1.50
Transition	$/t mined	2.05	2.05
Fresh Rock	$/t mined	1.87	1.87

A geotechnical study was completed on the Malikoundi and Boto 5 deposits. This work was completed by Absolute Geotechnics Pty LTD (AG). The slope domains were coded to the mining block model with their corresponding overall slope angles. The pit slope angles used for the optimization process are presented in Table 15.2.

When the mine cases mining, there will be a significant amount of low-grade stock pile near the primary crusher. During this rehandling operation, the G&A WILL BE $2.62/Ton.

Table 15.2 Pit Optimization Slope Angles

Rock Domains Horizon	Sector	Slope Domain (GEOT)	Overall Slope Angle (degrees)
Malikoundi Pit			
Saprolite/Laterite	HW	11	33
Saprolite/Laterite	FW	12	33
Transition	HW	13	39.2
Transition	FW	14	36.5
Fresh	HW (A, B, and C) + South wall	15	44.3
Fresh	FW A +FW B	16	37.3
Fresh	FW C	17	42.3
Malikoundi North Pit			
Saprolite/Laterite	HW	1	33
Saprolite/Laterite	FW	2	33
Transition	HW	3	39.2
Transition	FW	4	36.5
Fresh	HW	5	55.3
Fresh	FW	6	25.8
Boto 5 Pit			
Saprolite/Laterite	FW	1	30.8
Transition	FW	2	23.6
Saprolite/Laterite	Southwest Wall	3	30.8
Transition	Southwest Wall	4	27.4
Saprolite/Laterite	HW West	5	21.4
Transition	HW West	6	30.5
Saprolite/Laterite	HW East	7	23.4
Transition	HW East	8	36.5
Fresh	All	9	41.6

The detailed ultimate pit designs utilized the pit shells developed. The geotechnical parameters used were per the geotechnical recommendations. For the Malikoundi deposit, two pits were designed: Malikoundi and Malikoundi North. For the Boto 5 deposit, a single pit was designed: Boto 5.

15.2.1 Cut-off Grade

For determining the tonnes and grade in the pit, cut-off grades were varied by pit area and weathering type. The cut-off grades are shown in Table 15.3.

Table 15.3 Cut-off Grades

Weathered Material	Malikoundi (g/t) Au	Boto 5 (g/t) Au
Laterite	n/a	n/a
Saprolite	0.42	0.41
Transition	0.43	0.43
Fresh	0.58	0.58

15.2.2 Dilution and Ore Loss

The geologic block models developed for the optimization study were whole block fully diluted models. Additional contact dilution was integrated in the mining block model to better reflect expected results with mining practices.

Preliminary analyses of contact dilution were estimated using the following steps:

- For the Malikoundi deposit, the percentage of dilution was calculated for each contact side assuming 0.5 m contact dilution distance. If one side of an ore cell was adjacent to a waste cell, it was estimated that a dilution of 10% would result. If two, three and four sides were adjacent, it would rise to 15%, 20% and 30% respectively. The grades of the adjacent waste cells were considered.

- For the Boto 5 deposit, the percentage of dilution was calculated for each contact side assuming 0.5 m contact dilution distance also. If one side of an ore cell was adjacent to a waste cell, it was estimated that a dilution of 10% would result. If two, three and four sides were adjacent, it would rise to 20%, 30% and 40% respectively. The grades of the adjacent waste cells were considered.

Following these preliminary analyses, average dilution percentages were determined and applied to the pit optimization block models. The average dilution percentages are presented in Table 15.4.

Table 15.4 Average Dilution

Rock Type	Dilution (%)
Saprolite	6.0%
Transition	6.0%
Fresh Rock	5.5%

The dilution grade was assumed to be 0.0 g/t. The tonnes and grades for the pit designs are reported with the diluted tonnes and grades. Consequently, some cells that had in-situ grades above their respective cut-off grade may have diluted grades lower than their corresponding cut-off grade. Thus, these cells were considered as waste. No other ore loss was considered.

15.2.3 Mineral Reserves Statement

The reserves for the Project are based on the conversion of the Indicated resources to Probable reserves within the current Technical Report, which demonstrates their economic viability. No Measured resources are present in the current models. Indicated resources are converted directly to Probable Reserves.

Subsequent to the 2018 Feasibility Study Mineral Reserve estimates declaration, an update was made to the mining block model. The update of the reserve block model resulted in a modest reserve decrease compared to the 2018 FS.

Table 15.5 Boto Gold Project Mineral Reserves – December 31, 2019

Classification	Tonnes (000)	Grade (g/t Au)	Contained Ounces (000)	Attributable Contained Ounces (000)
Probable	29,040	1.71	1,592	1,432
Waste within Designed Pit	218,300			
Total Tonnage within Designed Pit	247,340			

Table 15.6 Proven and Probable Reserves by Pit Area

	Proven			Probable			Total		
Pit Area (Cut-off g/t)	**Tonnes (kt)**	**Grade (g/t)**	**Au (oz)**	**Tonnes (kt)**	**Grade (g/t)**	**Au (oz)**	**Tonnes (kt)**	**Grade (g/t)**	**Au (oz)**
Malikoundi									
Saprolite	-	-	-	916	1.49	43,774	916	1.49	43,774
Transition	-	-	-	986	1.92	60,980	986	1.92	60,980
Fresh Rock	-	-	-	24,777	1.67	1,326,687	24,777	1.67	1,326,687
Total Malikoundi	-	-	-	26,680	1.67	1,431,440	26,680	1.67	1,431,440
Malikoundi North									
Saprolite	-	-	-	526	1.97	33,410	526	1.97	33,410
Transition	-	-	-	156	1.85	9,301	156	1.85	9,301
Fresh Rock	-	-	-	331	2.31	24,566	331	2.31	24,566
Total Malikoundi North	-	-	-	1,014	2.06	67,277	1,014	2.06	67,277
Boto 5									
Saprolite	-	-	-	890	2.08	59,480	890	2.08	59,480
Transition	-	-	-	431	2.29	31,743	431	2.29	31,743
Fresh Rock	-	-	-	26	3.13	2,647	26	3.13	2,647
Total Boto 5	-	-	-	1,347	2.17	93,870	1,347	2.17	93,870
Boto Gold Project									
Saprolite	-	-	-	2,332	1.82	136,664	2,332	1.82	136,664
Transition	-	-	-	1,574	2.02	102,023	1,574	2.02	102,023
Fresh Rock	-	-	-	25,134	1.68	1,353,900	25,134	1.68	1,353,900
Total Boto Gold Project	-	-	-	29,040	1.71	1,592,587	29,040	1.71	1,592,587

1. *Reserves estimated assuming open pit mining methods.*
2. *Mineral Reserves are estimated using a long-term gold price of 1,200 US$/oz.*
3. *Average weighted process recovery of 89.4%.*
4. *Quantity of gold payable is 99%.*
5. *Transportation and refining costs estimated at 3.04 US$/oz.*
6. *Royalty and other charges of 4.0% are applied to the gold metal value.*
7. *Processing costs estimated at 10.82, 11.28 and 15.61 $/t milled for saprolite, transition and fresh rock material respectively.*
8. *G&A costs estimated at 4.29 $/t milled for saprolite for the Malikoundi deposit and 4.29 $/t milled for all other material at the Malikoundi deposit and for the Boto 5 deposit.*
9. *The cut-off grades for the Malikoundi deposit are 0.42 g/t for saprolite, 0.43 g/t for transition rock and 0.58 g/t for hard rock.*
10. *The cut-off grades for the Boto 5 deposit are 0.41 g/t for saprolite, 0.43 g/t for transition rock and 0.58 g/t for hard rock.*
11. *The tonnes and grades are diluted.*
12. *Numbers may not add due to rounding.*

16.0 MINING METHODS

16.1 Introduction

This mine plan is based on the pit optimizations described in Section 15. With current metal pricing levels and knowledge of the mineralization, open pit mining offers the most reasonable approach for development. The potential for underground development beneath the open pit has not been examined as part of this technical report.

The Project is located to the west of the Falémé River, which also represents a border with the Republic of 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. No mining has been conducted on the Malikoundi part of the project, but artisanal mining is ongoing at Boto 5.

16.2 Geologic Model Importation

16.2.1 Geotechnical & Hydrogeological Considerations

Absolute Geotechnics Pty LTD (AG) was engaged to undertake a geotechnical assessment of open pit slopes for the 2018 FS. This geotechnical assessment has been undertaken in line with CSIRO's best practice document Guidelines for open pit slope design.

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.

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.

Table 16.1 Geotechnical Slope Design Parameters - (Malikoundi (constraining design criteria emboldened)

Horizon	Domain	Approximate height (m)/ Design 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 (°)
Laterite caprock - ferricrete	All	Up to 5 m	Vertical	-	-	-	-
Transported and Saprolite	All	Up to 35 m, without ramp	FW – 55 HW – 60	5	FW - 4.2 HW – 4.8	33	-
		Up to 35m, with ramp	FW – 55 HW – 60	5	FW - 5.2 HW – 5.75	30	-
Transition	Footwall	Up to ~12 m	55	10	6.5	36.5	-
	Hangingwall	Up to ~20 m for phase 3 (Phase 1 & 2 - 15 m to 30 m)	60	10	6.5	39.2	-
Geotechnical berm at base of weathering	All	FW ~ 130 mRL HW ~ 115 mRL	-	-	12 to 20m	-	-
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 10 m berms)	46° within Fresh Pelite, including ramps
	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
	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	-

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 16.2 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, °)
Laterite caprock - ferricrete	All	Up to 5m	Vertical	-	-	-
Transported and Saprolite	All	Up to 35 m, without ramp	FW – 55 HW – 60	5	FW - 4.2 HW – 4.8	33
		Up to 35 m, with ramp	FW – 55 HW – 60	5	FW - 5.2 HW – 5.75	30
Transition	Footwall	Up to ~20 m	55	10	6.5	36.5
	Hangingwall	Up to ~20 m	60	10	6.5	39.2
Geotechnical berm at base of weathering	All	~129mRL	-	-	12m	-
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
		~30m, with or without ramp - steeper option	60	20	8.5	44.9

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 16.3 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, °)
All	Laterite caprock - ferricrete	Up to 5m	Vertical	-	-	-
Footwall	Saprolite, up to ~40m	With ramp	50	5	4.1	29
		Without ramp	50	5	4.8	31
	Transition	With or without ramps	55	10	6.5	36.5
	Fresh		55	20	8.5	41.6
			60	20	8.5	44.9
Southwest wall	Saprolite, up to ~40m	With ramp	55	5	5.5	29
		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
		Without ramp	55	5	5.5	29
	Saprolite – Hard to base of pit (69mRL)	With ramp	60	5	4	36
Hangingwall West	Saprolite – Soft to 50m	With ramp	55	5	6.75	26
		Without ramp	55	5	5.5	29
	Geotechnical berm at 150m RL	All	-	-	30	
	Saprolite – Hard to base of pit (69mRL)	With ramp	60	5	5.5	31
	Geotechnical berm at 100mRL	All	-	-	30	-

Notes: - Recommended that batter heights in Saprolite are limited to between 6 and 8m to limit erosion. The 5 m 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

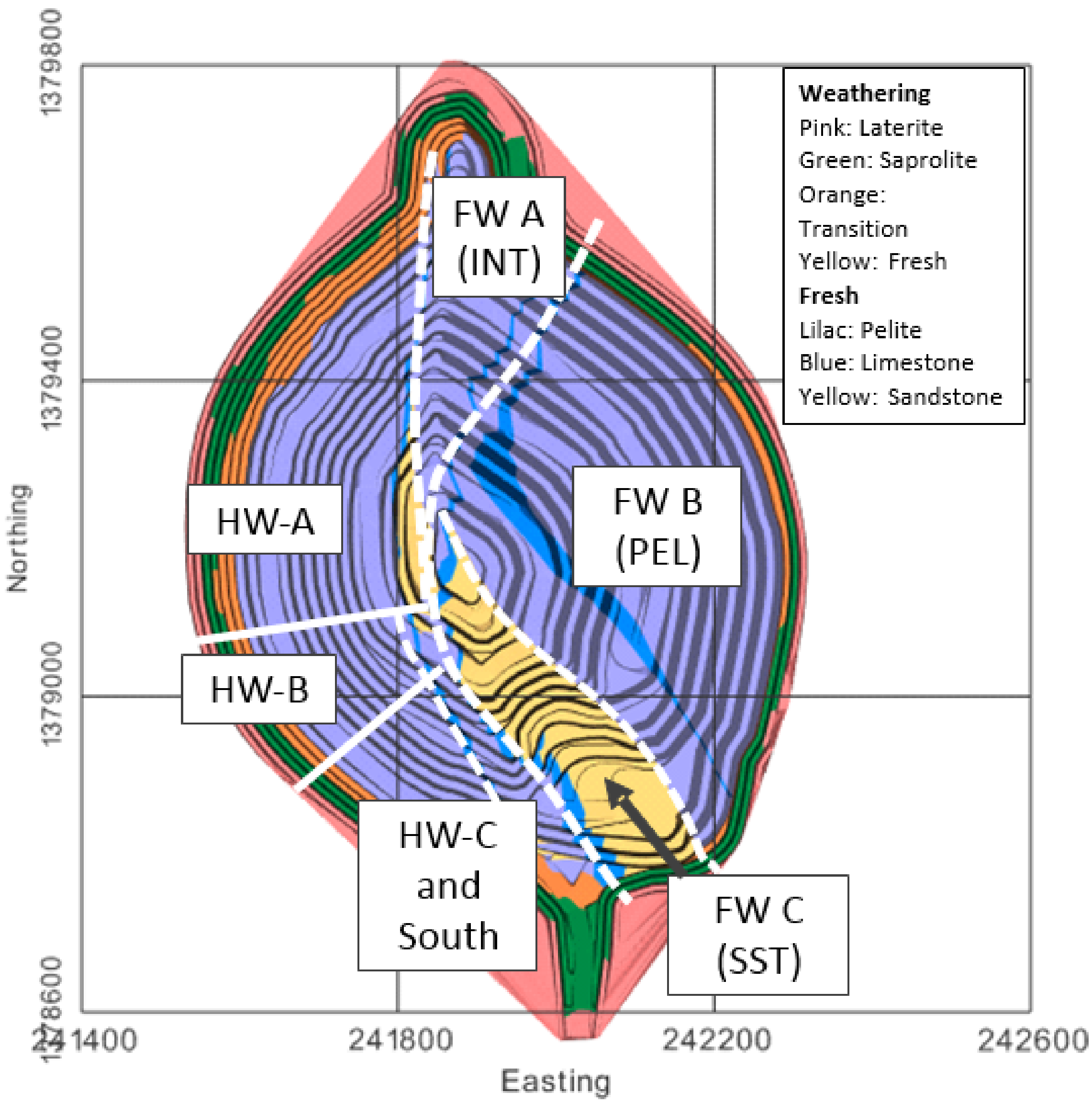


Figure 16.2 **Malikoundi North PFS Pit Design Coloured by Weathering Grade/Geology and showing Slope Design Sectors**

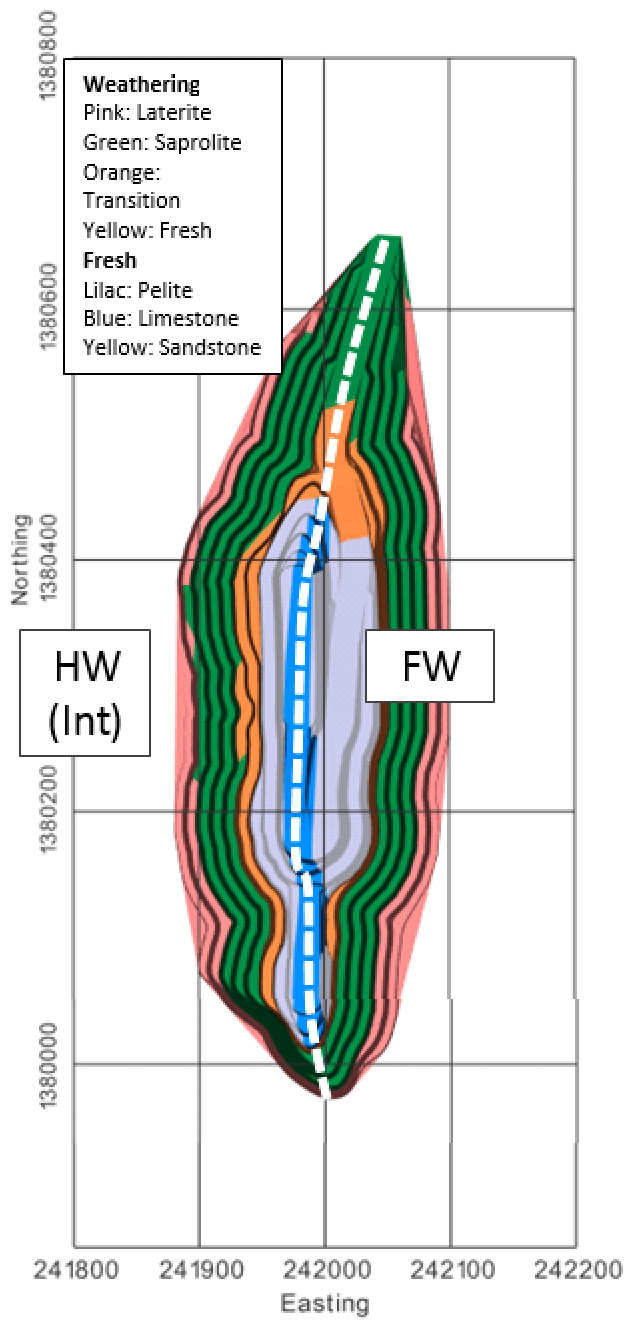
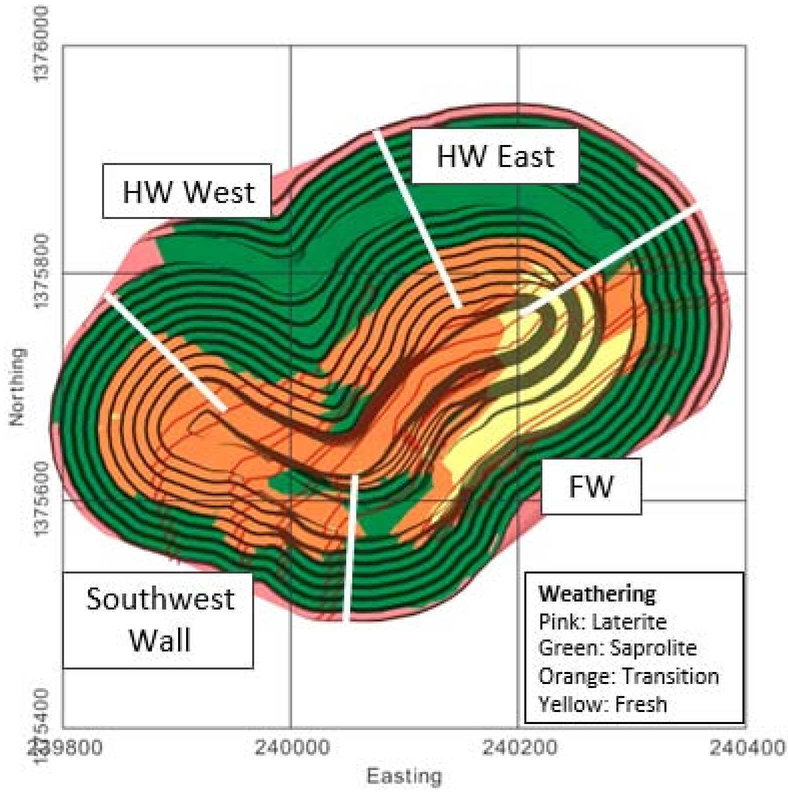


Figure 16.3 Boto 5 PFS Pit Design Coloured by Weathering Grade/Geology and showing Slope Design Sectors



16.2.2 Dewatering Considerations

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.

16.3 Mine 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 16.2 were used for each of the weathering zones and are shown again in Table 16.4.

Table 16.4 Pit Slope Parameters for Detail Design

Rock Domains Horizon	Sector	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	11	60	5	4.8	33	0	30	33
Saprolite/Laterite	FW	12	60	5	4.8	33	0	30	33
Transition	HW	13	60	10	6.5	39.2	0	20	39.2
Transition	FW	14	55	10	6.5	36.5	0	20	36.5
Fresh	HW (A, B, and C) + South wall	15	75	20	8.5	55.3	3	280	44.3
Fresh	FW A +FW B	16	60	20	8.5	44.9	3	300	37.3
Fresh	FW C	17	70	20	8.5	51.7	3	300	42.3
Malikoundi North Pit									
Saprolite/Laterite	HW	1	60	5	4.8	33.0	0	40	33
Saprolite/Laterite	FW	2	60	5	4.8	33.0	0	40	33
Transition	HW	3	60	10	6.5	39.2	0	20	39.2
Transition	FW	4	55	10	6.5	36.5	0	20	36.5
Fresh	HW	5	75	20	8.5	55.3	0	30	55.3
Fresh	FW	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	Southwest Wall	3	55	5	4.9	30.8	0	40	30.8
Transition	Southwest 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

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.

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. 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 four phases within the main pit. Malikoundi North is designed with two phases. Boto 5 is a single-phase pit. The final design phase tonnages and grades are shown in Table 16.5.

Table 16.5 Final Design – Phase Tonnages and Grades

Pit	Ore (Mt)	Au (g/t)	Waste (Mt)	Total (Mt)	Strip Ratio
Malikoundi Phase 1	5.0	1.98	20.8	25.8	4.1
Malikoundi Phase 2	7.8	1.55	59.7	67.5	7.7
Malikoundi Phase 3	7.5	1.54	71.1	78.5	9.5
Malikoundi Phase 4	6.4	1.73	43.0	49.4	6.8
Malikoundi North Phase 1	0.5	2.02	6.8	7.3	12.8
Malikoundi North Phase 2	0.5	2.12	5.2	5.7	10.8
Boto 5	1.3	2.17	11.8	13.1	8.7
Total	29.0	1.71	218.3	247.3	7.5

The phase designs are presented in Figure 16.4 to Figure 16.8.

Figure 16.4 Malikoundi Phase 1 and Malikoundi North Phase 1

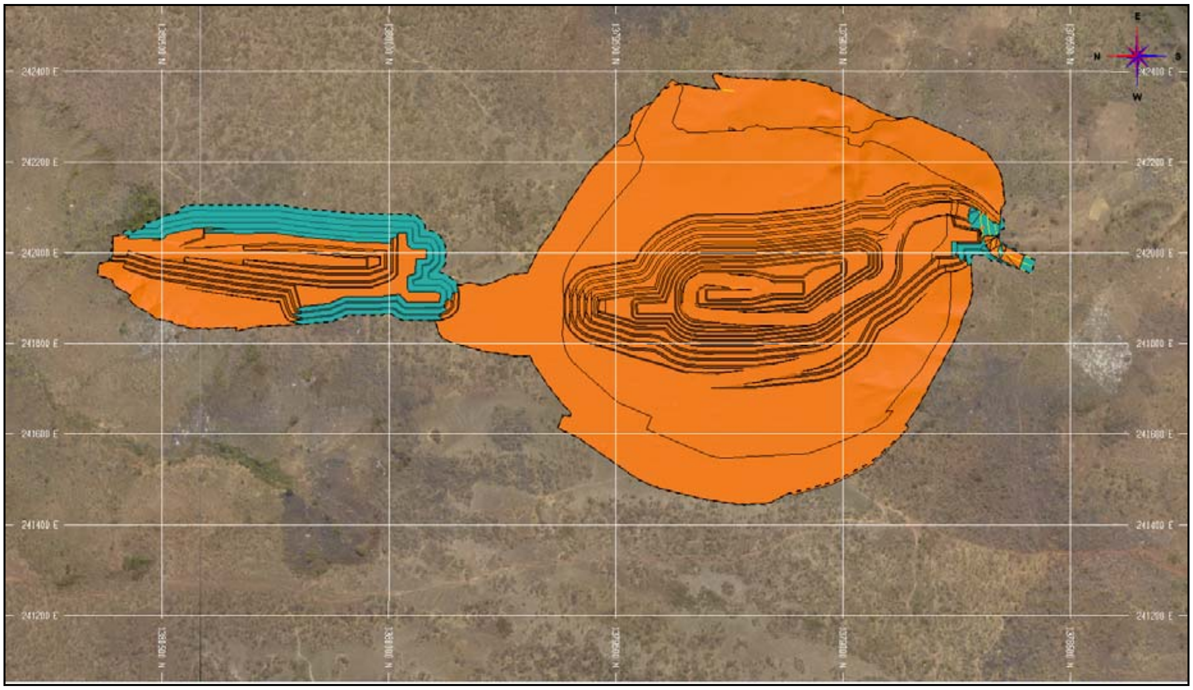


Figure 16.5 Malikoundi Phase 2 and Malikoundi North Phase 1

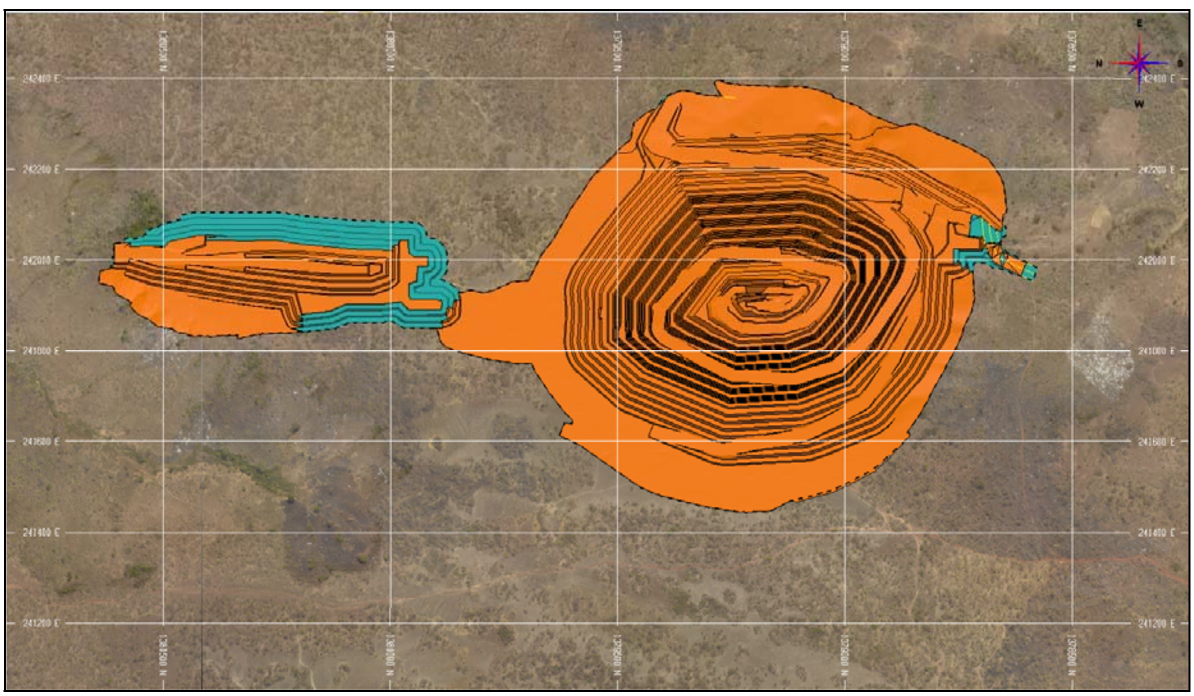


Figure 16.6 Malikoundi Phase 3 and Malikoundi North Phase 2

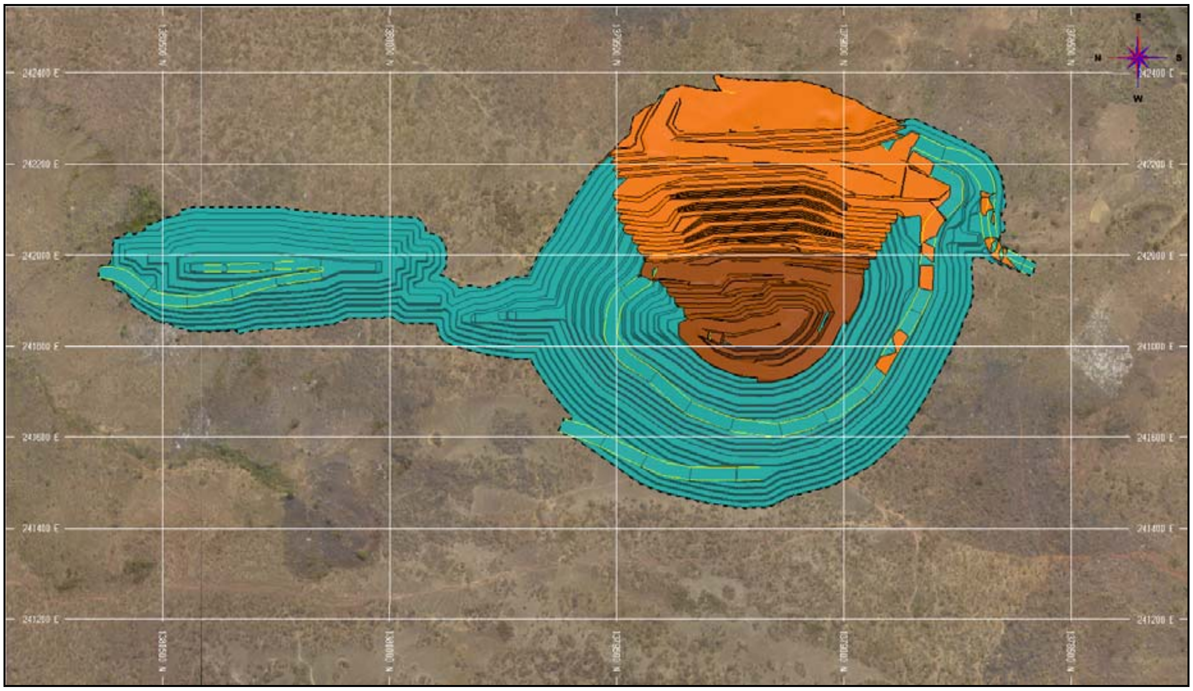


Figure 16.7 Malikoundi Phase 4 and Malikoundi North Phase 2

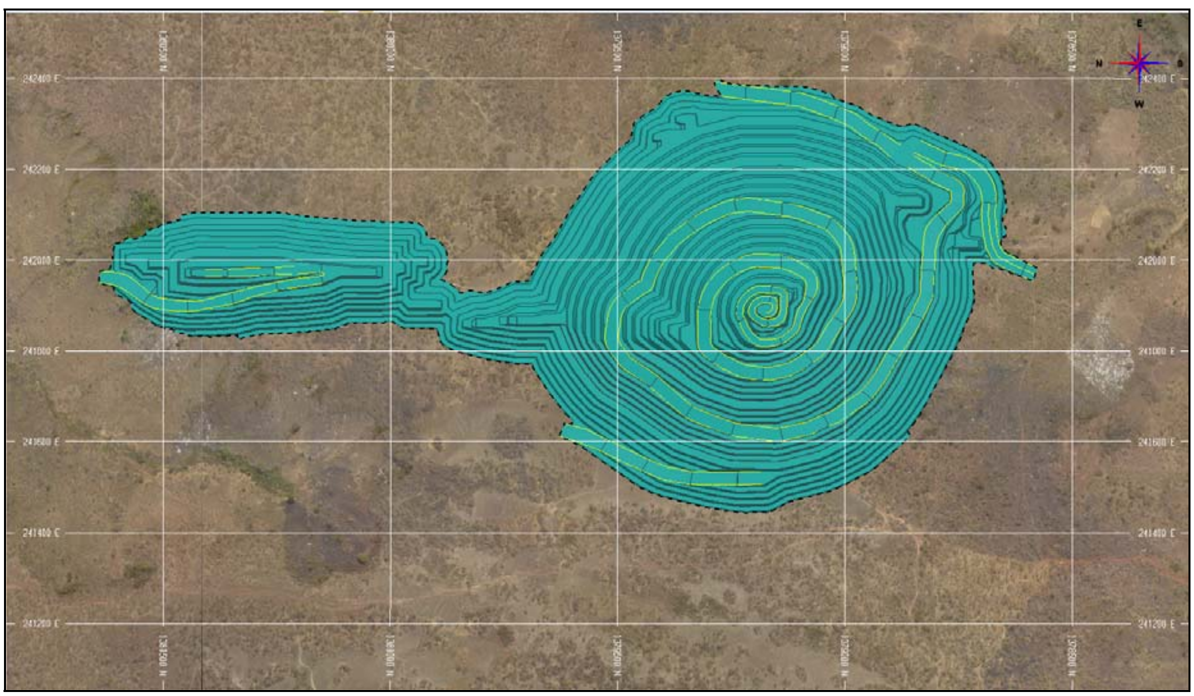
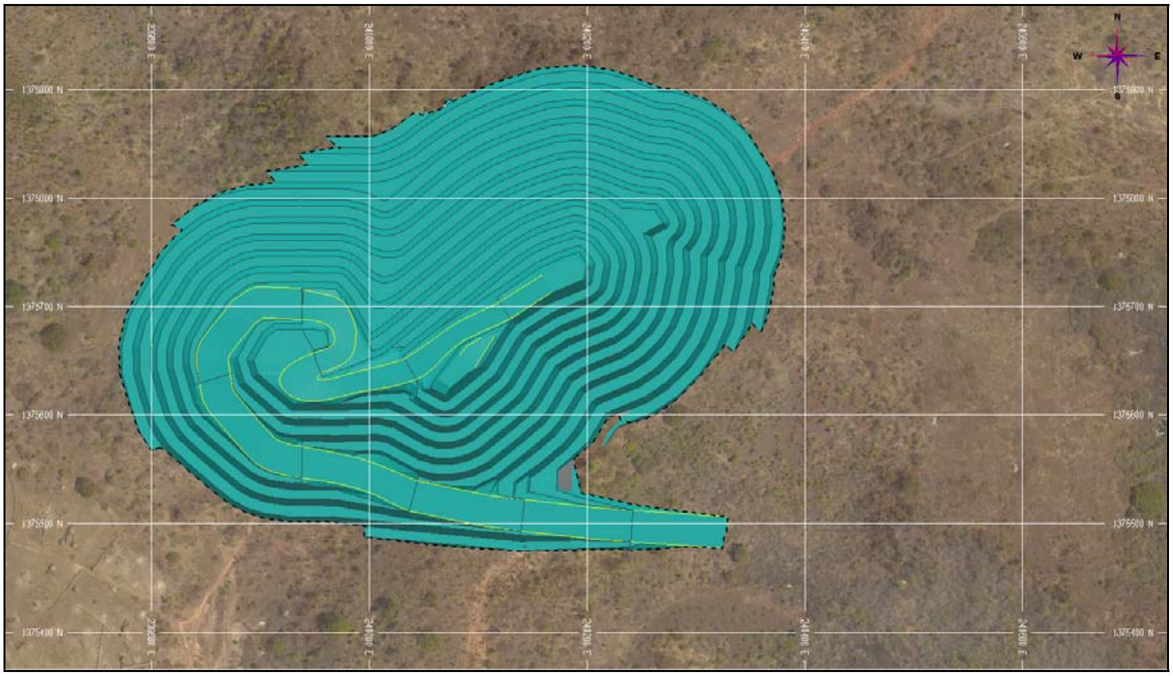


Figure 16.8 Boto 5



16.4 Storage Facilities

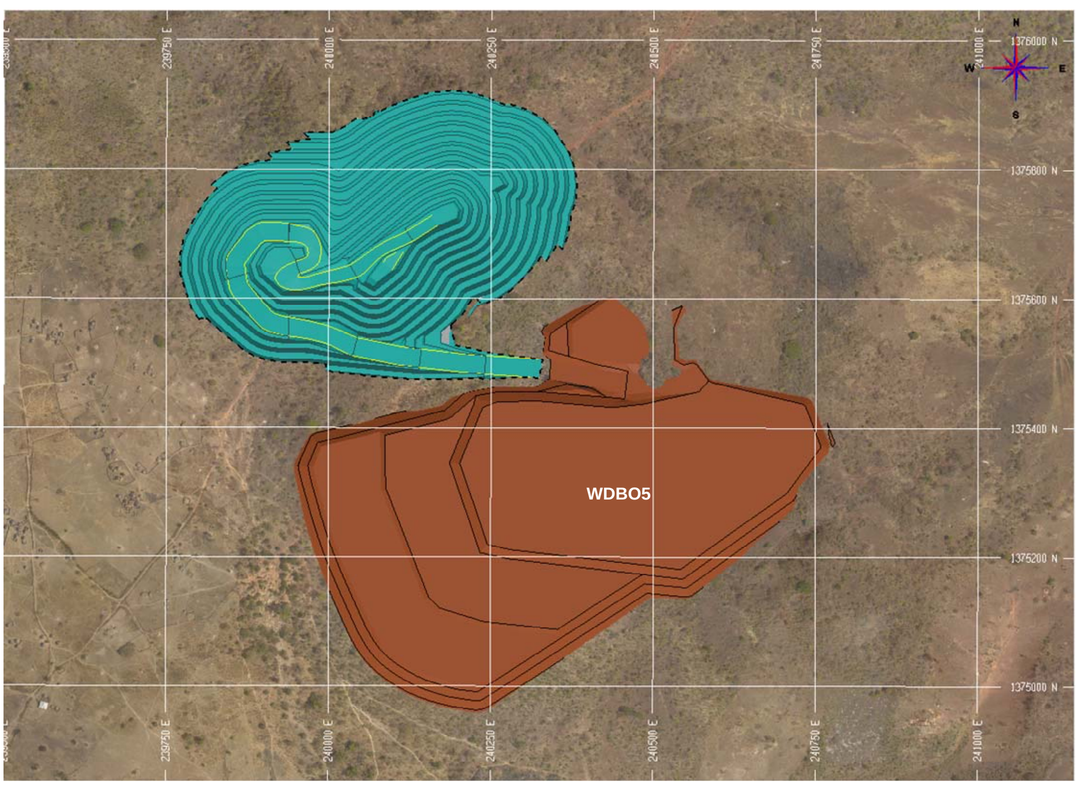
16.4.1 Waste Management Facilities

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. 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 (WMF).

Certain portions of the material will be directed to the TMF for embankment construction. In addition, there will be four waste storage areas. These are shown in Figure 16.9.

Figure 16.9 Waste Management Facilities at Malikoundi

Figure 16.10 Waste Management Facilities at Boto 5



The design and required capacities and the maximum heights are presented in Table 16.6.

Table 16.6 Waste Management Facility Parameters

Parameter	Units	WDNW	WDNE	WDBO5
Design Capacity	Mm^3	23.3	61.9	6.1
Maximum Elevation	Masl	220	190	195

The facilities will be built to respect an overall slope angle of 2H:1V. The uphill ramp gradients are 6%. Waste management facilities will be progressively reclaimed.

Drainage from each of the WMF will be diverted to sedimentation ponds to ensure the sediments are captured before the water exits 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.

16.4.2 Ore Stockpiles

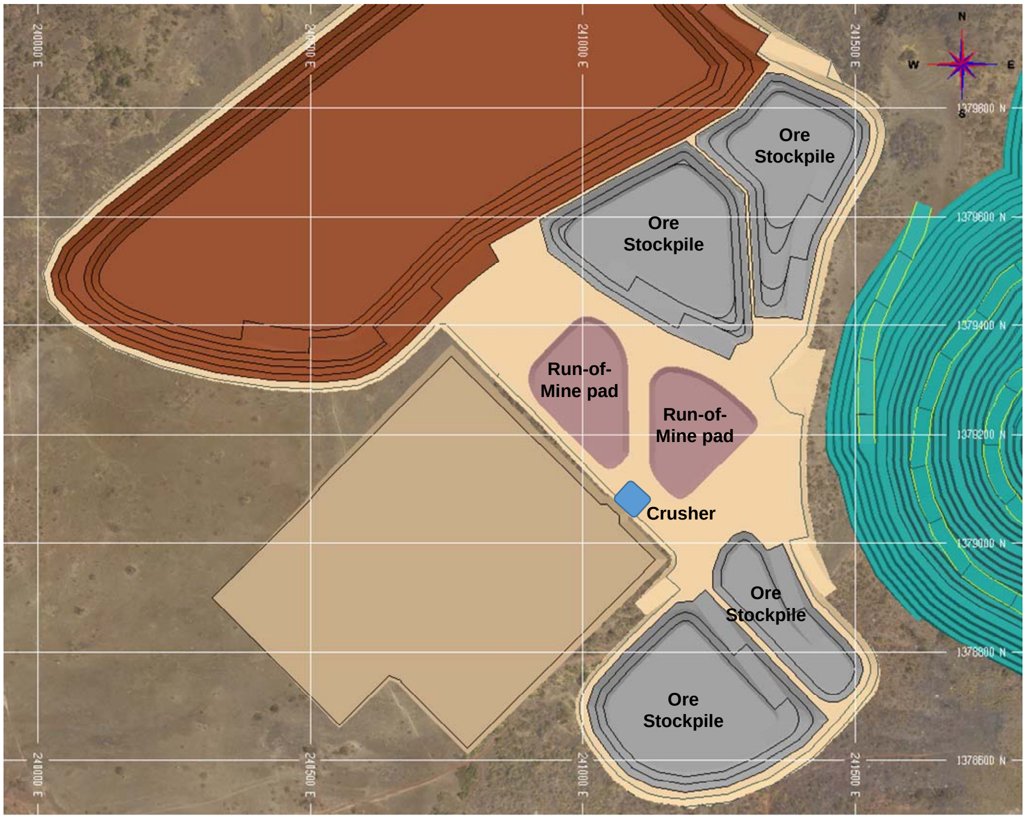
Several ore stockpiles for the different material types and grade bins will be developed throughout the mine life in order to optimally feed the mill. The stockpiling strategy by material type and grade bin are presented in Table 16.7. The stockpiles shapes and sizes will fluctuate throughout the project life. Furthermore, two run-of-mine pads will be located near the crusher. At the end of the mine life, all of these stockpiles will be depleted.

Table 16.7 Ore Stockpiling Strategy by Material Type and Grade Bin

Stockpile	Saprolite		Transition		Fresh Rock	
	From (g/t)	To (g/t)	From (g/t)	To (g/t)	From (g/t)	To (g/t)
LG	COG	0.66	COG	0.66	COG	0.76
MG	0.66	1.04	0.66	1.04	0.76	1.04
HG	1.04	+	1.04	+	1.04	+

The ore stockpiles are presented in Figure 16.11.

Figure 16.11 Location of Ore Stockpiles



16.5 Mine Schedule

The mine schedule delivers 29.0 Mt of ore grading 1.71 g/t Au to the mill over a mine life of approximately 11 years, including 12 months of pre-production. 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. The maximum mining capacity is 38 Mtpa. The pit phasing and ore stockpiling strategy will ensure that sufficient mill feed is available during the rainy season. Phases will be advanced quickly in the dry season to provide temporary water storage after a rainfall event. Dewatering pumps will evacuate the water from the pits during the wet season.

Project activities in the pre-production period include haul road construction, FWP construction, TMF material placement, initiation of mining in Malikoundi Phase 1 and development of an ore stockpile near the processing plant.

The mill's production will ramp-up over a three (3) month period before it achieves its nameplate capacity; 60% for the first month, 80% for the second month, and 90% for the third month. Thereafter, the mill will operate at 100%. Throughout the mine life, a blending strategy will be used to feed the mill optimally with soft and hard rock material.

The Malikoundi pit will be mined from the beginning of mining operations until Year 8. The Malikoundi North pit will be mined from Year 1 to Year 4. The Boto 5 pit will be mined from Year 1 to Year 2. From years 9 to 11, the mill will be fed exclusively from the ore stockpiles until they are completely depleted. The mine plan is presented in Table 16.8 and Figure 16.12 and Figure 16.13.

Table 16.8 Mine Plan

Mine Plan	Unit	-1	-2	1	2	3	4	5	6	7	8	9	10	11	Total
Material Mined - Malikoundi	(kt)	800	14,175	25,023	26,174	36,995	32,735	31,102	31,001	17,464	5,784	0	0	0	221,253
Waste	(kt)	800	12,931	22,089	23,942	33,695	28,219	28,825	27,581	13,667	2,825	0	0	0	194,574
Ore	(kt)	0	1,244	2,934	2,232	3,300	4,516	2,277	3,420	3,797	2,959	0	0	0	26,680
Material Mined - Malikoundi North	(kt)	0	0	3,999	4,737	0	4251	0	0	0	0	0	0	0	12,987
Waste	(kt)	0	0	3,884	4,302	0	3,787	0	0	0	0	0	0	0	11,974
Ore	(kt)	0	0	115	435	0	464	0	0	0	0	0	0	0	1,014
Material Mined - Boto 5	(kt)	0	0	7,000	6,100	0	0	0	0	0	0	0	0	0	13,100
Waste	(kt)	0	0	6,776	4,977	0	0	0	0	0	0	0	0	0	11,753
Ore	(kt)	0	0	224	1,123	0	0	0	0	0	0	0	0	0	1,347
Material Mined - Total	(kt)	800	14,175	36,023	37,012	36,995	36,986	31,102	31,001	17,464	5,784	0	0	0	247,341
Waste	(kt)	800	12,931	32,749	33,221	33,695	32,006	28,825	27,581	13,667	2,825	0	0	0	218,300
Ore	(kt)	0	1,244	3,273	3,790	3,300	4,980	2,277	3,420	3,797	2,959	0	0	0	29,040
Average Mined Grade	(g/t)	0.66	2.18	1.88	1.82	1.43	1.69	1.26	1.62	1.67	1.98	0.00	0.00	0.00	1.71
Material Reclaimed	(kt)	0	0	1,318	521	770	240	1 352	186	338	348	2,750	2,700	1,990	12,514
Average Reclaimed Grade	(g/t)	0.00	0.00	2.37	1.80	2.72	2.20	1.99	1.00	1.05	0.88	0.65	0.76	0.88	1.26
Total Material Moved	(kt)	800	14,175	37,341	37,532	37,766	37,226	32,454	31,187	17,802	6,132	2,750	2,700	1,990	259,855

Figure 16.12 Mined Material by Rock Type

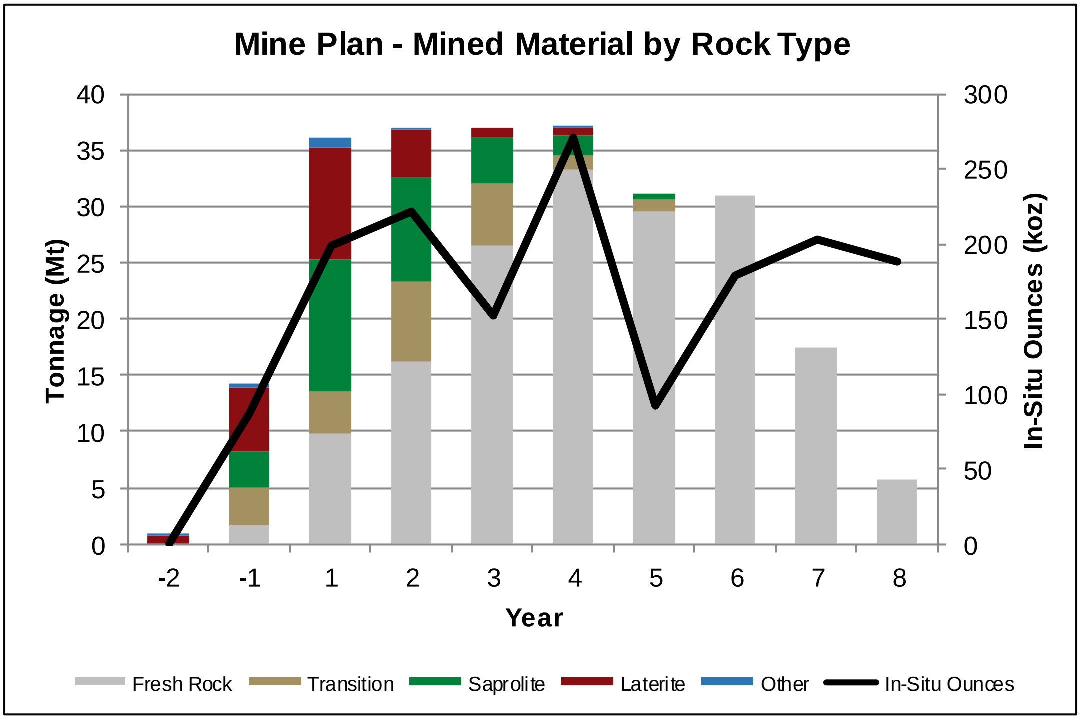
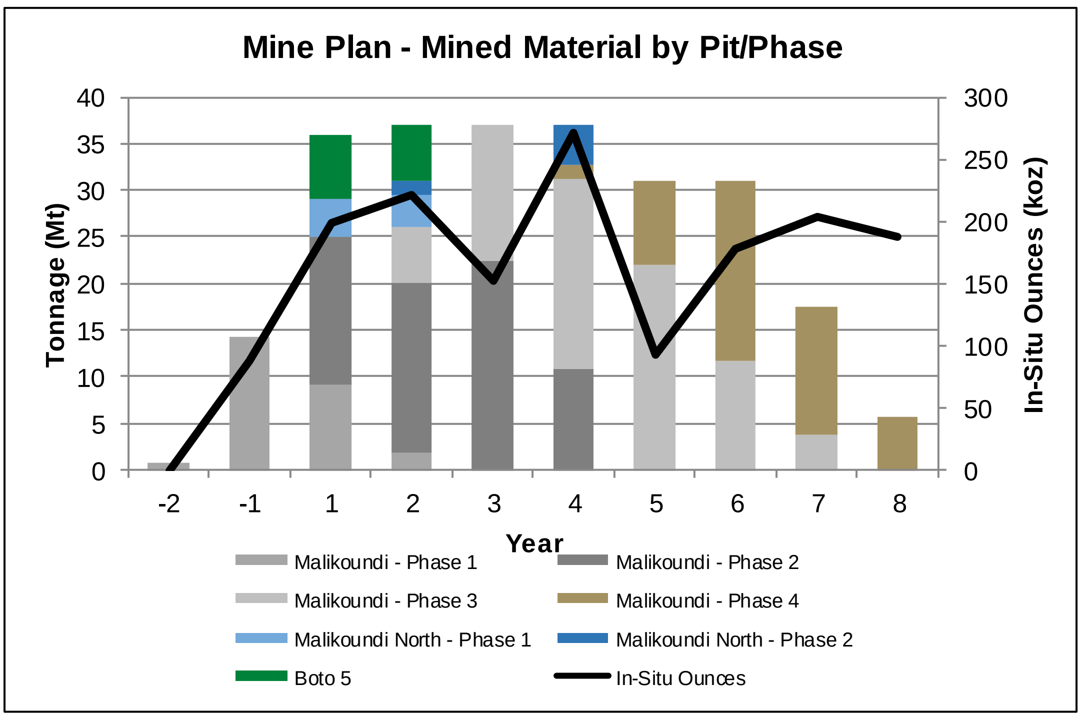


Figure 16.13 Mined Material by Pit/Phase



The mill plan is presented in Figure 16.14 and Figure 16.15.

Figure 16.14 Mill Feed by Material Type

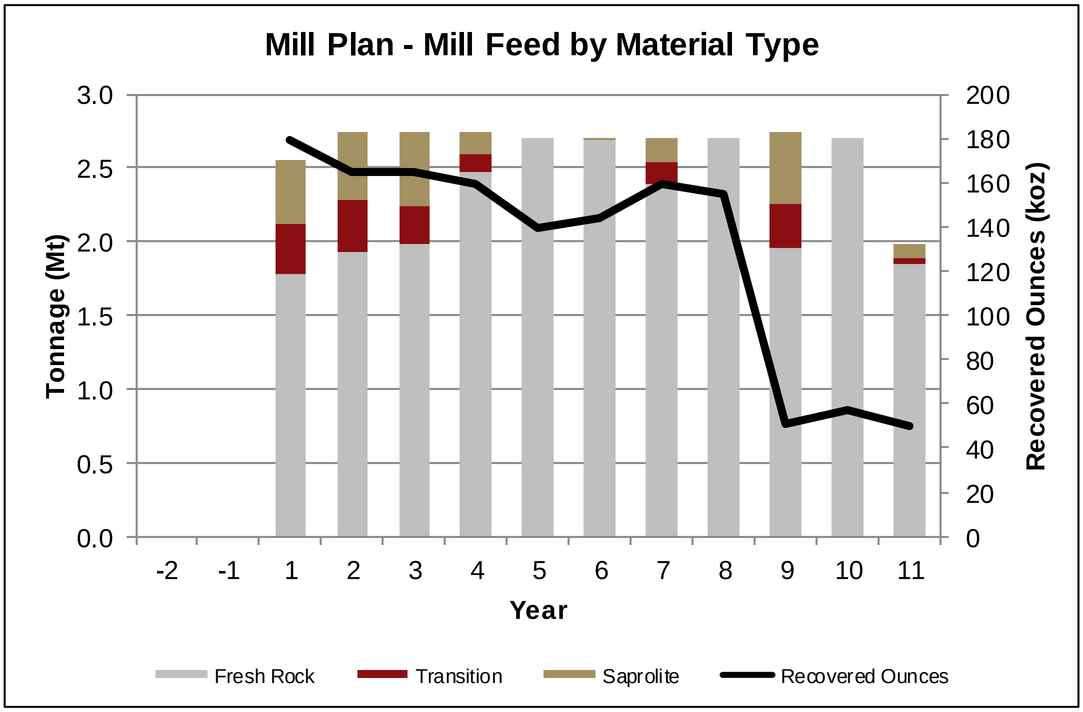
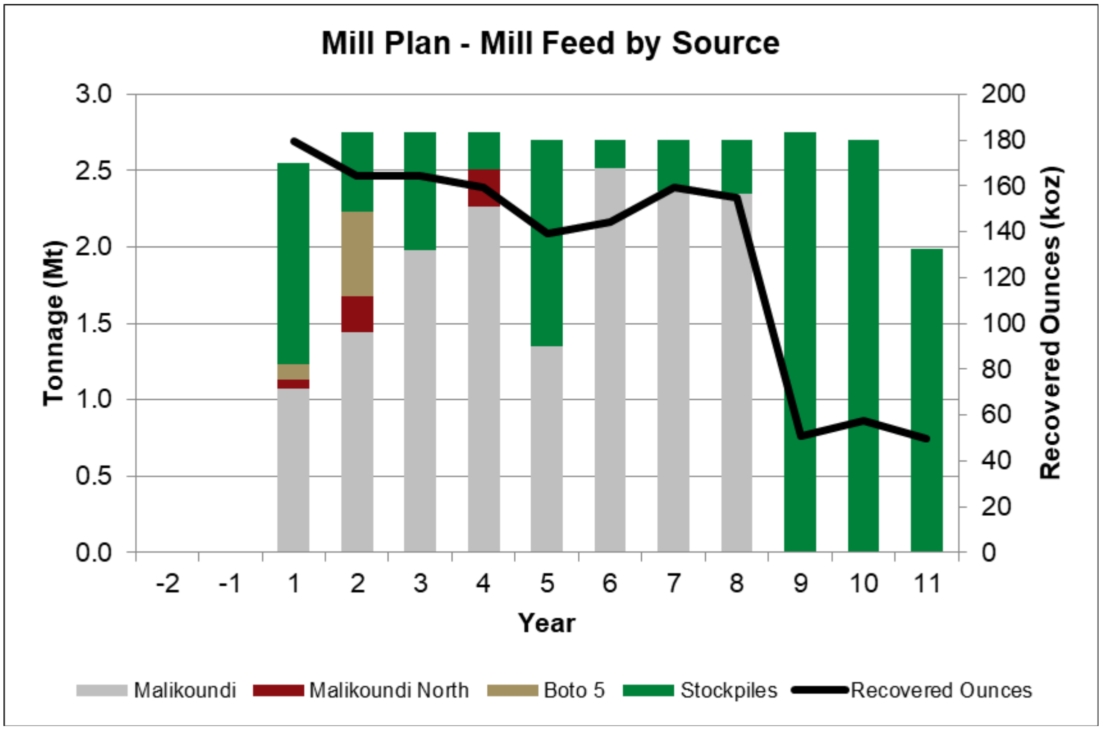


Figure 16.15 Mill Feed by Source



16.6 Mine Operations

The mine will operate 365 days/year, with two 12-hour shifts/day. The following sections describe the various mine related activities.

16.6.1 Grade Control

In order to reduce dilution, reverse circulation drilling will be completed. This information is then built into the short-range models and used to guide the loading equipment. All grade control drilling will be executed by a dedicated drill rig and crew. All material profiles, including the saprolite, will be drilled.

16.6.2 Production Drilling & Blasting

While saprolite material can be freely dug by the loading equipment, all transition and fresh rock material will be drilled and blasted. A fleet of down-the-hole (DTH) drills will complete all production drilling. The drill and blast patterns are presented in Table 16.9.

Table 16.9 Drill and Blast Patterns

Material		Transition		Fresh Rock	
		Waste	Ore	Waste	Ore
Average Density	(t/m^3)	2.17	2.17	2.76	2.76
Hole Diameter	(mm)	165	165	165	165
Burden	(m)	5.0	5.0	4.5	4.5
Spacing	(m)	5.2	5.2	4.8	4.8
Bench height	(m)	10.0	10.0	10.0	10.0
Powder Factor	(kg/t)	0.32	0.32	0.31	0.31

The explosives provider will be responsible for all explosives related activities (loading, personnel, product management, etc.). All holes will be loaded with a bulk emulsion and detonated with an electronic detonating system. Electronic detonators provide greater flexibility to the blasting sequence and reduce mining dilution.

Pre-split holes will be drilled along the perimeter pit phase walls in transition and fresh rock. These holes will be loaded with packaged emulsion and blasted prior to mining.

16.6.3 Loading

The primary loading equipment selected for the project is a $15m^3$ bucket capacity hydraulic front shovel. For additional loading capacity and operational flexibility, $13m^3$ bucket capacity front end loaders will be used for loading in the pit as needed and for all rehandling work. Additional loading capacity was considered for rehandling material near the crusher. This equipment will be used to optimally feed the mill, as previously described. Loading productivities were evaluated using the Talpac truck haulage simulation software.

16.6.4 Hauling

The primary hauling equipment is a 95 t capacity haul truck. These trucks will complete all movement of mined and reclaimed material. Haul truck requirements are based on simulations for haul cycle times with Hexagon's MSHaulage software and RPM's Talpac truck haulage simulation software.

16.6.5 Support & Auxiliary Equipment

The mining operation will be supported by a variety of equipment, including bulldozers, motor graders, backhoe excavators, dump trucks, water trucks, wheel loaders and compactors. This equipment will work to maintain benches around mining activity, haul roads and the various stockpiles.

Furthermore, other support equipment will be used by the maintenance department to service equipment in the field and near the maintenance workshop. These support equipment will include pump trucks, mechanic trucks, welding trucks, integrated tool carriers, and low-boy trailers.

A variety of auxiliary equipment will also be used, including pick-up trucks and lighting plants.

The quantities of all mining equipment per period are presented in Table 16.10.

Table 16.10 Mine Equipment Requirements

Equipment	Description	Year												
		-2	-1	1	2	3	4	5	6	7	8	9	10	11
Production Drill Rig	DTH	0	3	4	6	8	8	8	8	5	2	0	0	0
Grade Control Drill Rig	DTH RC	0	1	2	2	2	2	2	2	2	2	0	0	0
Hydraulic Shovel	$15m^3$ capacity	0	2	5	5	5	5	5	5	2	0	0	0	0
Front End Loader	$13m^3$ capacity	0	1	3	3	3	3	3	3	3	3	1	1	1
Haul Truck	95 t capacity	0	10	23	25	27	27	27	27	26	9	1	1	1
Bulldozer	$22m^3$ blade	0	3	5	5	5	5	5	5	5	2	1	1	1
Motor Grader	4.9m blade	0	1	2	3	3	3	3	3	3	2	1	1	1
Support Backhoe	$7m^3$	0	1	1	1	1	1	1	1	1	1	0	0	0
Water Truck	$50m^3$	0	1	2	2	2	2	2	2	2	2	1	1	1
Backhoe & rock hammer	$2\ m^3$	0	1	1	1	1	1	1	1	1	1	0	0	0
Compactor	2m width	0	1	1	1	1	1	1	1	1	1	0	0	0
Pump Truck		0	1	1	1	1	1	1	1	1	1	1	1	1
Crushed Pile Loader	$7m^3$ capacity	0	0	1	1	1	1	1	1	1	1	0	0	0
Mechanic Truck		0	2	2	2	2	2	2	2	2	2	1	1	1
Welding Truck		0	1	1	1	1	1	1	1	1	1	1	1	1
Integrated Tool Carrier		0	2	2	2	2	2	2	2	2	2	1	1	1
Light Plants		0	11	11	11	11	11	11	11	11	11	6	6	6
Pick-up Truck		0	20	25	25	25	25	25	25	25	25	15	15	15
Dump Truck		0	1	1	1	1	1	1	1	1	1	1	1	1
Low-boy Trailer		0	0	1	1	1	1	1	1	1	1	1	0	0

**Values presented are the maximum for the entire year.*

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 Boto mine plan calls for the processing of blends containing approximately 30% saprolite during the initial years of operation compared to a LOM average of 13% saprolite. The testing of saprolite samples showed that it is significantly softer than fresh rock and demands less grinding energy to achieve a specified grind. The implication is that a grinding mill, operating at a constant power draw, can process ore at a faster rate if the feed blend contains more saprolite. The above average gold grade of the initial saprolite, combined with its potential for increased throughput, presents an opportunity to maximize early revenues. To exploit this opportunity a design factor of approximately 10% was applied to the nominal throughput of 2.75 Mtpa for process plant equipment sizing.

Saprolite can be difficult to process as it can result in materials handling challenges with sticky fines leading to choked chutes and crusher blockages and high proportions in slurries causing viscosity and settling problems.

To mitigate the materials handling risk, transfer chute design allows for the nature of saprolitic material and a vibrating grizzly was added ahead of the crusher to allow fines to bypass the jaw crusher. Lycopodium has designed and constructed multiple projects in West Africa treating variable proportions of saprolitic ores and proven design principles have been incorporated into the Boto design.

Rheology testing performed at 20% saprolite content indicates that the elevated saprolite content is not predicted to cause viscosity related problems at the design pulp densities. Confirmatory rheology testing at 30% saprolite is recommended in the next phase of the project in order to provide additional validation for the tailings delivery system design.

The key project design criteria for the plant are:

- Nominal throughput of 2.75 Mtpa ore based on the 85th percentile of hardness for the design blend of 80% bedrock and 20% saprolite/saprock.
- 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, but allowing manual override and control if/when required.
- A design gold head grade of 2.83 g/t and a LOM head grade of 1.71 g/t.

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

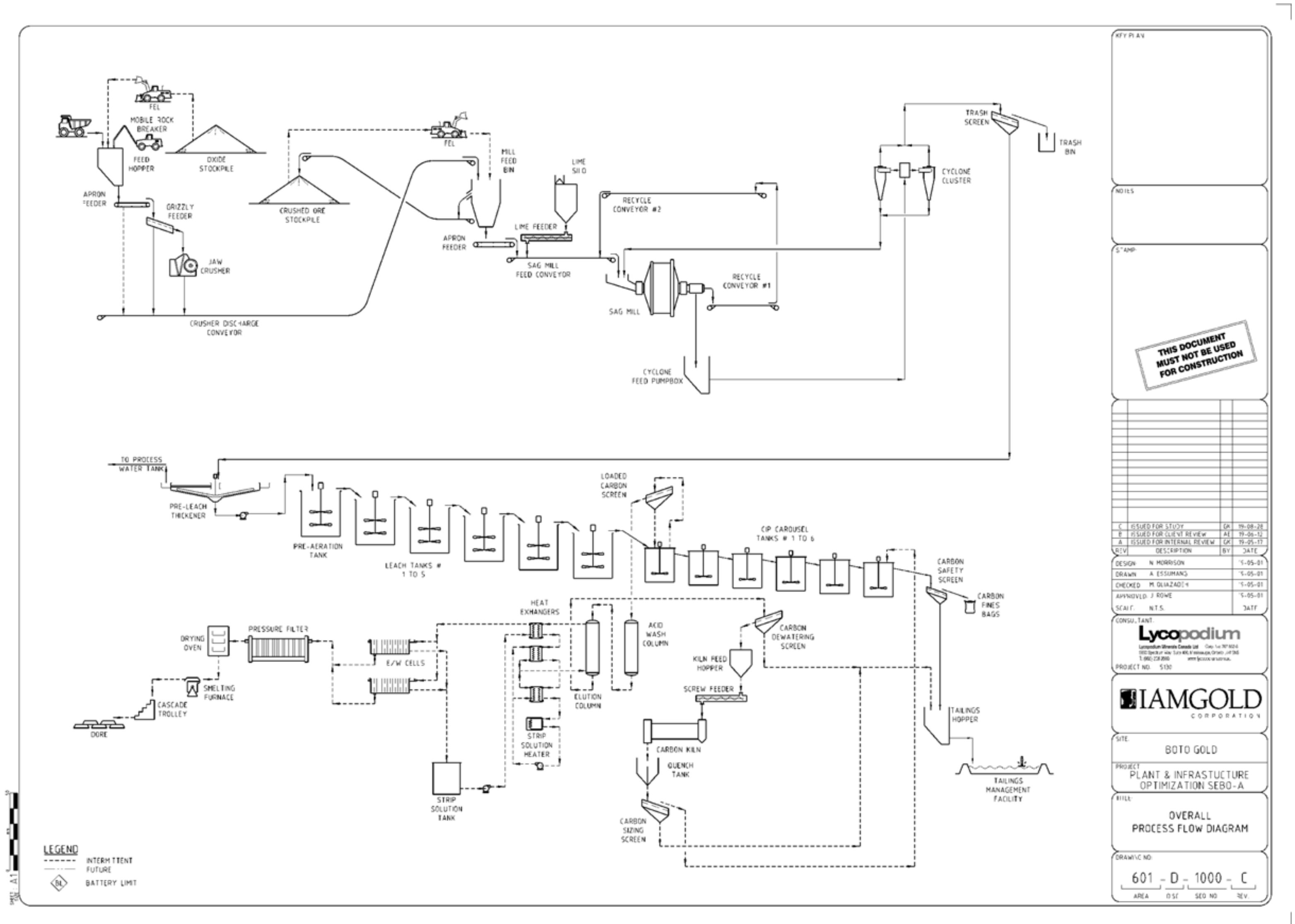
The treatment plant design incorporates the following unit process operations:

- Single stage primary crushing with a jaw crusher to perform the first comminution stage.
- A vibrating grizzly between the ROM apron feeder and the primary jaw crusher to allow potentially sticky fines to bypass the jaw crusher directly to the primary crusher discharge conveyor.
- Coarse ore storage (Mill feed surge/overflow bin, stockpile and extended stockpile) and ROM oxide stockpile.
- Single Stage Semi-Autogenous (SSAG) circuit.
- Hydrocyclones to promote better particle size separation efficiency.
- Trash screen to remove foreign materials prior to downstream processing.
- Pre-leach thickener to increase slurry density to the leach circuit, minimize leach tank volume requirements and reduce overall reagent consumption.
- Pre-oxidation ahead of leaching to minimize cyanide consumption and improve downstream leach kinetics.
- Leach circuit with five tanks to provide the slurry residence time required for gold dissolution.
- Six-stage CIP carousel circuit for recovery of gold dissolved in the leaching circuit.
- Pressure Zadra elution circuit including an acid wash column to remove inorganic foulants from the carbon with hydrochloric acid, followed by an elution column for gold stripping.
- Electrowinning cells and a smelting furnace for gold recovery from eluate solution and to smelt sludge to produce doré.
- 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 (LOM)	g/t Au	1.71	IAMGOLD
Head Grade (design)	g/t Au	2.83	IAMGOLD
Crushing Plant Availability	%	75.0	Lycopodium
Plant Availability	%	92.0	Lycopodium
Crushing Work Index (CWi, ave)	kWh/t	12.9	Testwork
Bond Ball Mill Work Index (BWi, ave)	kWh/t	17.5	Testwork
SMC Axb[1] (ave)		41.6	Testwork
Bond Abrasion Index (Ai, ave)		0.39	Testwork
Grind Size	µm	75	Testwork
Pre-Leach Thickener Solids Loading	$t/m^2.h$	1.4	Testwork
Leach Circuit Residence Time	hours	33.5	Testwork
Adsorption Circuit Residence Time	hours	1.5	
Leach Slurry Density	% solids (w/w)	49.4	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	0.27	Testwork
Lead Nitrate Addition	Kg $PbNO_3$/t ore	0.2	Testwork
Dissolved Oxygen Level in Leach	ppm	>15	Testwork
Lime Addition[2]	kg/t	1.92	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 90% 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 8% saprolite overlaying a layer of approximately 5% transition material (also referred to as “saprock”) followed by the remaining 87% 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 is 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 80% bedrock and 20% saprolite/transition material. The process plant was designed to be fit-for-purpose with no allowance 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, subject to availability, operate for 365 days/annum, 24 hours/day. On this basis, and at a design operating availability of 75%, the crushing circuit will operate for 6,570 hours/annum. This equates to a crusher circuit throughput of 419 t/h.

Downstream of the crushing circuit, the grinding, leach, adsorption and tailings disposal circuits will operate for 365 days/annum, for a nominal 24 hours/day. On this basis, and at a design operating availability of 92%, these circuits will operate for a nominal 8,059 hours/annum. This equates to a nominal circuit throughput of 341 t/h.

The grinding circuit will consist of a single 36’ (10.97 m) diameter by 18.3’ (5.57 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 comprise of a single pre-oxygenation tank followed by five leach tanks, providing a nominal leach residence time of 33.5 hours. The CIP carousel adsorption circuit will comprise of six adsorption contactors. These contactors will operate with a carbon concentration of 42 g/L on a one-day cycle. The total slurry residence time in the CIP section is 1.5 hours.

The gold recovery and carbon handling circuit will operate on a batch basis at a carbon throughput rate of 5 t 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 t 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 if required. The ROM stockpile will be primarily utilized for ore blending to optimize mill power consumption, grade and to equalize, as much as possible, the saprolite content 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 aperture), 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, which will feed a vibrating grizzly. Undersize from the grizzly will report directly to the primary crusher discharge conveyor. Grizzly oversize will feed 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 and grizzly undersize to the mill feed bin. The product from the crusher area will be at a P_{80} of 138 mm.

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 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 Ore Storage (Coarse and Blending Capacity)

The mill ore feed bin will have a live capacity of 108 t (equivalent to approximately 19 minutes 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 5,500 t (providing 16 hours of plant feed). Crushed ore will be reclaimed from the stockpile, to the ore bin, via a front-end loader.

During extended periods of up to four 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. Additional surge and blending capacity is provided by the ROM oxide stockpile at the truck tipping area.

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, with the instantaneous tonnage used for controlling the speed of the apron feeder and to totalize the feed to the mill for metallurgical accounting purposes.

Quicklime will be added directly to the SAG mill feed conveyor from a storage silo, 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 101 1t lime bags will be kept in storage, providing a 7-day contingency supply.

Grinding media (125 mm or 105 mm steel balls) will be added to the crushed ore bin using a front-end loader.

Solid spillage generated, within the reclaim area, will be manually recovered and transported to the mill feed bin. Water will be directed to a sump from where it will be pumped to the mill discharge pumpbox.

Auxiliary equipment for the reclaim area will include:

- Crushed ore stockpile dust suppression sprays.
- Lime silo and lime hopper dust collector.
- Lime hoist and lifting frame.
- Weightometer calibration chain.

17.2.4 Grinding and Classification

The grinding circuit will be a SSAG circuit comprising of a single, variable speed SAG mill. The SAG mill will operate in closed circuit with hydro-cyclones to achieve an overflow slurry density of 28.7 %w/w solids, 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 have a P_{80} of 75 µm.

To achieve the required leach product size when treating ore at the 85th percentile of hardness, a 10.97 m x 5.57 m SAG mill (36 ft x 18.3 ft; 13.5 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 pulp density in the mill. The SAG mill will be fitted with discharge grates, which will allow slurry to pass through the mill and will 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 purposes, 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.

Slurry spillage within the grinding circuit will be directed to a central sump fitted with a sump pump. Slurry from this sump will be discharged into the mill discharge pumpbox.

Auxiliary equipment within the grinding area will include:

- SAG mill drive lubrication system.
- SAG mill liner handler and relining equipment.
- 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 49.8% solids, will be pumped by the thickener underflow pumps to the leach circuit feed distribution box. This underflow stream will be sampled for metallurgical accounting before reporting to the leach circuit.

A sump pump will service the pre-leach thickener area. Spillage and wash down collected by the sump pump will be returned to the pre-leach thickener 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.

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. Each leach tank will have a live volume of 3,150 m^3. This provides a leach residence time of 33.5 hours at a slurry feed rate of 470 m^3/h to 49.4 %w/w solids.

Should a leach tank be off-line for maintenance the tank can be bypassed using pneumatic gates located within the leach inter-stage launders. One gate will shut off slurry to the leach tank taken off-line while the second gate will allow slurry diversion to the next leach tank.

Cyanide, for gold dissolution, will be added to the leach circuit by the cyanide dosing pumps. The primary cyanide dosing point will be the first leach tank, with further addition points located down the leach train for use as required. 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 delayed 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. This will be used in emergencies only and will be monitored to limit unnecessary or accidental caustic consumption.

To sustain the desired dissolved oxygen levels, oxygen will be supplied by a Pressure-Swing-Adsorption (PSA) oxygen plant. Oxygen will be injected into each Leach tank via dedicated oxygen spargers and external oxygen contactors will be installed on the first two-leach tanks to increase the oxygen transfer rate. Lead nitrate will also be added to the leach feed distribution box.

Leach circuit tailings slurry will flow, by gravity, to the carousel circuit feed distribution launder where the solubilised gold will be recovered by carbon adsorption within the Pumpcell® carbon-in-pulp carousel circuit.

The leach area will be serviced by two sump pumps. The sump pump closer to the feed-end will return spillage to the leach feed distribution box. The sump pump closer to the last tank will discharge spillage either to the final leach tank or to the trash screen as an alternative.

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 current 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 120 m^3. This equates to a total slurry residence time of 1.5 hours at a feed rate of 473 m^3/h. The tanks will operate with a carbon concentration of 42 g/L. The adsorption tanks will operate on a daily cycle with the single stage inventory of 5 t carbon recovered and transferred 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 is 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 feed advanced counter current to the carbon adsorption stages. Slurry will flow from the current Tank 1 through to Tank 6, with the slurry 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 adsorption tank, 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 tank from which carbon was previously removed, with excess water and carbon fines flowing to the carbon safety screen. Feed to the carousel will be advanced to the previous Tank 2 (now Tank 1) with the tank containing fresh carbon now becoming Tank 6 in the carousel.

Slurry discharging the last adsorption tank will flow by gravity to a transfer hopper, from where it will be pumped to the carbon safety screen. The carbon safety screen will recover undersized carbon exiting the adsorption circuit and protect against loss of carbon through a damaged Pumpcell® screen. The safety screen oversize will report to a fine carbon bin stream, while the undersize will flow to the tailings pumpbox. A sampler, installed on the carbon safety screen feed will periodically collect a sample of the adsorption tailings 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 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 have separate 5 t capacity acid wash and elution columns. Following a cold acid wash to remove calcium and magnesium salts, gold will be eluted from the carbon, using the pressure Zadra elution process. As the CIP circuit operates on a daily (24-hour) cycle, one elution will be completed every day. The elution circuit can, however, complete a strip cycle in less than 24 hours providing some capacity to deal with periods of high mill head grades. At the design ROM head grade of 2.83 g/t and with a carbon gold loading of about 4,300 g/t the required daily carbon movement is 5 t.

Acid Wash

The cold acid wash sequence will be required to remove accumulated calcium 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. Once the Acid Wash Column is filled to the required level, the carbon fill sequence will be stopped.

The acid wash cycle will use a 3% w/v hydrochloric acid solution. This dilute acid will be prepared by mixing raw water and neat commercial grade (32%) hydrochloric acid, in the acid dilution tank. The diluted acid will be injected into the acid wash column via the feed manifold located beneath the column. Once the column is full the carbon will be allowed to soak in the acid for a period of one hour.

Upon completion of the acid soak, the dilute acid solution will be circulated through the column for a period of 2 hours. After acid washing the carbon will be rinsed with four bed volumes (4 BV) of water, at a rate of 2 BV/h. 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 plant tailings 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 addition of make-up raw water into the strip solution tank, along with the simultaneous injection of cyanide and caustic solution. Cyanide and sodium hydroxide (caustic) will be added to achieve 1% w/w NaOH and 0.2% w/w NaCN in the strip solution (barren eluate).

The elution column will be filled with barren eluate and pre–heating period will commence with the eluate circulated through the primary heat exchanger to pre-heat it and the elution column to 95°C. Upon completion of pre-heating, the elution sequence will commence and additional barren eluate, from the strip solution tank, will be pumped, through a heat recovery heat exchanger and the primary heat exchanger through the elution column.

The incoming eluate, heated to 135°C, 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. Eluate emerging from the heat exchanger will be directed to the pregnant eluate tank. Pregnant eluate will be pumped through the electrowinning cells for gold recovery before returning to the strip solution tank.

A total of 28 bed volumes of strip solution will be cycled through the elution column to complete the stripping of gold from the carbon. Upon completion, heating will cease and cooling water will be injected into the circulating stream for a period of 1 hour. 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 a sump pump. Elution area spillage will be pumped to the CIP feed launder. Auxiliary equipment within the acid wash and elution circuits will include:

- Strip solution heater and ancillary equipment.
- Acid wash and elution column discharge strainers.

Carbon Regeneration

Eluted 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, using a 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 and restore 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 can be added, as required, 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. One rectifier per electrowinning cell will supply the DC current necessary 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.

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, using a pressure filter.

The gold bearing filter cake will be 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 metals 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.

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.
- Safe and strongroom.

17.2.10 Tailings Disposal

Slurry from the CIP circuit will be pumped to a carbon safety screen by the CIP tailings 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 TMF 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:

- Lime (quicklime, 90% 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 circuits.
- Fluxes for smelting.

Quicklime

Quicklime, at 90% CaO purity, will be delivered to site in bulk in 36 t road tankers or 1,000 kg bulk bags. Bulk tankers will off-load directly to the lime storage silo using a 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 54 t or 3.4 days usage.

The quicklime will be withdrawn from the silo and deposited directly onto the SAG Mill feed conveyor by a variable speed feeder.

Sodium Cyanide (NaCN)

Sodium cyanide will be delivered to site in 1 t boxes of briquettes. The bulk bag of briquettes within the box will be lifted, using the cyanide hoist, to a bag breaker above the cyanide mixing tank.

Process water will be added to the desired level in the mechanically agitated mixing tank. 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.

The required quantity of cyanide briquettes will be discharged into the partially filled mixing tank from the bag breaker. After complete dissolution of a batch of cyanide, the solution will be topped up to achieve a 20% w/v NaCN strength and transferred into the cyanide storage tank.

From the storage tank cyanide solution will be distributed, via the 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 in a bunded area serviced by a common sump pump. Spillage generated within this area will be pumped to the cyanide mixing tank or leach feed distribution box, or, alternatively, to the tailings 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, using 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 fill the caustic soda mixing tank to a predetermined level before caustic pearls are added, in a controlled manner to prevent local heating, into the agitated tank. Sufficient water will be used to achieve a mixed solution with the desired caustic concentration (20% w/v). The mixing tank will be mechanically agitated to assist with dissolution of the pearls. 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). An acid proof drum pump will be used to transfer acid, from the bulk containers, to the acid mixing and storage tank for dilution. From the acid mixing and storage tank, the hydrochloric acid dosing pump will transfer the requisite amount of dilute acid to the acid wash column.

The hydrochloric acid storage area will be serviced by a sump pump. 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, using 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 into raw water in the mixing/storage 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. 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 vendor packaged 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 a mixing/aging 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. 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 for use in the gold room; borax ($Na_2B_4O_2$), sodium nitrate (nitre, $NaNO_3$), sodium carbonate (soda ash, Na_2CO_3) and silica powder (SiO_2).

17.2.12 Water Services

The process plant will have three circuits distributing fresh water, filtered water and process water.

Run-off Water, Ponds and Water Management

Containment ponds will be provided to manage site water and contain run-off. Water from the open pit de-watering station will be pumped to the fresh water pond (FWP). Waste dump run-off will be collected within dedicated sediment ponds. From these sediment ponds, the water will be pumped to the FWP. From the FWP, water will be pumped to the raw water tank (RWT).

The FWP will be the preferred source of make-up water for the process water circuit.

Process Water

Process water will predominantly be comprise of pre-leach thickener overflow and TMF reclaim water. Process water will be stored in a 1,500 m^3 process water tank (PWT), which provides 1.7 hours of surge capacity. Provision has been made to top-up the PWT from the FWP should this be required. From the process water tank, process water will be reticulated by the process water main using duty/standby single stage process water pumps. The predominant use for process water is the mill feed dilution water.

Fresh Water and Fire Water

Fresh water, for the process plant and mining operation, will be harvested from various runoff collection ponds around the site. This will be supplemented by river water in emergency cases only. Fresh water from the various sources will be stored within the 1.5 Mm^3 FWP, which provides 35 months of water storage capacity. Water will be pumped to a raw water tank (RWT) at the plant site. The live volume of this RWT will be 908 m^3, which comprises of 288 m^3 fire water reserve and 620 m^3 raw water storage (4 hours predicted consumption). The raw water suction take-off will be elevated above the bottom of the tank to ensure a minimum volume of fire water reserve is always available in the RWT.

Raw water from the RWT will be reticulated through the plant by duty/standby raw water pumps. Significant points of use include:

- Process water make-up.
- Dust suppression (crushing area).
- Carbon regeneration.
- Flocculant dilution at the pre-leach thickener.
- Reagent make-up.
- Workshops and mine services.

Firewater will be reticulated through a fire water main. The firewater system will comprise:

- An electric jockey pump to maintain pressure in the ring main.
- An electric firewater pump.
- A diesel firewater pump should the emergency event interrupt power to the electric pump.

The electric fire water pump will automatically start on sensing a drop in fire main 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 uses require water with a low suspended solids content (mill cooling water circuit, acid wash circuit and elution circuit, gland seal water). To satisfy this need, a portion of the raw water will be treated by filtration. Filtered water will be stored in a filtered water storage tank from where it will be pumped to point of use by duty/standby pumps.

Gland water will be supplied from the filtered water storage tank by high-pressure gland water pump (duty and standby).

Potable Water

Potable water will be sourced from 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. The air system includes three plant 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 256 m^3/h of decant return water is expected to be available for recycle from the TMF to the process plant. This would satisfy most of the process water requirements with only 49 m^3/h of make-up water required from the raw water system. These flows will vary significantly due to seasonal variation of the precipitation and evaporation rates.

Fresh water consumption is estimated at 59 m^3/h. The water balance shows that there will be an excess of 77 m^3/h in an average year that would need to be decanted from or stored in the FWP. Given the large positive water balance, no extraction from the river is anticipated. The flow rates quoted are aligned with the plant nominal flows at a utilization of 92%.

17.3.2 Energy Consumption

Power for the project will be provided by an 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 peak power demand or spinning reserve required for equipment start-up.

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	23	198,524
Area 551 – Fire Protection	92	2.3	19,609
Area 552 – Sewage Treatment and Disposal	79	47	410,581
Area 604 - Crushing	397	220	1,923,085
Area 605 – Ore Handling	180	87	765,901
Area 610 - Grinding	14,487	9,643	84,475,639
Area 620 – Pre-Leach Thickener	242	105	920,177
Area 625 - Leaching & Adsorption	1,087	652	5,712,777
Area 630 – Acid wash and Elution	57	18	159,432
Area 631 – Carbon regeneration	25	2	16,031
Area 632 – Electrowinning and Refining	104	37	321,842
Area 645 – Tails Pumping	1,000	361	3,162,925
Area 650 – Reagent Preparation and Storage	129	26	229,687
Area 655 - Air Services	1,000	494	4,329,455
Area 660 - Water Services	666	265	2,319,648
Area 805 - Tailings Management Facility	15	10	89,527
Area 810 – Reclaim Water	220	87	757,740
Area 820 - Raw Water	310	88	773,158
Area 830 - Event Pond	19	1	7,709
Area 420 – On-site Power Distribution - LSP	703	198	1,730,153
Area 500 - Infrastructures and Buildings	475	143	1,253,939
Area 627- Fuels and Lubricants	100	58	509,598
Area 300 - Mining	345	158	1,386,991
Area 515 - Permanent Camp	335	142	1,247,862
Total	**22,144**	**12,868**	**112,721,989**

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)	13 sets
SAG Mill Liners	1.9 sets
SAG Mill Grinding Media	3,011 t
Quicklime	5,280 t
Sodium Cyanide	743 t
Lead Nitrate	550 t
Activated Carbon	96 t
Sodium Hydroxide (Caustic)	155 t
Hydrochloric Acid	218 t
Flocculant	41 t
Borax	4.7 t
Sodium Nitrate (Nitre)	0.63 t
Soda Ash	0.63 t
Silica Sand	2.0 t
Smelting Furnace Crucibles	6.1 units
Diesel Fuel (plant usage only)	855 m^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 PCS 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 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-energized 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 connect 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) on 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 utilizing Pulse Width Modulated (PWM) technology. The drive will be mounted in a freestanding 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

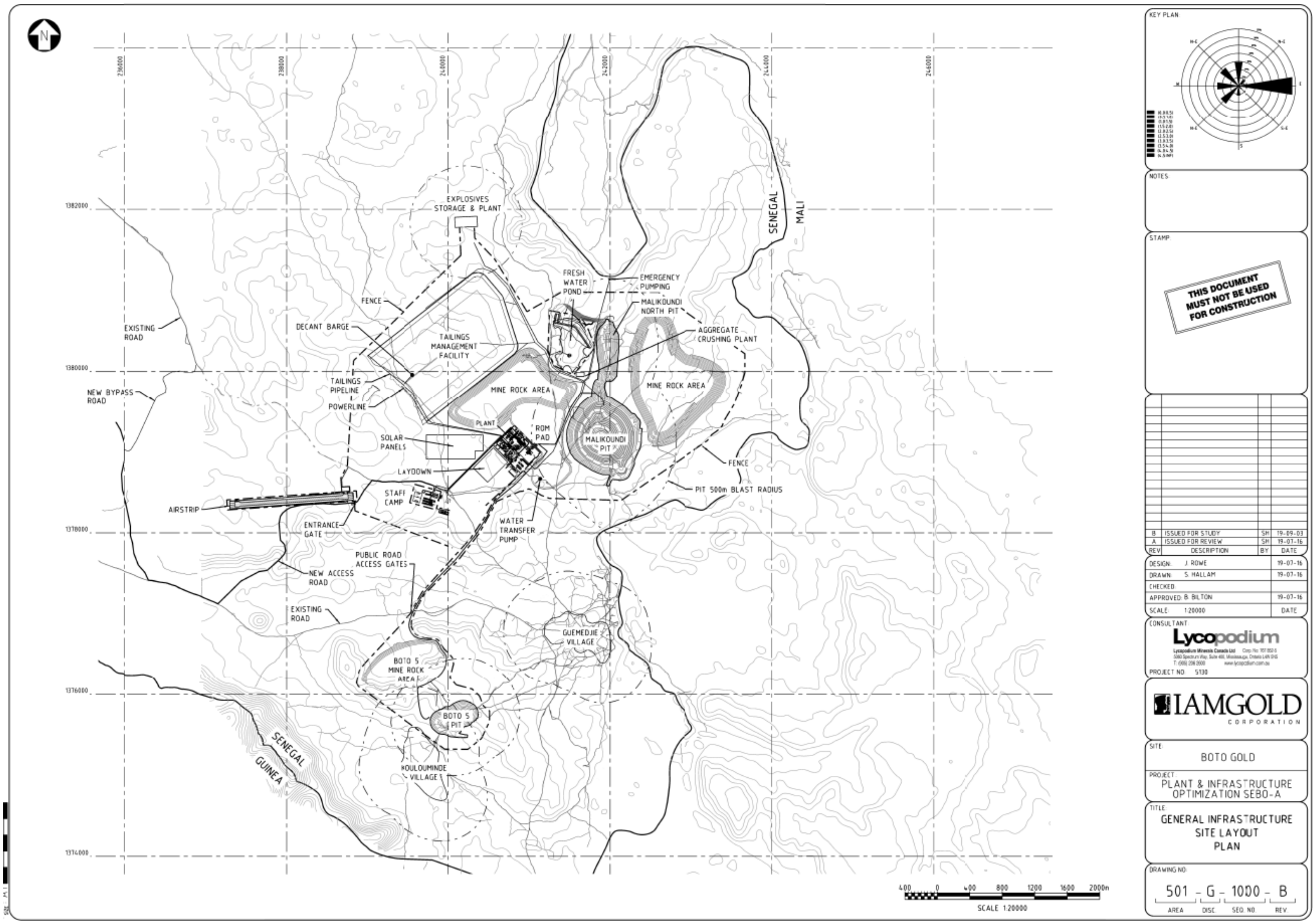
18.1 Overall Site

The overall site plan for the Project (refer to Figure 18.1) includes the main facilities including the open pit mines (Malikoundi, Malikoundi North and Boto 5), waste dumps, process plant, Tailings Management Facility (TMF), Fresh Water Pond (FWP), staff camp, airstrip and site access road. Other onsite facilities, not labelled on the figure include a power plant and bulk fuel storage are also provided. 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 pit. 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, including 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 and the airstrip. High security fencing will surround the process plant.

Figure 18.1 Boto Project 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. A new bridge currently under construction at Saroudia, funded and constructed by the government of Senegal, will be the link to site. 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 it is anticipated that the government bridge at Saroudia will enable all weather access to site.

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

A new 1400 m long airstrip will be constructed immediately adjacent to the plant site and within the fenced boundary, to enable turboprop aircraft to access the site. 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 at 11 kV to the process plant HV switchboard from which power will be distributed across the site. Power distribution to infrastructure outside the process plant will be by overhead lines. The power plant has been sized to accommodate 19.1 MW of connected load with a predicted peak load of 16.9 MW and average running load of 12.9 MW. The power plant will have 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 MW solar units.

HFO units will satisfy the average running load and diesel units will provide supplementary power to start large drives, meet peak load requirements and for when one of the HFO units is offline for maintenance. The solar units will feed power into the system during the day with one or more of the generators providing 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 415 V, 50 Hz working voltage. The 11 kV feeder from the power plant will feed the site distribution 11 kV switchboard. For local use the 11 kV supply will be stepped down from 11 kV to 415 V at each switchroom or point of use using 11 kV/415 V distribution transformers.

The following switchrooms will be provided in the plant:

- Primary crushing area.
- Grinding area.
- Leaching and CIP area.

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 MCC or VSD panel or distribution board local to each equipment area.

The staff camp power will be supplied from a local transformer/distribution board 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 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.

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 other 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 sterilization 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 Camp

18.8.1 Staff Camp

A 209-bed camp will be located west of the process plant. The camp will be designed and constructed by a building contractor and catered and managed by a separate provider.

The camp will be erected prior to the main plant construction period and during construction will be used by the owner's team, EPCM contractor's team, contractor superintendents and vendor representatives.

During operations the camp will provide accommodation for staff not originating from the local area. The catering provider will also provide meals for all personnel during the day.

The camp will be fenced and consist of:

- One accommodation block for senior personnel with 17 bedrooms.
- Four accommodation blocks for junior personnel with 32 bedrooms in each block and ensuite bathrooms.
- One accommodation block for contractor and visitor overflow with 32 rooms and 64 beds, with ensuite bathrooms.
- Guard house.
- Kitchen and dining hall building.
- Recreation facilities.
- Laundry.

The 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 buildings 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 buildings 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 buildings 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.

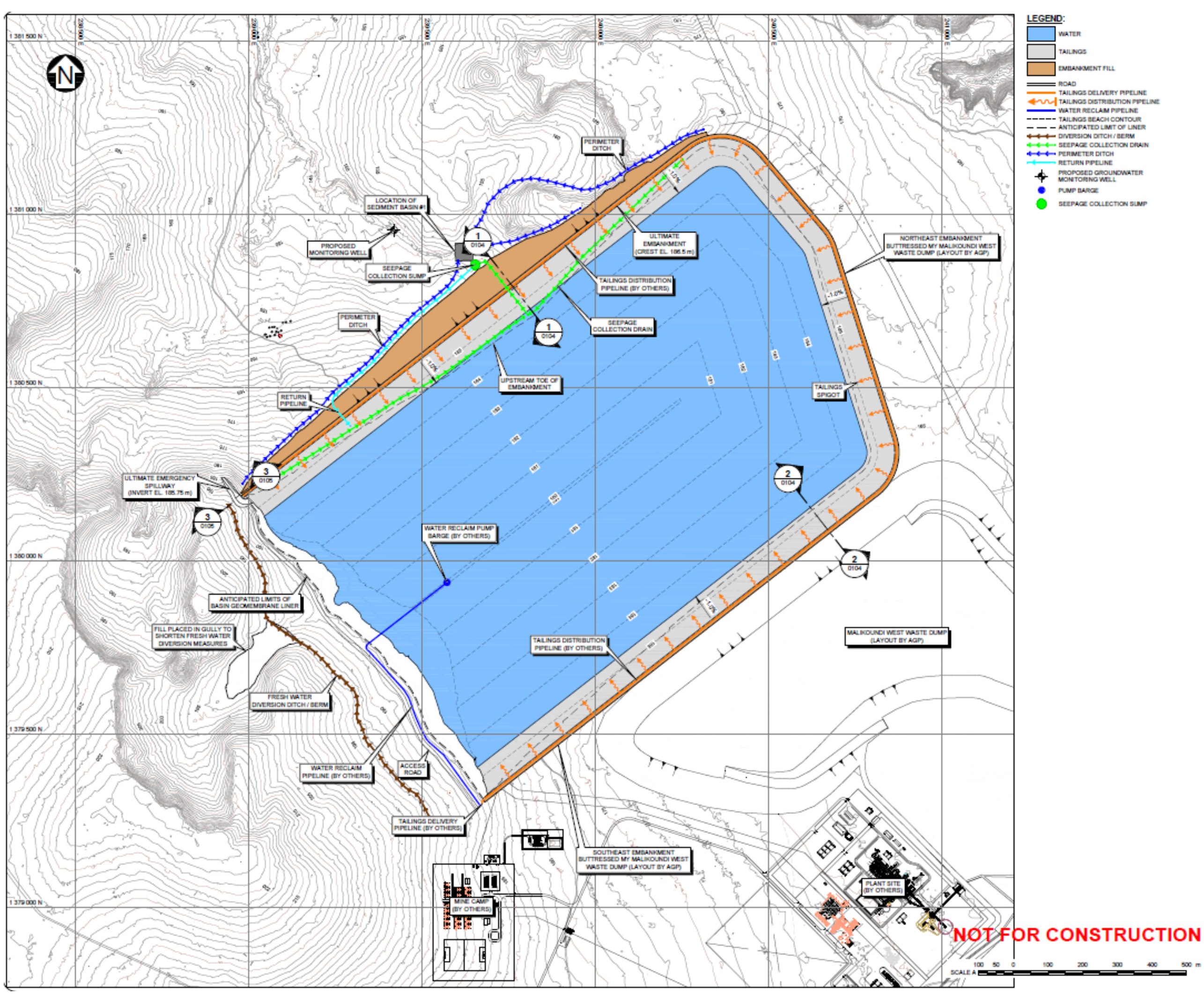
18.10 Tailings Management Facility

18.10.1 Introduction

KP has completed a feasibility level design for the TMF. The TMF will provide secure storage for tailings and process water, and protect groundwater and surface waters during operations and post closure. The TMF has been sized to permanently store 29.0 Mt of tailings, or 21.4 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.

  Achievable field densities 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 safe and 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 considered the following requirements:

- 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

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 four subsequent embankment raises (i.e. Stage 2 to Stage 5) 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

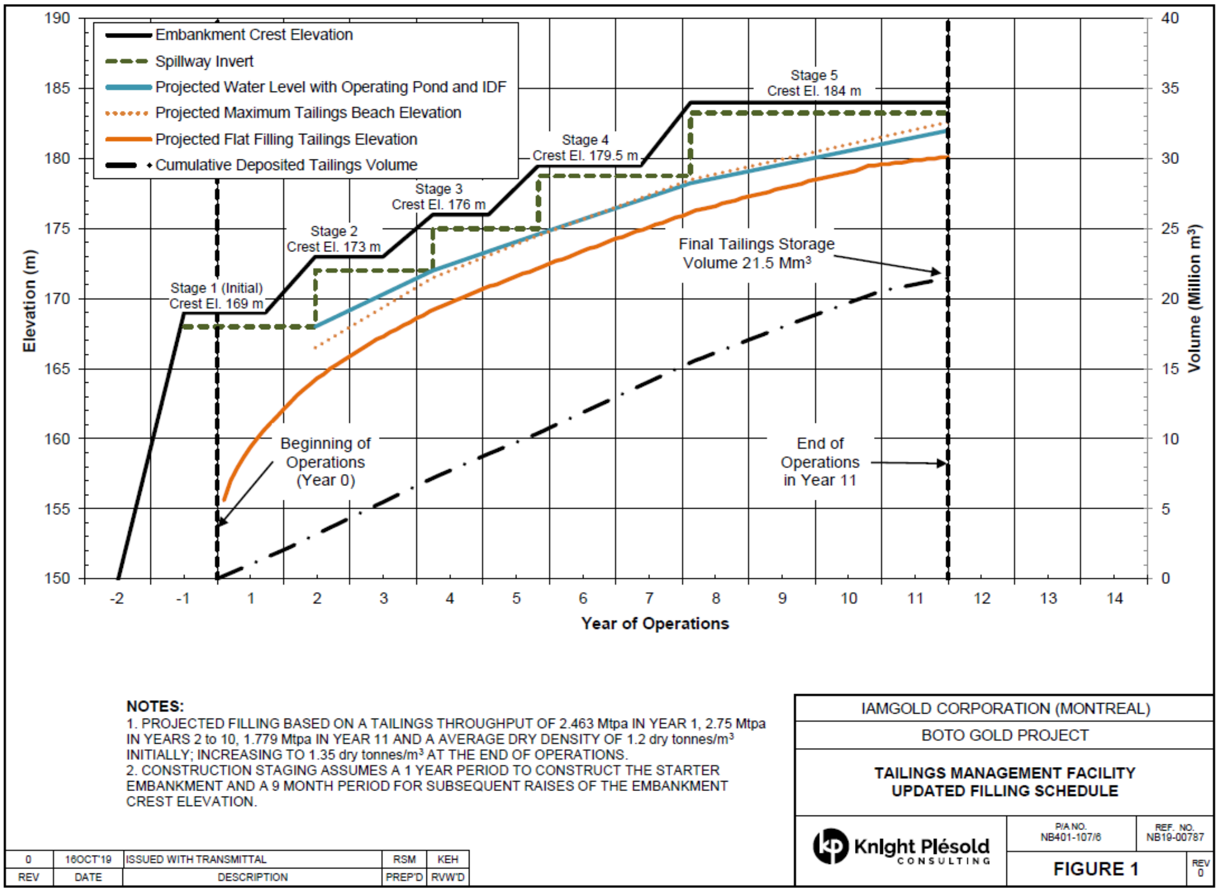
The capacity and filling schedule of the TMF are based on the following:

- 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 tonnage 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

Ferricrete/duracrust and colluviual deposits are typically present near surface within the TMF basin foundation. The TMF embankment and basin footprint will be cleared, grubbed, and stripped of surface soils and unsuitable materials to expose stiff to hard ferricrete/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 19 m along the plateau for the East Embankment that runs along the northeast and southwest side of the basin to 34 m in the valley along the Northwest Embankment.

The fill for the embankments will be obtained from the open pit waste materials. 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 overlying HDPE geomembrane.

- **Transition - Soil (i.e. laterite) for Embankment Fill (300 mm lifts) -** The Transition 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. ferricrete) 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 envelope.

- **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 five 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 5 Northwest Embankment, surface movement monuments installed every 250 m along the embankment crest following the completion of each raise of the Northwest Embankment (73 in total) and 3 slope inclinometers installed in the Stage 5 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 5 (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 and 5. The FoS targets 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 5 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 typically several orders of magnitude less than the leakage due to geomembrane defects. The total estimated leakage for the Stage 5 TMF was calculated for the following:

- Leakage through the basin floor.
- Leakage through the basin side slopes and embankment for Stage 5.

The total seepage from the TMF is estimated to be approximately 6.5 m^3/hr.

Tailings Management

Tailings will be pumped as a conventional slurry from the plant site to the TMF via pipline(s) 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 tailings beach in a location, 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 47% solids by weight. The tailings slurry will be pumped 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 80,000 m^3 to 1.9 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.
- At no time during operations, will there be an excess of water in the TMF that would need to be treated and released to the environment. The facility will be a zero-discharge facility.

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 1,150,000 m^3 for Stage 5. 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 southwest abutments of the Northwest 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 5 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 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 is a water retaining structure designed to store fresh water for operational water needs of the mine and process plant. 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 FWP has been designed to provide approximately 1.5 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.

The design basis and operating criteria for the FWP have been adopted based on accepted national and international standards for mining dam design and operation, available site information, and the operational requirements developed in consultation with Lycopodium, AGP Mining Consultants 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 an Extreme dam hazard classification based on the foreseeable consequences.

18.11.3 Embankment Section and Basin Lining System

General

The general arrangement of the FWP consists of a cross valley type zoned earthfill embankment and upstream water retaining basin. The FWP embankment and buttress will be constructed from waste overburden (i.e. mix of ferricrete and laterite) from the open pit mining operations (i.e. Malikoundi Pit and/or Malikoundi North Pit), which will be hauled, placed and compacted in the embankment.

Embankment Geometry

The embankment of the FWP will be constructed to an elevation of 151 m (approximately 27 m in height) with a 10 m wide crest, 3H:1V downstream slope and a 3.5H:1V upstream slope. A 20 m wide buttress constructed to an elevation of 135 m is required along the downstream toe to stabilize the downstream slope. The upstream slope includes a 3 m internal bench at an elevation of 144 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 embankment to collect any potential seepage from the embankment or basin of the FWP and safely route it to the downstream embankment toe.

Lining System

The FWP basin and upstream embankment face will be lined with a geosynthetic lining system to minimize 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 any seepage.

18.11.4 Water Management

The primary function of the FWP is to store fresh water during the initial construction and commissioning of the Project and subsequent mining operations. Fresh water, make-up reclaim water (i.e. when not all of the required reclaim water is available from the TMF), and water for dust suppression along haul and access roads (when required) will be taken from the FWP. During operations, fresh water will be obtained from direct precipitation on the pond (when present) and from diverting stormwater runoff from the surrounding catchment area, as well as from waste dumps and open pits and groundwater inflow into the open pits that is sent to the FWP. In addition to providing water for the mining, process plant and dust suppression, the facility will serve as a settling basin to reduce the total suspended solids in the runoff reporting to it, prior to any discharge to the environment (KP, 2018c).

The FWP has been sized to manage runoff originating from the adjacent upstream catchment areas as well as collected runoff and groundwater pit inflow pumped to it from other project areas (when required for use in the mining process and for dust suppression). Any excess water above the maximum operating pond level will be released through the Overflow Spillway. The FWP includes an Overflow Spillway to route excess water during normal operating conditions and extreme precipitation events through the FWP basin. The Overflow Spillway consists of a two-staged trapezoidal channel through the western abutment of the FWP embankment which will discharge away from the downstream toe of the embankment. The low flow portion of the two-stage channel has been designed to discharge excess water from normal operating conditions up to and including the EDS. The high flow portion has been 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 $1,350/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.

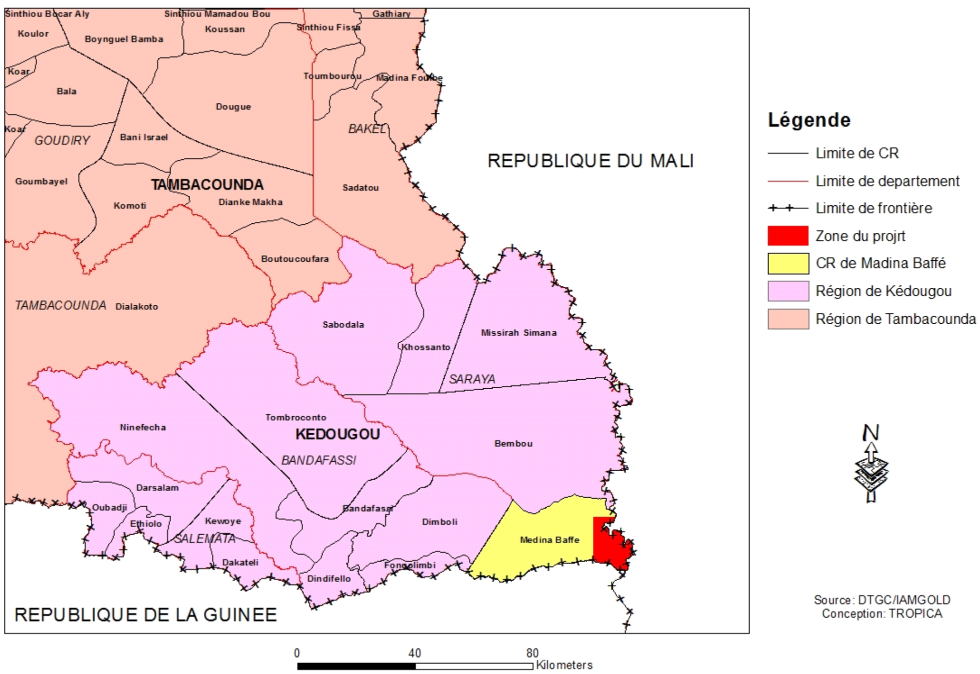
19.2 Contracts

There are no material contracts or agreements in place as of the effective date of this report that impact the Boto Project. Refining contracts are put in place with well recognized international refineries. Refining 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 refining contract in place for the sale of doré prior to start of production is not seen as a risk to the Project. IAMGOLD sells refined doré (gold bullion) from its producing mines on the open market based on prevailing market prices. It has been assumed that gold bullion refined from Boto doré production, will follow a similar process.

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 Environment Code regulates the environmental impact assessment by the provisions presented in articles L48 to L54 and articles R38 to R44 of Decree No. 2001-282 of April 12, 2001 and various additional regulatory documents, in particular:

- Order No. 009468 of 28 November 2001 regulating public participation in the impact study.

- Order No. 009469 of 28 November 2001 on the organization of the functioning of the Technical Committee.

- Order No. 009471 of November 28, 2001 defining the content of terms of reference of the impact studies.

- Order No. 009472 of November 28, 2001 defining the content of the impact study report.

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 Environment 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.

An environmental compliance certificate was issued by the Senegalese Governement in October 2018 followed by a decree in November 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

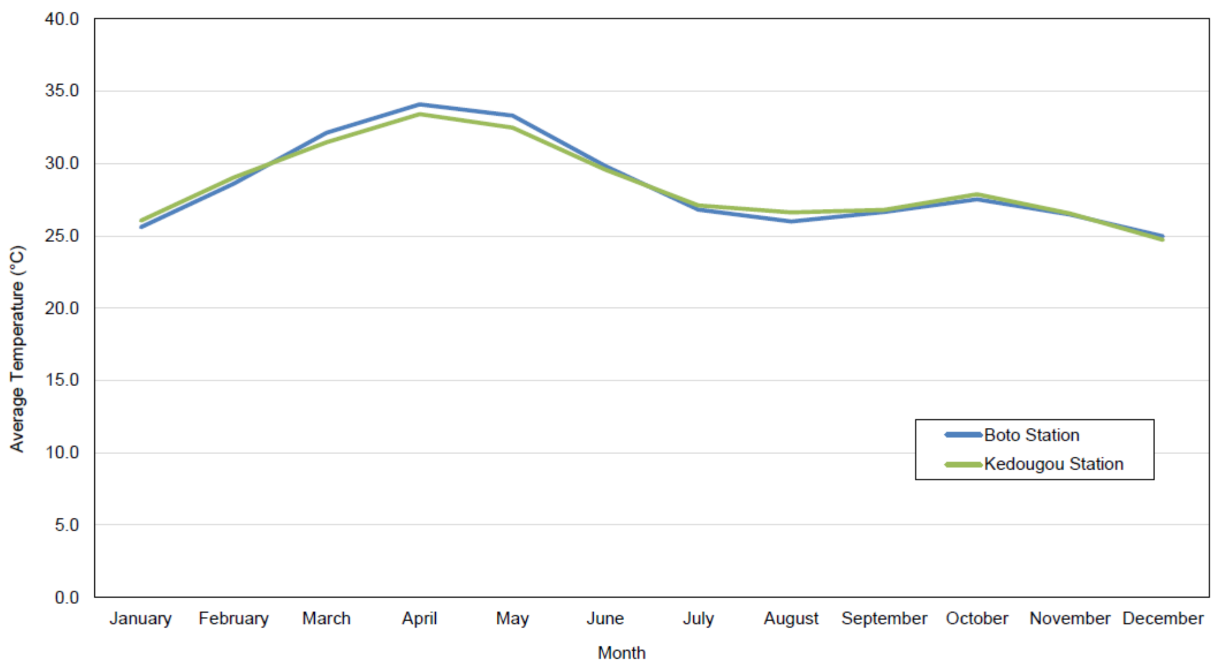
To obtain climate data specific to the Project's site, a latest generation meteorological station, operated by IAMGOLD, was installed in early 2016. Approximately three years of climatological data have been recorded by the station, starting in February 2016. There are several climate stations in the general Project area, with the datasets for Kédougou and Saraya considered to be the most relevant to the Project site.

For the purposes of the study, IAMGOLD acquired the most recent data from the National Agency of Civil Aviation and Meteorology (ANACIM) for the Kédougou and Saraya stations. Temperature, relative humidity, wind and insolation were obtained for Kédougou for the period from 1985-2017.Evaporation and precipitation data have longer periods of record (1970-2018 (Kédougou) and 1941-2018 (Saraya)/1950-2017 (Kédougou), respectively). 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.

Temperature

The analysis of monthly temperatures recorded at Kédougou for the 1985-2017 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. Temperatures recorded at the Boto site station show consistent results with temperatures recorded at Kédougou (see Figure 20.5). No temperature data was available for Saraya at the time of this study.

Figure 20.5 Monthly Temperature Distribution for Kédougou (1985-2017) and Boto Stations (2016-2019)



Notes:

1. *Data obtained from the Boto station for the period from January 2016 through March 12, 2019.*
2. *Data obtained from the Kédougou station for the period from January 1985 through October 2017. No data was provided for 2015. Average temperature was taken as the average of the maximum and minimum temperatures each month.*

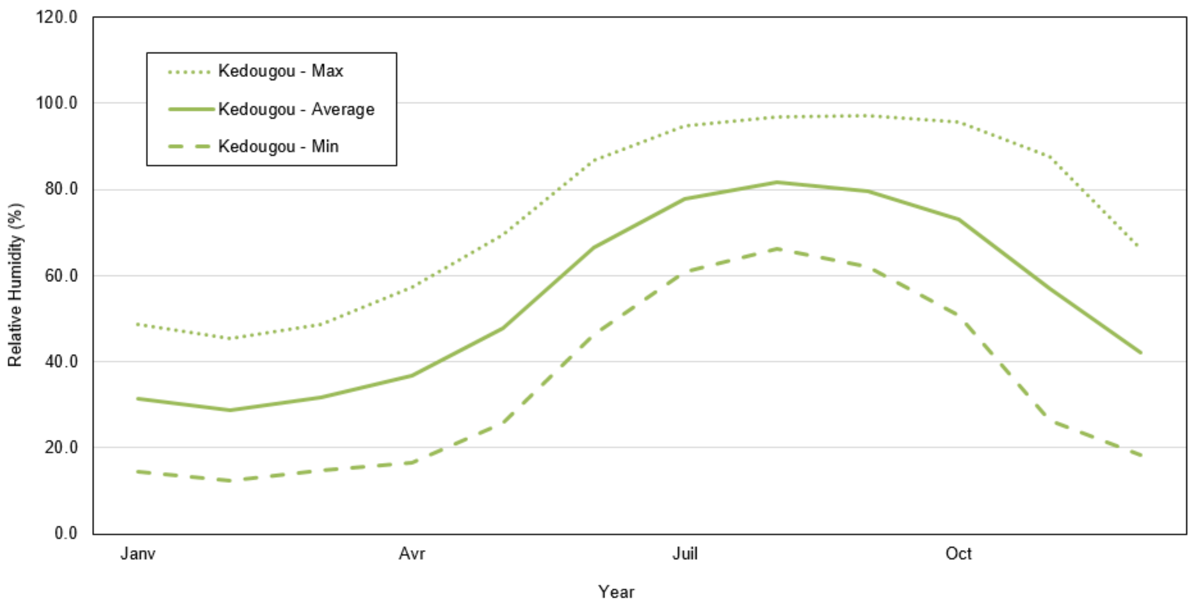
Kédougou's and Boto's average annual temperature is 28.5°C, and the annual temperature range is 8°C to 9°C, respectively, on average. Diurnal variations for Kédougou are between 14°C (April) and 17°C (December). Kédougou's temperature extremes during the 1985-2017 period are 42.4°C (April 2017) and 14.2°C (December 2002 and January 2003).

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.

For the data recorded at Kédougou, average monthly relative humidity is at its highest in August (81.5%) and at its lowest in February (28.8%). Maximum relative humidity is at its highest in September (97%), on average, and at its lowest in February (45.3%). As for minimum relative humidity, it varies between 66.3% in August and 12.4% in February, on average.

Figure 20.6 Monthly Relative Humidity (%) in the Kédougou Region (1978-2007)



NOTES:

1. *Data obtained from the Kédougou station for the period January 1985 through October 2017.*
2. *Average relative humidity for Kédougou station was taken as the average of the maximum and minimum values recorded each month.*

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 generally does not exceed 3 m/s. But the wind can reach speeds of more than 4 m/s in April and May, sometimes decreasing to less than 2 m/s in August and September.

Insolation

Insolation is the factor that directly affects temperature. In the region studied, over the period of 1985 to 2017, average insolation is at its highest in March (9.4 h/d) and April (9.6 h/d), and drops to its lowest in August (3.6 h/d), September (3.9 h/d) and July (4.2 h/d). The monthly average rarely exceeds 10 h/d, especially during the dry season.

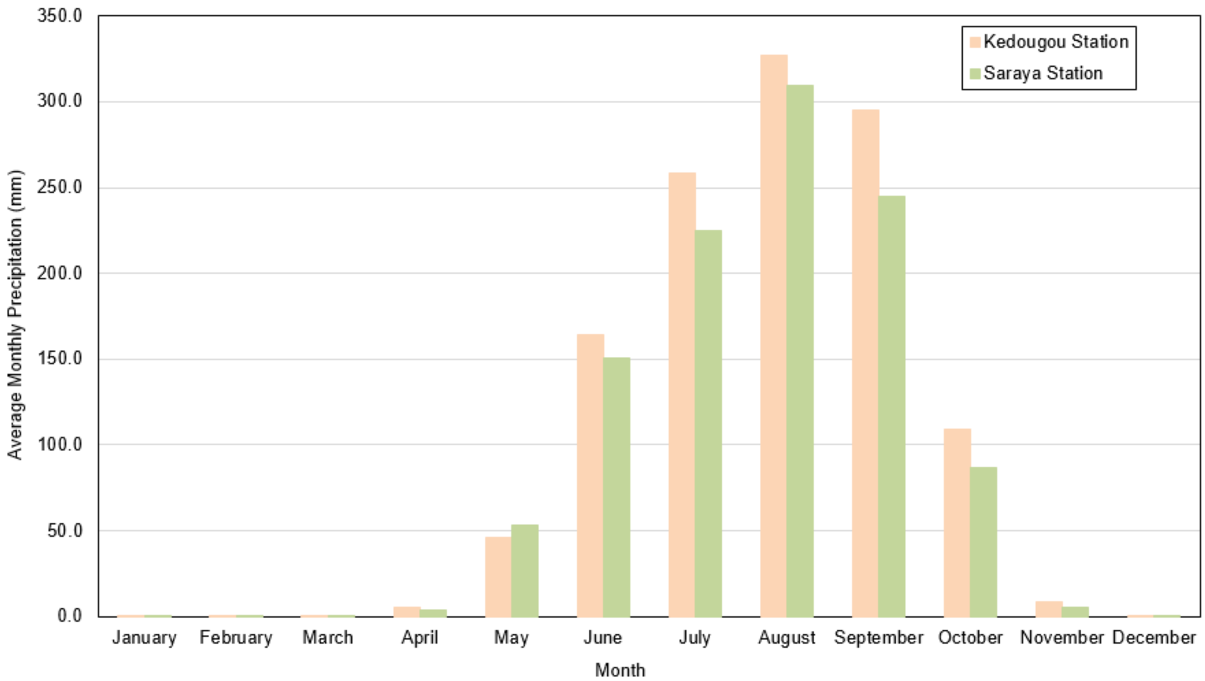
Evaporation

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 1970 to 2018, the monthly average for evaporated water varies similarly to insolation and temperature. It is at its highest in March with 218 mm and at its lowest in August and September with 35 mm. The annual average during this period is 1,461 mm.

Rainfall

Rainfall in the region hosting the permit is distinctive of the Sudanese zone with a two-season regime: a humid season from May through October, and a dry non-rainy season from November to April. Rainfalls are generally 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 rainfall varies between 780 mm and 2,160 mm for Kédougou, and between 380 mm and 2,180 mm for Saraya. The average annual rainfall for Kédougou for the period of 1941-2018 is 1,216 mm, while for Saraya for the period of 1950-2017 is 1,080 mm. The highest annual rainfall recorded at Kédougou was in 1954 (2,159 mm), and the lowest, in 2007 (787 mm). The highest annual rainfall recorded at Saraya was in 1951 (2,183 mm), and the lowest, in 2000 (382 mm).

Figure 20.7 Average Monthly Rainfall Distributions



NOTES:

1. *Data obtained from the Kédougou station for the period from January 1941 through December 2018.*
2. *Data obtained from the Saraya station for the period from January 1950 through December 2017 (excluding the years 1993, 1996, 2002, 2004 and 2016 because of insufficient data).*

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

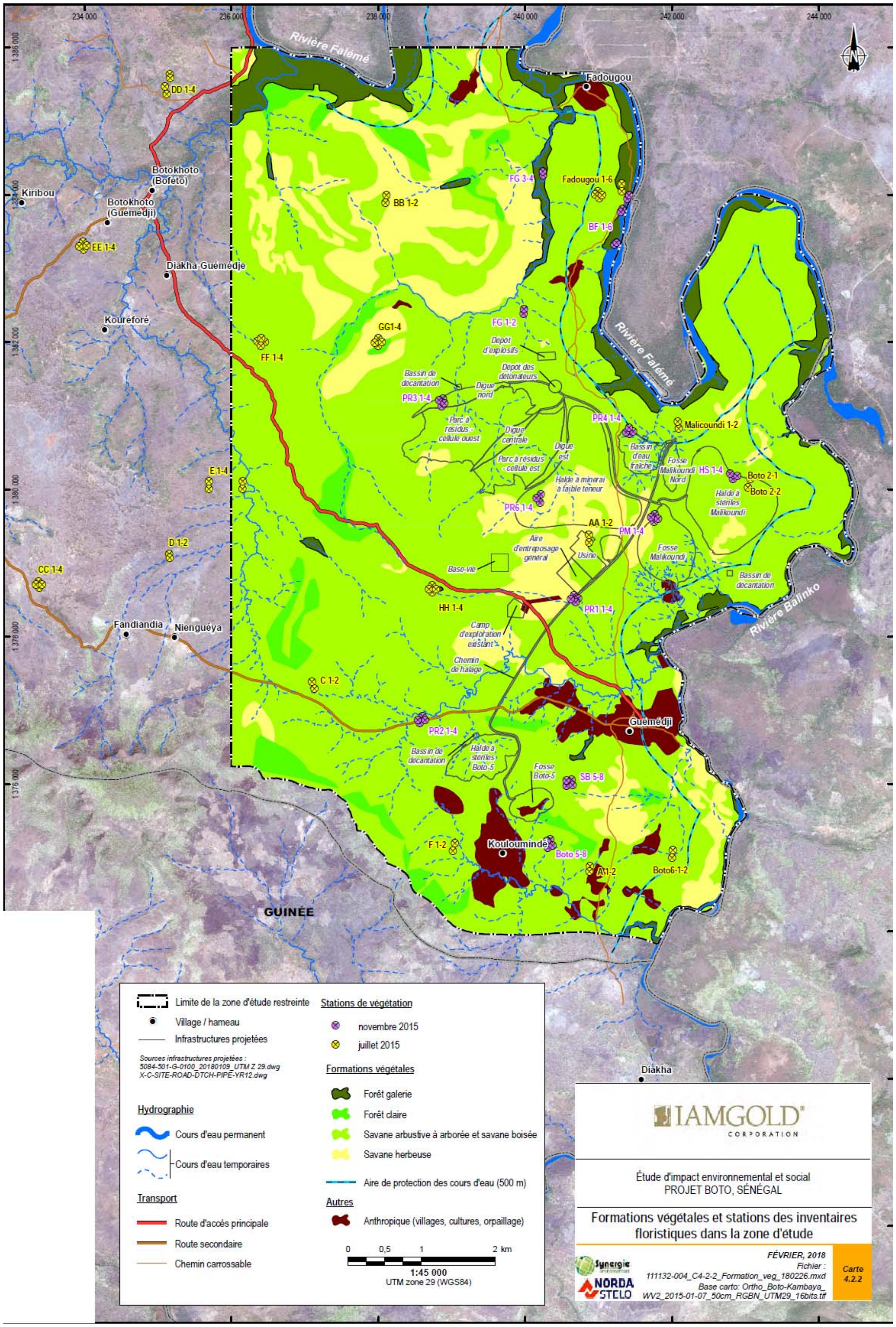
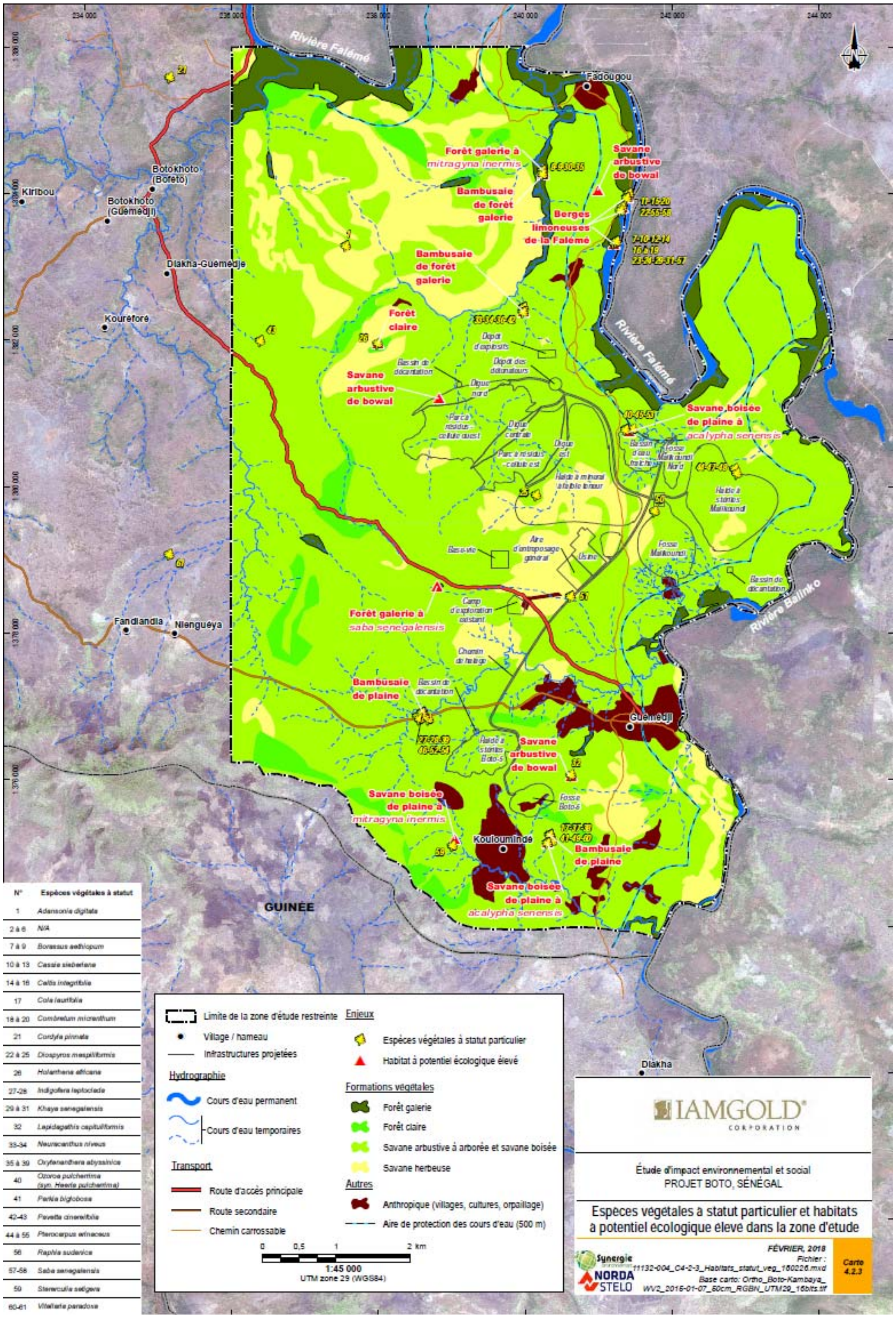


Figure 20.9 Locations where Special Status Plant Species were identified



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.

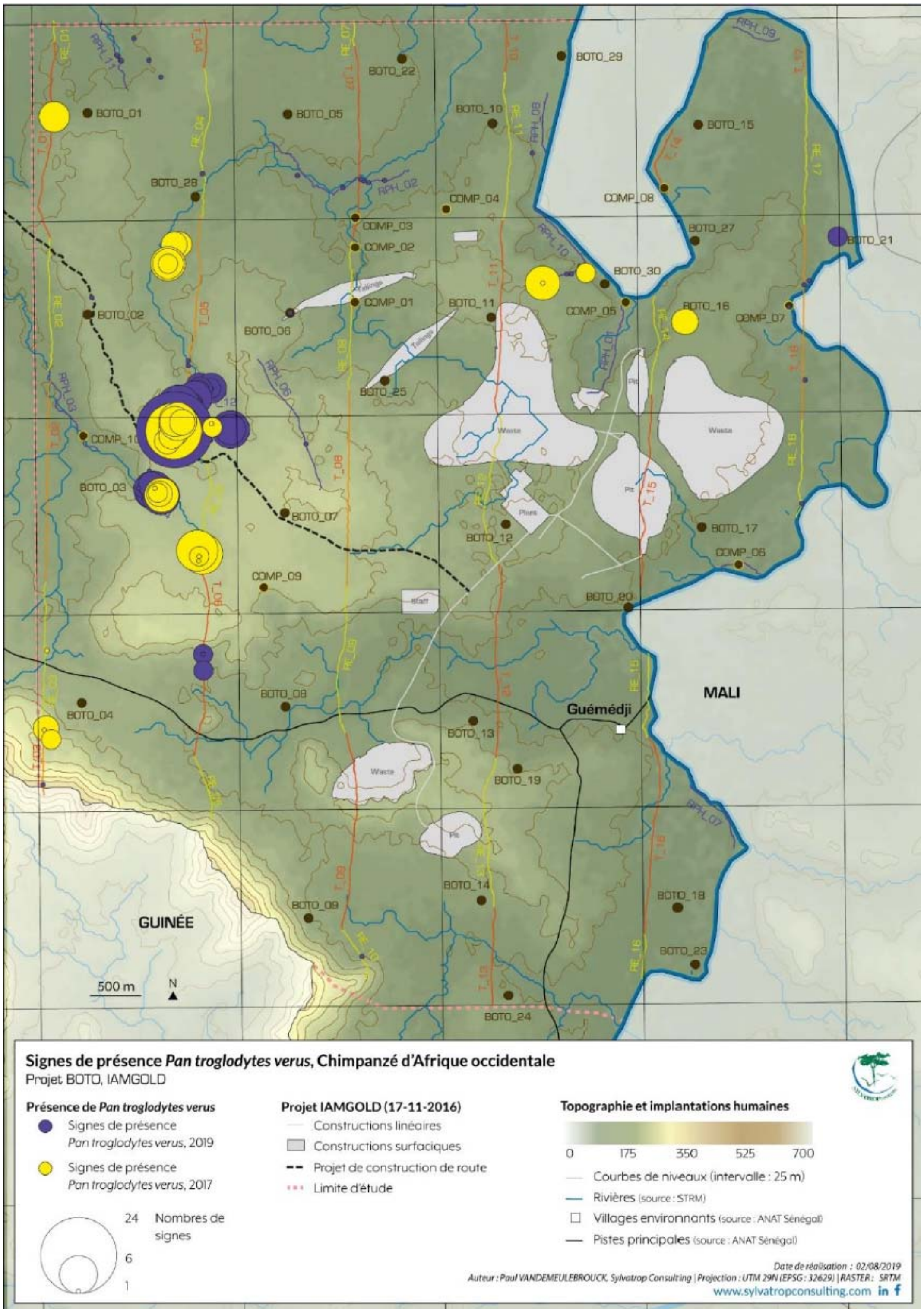
According to IUCN (2016), Senegal has 187 mammal species, including sea mammals (whales, dolphins, manatees). Out of those species, one is extinct in the wild (scimitar oryx), one is critically endangered (dama gazelle), four are endangered, twelve are vulnerable and five are near threatened, based on 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 made in 2017 with the local communities 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), feces, and scraps of food. Another study done in 2019 using the normal survey techniques as well as the use of camera traps showed that, whatever the season, chimpanzees mainly frequent the western part of the permit, and more particularly a narrow "forest corridor" along a west/north-east axis to the edge of the Falémé River. Groups of fresh nests found (around 210 in total) during surveys show the presence of a group of at least five adult individuals in this part of Boto permit. The signs of presence observed (numbers of nests, groups of nests, photos, etc.) and the permanent occupation of the territory by the chimpanzees are potential indications that this zone is occupied by several adult males and females, as well as by young individuals who would evolve over a large part of the mining area. However, at this stage, it has yet to be proven that they form part of a single group. The ongoing genetic analysis of 22 faecal samples collected at both ends of the area frequented by chimpanzees will allow to evaluate the right number of individuals.

The Western and Northern portions of the permit are virtually not retained to house the project infrastructures. This may be an important element in IAMGOLD's future environmental monitoring plan as it helps preserving much of the home range of chimpanzees in the area.

Figure 20.10 Representation of Land use by Chimpanzees in all Seasons (Sylvatrop Consulting data 2017/2019)

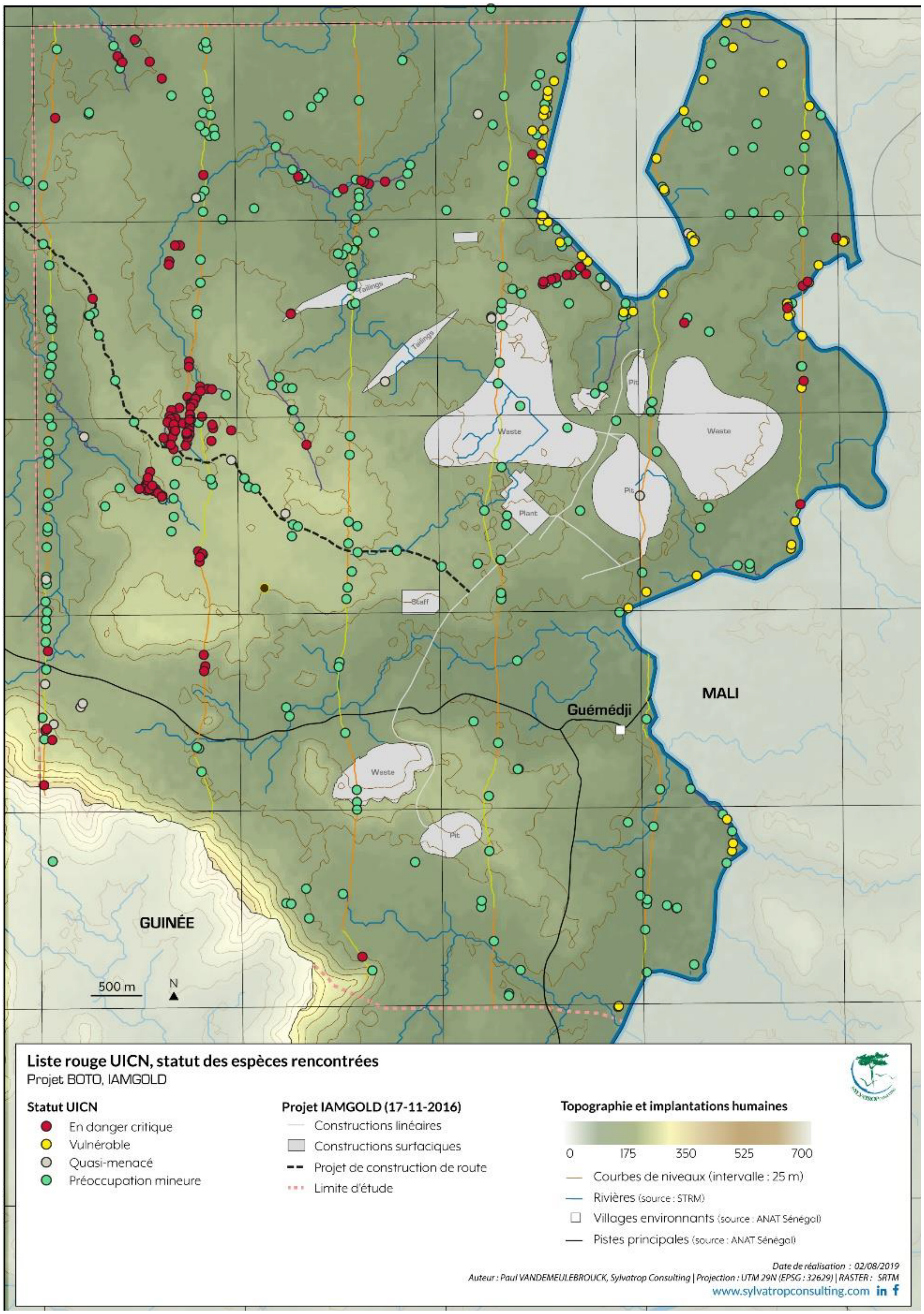


Additionally, images obtained from camera traps confirmed the presence of 19 species of mammals in the study zone, including the chimpanzee, and two other species with a precarious status: the common hippopotamus and the Guinea baboon. Based on the results from camera traps, gallery forests represent 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, 5 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).

Fieldwork in April 2019 and photographic trapping during the period April - June 2019 identified 10 new mammal species and confirmed the presence of the Gambia squirrel (Heliosciurus gambianus). Out of these species, several are listed as Threatened on the IUCN Red List: the critically endangered West African Chimpanzee (Pan troglodytes versus), the amphibious Hippopotamus (Hippopotamus amphibius) classified as Vulnerable, the Golden Cat (Profelis aurata or Caracal aurata).) classified as Vulnerable. These species will therefore be taken into account by the project as priority species for conservation. The Guinea Baboon (Papio papio) is classified as Near Threatened and will be subject to a minimum of follow-up in the future.

The figure hereunder shows the location of species encountered on the Boto permit according to their status. The red dots on the map indicate the presence of Critically Endangered Chimpanzees (CR), yellow is for Vulnerable Species (VU), which is the Amphibious Hippopotamus, and the suspicion of a Golden Cat, in gray the Guinea Baboon considered as nearly threatened (NT), and in green all other species considered as minor concern (LR). The advantage of this map is that it can easily identify critical habitats where sensitive species live so that they can easily locate them and take the necessary measures to preserve these ecosystems. It shows us in fact that the project of exploitation in itself at the level of the quarries, waste dumps, process plant and camp does not encroach on these habitats, but that the location of the tailings In addition, the current route of the road project poses problem, as it crosses the area most inhabited by chimpanzees and many other species.

Figure 20.11 Status of Species Encountered according to the IUCN Red List



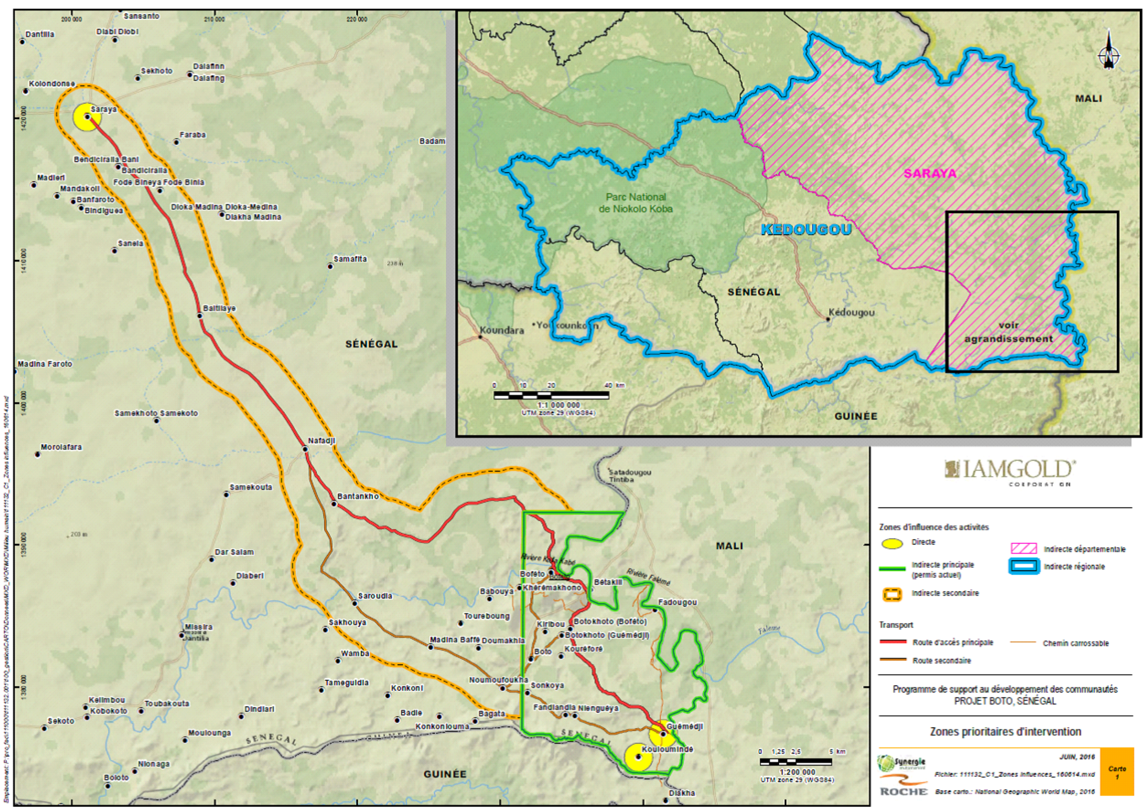
20.2.3 Human Environment

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 20.12 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 20.13 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 archeological potential was conducted as part of the ESIA. No archeological site was identified. Should archeological site be found during operations and before the start of any activities, it will be decided whether excavations or relocation are necessary in consultation with the relevant authorities and local communities.

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 characterization testwork was carried out from 2015 to 2017 on 46 samples from the Malikoundi deposit and 29 samples from the Boto-5 deposit. Four composite samples, two from each deposit, were selected for kinetic testing (humidity cell testing). Mining within the Boto-5 deposit will occur within the saprolite and within a portion of the transition zone. It should be noted that the historic geochemical studies were inclusive of these lithologies and others throughout both deposits.

Samples from the Malikoundi deposit are all considered Non-PAG, indicating that the materials tested have no acid generating potential. Samples from the Boto-5 deposit demonstrated some acid generating potential. Approximately one third of the tested samples had a sulphide content greater than 0.3%, which is considered to have acid generating potential. The majority of the samples had little to no neutralization potential. The materials from Boto-5 are more weathered than that of Malikoundi, as such it is possible that the minerals that offer neutralizing potential were consumed during the weathering processes. Additional spatial and lateral testing will be completed, especially at Boto-5, to assess the overall acid generating potential from the updated mining blocks.

Based on Inductively Coupled Plasma Mass Spectrometry (ICP-MS) testing, the waste from the Malikoundi deposit has elevated antimony, bismuth, and selenium concentrations, however neither the short-term leaching or kinetic testing had exceedances above the water quality objectives. Metal leaching is not likely a concern from the Malikoundi deposit. The waste from the Boto-5 deposit had elevated antimony, arsenic, bismuth, molybdenum, selenium, and copper from ICP-MS testing, however the leaching testing demonstrated exceedances of arsenic, copper, and nickel with minor cadmium, selenium, and zinc. The data suggests that there is a correlation between the acid generating potential and metal leaching from the Boto-5 deposit. Additional testing will be completed to characterize the overall metal leaching potential from the updated mining blocks.

If acid mine drainage is generated, then mitigation measures to reduce and or manage the drainage would be implemented, in accordance with the best environmental practices recognized in the mining industry.

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.

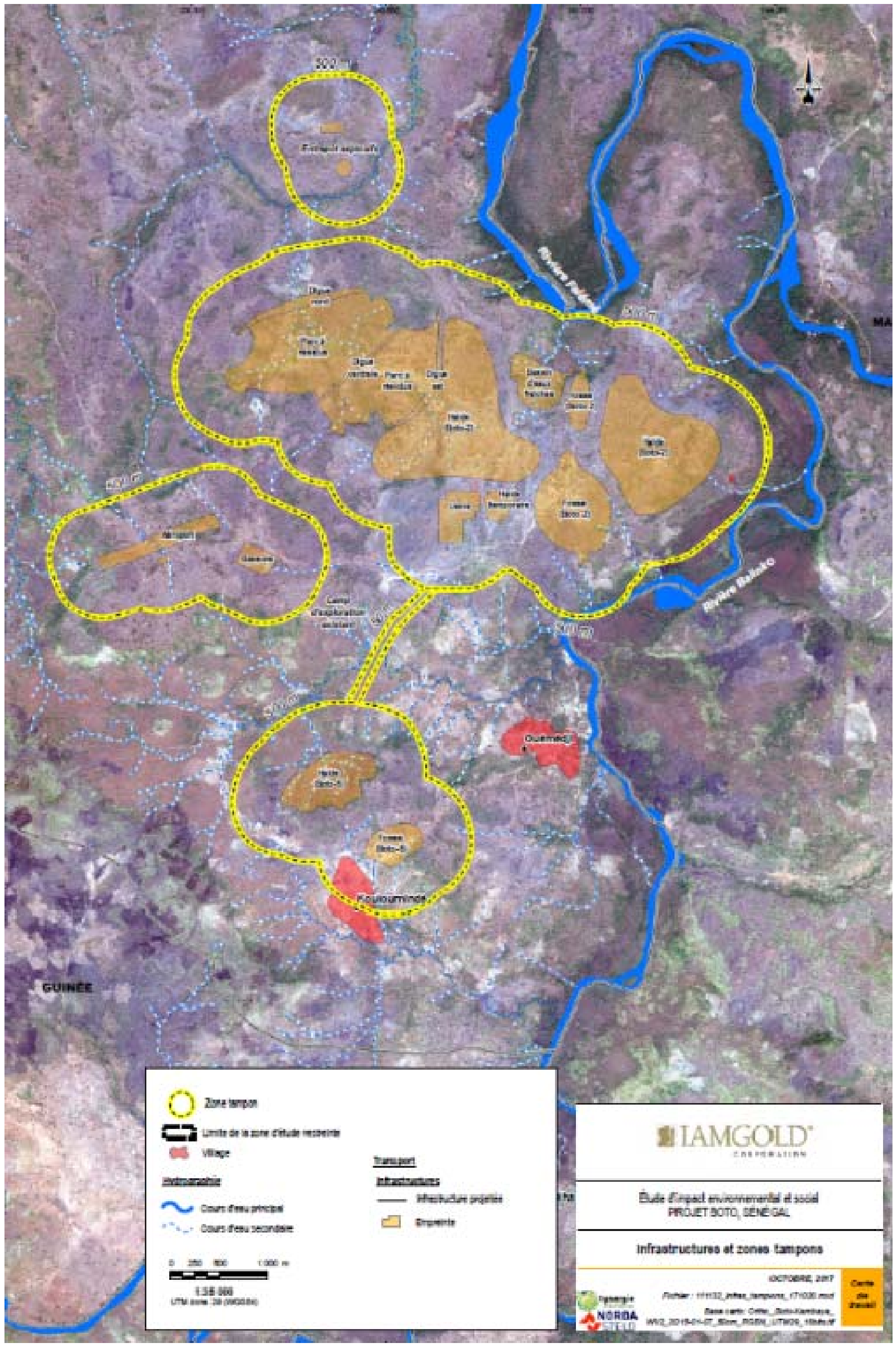
The Zero Harm vision is not only a corporate commitment, it is also the vision that guides the development of the Project and the design of its various components. Thus, the Project was developed in order to:

- 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.

Figure 20.14 Safety Radius around the Infrastructures and the Project's Components



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 within the 500 m buffer zone, IAMGOLD is investigating the best solution to mitigate the proximity of Koulouminde (see hereunder in section 20.4.2.)

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 mitigate 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.

The Project's ESIA report states that the Project is not expected to result in the loss of any permanent residences, though seasonal structures may be affected and the construction of site infrastructure will likely result in a loss of access to agricultural land and artisanal mining sites.. The Project's ESIA report commits the Project to managing any displacement impacts in accordance with national requirements and the Performance Standards of the International Finance Corporation (IFC).

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, IAMGOLDintends 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.

IAMGOLD 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.

IAMGOLD'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. The official version of the ESMP is currently under development and will be implemented during all phases of the project.

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 was March 4, 2019. A request for a mining permit or a mining concession permit was filed on October 22, 2018. The request was supported by, among other things, a FS, an ESIA, a development and start-up plan and an investment plan. The mining permit was granted to AGEM by Presidential decree on December 16, 2019.

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 IAMGOLD) 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.

All costs are expressed in United States Dollars unless otherwise stated and are based on Q3 2019 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 (Q3 2019, ±15%)

Area	M$ (Excluding Duties and Taxes)
Direct Costs	
Site General	29.7
Mining	57.3
Power Supply	2.6
Process Plant	64.7
Tailings & Water Management	14.0
Sub-Total Direct Costs	**168.3**
Indirect Costs	
Construction In-directs	23.2
Owner's Costs	63.1
Contingency	16.7
Sub-Total Indirect Costs	**103.0**
Total Initial Capital Cost	**271.3**
Sustaining Capital Cost	$68.5
Total Project Capital Cost	**339.8**

The sustaining costs for the Project are provided in Table 21.2 and include mine equipment purchases and replacement, and stage development of the TMF.

Table 21.2 Summary of Sustaining Capital Costs (Q3 2019, ±15%)

	Total	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Mining Capital ($M)	**$58.05**	$39.26	$6.04	$4.16	$1.70	$4.29	$1.61	$0.28	$0.17	$0.55
Ore Handling & Processing ($M)	**$0.16**	-	-	-	$0.16	-	-	-	-	-
TMF Capital ($M)	**$10.33**	$2.68	-	$2.22	$2.86	-	$2.56	-	-	-
Total	**$68.53**	**$41.94**	**$6.04**	**$6.38**	**$4.72**	**$4.29**	**$4.17**	**$0.28**	**$0.17**	**$0.55**

The foreign exchange rates in Table 21.3 have been used in the compilation of the estimate.

Table 21.3 Currency Exchange Rates

Currency	$USD
AUD	0.70
USD	1.00
Euro	1.20
Rand	0.07
CAD	0.77
CFA	0.001829
YEN	0.009

21.2 Mine Capital Cost

The capital costs for the mine are summarized in Table 21.4. These costs exclude contingency, duties and taxes.

Table 21.4 Capital Cost Summary – Mining

Capital Category	Initial Capital $M	Sustaining Capital $M	Total Capital $M
Mining Equipment	$27.4	$54.9	$82.3
Pre-Production Stripping	$26.4	-	$26.4
Miscellaneous Mine Capital	$3.6	$3.1	$6.7
Total	**$57.3**	**$58.0**	**$115.4**

Initial capital requirements (pre-production) are estimated to be $57.3M 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 study.

The distribution of the mining equipment 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 miscellaneous mine capital includes various separate line items in the costing, such as engineering office equipment, mine dispatch system, and the dewatering system.

The mine is scheduled to initiate pre-production stripping in Year -2 in the Malikoundi Phase 1 for a total of 13 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 1.2 Mt of ore stockpiled.

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 21.5 and estimate methodology is provided in Table 21.6.

Table 21.5 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 21.6 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
Access Roads	Quantities estimated from partial Detailed Engineering based on surface and images extracted from Google Earth.
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.

Description	Basis
Site Establishment	Requirements estimated using base rates.
Construction Facilities	Allowance based on projects of a similar size.
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 21.7.

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 21.7 Derivation of Quantities - Bulks

Classification	Quantity	Unit	Preliminary Engineering %	Estimated %	Factored %
Concrete (excluding blinding)	9,855	m^3	100%	-	-
Structural Steel	760	t	100%	-	-
Chutes/Hoppers/Bins	392	t	100%	-	-
Field Erected Tanks	608	t	100%	-	-
Piping Bulks	-	-	-	-	100%
Overland Pipeline	28.85	km	100%	-	-
Electrical and Instrumentation	-	-	62%	-	38%
Modular Buildings	5,484	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 21.8 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 21.8 Source of Pricing

Classification	Budget Quotation %	Database %	Estimated %	Factored / Allowance %
Concrete	99%	1%	-	-
Structural Steel	100%	-	-	-
Platework / Tanks	99%	1%		
Mechanical Equipment	90%	9%	-	1%
Piping - Plant	-	-	-	100%
Overland Pipeline	30%	41%	29%	-
Electrical and Instrumentation	25%	10%	15%	50%
Buildings	48%	28%		24%

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.

Access Roads quantities were estimated from partial Detailed Engineering based on surface and images extracted from Google Earth. Rates were derived from a combination of responses to Budget Quotation Requests (BQRs) from local contractors and Lycopodium Historical Data.

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 a combination of responses to BQRs from regional subcontractors with experience on this kind of work and capacity to perform the works and Lycopodium historical data.

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.

Withholding Tax applicable to the offshore technical services component of the EPCM has been calculated and 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 2019. 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) including withholding tax.

- 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.

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.
- 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 21.9.

Table 21.9 Life of Mine Operating Costs (Q2 2019)

	Total Cost ($M) from first gold pour	$/t Processed
Mining	$487	$16.76/t
Processing	$396	$13.65/t
G&A	$108	$3.70/t
Total Cash Cost	**$991**	**$34.11/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 are presented in Table 21.10.

Table 21.10 Mine Operating Costs ($/t Mined)

Operating Category	Unit	Year 1	Year 5	Year 8	Years -2 to 11 Average Cost
Salaries	$/t mined	0.24	0.30	1.06	0.33
Drilling and Blasting	$/t mined	0.23	0.65	0.85	0.51
Loading and Hauling	$/t mined	0.46	0.67	0.78	0.56
Support	$/t mined	0.10	0.13	0.21	0.12
Grade Control	$/t mined	0.02	0.02	0.13	0.03
Contract Services	$/t mined	0.37	0.49	1.22	0.53
Total	$/t mined	1.43	2.26	4.24	2.07

The mine operating costs have been estimated from first principles based on equipment hourly operating costs, equipment usage models and productivity assumptions. The average LOM operating cost is estimated at $2.07 /t mined, which includes costs associated with re-handling from stockpiles.

Labour costs for the various job classifications were developed by IAMGOLD. These rates 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 period from Years 3 to 6 is the peak in manpower. The manpower requirements are presented in Table 21.11.

Table 21.11 Mine Manpower Requirements

Manpower		Yr -1	Yr -2	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11
Operations	Staff	2	11	11	11	11	11	11	11	11	11	7	7	7
	Hourly	0	90	168	189	207	207	207	207	177	90	21	21	21
Engineering and geology	Staff	5	52	52	52	52	52	52	52	52	49	12	12	12
	Hourly	0	15	15	15	15	15	15	15	15	15	0	0	0
Maintenance	Staff	1	20	20	20	20	20	20	20	20	20	4	4	4
	Hourly	0	18	36	40	43	43	43	43	38	18	4	4	4
Total	Staff	8	83	83	83	83	83	83	83	83	80	23	23	23
	Hourly	0	123	219	244	265	265	265	265	230	123	25	25	25
Total		8	206	302	327	348	348	348	348	313	203	48	48	48

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 Q4 2019 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:

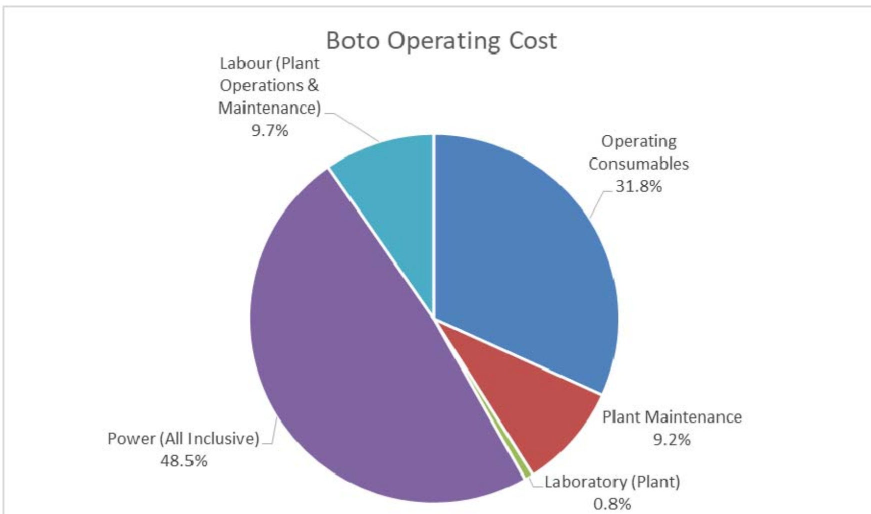
- All taxes and import duties.
- Any impact of foreign exchange rate fluctuations.
- Any business interruption costs.
- Any escalation beyond the date of the estimate.
- First fill and opening stocks costs (included in the capital cost estimate and financial model).
- Tailings storage, rehabilitation 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 in Figure 21.1:

1. Operating Consumables.
2. Plant Maintenance.
3. Power.
4. Plant Laboratory.
5. Process Plant Labour (Operation and Maintenance).

Figure 21.1 Boto Operating Cost



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 21.12. 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.

- Average fuel costs of CFA 385.03/L for diesel and CFA 305.52/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 21.12 Consumables Cost Summary

Area	Consumption Rate	Consumables Cost $/y	Consumables Cost $/t
SAG Mill Liners	0.21 kg/t	$2,129,288	$0.77
SAG Mill Balls	1.10 kg/t	$3,556,763	$1.29
Sodium Cyanide	0.27 kg/t	$1,706,835	$0.62
Quicklime	1.92 kg/t	$2,112,000	$0.77
Other Consumables and Reagents		$3,224,913	$1.17
Total		**$12,729,800**	**$4.63**

Plant 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 supply costs only and excludes installation 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 $3.7 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 21.13.

Table 21.13 Process Plant Maintenance Cost

Area	Maintenance Cost $/y	Maintenance Cost $/t
Process Plant spares	$2,293,254	$0.83
Mobile equipment lease and maintenance	$532,906	$0.19
Maintenance systems	$177,000	$0.06
Contracted maintenance	$688,655	$0.25
Total	**$3,691,815**	**$1.34**

Power

Electricity is generated on site. The power unit cost for the first five years of operation of $ 0.172/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 $0.163/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 21.14. The estimated annual power cost for the site during the initial years of operation, including infrastructure and the permanent camp.

Table 21.14 Site Power Cost by Area

Area	Average Power (kW)	Annual Power Consumption (kWh)	Total Annual Power Cost ($)	Cost, $/t
Grinding	9,643	84,475,639	14,557,157	5.29
Leaching/Adsorption	757	6,632,954	1,143,015	0.42
Tailings Disposal	361	3,162,925	545,047	0.20
Other Processing	1,334	11,685,506	2,013,690	0.73
Site Power	772	6,764,965	1,165,764	0.42
TOTAL	**12,868**	**112,721,989**	**19,424,674**	**7.06**

The overall average power consumption is estimated at 12,868 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 $3.881 M or $ 1.41/t ore.

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 21.15 below summarizes the estimated annual Plant operating cost linked to the mine plan.

Table 21.15 Annualized Plant OPEX by Year

Year	Tonnes Treated	% Saprolite and Transition	Grinding Steel Cost ($/yr)	Plant Power Cost ($/yr)	Total Annual Cost $ /y	Total $/t
1	2,550,000	30.1	2,970,348	16,068,618	34,981,457	13.72
2	2,750,000	30.0	3,205,130	15,954,080	35,642,643	12.96
3	2,750,000	28.1	3,267,363	16,100,628	35,756,620	13.00
4	2,750,000	10.1	3,899,224	17,714,380	37,691,672	13.71
5	2,700,000	0.0	4,173,930	18,176,746	38,185,632	14.14
6	2,700,000	0.4	4,160,163	18,824,173	38,647,600	14.31
7	2,700,000	11.3	3,788,429	18,024,607	37,508,673	13.89
8	2,700,000	0.0	4,173,930	18,878,585	38,412,422	14.23
9	2,750,000	29.1	3,234,200	16,929,733	35,441,517	12.89
10	2,700,000	0.0	4,173,930	18,902,155	38,026,388	14.08
11	1,990,136	7.5	2,887,055	12,433,939	26,173,510	13.15

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 $3.70/t processed.

22.0 ECONOMIC ANALYSIS

22.1 Introduction

The following section presents the economic assessment of the Boto Project. The cash flow was completed and prepared by IAMGOLD. The cash flow is estimated on the project base. No adjustment for future increases in costs is planned, no provision is made for inflation nor increase in gold price, which means that the financial analysis remains in constant dollars. Cash flows were estimated on an annual basis and excludes debt financing (except interest tax impact) neither a potential financing on equipment of $23M.

A discount rate of 6% was applied to determine the present value of the project. The internal rate of return, the net present value, the All-In Sustaining Costs (AISC) and the payback period were used to determine the economic viability. All the amounts are in constant 2019 dollars and expressed in United States dollars, unless otherwise indicated.

22.2 Assumptions

The cashflow presented in this document was prepared according to the assumptions in Chapter 21 and the following assumptions stated below:

22.2.1 Metal Price

The price of gold considered for the revenue base is at $1,350/oz. It is aligned with the IAMGOLD's long-term corporate assumptions. No price variation was considered for this model.

22.2.2 Fiscal Regime and Income Taxes

The corporative fiscal regime is specified by the Mining Code 2003-36 and in the Mining Convention, and the 2019 amendment to the mining convention between AGEM and the Government of the Republic of Senegal, which states the advantages during construction and operation phases.

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. No other shareholder was considered.

According to the mining convention, during the investment period up to commercial production, the company benefit from an exoneration of all import taxes and duties (except statistical fee). Those exonerations starts of the delivery date of ''permis d'exploitation'' and are valid for a maximum of four years.

Finally, a seven years total tax exoneration is applied starting the first year of construction (Year -2) of the project, which is compliant with the mining convention. This exoneration applies on all corporate income tax as well as VAT on good and services.

In the financial model, the corporate taxes are at 30% and applied on taxable benefits starting year 8 of project starts (equivalent Year 6 of commercial production) of production on the 5 years tax exoneration. The corporate taxes for this project stands at $40.3 M.

Previous costs related to the valuation of the project are estimated at $64.4M and are considered in the financial analysis in terms of future tax depreciation only.

22.2.3 Royalties & Other Taxes

Royalties are applied to gross revenues. The following rates vary depending on prevailing gold price: up to $1,250 = 3.0%, greater than $1,250 up to $1,350 = 4.0% and higher than $1,350 = 5%. The financial model use gold price at $1,350 applying a rate of 4.0% for a total estimate of $76.9M.

Tax on labour included in operating costs and Import Custom Tax included in sustaining capital are respectively estimated at $66.0M and $35.4M.

Other Taxes as Environmental Tax, Patent Tax and Development Support Fund are estimated at $32.1M.

22.2.4 Working Capital

The working capital calculation take into consideration variation on receivables, payables, inventory along the life of mine in addition to a minimum cash requirement. An initial value of $10.0M was used in the financial model.

22.3 Financial Analysis

The life of mine capital cost required for the project is estimated at $339.8 M, with an initial capital expenditure of $271.3 M. Table 22.1 presents a summary of the production parameters that form the basis of this financial model.

22.3.1 Project Financial Summary

The scenario presented in Table 22.1 shows a robust project under the financial environment reflected in this study. The mine would operate approximately 11 years at a throughput of 2.75 Mtpa averaging an annual production of 129 koz Au.

Table 22.1 Parameters Summary

	Value
Ore milled	29.0 Mt
Total tonnes mined	247.3 Mt
Average head grade	1.71 g/t Au
Contained gold in material	1,593 koz
Total gold produced	1,424 koz
Average gold recovery	89.4%
Production life (processing)	10.7 years
Nominal annual processing rate	2.75 Mtpa

The revenue is $1,922.1M over the LOM. The operating costs over the same period are $999.1M (including import custom tax and labour tax). Initial capital expenditures of $271.3M and sustaining of $68.5M over the LOM would complete the expenditures. Royalties and other taxes account respectively for $76.9M and $32.1M. The rehabilitation and retrenchment are $21.7M excluding the salvage value of $6.6M. A pre-tax cash flow of $459.0M would be generated. Income taxes accounts for $40.3M. This would result in a net cash flow after taxes of $418.7 M. At the selected discount rate of 6%, this represents a net present value after taxes and interest expense of $218.7 M. Table 22.2 summarizes the discounted cash flow at different rates.

Table 22.2 After Tax Cash Flows (M$)

Discount Rate	After-Tax CF (M$)
0%	$ 418.7
5%	$ 244.7
6%	$ 218.7
10%	$ 135.0

22.3.2 Cashflow

Table 22.3 summaries the annual cash flow and the cumulative cash flow after tax. The payback period is three years (3) and two (2) months that would be necessary to reach the break-even and repay the investment in full. The internal rate of return is 22.6% and the average All-in sustaining cost is $842/oz.

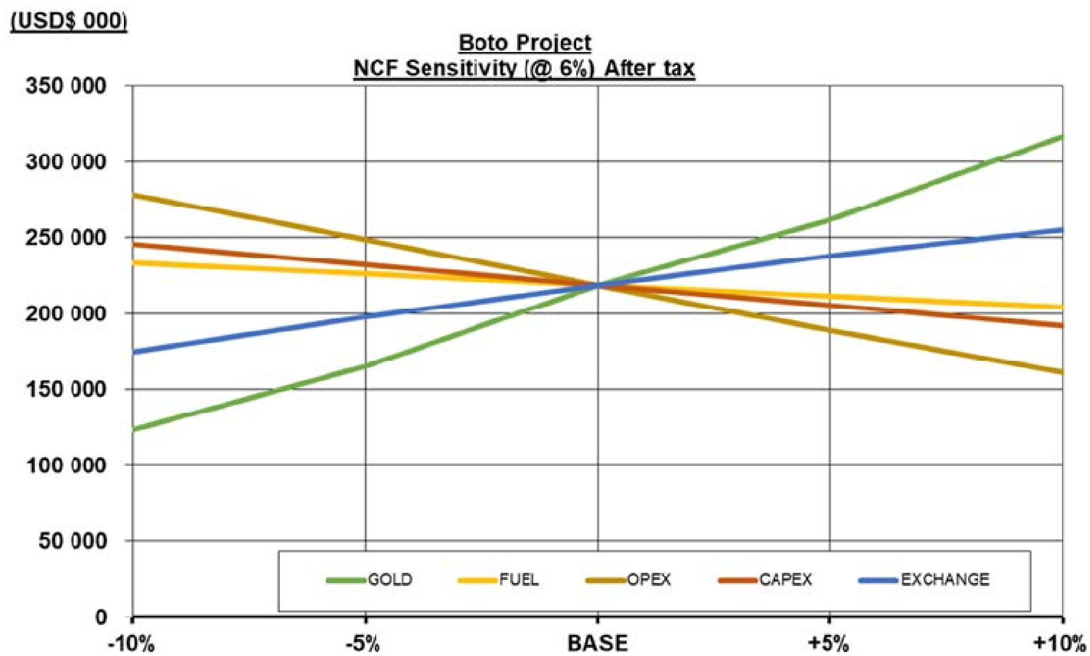
Table 22.3 Summaries

	Unit	Total	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	Year 14
Mined tonnes	kt	247,341	800	14,175	36,023	37,012	36,995	36,986	31,102	31,001	17,464	5,784	-	-	-	-	-	-
Ore tonnes	kt	29,040	0	1,244	3,273	3,790	3,300	4,980	2,277	3,420	3,797	2,959	-	-	-	-	-	-
Waste tonnes	kt	218,300	800	12,931	32,749	33 221	33 695	32 006	28,825	27,581	13 667	2,825	-	-	-	-	-	-
Milled tonnes	kt	29,040	-	-	2,550	2,750	2,750	2,750	2,700	2,700	2,700	2,700	2,750	2,700	1,990	-	-	-
Recovery	%	89.4%	0.0%	0.0%	90.5%	90.3%	90.5%	89.4%	88.8%	88.9%	89.3%	89.0%	88.3%	87.0%	87.7%	0.0%	-	-
Recovered Ounces	koz	1,424	-	-	179	164	164	159	139	144	159	155	51	57	50	-	-	-
After-Tax CF	k$	418,704	69,139	212,156	80,378	99,502	89,729	77,416	55 649	46 675	79 511	95 561	23,187	25,326	31,119	6,260	2,258	8,057
Cumulatif After-Tax CF	k$		69,139	281,295	200,917	101,415	11,686	65,730	121,379	168,055	247,566	343,127	366,314	391,641	422,759	429,019	426,761	418,704
After-Tax CF Discounted @6%	k$	218,648	65,225	188,818	67,487	78,815	67,051	54,576	37,010	29,285	47,062	53,361	12,215	12,586	14,590	2,769	942	3,172

22.4 Sensitivity Analysis

Sensitivity analysis of the gold price, exchange rates, CAPEX, OPEX and fuel price show that the project is relatively robust in an environment ranging between -10% to +10%. Results are presented in Figure 22.1 presented below.

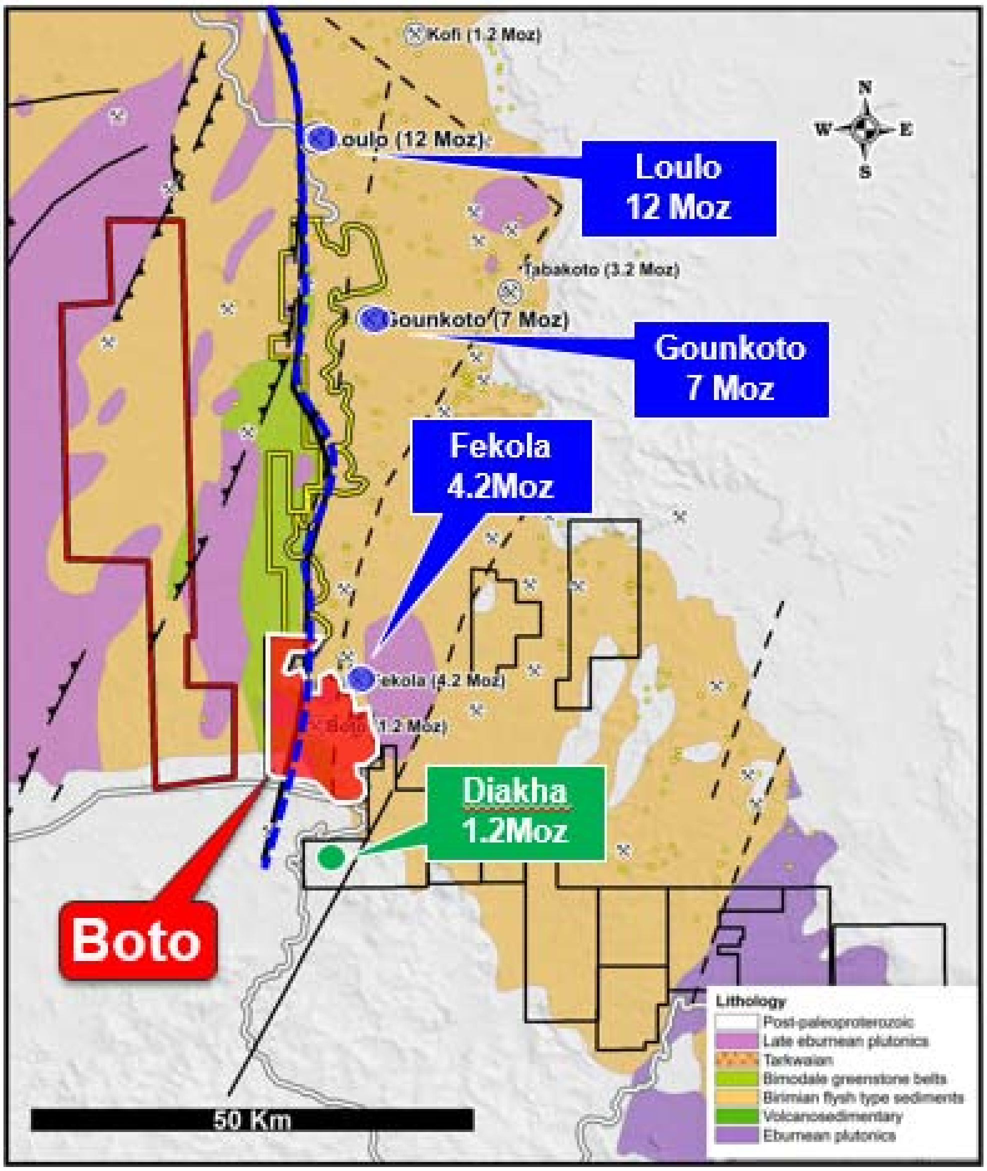
Figure 22.1 NPV Sensitivity Analysis (After-tax @6.0% Discount)



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

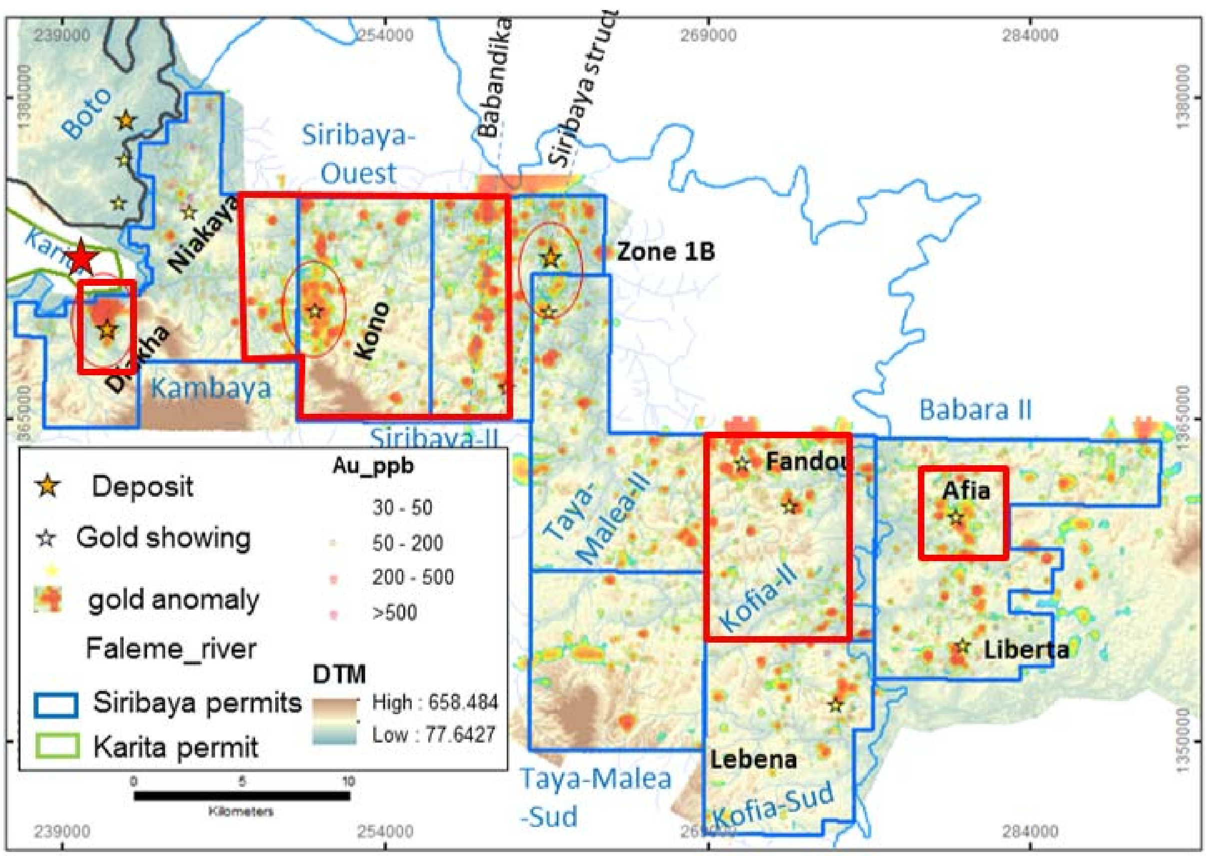


The most proximal and significant property to the Project is the operating Fekola gold mine located in southwest Mali and is owned and operated by B2Gold. The Fekola gold mine is situated approximately 6 km north and along strike with the Malikoundi/Boto 2 deposit.

In Senegal, IAMGOLD also holds the Noumoufoukha permit, which is adjacent to the west to Boto permit. IAMGOLD is currently engaged in a joint venture with Oriole Resources on the Dalafin permit (further west), and with Barrick on the Bambadji property (adjacent to the north of Boto permit). All aforementioned permits are predominately in the green field exploration stage.

In Mali, IAMGOLD holds eight exploration permits covering 600 km^2 at the triple junction between Mali, Senegal and Guinea (Figure 23.2). A recent discovery has been made on the Fekola-Malikoundi trend, known as the Diakha Project with reported Indicated Resources of 18.0 Mt at 1.28 g/t Au and Inferred Resources of 232 Mt at 1.58 g/t Au (RPA, 2018). Exploration is still ongoing at the Diakha project with step out and infill drilling, as well as some sub-surface sampling through the Diakha project area.

Figure 23.2 IAMGOLD Properties in Mali



In Guinea, the Karita Gold Project is wholly owned by IAMGOLD and was acquired in 2017 as a granted exploration permit that covers approximately 100 square kilometres, located in Guinea between the Company's Boto Gold Project in Senegal to the north, and it's Diakha-Siribaya Gold Project in Mali to the south.

During 2019, a first pass drilling program totalling approximately 1,800 m of RC drilling was completed ahead of the rainy season to follow up on a previously identified termite mound geochemical anomaly interpreted to be a possible extension of the mineralized trend between the Boto and Diakha deposits. The Company announced assay results from the drilling program which confirmed a new discovery of mineralization along this portion of the Senegal-Mali Shear Zone and included the following highlights: 29.0 metres grading 2.96 g/t Au; 21.0 m grading 9.01 g/t Au; and 16.0 m grading 3.17 g/t Au (see news release dated October 2, 2019).

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. Major construction contracts will be 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.

IAMGOLD in collaboration with 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 project construction schedule is considered to be 24 months after the project full approval.

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 on the Project has been primarily focussed on the development of the Malikoundi/Boto 2, Boto 5 and Boto 6 deposits. The database for the Project contained 951 core and RC holes totalling 146,195.7 m. RPA conducted a site visit, reviewed property and deposit geology, exploration and drilling methods and results, sampling method and approach, sample and data handling, including chain of custody, and completed independent verification of the data. RPA evaluated the compilation of QA/QC data and is of the opinion that the sample preparation, security, and analytical procedures used by IAMGOLD and prior companies followed industry-standard procedures and the resulting analytical data are acceptable for use in the resource estimation.

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. The latest drilling campaign was focused on defining a mineralized structure parallel to the Malikoundi/Boto 2 main lenses, offset to the east. Additional infill holes were drilled in the south and centre of Malikoundi/Boto 2 deposit.

The resource estimate has been prepared using interpreted mineralized lenses from Malikoundi/Boto 2, Boto 5, and Boto 6 deposits. The Boto 4 deposit has been excluded from the resource estimate. Geovia GEMS™ 6.8.2 was used for drill hole database management, geological interpretation, mineralized wireframe modelling, and to generate the block model supporting the resource estimate. The inverse distance cubed (ID^3) interpolation method was used to estimate the block model gold grades.

The Mineral Resources for the Project are reported within optimized constraining shells using Hexagon Mining MineSight 3D software using a gold price of US$1,500/oz. Cut-off grades vary between 0.37 g/t Au and 0.50 g/t Au, and densities vary between 1.65 g/cm^3 and 2.75 g/cm^3, depending on weathering 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. The Mineral Resources have an effective date of December 31, 2019.

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 – December 31, 2019

Zone	Classification	Tonnes ('000 t)	Grade (g/t Au)	Contained Gold ('000 oz)
Malikoundi/Boto 2	Indicated	35,546	1.60	1,830
	Inferred	7,192	1.88	435
Boto 5	Indicated	1,731	2.00	111
	Inferred	235	1.27	10
Boto 6	Indicated	3,290	0.87	92
	Inferred	770	1.00	25
Boto 4	Indicated	-	-	-
	Inferred	-	-	-
Total	**Indicated**	40,567	1.56	2,033
	Inferred	8,196	1.78	469

***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 0.50 g/t Au depending on weathering zone.*
6. *Capping of grade outliers varies between 2 g/t Au and 25 g/t Au depending on interpreted mineralized zone and sub-domain.*
7. *The density varies between 1.65 g/cm^3 and 2.75 g/cm^3 depending on weathering zone.*

The QP is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate at the time of this report.

The optimized constraining shells for Malikoundi/Boto 2 and Boto 6 mineral resources are outside a 500 m buffer zone from the Balinko and Falémé Rivers. The Boto 5 deposit is located approximately 1 km west of the Balinko River. 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. The Boto 4 has adequate drill information to support a 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 the 500 m buffer zone along 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.

The QP is of the opinion that additional core and RC drilling is warranted at the Project.

25.2 Mining

Mining studies have been completed using the mineral resource estimates as of December 31, 2019 for both the Malikoundi and Boto 5 deposits, and include the following aspects:

- Pit optimization utilized the Pseudoflow algorithm to determine the ultimate pit limits. A metal price of \$1,150/oz Au was used to define the ultimate pit for Malikoundi and \$1,200/oz Au for Boto 5. Appropriate mining dilution estimations were considered.

- Final pits were designed for Boto 5, Malikoundi and Malikoundi North. Bench and overall pit slope designs were based on recommendations in the geotechnical report prepared by Absolute Geotechnics. The mineral reserve estimate is consistent with the CIM Definition Standards for Mineral Resources and Mineral Reserves.

- There are no Proven Mineral Reserves. Probable Mineral Reserves amount to 29.04 Mt at an average grade of 1.71 g/t Au. Total estimated Mineral Reserves amounts to 29.04 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 mines 247.3 Mt, including 29.0 Mt of ore and 218.3 Mt of waste. A total of 259.9 Mt of material will be moved, and the average project strip ratio is 7.5:1.

- The project duration includes approximately 13 months of pre-production, 8 years of production mining and 3 years of ore stockpile reclamation, resulting in a project life of approximately 11 years.

- This mine plan supplies the mill with an optimal feed of saprolite, transition and fresh rock material throughout the project life. The maximum mill feed throughput is 2.75 Mt.

- Grade control drilling will be provided by a separate fleet of reverse circulation drills working in advance of the active mine faces in the saprolite, transition and fresh rock horizons.

- Estimates of both mine capital and operating costs are made. Capital costs consider owner-operated mining 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.

- A ROM blend containing 20% saprolite and saprock is not expected to cause significant materials handling or slurry viscosity related problems.
- The optimum grind size was determined to be a P_{80} of 75 µm.
- The ore at Boto is not expected to exhbit preg-robbing properties.
- A pre-oxygenation step with oxygen sparging during leaching, combined with lead nitrate addition is essential 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, at an average at 1.73 kg CaO/t ore at the design ore blend. When accounting for 90% purity of the supplied lime an addition rate of 1.92 kg lime/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 yielded 2% to 3% higher gold extraction than the results at the coarser grind P_{80} of 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. 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 required to justify its inclusion in the flowsheet.

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 extractions to the leach tests on gravity tailings.

All the results combined provided no justification for inclusion of a gravity circuit in the Boto 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 reagents were not used. The leach kinetics will be more rapid and the cyanide consumption will be reduced significantly.

A trade-off study was conducted by Lycopodium to justify the inclusion of the oxygen plant and the lead nitrate system.

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 4 hours.
- Cyanide concentration of 0.5 g NaCN/L to be maintained in first leach tank.

26.0 RECOMMENDATIONS

26.1 Geology

Core and RC drilling should continue at the Project in order to advance the knowledge of the deposits. Both step-out and testing of potential targets, as well as in-fill drilling are recommended. Drill logging and sampling, assaying and QA/QC protocols and density measurements practice should be continued following the established protocols.

Specific for Malikoundi/Boto 2, deep and shallow drilling are necessary to support the geological model. Two phases of drilling are recommended, as follows:

Phase 1: Deep drilling in support of high-grade blocks at depth. Several drill holes with targets in the proximity of the deep high-grade intercepts in the median part of the deposit are needed in order increase the confidence in the geological model. The Whittle pit optimisation work identified a large resource sensitivity related to the deep high-grade blocks, in the Inferred category. The current drill hole spacing in the deep part of the deposit is sparse and the additional drilling will reduce significantly the drill spacing and help upgrade the classification. RPA recommends drilling five diamond holes, each with lengths of 600 m to 700 m, for a total length of approximately 3,000 m.

Phase 2: Infill RC drilling in the starter pit. The purpose of the RC drilling is to assess the short distance continuity of the mineralization. An area of approximately 500 m by 400 m, located in the starter pit, should be selected for this infill phase. The drilling should be 50 m to 70 m deep, for a total length of approximately 7,000 m, sampling several levels that will be mined in the first years of production. The pattern for the infill drilling should reduce locally the nominal spacing to 25 m, with a number of infill holes drilled at 10 m to 15 m. This will provide information in support of advancing the project towards production.

Additional resource sensitivity studies will be performed based on the core and RC data generated by the Phase 1 and Phase 2 drilling.

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 confirm the foundation conditions below the FWP and TMF embankments, Waste Dumps, significant, highly loaded or settlement sensitive structures at the Plant Site, any new or modified infrastructure locations, and the area between the FWP embankment and Malikoundi North Pit. 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

- Complete updated tailings testwork to characterize the settling and consolidation properties based on any significant changes to the mine plan (i.e. ore blend(s)) or solids content of the tailings slurry

- 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 an Operations, Maintenance and Surveillance (OMS) Manual to assist with the safe and secure operational management of the TMF facility and to provide appropriate maintenance and monitoring protocols.

- Development of a full closure plan for the TMF based on the final design configuration.

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 stability analysis between the FWP and Malikoundi North Pit.

- Develop an OMS Manual to assist with the safe and secure operational management of the FWP and to provide appropriate maintenance and monitoring protocols.

Water Balance

- The catchment areas contributing runoff to the Plant Site, open pits and waste dumps, need to be confirmed based on the ultimate mine plan and site layout and the amount of groundwater inflow to the open pits with time needs to be evaluated and updated.

- Collection of site specific meteorological and hydrological data and evaporation data from the Saraya climate station. This data will be used to refine seasonal runoff values, design storms, and to update the water balance in future detailed design work.

- Calibration of the water balance should be completed, based on actual site data, during early years of operation

- Further evaluate the pumping requirements of runoff and groundwater inflow from each of the open pits and runoff from the waste dumps to the FWP to better define when and how much water is required to be pumped from each area in order to optimize water takings while reducing excess water being sent to the FWP that would need to be released to the environment without being used in the process.

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, Malikoundi and Malikoundi North pits. This work demonstrates the potential for economic development of the Project. Nevertheless, there are still some areas that can be enhanced to assist in the mine development. The following presents the recommended work to be completed prior to the commencement of the project.

26.3.1 Geotechnical

Mine Pits and Phases

AG provided several recommendations for the project. For Boto 5, samples were not analyzed in the weathered material. Prior to the commencement of mining, samples should be analysed and the geotechnical parameters reviewed. Vibrating wire piezometers (VWPs) should be installed in the hanging walls and footwalls of all pits in order to monitor the water levels prior to and during the dewatering activities. Further tests could be conducted to better understand the hydrogeology, especially regarding the aquifers and the permeability of the materials. Lastly, the planned pit phasing, along with continuous monitoring of the pit walls during the operation of the mine, will provide additional geotechnical and hydrogeological data. This data should be used to optimize the pit slope parameters.

In July 2019, KP was mandated to review the geotechnical analysis. According to KP, additional geotechnical data are needed for the Boto 5 open pit as well as several sectors of the Malikoundi open pit.

26.3.2 Ore Control Procedures

The optimization study went into detail with the specific methods of ore control to reduce dilution. However, reducing dilution could still represent a significant area of benefit to the overall mine operation. Additional studies are required to evaluate exact equipment for use in the grade control program, other methods of reducing dilution/ore losses (i.e. blast monitoring systems), and methodologies regarding appropriate sample sizes.

26.3.3 Dewatering

A detailed plan of the various ditches, basins, and pumping systems will be required prior to the commencement of mining. The construction of this infrastructure will need to be planned and updated as the work progresses.

Pit inflow estimates need to be evaluated and updated considering inflows to the Pits via geological structure and surface runoff.

26.3.4 Review of Equipment Selection

A review of all mining equipment and their options will be conducted in order to properly select the best options for this project, avoid any modifications to the equipment once on site, and to maximize security and productivity.

26.3.5 Maintenance Strategy

Prior to the commencement of the mining operation, a study will be completed to determine the best maintenance strategy for the project. The study will determine whether it would be advantageous to sign a maintenance-and-repair contract with the mining equipment providers, rather than an in-house maintenance team.

26.3.6 General Mining

Mine operating costs were optimized through various processes, namely the selection of the mining equipment fleet, the location of the storage facilities, and the utilization of the loading equipment. These assumptions need to be applied during the operation.

26.4 Processing

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.

Additional sedimentation and rheology testing should be performed on blends containing elevated (> 30%) soft rock to assess its potential impact on operability.

During plant operations, the following items are recommended:

- Observations made during the test program indicated that an abundance of saprolite in the plant feed might 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.

27.0 REFERENCES

ASCE/SCI 7-10. *Minimum Design Loads for Buildings and Other Structures.* American Society of Civil Engineers (ASCE). 2010.

Canadian Dam Association (CDA), 2014. *Technical Bulletin - Application of Dam Safety Guidelines to Mining Dams*.

Canadian Dam Association (CDA), 2013. 2007 Dam Safety Guidelines. 2013 Revision.

Chaisson, Guillaume. "An Investigation into the Grindability Characteristics of Forty-Two Samples from the Boto Project", Project 15080-001, November 2015. SGS Mineral Services.

Delaney, Vivien. "An Investigation into the Recovery of Gold from the Boto Project", Project 14573-001, December 2014. SGS Mineral Services.

Desharnais, Guy. "Sample Selection Report for Boto Deposit – Senegal", June 2015. SGS Geostat.

Halliday, Matthew. "Sample Selection Report for Boto Deposit – Senegal", June 2017. SGS Geostat.

Hendry, Lesley. "An Investigation into the Recovery of Gold from the Boto Project", Project 14037-001, August 2013. SGS Mineral Services.

HydroCAD Software Solutions LLC (HydroCAD®), 2015. HydroCAD®. Version 10. Chocorua, New Hampshire.

IAMGOLD, 2018, ''*Tome 1 et Tome 2 – Rapport Final Étude d'impact environnemental et social, Projet de Mine d'Or de Boto mai 2018*''.

Jackman, Rene. "An Investigation into the recovery of Gold from the Boto Project Samples", Project 15080-001, November 2016. SGS Mineral Services.

Knight Piésold Ltd. 2019d. Letter to: Luc Bernard Denoncourt Re: Boto Gold Project - Updated Feasibility Level Site Wide Water Balance, September 11, 2019. North Bay, ON. Ref. No. NB19-00493 (NB401-107/06)

Knight Piésold Ltd. (KP), 2018a. Tailings Management Facility Feasibility Design. September 13. North Bay, Ontario. Ref. No. NB401-107/3-2, Rev 0.

Knight Piésold, 2019. Letter to: Luc-Bernard Denoncourt, IAMGOLD Corporation. Re: Updated Fresh Water Pond Design (1.5 M m^3) Summary. August 160. North Bay, Ontario. Ref. No. NB19-00628, (NB401-107/06).

Knight Piésold Ltd. (KP), 2018b. Letter to: Neil Lincoln, Lycopodium Minerals Canada Ltd. Re: *Tailings Physical Testing Results*. July 5. North Bay, Ontario. Ref. No. NB17-00266 (NB401-107/3).

Knight Piésold, 2018b. Letter to: Mr. Neil Lincoln, Lycopodium Minerals Canada Ltd. Re: Fresh Water Pond – Slope Stability Analysis. July 17. North Bay, Ontario. Ref. No. NB18-00439, (NB401-107/03).

Knight Piésold Ltd. (KP), 2017. Letter to: Neil Lincoln, Lycopodium Minerals Canada Ltd. Re: *Boto Gold Project - Geochemical Waste Characterization Update*. November 9. North Bay, Ontario. Ref. No. NB17-00617 (401-107/2).

Knight Piésold, 2018c. Letter to: Mr. Neil Lincoln, Lycopodium Minerals Canada Ltd. Re: Boto Gold Project – Feasibility Level *Site Wide Water Management Plan*. September 11. North Bay, Ontario. Ref. No. NB18-00573, (NB401-107/03).

Knight Piésold Ltd. (KP), 2018d. Memo to: Neil Lincoln. Re: *Boto Gold Project - Humidity Cell Testing Summary - Updated.* August 31. North Bay, Ontario. Ref. No. NB18-00569 (401-107/3)

Knight Piésold Ltd. (KP), 2018e. Memo to: Neil Lincoln. Re: *Boto Gold Project Preliminary Study Seismic Hazard Assessment.* June 12. Perth, Australia. Ref. No. PE18-00540 (NB401-107/3).

Knight Piésold Ltd. (KP), 2018f. Letter to: Neil Lincoln. Re: *Tailings Management Facility - Slope Stability Analysis.* July 6. North Bay, Ontario. Ref. No. NB18-00421 (401-107/3).

Knight Piésold Ltd. (KP), 2018g. Letter to: Neil Lincoln. Re: *Boto Gold Project - Feasibility Level TMF Water Balance Summary.* July 23. North Bay, Ontario. Ref. No. NB18-00472 (401-107/3).

Knight Piésold Ltd. (KP), 2018. Letter to: Neil Lincoln. Re: *Boto Gold Project – Waste Dump Slope Recommendations.* July 9. North Bay, Ontario. Ref. No. NB18-00429 (401-107/3).

Knight Piésold Ltd. (KP), 2018. Letter to: Neil Lincoln. Re: Plant Site Foundation Recommendations. May 18. North Bay, Ontario. Ref. No. NB18-00274 (401-107/3).

Knight Piésold Ltd. (KP), 2017a. Letter to: Neil Lincoln, Lycopodium Minerals Canada Ltd. Re: Boto Gold Project - 2017 Pre-Wet Season Hydrogeological Site Investigations. August 31. North Bay, Ontario. Ref. No. NB17-00496 (NB401-107/2).

Knight Piésold Ltd. (KP), 2018a. 2017 Geotechnical Site Investigation Summary. May 25. North Bay, Ontario. Ref. No. NB401-107/3-1, Rev 0.

Knight Piésold Ltd. (KP), 2018. Memorandum to: Mr. Neil Lincoln. Re: 2017 Site Investigation - Laboratory Test Results - Geolabs. July 6. North Bay, Ontario. Ref. No. NB18-00427 (NB401-107/3).

Lycopodium Minerals Canada, 2017, Rapport Technique NI43-101 Etude De Prefaisabilite – Boto - Senegal, Effective Date: December 31, 2017.

MacDonald, James. "An investigation into Recovery of Gold from Boto Project Samples", Project 15080-003, July 2018. SGS Mineral Services.

MacDonald, James. "Rheology Results Summary", Project 15080-003, July 26, 2019. SGS Mineral Services.

Makdisi, F.I. and Seed, H.B., 1978. *Simplified Procedure for Estimating Dam and Embankment Earthquake-Induced Deformations*. Journal of the Geotechnical Engineering Division, American Society of Civil Engineers. Vol 104, No. GT7. pp 849-867.

Thomas R., 2018. "IAMGOLD Corporation Boto Project DFS, Geotechnical Assessment – Open Pits", Absolute Geotechnics Pty Ltd. Report Number: IAM002-REP04-REV0. July 2018.

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.

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.