NI 43-101 TECHNICAL REPORT
FEASIBILITY STUDY OF THE
BOMBORÉ GOLD PROJECT
BURKINA FASO

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

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Project Name:	Bomboré Gold Project
Title of Report:	NI 43-101 Technical Report
Location:	Burkina Faso
Effective Date of Report:	26 June 2019

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Manochehr Oliazadeh, P.Eng. (Lycopodium Minerals Canada Ltd.) Oliazadeh M.	12 August 2019
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1.0 SUMMARY

1.1 Introduction

This Technical Report was compiled by Lycopodium Minerals Canada Ltd (Lycopodium) for Orezone Gold Corporation (Orezone) from contributions from Qualified Persons as set out in Table 1.1 to support the Company's press release dated June 26, 2019, and to summarize the results of the July 2019 Feasibility Study of the Bomboré Gold Project. This Technical Report expands on the Feasibility Study and Technical Report issued in 2018 (Feasibility Study of the Bomboré Gold Project, Burkina Faso, August 23, 2018) to incorporate lower transition and fresh rock material into the mine plan and mill feed. This Technical Report was prepared in compliance with the disclosure requirements of NI 43-101 and in accordance with the requirements of Form 43-101 F1.

Table 1.1 Study Contributors

Contributor	Scope
Lycopodium Minerals Canada Limited (Lycopodium)	Metallurgical testwork interpretation, process plant, project infrastructure, project development plan, compile CAPEX and OPEX, financial modelling, coordination and compiling of report
Roscoe Postle Associates Inc. (RPA)	Geology, mineral resources
AMC Consultants (AMC)	Mining, reserve statement
Knight Piésold Consulting (KP)	Tailings storage facility, water management and supply
Antea® Global (Antea)	Environment, permitting, community relations

The study on which this Technical Report is based incorporates the mineral resource update completed by RPA and dated January 5, 2017.

1.2 Project Description and Ownership

The Bomboré Gold Project property (the Property) comprises a block of contiguous permits totalling 15,029 ha located in Ganzourgou Province, Burkina Faso, approximately 85 km east of the capital city of Ouagadougou (Figure 1.1). The Property is easily accessible by the paved road, national highway N4 from Ouagadougou. Orezone, has recently upgraded the 5 km unsealed road from N4 to the accommodation camp at the north of the Property, including installation of culverts to allow all weather access.

All of the Property is accessible in the dry season. Access in the wet season can be restricted by the flooding of local watercourses but, as part of the project development, permanent bridges will be installed providing year-round access for light and heavy vehicles.

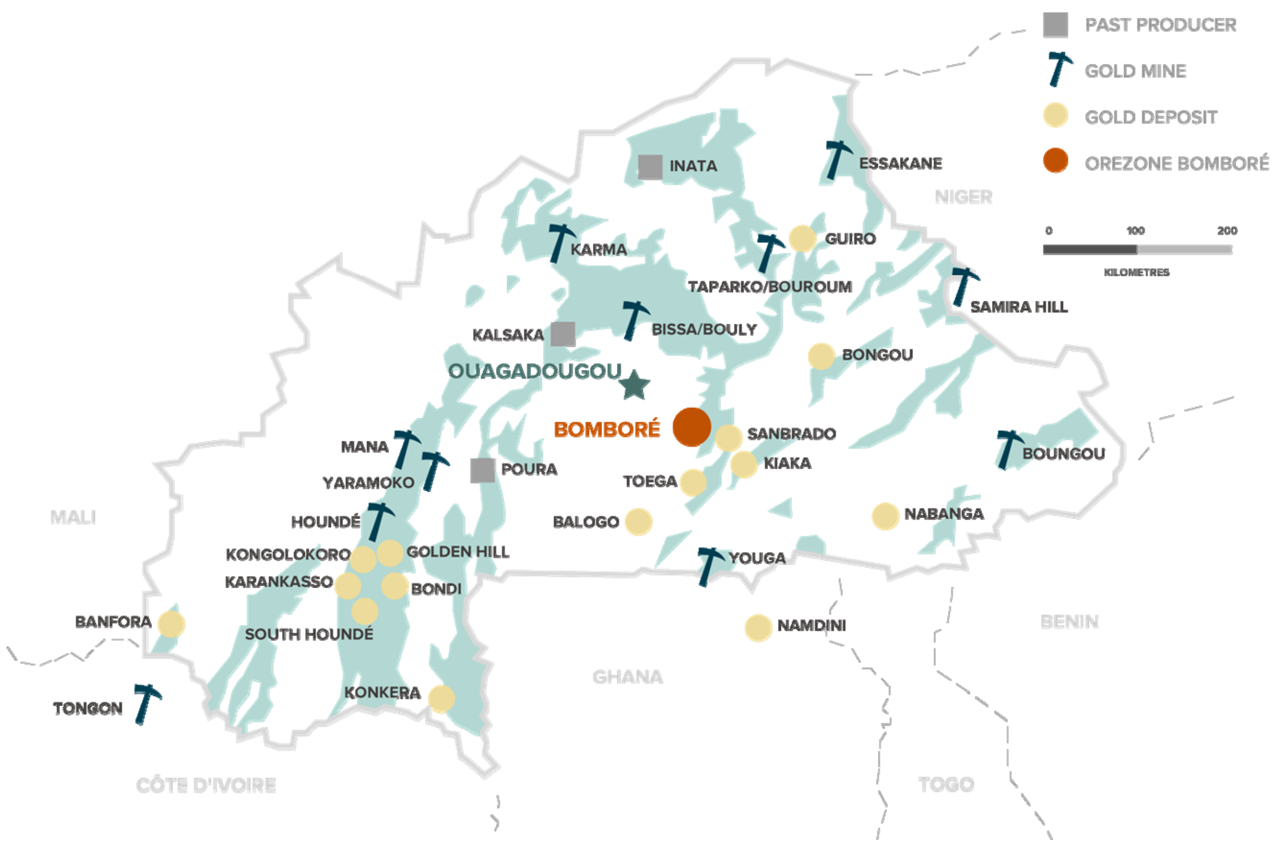
The Property is within 15 km of the regional town of Mogtédo, with a population greater than 15,000. The town is developing rapidly with many substantial multi-storey concrete block buildings established or under construction.

The local climate consists of dry and wet seasons. It is common for rain to occur from April through October, however, the highest concentration of rainfall events occurs between late June and late September. On average, approximately 800 mm of rainfall occurs annually, typically in short bursts of heavy rain. Construction and mining operations can be scheduled year-round, with short delays during heavy rainfall events expected.

Temperatures range from a low of about 10°C in December and January to highs of about 43°C in March and April with average daily temperatures in the range of 23° to 33°. Between the end of the wet season and March the north-easterly trade winds bring dust down from the Sahara (the Harmattan) resulting in reduced visibility.

The Universal Transverse Mercator (UTM) co-ordinates for the approximate centre of the Property are 1,348,800mN, 728,100mE (Zone 30, Clarke 1880 ellipsoid, Adindan datum). The geographic co-ordinates for the approximate centroid of the currently defined Bomboré gold deposit are 12°12'N Latitude and 0°12'W Longitude.

Figure 1.1 Project Location



The Property covers an area of 15,029 ha and consists of one Industrial Operating Permit (the Bomboré Mining Permit) of 2,500 ha, surrounded by four Mining Exploration Permits: the Toéyoko Exploration Permit of 4,669 ha, the Bomboré II Exploration Permit of 1,815 ha, the Bomboré III Exploration Permit of 4,810 ha and the Bomboré IV Exploration Permit of 1,235 ha.

The Bomboré Mining Permit is registered in the name of Orezone Bomboré S.A. (OBSA), a 90%-owned subsidiary of Orezone Inc. S.A.R.L (OSARL), itself a 100%-owned subsidiary of Orezone Inc. (OINC), which is 100% owned by Orezone. The Bomboré Mining Permit was granted to OBSA by way of Decree No. 2016-1266/PRES/PM/MEMC/MINEFID/MEEVCC dated December 30, 2016 and is valid for an initial tenure of 10.7 years but can be extended if the mine life is extended beyond what was initially applied for.

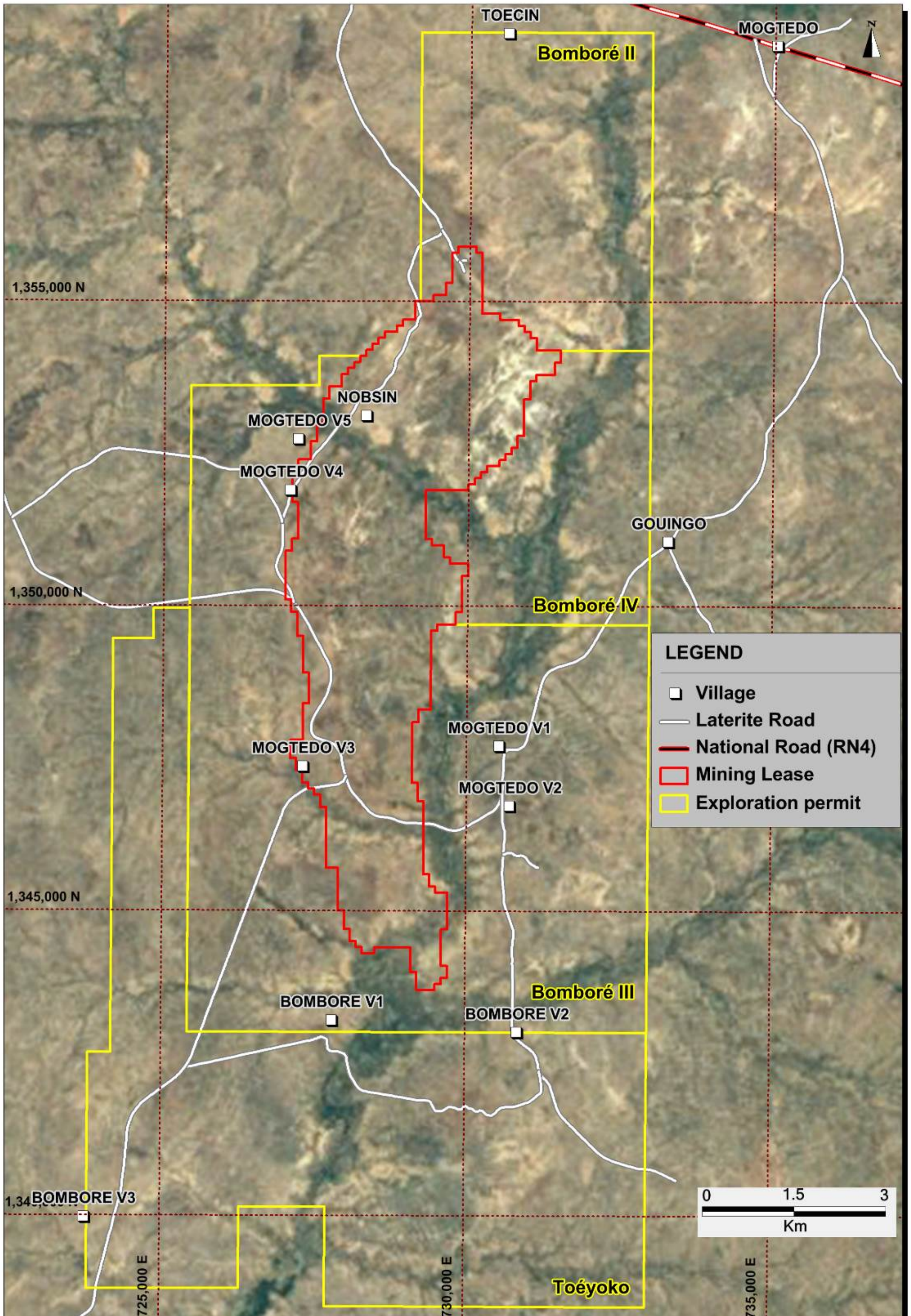
The Toéyoko Exploration Permit is currently registered in the name of OSARL. It was granted to OINC in July 2011 and is valid until July 13, 2020 when it will be renewable for one last exceptional three-year additional term.

The Bomboré II, Bomboré III and Bomboré IV Exploration Permits are registered in the name of OSARL. They were granted to OSARL on January 17, 2017 and are valid until January 17, 2020 when they will be renewable for the first of three possible three-year additional terms.

The Mineral Resources reported in this Report are essentially located within the Bomboré Mining Permit (Figure 1.2), with one small deposit on the Toéyoko Exploration Permit (P17S) and one small deposit on the Bomboré III Exploration Permit (P17N).

All mining ventures in Burkina Faso are subject to a 10% free carried interest and a royalty on gold sold in favour of the Government of Burkina Faso, once a mining convention is signed and an operating permit is awarded by the government.

Figure 1.2 Bomboré Tenements



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OREZONE ; Datum : WGS 84 UTM Zone 30N

1.3 Geology and Mineralization

The Project covers part of a northeast-southwest trending greenstone belt extending for 50 km from the southwest corner to the village of Meguet in the northeast. The permit area is underlain mainly by a meta-sedimentary flysch-type sequence dominated by meta-sandstones with subordinate carbonaceous meta-pelites and polymictic meta-conglomerates. This metasedimentary sequence is intruded by early meta-gabbroic and ultramafic intrusives and then syntectonic granodioritic intrusives. Late-tectonic quartz-feldspar porphyries occur as dikes and larger bodies within the greenstone belt. Large biotite granite intrusives are present on the Property to the west and to the south of the greenstone belt that is also moulded on a large quartz diorite intrusive located along the eastern limit of the Project. A syenitic intrusion referred to as the Petite Suisse is exposed in the west portion of the Property.

The Bomboré Shear Zone (BSZ) is a major, one to three-kilometre thick structure that contains the Bomboré gold mineralization and represents the dominant structural feature of the area. The Bomboré gold mineralization trend is defined by a gold-in-soil anomaly exceeding 0.1 g/t Au, as well as by the presence of numerous gold showings and *orpaillage* (artisanal miners) sites. The Bomboré anomaly measures 14 km in length, is several hundreds of metres in width, and occurs within the BSZ.

Surface weathering has affected the rocks to an average depth of 35 m to 50 m but can be as deep as 100 m on the P8/P9 and CFU hanging wall, and as shallow as 5 m to 10 m in the P17 area.

The gold mineralization on the Property is hosted in the BSZ, a major north-northwest to north-northeast trending structure. This shear zone has an arcuate shape and extends over tens of kilometres beyond the limits of the Property. It is interpreted as a secondary structure to the Tiébélé-Dori-Markoye Fault, a regional north-northeast trending sinistral fault that represents a major discontinuity in the Birimian rocks, across which regions of contrasting structural styles are juxtaposed.

Generally, the gold occurs as fine grain electrum (< 10 µm) but can be visible in outcrop. Artisanal mining over the 1990-2016 period attests to the existence of coarser gold locally. Gold occurs as free gold and is mainly associated with pyrite, pyrrhotite, chalcopyrite, and arsenopyrite. Most sulphides occur as disseminations and fine stringers sub-parallel to the foliation fabric suggesting development in active shear zone or re-mobilization. Magnetite and graphite are present locally. Although the sulphide content can be as much as 5%, it is on average only 1% to 2% in fresh (i.e., non-weathered) mineralized rocks.

Gold mineralization is most commonly hosted in the biotite schist (meta-gabbro) and its host rocks (typically the meta-sandstones) and the granodiorite dikes that intrude the gabbros, although in Maga north, P16 and P17N areas the meta-argillites are the main host. The syn-tectonic granodioritic intrusives are also mineralized, although to a lesser degree than the biotite schist and the meta-argillites. The meta-conglomerate and meta-peridotite are unfavourable hosts. The meta-gabbro might represent the best chemical trap given its high iron content if gold was transported as a thio-complex, as suggested by the pervasive fine pyritic assemblage that is associated with the gold mineralization in the sulphide zone. Although much of the gold resources defined within the Project area are hosted in the meta-gabbro unit, the deformed granodiorite and its contact zone with the meta-gabbro host is where the better-grade mineralization is concentrated.

At a cut-off grade of approximately 0.2 g/t Au, the gold mineralization exhibits reasonable continuity over a strike length of approximately 10 km. At this cut-off grade, the gold mineralization forms more restricted corridors (500 m to 1,000 m in length and 10 m to 100 m in width) defining anastomosing patterns, parallel and slightly oblique to the general trend of the BSZ.

These higher-grade corridors formed the basis for defining geostatistical domains within each litho-domain considered for resource estimation. One of the benefits of the 2010 to 2013 infill drilling programs was the delineation of higher-grade sub-domains based on a cut-off grade of 0.5 g/t Au with the broader low-grade domains based on a lower cut-off grade of 0.2 g/t Au. The higher-grade sub-domains have a strike length of up to 500 m and a width typically between 5 m and 30 m.

1.4 Exploration Status

The Property area was first explored in 1989, and between 1989 and 2000, various phases of mineral exploration were completed by La Générale des Mines et des Carrières (GMC), Channel, Solomon, and Placer Dome. A total of 1,271 boreholes (combined core, reverse circulation (RC) and rotary air blast (RAB)) were drilled and geochemical, geophysical, and trenching surveys were also conducted during this time. Two preliminary resource estimates were made in 1997 and 1998 by Channel (non-compliant, pre- NI 43-101).

Channel drilled 10 diamond holes (1,080 m), 261 RC (19,501 m) and 1,000 rotary air blast (34,249 metres) boreholes on the Property during the period 1994 to 2000. There are no records describing the drilling procedures used by Channel in their exploration program.

Since acquisition of the Property in 2003, Orezone has carried out systematic mapping, prospecting, sampling, and gold assaying of outcrops and gold workings. Several airborne and ground magnetic and induced polarization/resistivity surveys as well as core, RC, and auger drilling campaigns have also been completed that support the geological model used for the current Mineral Resource estimate. Between 2003 and 2018, Orezone completed 1,172 core holes for approximately 168,000 m, 5,361 RC holes for approximately 318,000 m, and 4,221 auger holes for approximately 20,000 m. The Mineral Resource estimate is based only on data from core and RC drilling.

Data from 276 new holes totalling 15,387 m, within the resource area, were received after the resource database was finalized for the January 5, 2017 resource statement. RPA reviewed the results and is of the opinion that the 2017 resource model is still appropriate to be used as the basis for this Technical Report and that the effective date should remain at January 5, 2017.

1.5 Mineral Resources

Gold mineralization has been defined at shallow depths by reverse circulation (RC) drilling, diamond drilling, and trenching along a strike length of over 12 km. The gold mineralized zones have been modelled as a large number of sub-parallel, tabular zones that gradually change in strike from north-northwest, to northeast. Most of the mineralization wireframes are interpreted to dip moderately to the east or southeast. Review of the lithologic models shows that gold values are contained within all host rock types and can be seen to follow a stratiform orientation.

In order to keep the size of the various block model files within functionally manageable limits, the gold mineralization has been split into five separate block model areas, referred to as the North, South, P16, P17 and P17S areas. Together, the North and South block models contain the majority of the Mineral Resources. Low grade mineralized wireframe models were created for the 2016 Mineral Resource estimate using a grade threshold of approximately 0.20 g/t Au, and high-grade mineralized wireframe models that were created using a grade threshold of approximately 0.45 g/t Au.

Following completion of the 2016 Mineral Resource estimate, an additional set of low-grade mineralized wireframes was created for the North and South model areas using only the lower-grade threshold of 0.20 g/t Au to capture material remaining outside the 2016 Mineral Resource estimate wireframes. There was also a further grade estimate completed for selected material outside all wireframes on an unconstrained basis ("third domain") for the North, South, P16, and P17 model areas. The low-grade mineralized wireframe models and the third domain were used to extract a total of 3,207 and 146,372 assay results, respectively, from the four drill hole databases (North, South, P16, and P17) for analysis. The P17S high-grade deposit was modelled and interpolated in December 2018 following additional drilling and mineralization wireframe interpretations. The P17S drill hole database includes 108 diamond drill holes (16,423 m) and 54 reverse circulation holes (1,979 m), totalling 162 drill holes (18,402 m), and seven channels totalling 23.4 m.

Orezone has elected to use the capping method to reduce the influence of high-grade assay values. The selection of the various capping values was guided by the goal of achieving a target coefficient of variation (CoV) of less than approximately 2.0. This resulted in capping values that ranged from 1.50 g/t Au to 48.97 g/t Au for the low and high-grade mineralized wireframe domain assays for the 2016 Mineral Resource estimate. A universal value of 5.00 g/t Au was used by RPA for both the January 2017 estimate low-grade mineralization wireframe domain and third domain assays. Capped assays were composited within the domain boundaries at 1.5 m length. Parts of P17 were composited at 1 m length. For the P17S resource, assays were composited to 1 m lengths with a minimum length of 0.25 m for each individual mineralization wireframe.

Gold grades within the 2016 Mineral Resource estimate mineralized wireframe models (low-grade, high-grade) for the North, South, and P16 areas were estimated using the ordinary kriging (OK) interpolation algorithm. The gold grades within the 2016 and 2017 Mineral Resource estimate wireframe models (low-grade, high-grade) for the P17 model area were estimated using the inverse distance squared (ID^2) interpolation algorithm. The gold grades inside the January 2017 additional low-grade mineralized wireframe models for the North and South areas were also estimated using the OK interpolation algorithm; there were no additional low-grade wireframe domain models in the P16 and P17 model areas. Hard boundaries were used to constrain the source composite files such that only those composite samples that are present within a specified wireframe were used to estimate block grades. Similarly, hard boundaries were used to constrain coding of the block model where only those blocks that are contained within the specified mineralized wireframe model were permitted to receive estimated gold grades. Gold grades for the January 2017 estimate third domain in all model areas were estimated using a two-step process using the inverse distance cubed (ID^3) interpolation algorithm. The first step used only composites outside wireframes and above 0.20 g/t Au to flag blocks with a grade above 0.00 g/t Au from a minimum of two composites, then on the second step used all composites outside wireframes to estimate the gold grade of the previously flagged blocks. Gold grades for the P17S December 2018 estimate were estimated within the wireframe models by ID^2.

Data from 276 new holes totalling 15,387 m, located within the resource area but outside P17S, were received after the resource database was finalized for the January 2017 Mineral Resource statement. RPA reviewed the results and is of the opinion that the resource model is still appropriate to be used as the basis for this Technical Report and that the effective date should remain at January 5, 2017.

Measured Mineral Resources comprise that mineralized material that has been outlined with a drill hole density of at least 25 m x 25 m. Indicated Mineral Resources comprise that mineralized material that has been outlined with a nominal drill hole density of 25 m x 50 m. Inferred Mineral Resources comprise the mineralized material that has been outlined with a nominal drill hole density of 100 m x 100 m and to within a depth of 100 m below the bottom of the drill hole coverage. Clipping polygons representing the various Mineral Resource categories were created for each of the oxidation layers to ensure the continuity and consistency of the classification category. These clipping polygons were used to code final classification into each of the four block models.

A number of cut-off grades were developed for the Project that reflect the varying processing costs and metallurgical recoveries of the different oxidation layers and the additional transportation costs for mineralized material that is located distant to the proposed processing plant. A gold price of US$1,400/oz was used for all cut-off grades for reporting of the Mineral Resources. To fulfill the NI 43-101 requirement of "reasonable prospects for eventual economic extraction", RPA prepared a preliminary open pit shell to constrain the block models for resource reporting purposes. Additional criterion to constrain the Mineral Resource report included several "non-permitted" areas related to environmentally sensitive areas and mineralized areas being set aside for the benefit of local artisanal miners.

RPA provided an updated Mineral Resource estimate with an effective date of January 5, 2017 ("2017 Mineral Resources") by incorporating the oxide material within the previously excluded "Restricted Zones", the sulphide resources comprising lower transition and fresh layers and all drilling completed to December 2018 on the high-grade P17S deposit.

The Mineral Resource estimate for the P17S area has an effective date of December 21, 2018. RPA notes that the effective date of the deposit as a whole remains January 5, 2017 since the bulk of the Mineral Resources (North, South, P16, and P17) has not been updated since that estimate. A fifth block model has been added for the P17S deposit. The P17S deposit is described in detail in section 14.15.

The updated 2017 Mineral Resource estimate comprises five separate block models, which have been combined into a global resource as shown in Table 1.2. The 2017 Mineral Resource estimate conforms to the Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM (2014) definitions).

Table 1.2 Summary of the Mineral Resources as of January 5, 2017

Classification		Measured			Indicated			Measured + Indicated			Inferred		
	Cut-off	Tonnage	Grade	Contained	Tonnage	Grade	Contained	Tonnage	Grade	Contained	Tonnage	Grade	Contained
	g/t Au	000 t	g/t Au	Au koz	000 t	g/t Au	koz Au	000 t	g/t Au	Au koz	000 t	g/t Au	koz Au
Oxides	0.20	31,600	0.62	628	75,300	0.53	1,273	106,900	0.55	1,901	20,900	0.40	265
Sulphides	0.2/0.38	9,000	0.90	260	113,600	0.79	2,894	122,600	0.80	3,154	32,400	0.81	842
TOTAL		**40,600**	**0.68**	**888**	**188,900**	**0.69**	**4,167**	**229,400**	**0.69**	**5,055**	**53,300**	**0.65**	**1,107**

Notes:

1. *CIM (2014) definitions were followed for Mineral Resources.*
2. *Mineral Resources are inclusive of Mineral Reserves.*
3. *Oxide resources are made up of the regolith, saprolite, and upper transition layers reported at a cut-off of 0.2 g/t Au.*
4. *Sulphide resources are made up of lower transition and fresh layers reported at 0.2 g/t Au and 0.38 g/t Au respectively.*
5. *Mineral Resources have been constrained within a preliminary pit shell generated in Whittle software.*
6. *Mineral Resources are estimated using a long-term gold price of US$1,400/oz.*
7. *A minimum mining width of approximately 3 m was used.*
8. *Bulk densities vary by material type.*
9. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
10. *Numbers may not add due to rounding.*

RPA is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the 2017 Mineral Resource estimate.

1.6 Mineral Reserves

The Mineral Reserve Estimate is based on the updated 2017 Mineral Resource estimate prepared by RPA (with an effective date of January 5, 2017), incorporates the oxide material within the previously excluded “Restricted Zones” and includes all drilling completed to that date in the P17S deposit.

AMC used the following four separate resource block models.

- North model including the Maga, CFU and P8P9 deposits.
- South model including the P11, Siga E, and Siga W deposits.
- P16 model, a standalone deposit at the southern end of the Project.
- P17S model, a standalone deposit at the south-east end of the Project.

This Technical Report incorporates all available Measured and Indicated Mineral Resources material in the 2017 Mineral Resource Estimate within the oxide, transition, and sulphide horizons. AMC developed mine models by applying modifying factors to the resource block models using Datamine’s™ Studio OP software (Datamine). Pit optimizations were conducted on the mine models using Gemcom’s Whittle™ 4.X software (Whittle). The pit optimization was then used as basis for producing practical mine designs.

The weathered saprolite and upper transition (UT) horizons, which reach a thickness of up to 90 m across the site, can be excavated without the need for prior blasting (free-dig material).

AMC assumed that 70% of the Lower Transition (LT) material would require ripping prior to being loaded onto the haul trucks, while the remaining 30% will have to be blasted.

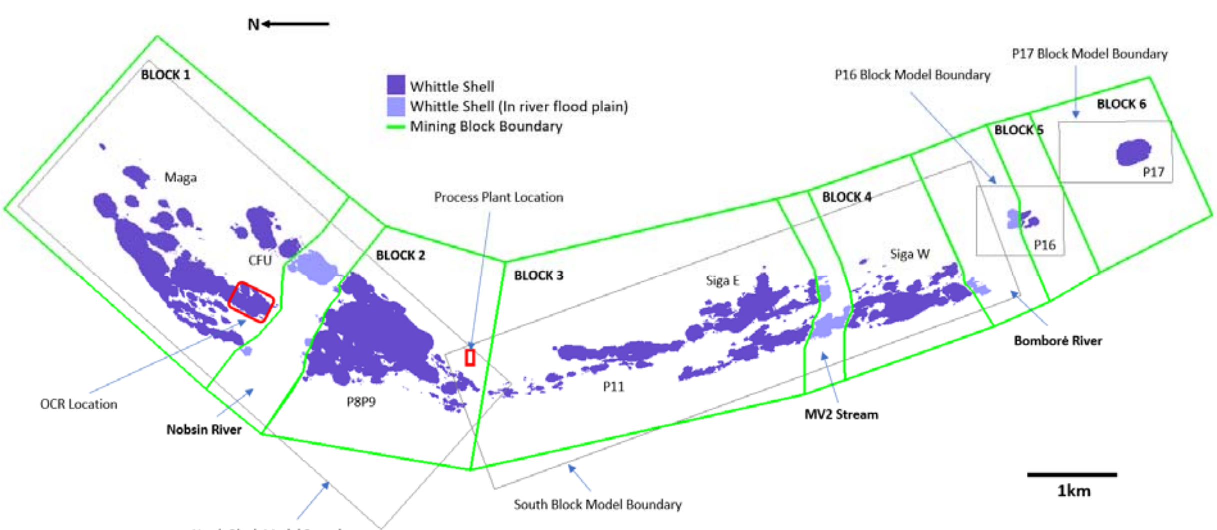
The sulphide material below the weathered horizons requires drill-and-blast.

AMC validated the RPA resource block models and divided them into seven mining blocks separated geographically as follows:

- Block 1 – North model: North of the Nobsin River.
- Block 2 – North model: South of the Nobsin River.
- Block 3 – South model: North of the MV2 Stream.
- Block 4 – South Model: South of the MV2 Stream.
- Block 5 – P16 Model.
- Block 6 – P17 Model.
- Restricted Zones - Material contained in the Nobsin, MV2, and Bomboré river flood plains.

Figure 1.33 shows a plan view of the mining block boundaries.

Figure 1.3 Plan View of Mining Block Boundaries



The pit optimization included both oxide and sulphide horizons, with inputs varied depending on the proposed mining method. Inferred Mineral Resources were treated as waste, and only Measured and Indicated Mineral Resources were considered as feed to the processing plant.

Orezone provided the gold price of $1,250/oz and associated offsite charges.

Gold royalties in Burkina Faso are calculated as follows:

- Less than US$ 1,000/oz: 3% of the NSR + 1% Local Development Mining Fund ("FMDL") tax.
- Equal to or greater than US$ 1,000/oz and less than or equal to US$ 1,300/oz: 4% of the NSR + 1% FMDL tax.
- Greater than US$ 1,300/oz: 5% of the NSR + 1% FMDL tax.

Royalties are applied to the totality of the gold sold.

The Bomboré Mineral Reserve Estimate is summarized in Table 1.3. The Mineral Reserve includes diluted recovered Measured and Indicated Resources constrained by the ultimate pit design. The Mineral Reserve Estimate excludes 1.7 Mt of mineralized low-grade oxides that are stockpiled and not included in the current mill feed schedule.

Table 1.3 Summary Mineral Reserve Estimate – June 26, 2019

Classification	**Proven**			**Probable**			**Proven & Probable**		
	Tonnes	Gold Grade	Contained Gold	Tonnes	Gold Grade	Contained Gold	Tonnes	Gold Grade	Contained Gold
	000 t	g/t Au	000 oz Au	000 t	g/t Au	000 oz Au	000 t	g/t Au	000 oz Au
Material type									
Oxides	20,213	0.73	473	32,326	0.66	687	52,539	0.69	1,161
Sulphides	3,241	1.31	136	14,320	1.17	538	17,561	1.19	675
Total	**23,453**	**0.81**	**610**	**46,647**	**0.82**	**1,225**	**70,100**	**0.81**	**1,835**

Notes:
1. *Oxides include regolith, saprolite and upper transition material.*
2. *Sulphides include lower transition and fresh material.*
3. *Mineral Reserves have been estimated in accordance with the CIM Definition Standards.*
4. *Mineral Reserves are based on cut-off grades that range from 0.300 to 0.325 g/t Au for oxides, and 0.466 to 0.555 g/t Au for sulphides.*
5. *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability.*
6. *There are 1.7Mt of low-grade mineralized oxide material above cut-off grade remaining in the stockpiles that are not included in the Mineral Reserves Estimate.*
7. *Mineral Reserves are estimated at an average long-term gold price of US$ 1,250/troy oz.*
8. *Mineral Reserves are reported effective June 26, 2019.*
9. *Mining recovery factors estimated at 98% for oxides and 96%-100% for sulphides.*
10. *Processing recovery varies by grade, weathering unit and location.*
11. *Rounding of some figures may lead to minor discrepancies in total.*

Table 1.4 presents the Mineral Reserve Estimate by weathering unit.

Table 1.4 Mineral Reserve Estimate by Weathering Unit – June 26, 2019

Classification	Units	Block 1	Block 2	Block 3	Block 4	Block 5	Block 6	Restricted Zones	Total
Summary of Mineral Reserves									
Proven									
Ore	000 t	9,022	13,211	-	-	166	838	215	**23,453**
Gold Grade	g/t Au	0.79	0.74	-	-	0.95	1.99	0.91	**0.81**
Contained Gold	000 oz Au	229	316	-	-	5	54	6	**610**
Probable									
Ore	000 t	9,310	12,671	12,355	10,356	9	428	1,518	**46,647**
Gold Grade	g/t Au	0.81	0.79	0.77	0.90	0.66	1.61	0.71	**0.82**
Contained Gold	000 oz Au	242	320	308	299	0	22	35	**1,225**
Proven & Probable									
Ore	**000 t**	**18,333**	**25,883**	**12,355**	**10,356**	**175**	**1,266**	**1,733**	**70,100**
Gold Grade	**g/t Au**	**0.80**	**0.76**	**0.77**	**0.90**	**0.94**	**1.86**	**0.73**	**0.81**
Contained Gold	**000 oz Au**	**471**	**636**	**308**	**299**	**5**	**76**	**41**	**1,835**
Mineral Reserves by Material Type									
Proven									
Regolith									
Ore	000 t	375	945	-	-	29	1	22	**1,372**
Gold Grade	g/t Au	0.62	0.49	-	-	0.47	1.50	0.44	**0.53**
Contained Gold	000 oz Au	7	15	-	-	0	0	0	**23**
Saprolite									
Ore	000 t	6,353	8,317	-	-	115	5	154	**14,944**
Gold Grade	g/t Au	0.71	0.74	-	-	1.09	1.11	0.88	**0.73**
Contained Gold	000 oz Au	144	197	-	-	4	0	4	**350**
Upper Transition									
Ore	000 t	1,264	2,573	-	-	15	7	39	**3,897**
Gold Grade	g/t Au	0.91	0.74	-	-	0.84	1.55	1.32	**0.80**
Contained Gold	000 oz Au	37	61	-	-	0	0	2	**100**
Lower Transition									
Ore	000 t	550	1,011	-	-	6	16	-	**1,583**
Gold Grade	g/t Au	1.18	0.98	-	-	0.91	1.90	-	**1.06**
Contained Gold	000 oz Au	21	32	-	-	0	1	-	**54**
Sulphide									
Ore	000 t	480	365	-	-	1	810	-	**1,657**
Gold Grade	g/t Au	1.22	0.98	-	-	0.88	2.00	-	**1.55**
Contained Gold	000 oz Au	19	12	-	-	0	52	-	**83**

Classification	Units	Block 1	Block 2	Block 3	Block 4	Block 5	Block 6	Restricted Zones	Total
Probable									
Regolith									
Ore	000 t	452	666	838	424	9	1	77	**2,467**
Gold Grade	g/t Au	0.52	0.50	0.51	0.50	0.66	1.20	0.51	**0.51**
Contained Gold	000 oz Au	8	11	14	7	0	0	1	**40**
Saprolite									
Ore	000 t	5,798	6,488	7,068	3,438	-	2	1,009	**23,803**
Gold Grade	g/t Au	0.63	0.60	0.69	0.70	-	0.86	0.71	**0.65**
Contained Gold	000 oz Au	118	125	156	77	-	0	23	**499**
Upper Transition									
Ore	000 t	758	1,424	1,876	1,559	-	8	431	**6,056**
Gold Grade	g/t Au	0.73	0.77	0.77	0.76	-	1.13	0.74	**0.76**
Contained Gold	000 oz Au	18	35	47	38	-	0	10	**148**
Lower Transition									
Ore	000 t	331	778	823	610	-	10	-	**2,552**
Gold Grade	g/t Au	1.15	1.09	1.09	1.06	-	1.19	-	**1.09**
Contained Gold	000 oz Au	12	27	29	21	-	0	-	**90**
Sulphide									
Ore	000 t	1,971	3,316	1,750	4,325	-	407	-	**11,768**
Gold Grade	g/t Au	1.37	1.15	1.12	1.12	-	1.63	-	**1.19**
Contained Gold	000 oz Au	87	122	63	156	-	21	-	**449**
Subtotals Proven & Probable									
Regolith									
Ore	000 t	827	1,611	838	424	38	2	100	**3,839**
Gold Grade	g/t Au	0.56	0.50	0.51	0.50	0.51	1.32	0.49	**0.51**
Contained Gold	000 oz Au	15	26	14	7	1	0	2	**63**
Saprolite									
Ore	000 t	12,152	14,805	7,068	3,438	115	7	1,163	**38,747**
Gold Grade	g/t Au	0.67	0.68	0.69	0.70	1.09	1.03	0.73	**0.68**
Contained Gold	000 oz Au	262	322	156	77	4	0	27	**849**
Upper Transition									
Ore	000 t	2,021	3,997	1,876	1,559	15	15	471	**9,954**
Gold Grade	g/t Au	0.85	0.75	0.77	0.76	0.84	1.32	0.78	**0.78**
Contained Gold	000 oz Au	55	96	47	38	0	1	12	**248**
Lower Transition									
Ore	000 t	881	1,789	823	610	6	26	-	**4,135**
Gold Grade	g/t Au	1.17	1.03	1.09	1.06	0.91	1.63	-	**1.08**
Contained Gold	000 oz Au	33	59	29	21	0	1	-	**143**

Classification	Units	Block 1	Block 2	Block 3	Block 4	Block 5	Block 6	Restricted Zones	Total
Sulphide									
Ore	000 t	2,451	3,681	1,750	4,325	1	1,216	-	**13,425**
Gold Grade	g/t Au	1.34	1.13	1.12	1.12	0.88	1.88	-	**1.23**
Contained Gold	000 oz Au	106	134	63	156	0	73	-	**531**

Notes:
1. *Mineral Reserves have been estimated in accordance with the CIM Definition Standards.*
2. *Mineral Reserves are based on cut-off grades that range from 0.300 to 0.325 g/t Au for oxides, and 0.466 to 0.555 g/t Au for sulphides.*
3. *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability.*
4. *There are 1.7Mt of low-grade mineralized oxide material above cut-off grade remaining in the stockpiles that are not included in the Mineral Reserves Estimate*
5. *Mineral Reserves are estimated at an average long-term gold price of US$1,250/troy oz.*
6. *Mineral Reserves are reported effective June 26, 2019*
7. *Mining recovery factors estimated at 98% for oxides and 96%-100% for sulphides.*
8. *Processing recovery varies by grade, weathering unit and location.*
9. *Rounding of some figures may lead to minor discrepancies in totals.*

1.7 Mining

The Bomboré mine will be developed as an open pit operation mining oxide and sulphide material from over 60 separate pits of variable size and depth across a mineralized zone approximately 12.2 km long and 3 km wide. The oxides include the regolith, upper saprolite, lower saprolite, and upper transition weathering units. The oxide material can be readily excavated in situ (free-dig material). The sulphides include the lower transition and fresh rock weathering units, which will require a varying degree of drill and blast prior to being loaded onto trucks.

This Technical Report considers mining of the Restricted Zones which are mining areas located within the floodplains of the Nobsin River, MV2 Stream, and Bomboré River. Mining within the Restricted Zones targets the free-dig oxide material and will only take place during the dry seasons. Mining, backfilling, and rehabilitation of the pits within these areas is to be fully completed prior to re-establishing river flow by the start of the next wet season.

The production schedule is based on the Mineral Reserve Estimate described in Section 15. Mining is planned to span 13.3 years with run-of-mine (ROM) ore delivered to the plant, followed by processing of low-grade stockpiles at the end of the mine life.

The key project life of mine (LOM) highlights are:

- 236.2 Mt total material mined:
  - 71.8 Mt of mineralized material:
  - 70.1 Mt of ore at 0.81 g/t Au mined and processed, including 52.5 Mt of oxides at 0.69 g/t Au and 17.6 Mt of sulphides at 1.19 g/t Au.
  - 1.7 Mt of mineralized low-grade material remaining on stockpiles and not processed at the end of the mine life.

  - 164.4 Mt waste
  - 2.34 strip ratio.
- 13.3-year mine life.
- Pre-production mining of 1.5 years, including excavation of the Off-Channel Reservoir (OCR) for water storage and supply.
- Total production:
  - 54.5 Mt at 0.88 g/t Au ROM ore
  - 15.6 Mt at 0.60 g/t Au low-grade ore re-handled from stockpiles
  - 1.6 Moz Au produced.

Mining of ore and waste will be contracted out with an owner's team responsible for site management, grade control, and mine planning activities. Mining of oxides will be undertaken with 4.5 m^3 hydraulic excavators (i.e. Komatsu PC850) and 30 - 50 t highway dump trucks. The sulphides will be mined using a separate fleet (i.e. Komatsu PC1250 and 50 t Volvo FMX rigid body trucks) to account for the increased density, abrasion and hardness of the material.

ROM ore will be hauled to the process plant and low-grade material hauled to the low-grade stockpiles.

Separate waste rock dumps (WRD) will be constructed for oxides and sulphides. Testwork is currently underway to determine acid generating and metal leaching potential of fresh rock, however results to date do not indicate that metal leaching or acid rock drainage (ARD) control will be an issue. If, however, testwork indicates that there are any such zones, they will be dealt with then and within an updated closure plan. Approximately 53% of the oxide waste produced will be used in the construction of the Tailings Storage Facility (TSF) with the remainder being hauled to the oxide WRDs and four environmental barriers.

Figure 1.44 and Figure 1.5 show the process feed schedule of oxide and sulphide ore respectively.

Figure 1.4 Oxides Process Feed Schedule

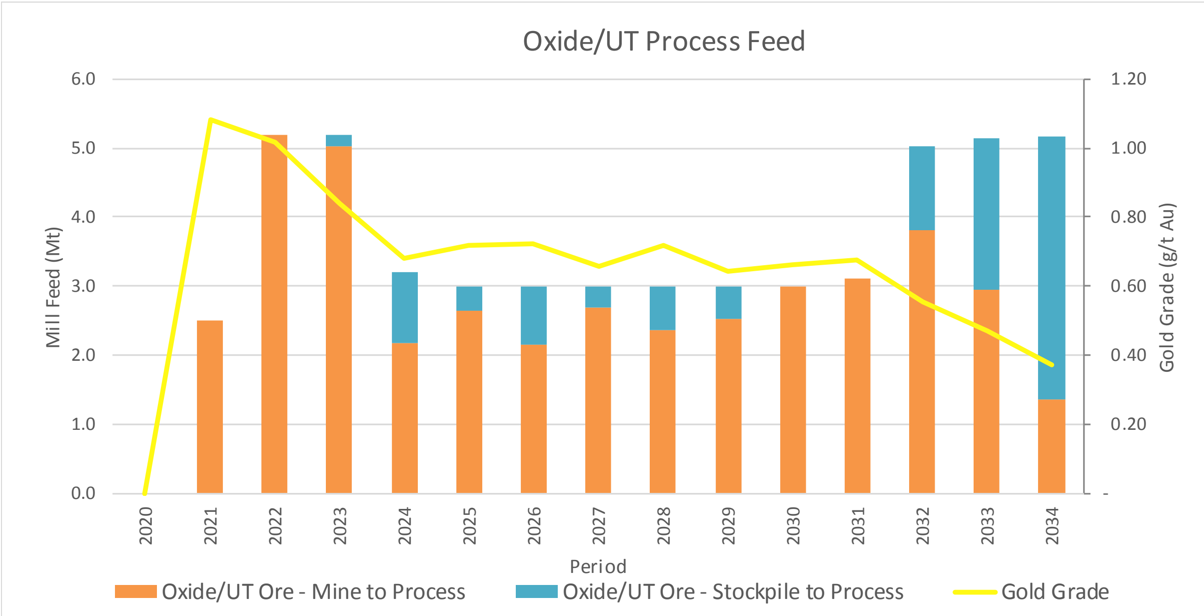
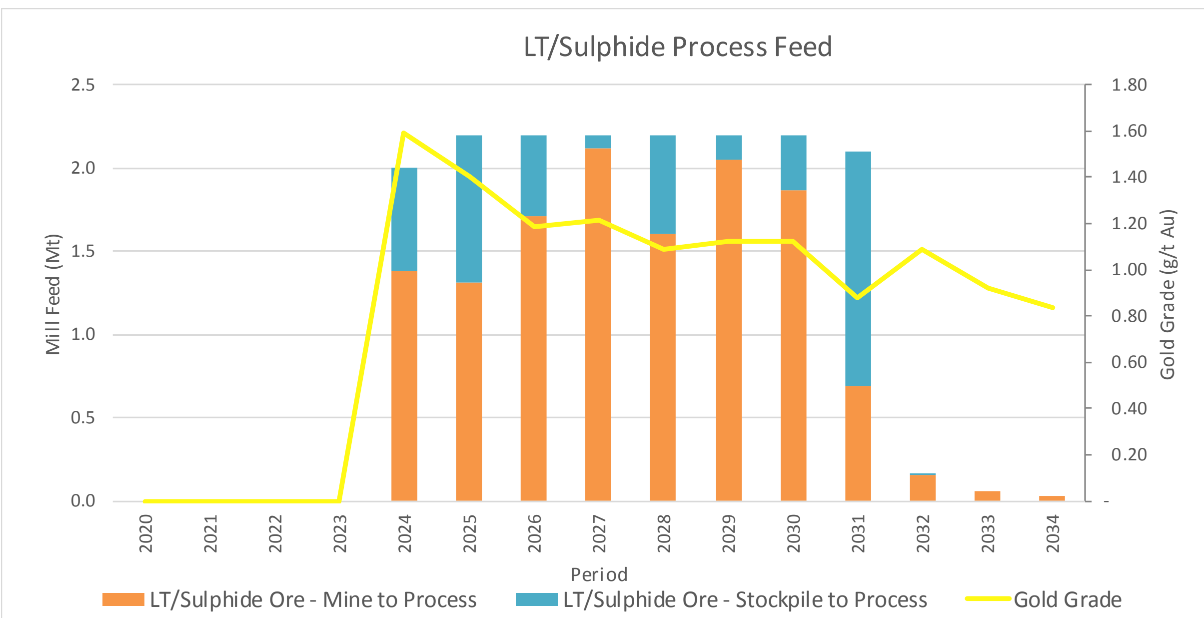


Figure 1.5 Sulphides Process Feed Schedule



The sulphide ore from year 2032 to 2034 will be crushed and processed through the oxide circuit, thereby eliminating the need to operate the sulphide SAG mill.

1.8 Metallurgy

Extensive testwork programs have been carried out at different laboratories for the Project with the first test program started in 1997 and the latest completed in 2019. The test programs were conducted on drill core composites, RC cuttings, and RAB drill samples considered representative of the ore deposit at the time of each test program. A summary list of the programs is included in Table 1.5.

Table 1.5 Summary of Testwork Programs

Program	Leachwell Recoveries	Head Analysis	Variability	Cyanidation	Gravity	Flotation	Carbon-in-Leach (CIL)	Carbon Adsorption & Equilib.	Column Leach (HL)	Comminution	Scrubbing	Gold Deportment	Petrography	Thickening / Rheology	Neutralization	Lime Demand	Acid Mine Drainage
SGS / ITS 1997			✓	✓									✓				
Osborne 2008			✓	✓													
AMMTEC 2009		✓	✓	✓	✓	✓	✓		✓	✓							✓
McClelland 2012*		✓	✓	✓	✓	✓	✓			✓			✓	✓	✓		
Phillips 2012										✓							
OREZONE Scrubbing 2012			✓	✓							✓	✓					
Met-Solve 2013											✓	✓					
SGS Lakefield 2013										✓							
COREM 2013				✓						✓			✓				
Met-Solve 2014				✓			✓				✓						
Consolidated Database 2013	✓																
Kappes 2014			✓	✓			✓		✓		✓	✓		✓	✓		
SGS Lakefield 2014										✓							
SGS Lakefield 2016				✓	✓	✓				✓			✓				
SGS Lakefield 2017/2018			✓	✓						✓						✓	
Outotec 2018														✓			
Base Metallurgical Lab 2019		✓	✓	✓						✓			✓	✓			
SGS Lakefield 2019								✓									

*****Includes Pocock report in appendix

A summary of the metallurgical inputs to the oxide plant and sulphide plant process design criteria, derived from the interpretation of the testwork, are presented in Table 1.6 and Table 1.7, respectively.

Table 1.6 Summary of Metallurgical Criteria for Oxide Plant

Criteria	Units	Design	Notes/Source
Plant Throughput	tpa	5,200,000	Orezone
Ore Type	-	Upper & Lower Oxide Upper Transition	Mine plan
Design Ore Blend - Upper & Lower Oxide - Lower Transition	% %	85 15	Mine plan Mine plan
Head Grade - Gold (Design) - Gold (LOM average)	g/t Au g/t Au	1.0 0.67	Lycopodium/Orezone Mine plan
Gold Recovery Estimation at 1 g Au/t - Upper Oxide - Lower Oxide - Upper Transition - Per Design Ore Blend	 % % % %	 92.2 92.0 91.7 92.0	 Recovery plan Recovery plan Recovery plan Calculated
Ore Specific Density Ore Bulk Density Crushing Work Index (CWi) Rod Mill Work Index (RWi) Bond Ball Mill Work Index (BWi) Bond Abrasion Index (Ai)	t/m^3 t/m^3 kWh/t kWh/t kWh/t g	2.8 1.65 7.7 5.8 4.8 0.031	Testwork Lycopodium/Orezone Testwork Testwork Testwork Testwork
Grind Size P_{80} CIL Circuit Residence Time CIL Slurry Density (for saprolitic ore)	µm hrs % solids	125 24 ~40%	Lycopodium Testwork Lycopodium
Thickener Solids Loading	$t/m^2{\cdot}h$	0.60	Testwork
Sodium Cyanide Addition	kg/t NaCN	0.28	Testwork/Calculated
Quicklime Addition	kg/t CaO	1.86	Testwork/Calculated

Table 1.7 Summary of Metallurgical Criteria for Sulphide Plant

Criteria	Units	Design	Notes/Source
Plant Throughput	tpa	2,200,000	Orezone
Ore Type	-	Lower Transition Upper & Lower Sulphide	Mine plan
Design Ore Blend - Lower Transition - Upper & Lower Sulphide	% %	24 76	Mine plan Mine plan
Head Grade - Gold (Design) - Gold (LOM average)	g/t Au g/t Au	1.25 1.22	Lycopodium/Orezone Mine plan
Gold Recovery at 1.25 g/t Au - Lower Transition - Upper & Lower Sulphide - Pit P17S Only* - Per Design Ore Blend	 % % % %	 88.0 82.6 94.95 84.6	 Recovery model Recovery model Recovery model Calculated
Ore Specific Density Ore Bulk Density Crushing Work Index (CWi) Rod Mill work Index (RWi) Bond Ball Mill Work Index (BWi) A x b Parameter Bond Abrasion Index (Ai)	t/m^3 t/m^3 kWh/t kWh/t kWh/t g	2.8 1.65 19.8 17.1 16.9 27.0 0.258	Testwork Lycopodium/Orezone Testwork Testwork Testwork Testwork Testwork
Grind Size P_{80} CIL Circuit Residence Time CIL Slurry Density	µm hrs % solids	75 24 ~50%	Testwork Testwork Lycopodium
Thickener Solids Loading	$t/m^2 \cdot h$	0.50	Orezone
Sodium Cyanide Addition	kg/t NaCN	0.78	Testwork/Calculated
Quicklime Addition	kg/t CaO	1.35	Testwork/Calculated

** Amount of total processed ore for sulphide plant coming from Pit P17 is 6% per mine plan.*

The following conclusions can be drawn from the metallurgical testwork:

- Oxide, transition and sulphide ores at Bomboré are readily amenable to CIL whole ore cyanidation.
- Oxide plant: Gold recoveries are predicted to be over 90% for head grades over 0.80 g/t Au, high 80%'s for head grades of 0.55 g/t Au to 0.80 g/t Au, and low 80%'s for head grades of 0.40 g/t Au to 0.55 g/t Au.
- Sulphide plant: Gold recoveries are predicted to be over 80% for head grades over 0.70 g/t Au, and stay in the high 70%'s even for lower head grades.
- Optimum grind size for the oxide plant was determined to be P_{80} of 125 µm based on grind size and recovery relationship.
- Optimum grind size for the sulphide plant was selected to P_{80} of 75 µm however, this still requires additional investigation.

- Leach extraction rates are essentially complete within 24 hours based on the observed leach kinetics.
- Oxygen addition is beneficial for sulphide ore leaching.
- Cyanide consumption rates are expected to be low, averaging about 0.19 kg/t NaCN for the oxide ore and about 0.37 kg/t NaCN for the sulphide ore.
- Lime consumption rates are expected to be moderate, averaging about 1.86 kg/t CaO for the oxide ore and about 1.35 kg/t CaO for the sulphide ore.

1.9 Process Plant

The process plant design is based on a robust metallurgical flowsheet developed for optimum recovery while minimizing initial capital expenditure and life of mine operating costs. The flowsheet is based on unit operations including crushing, milling, Carbon-in-Leach (CIL) leaching, Zadra elution, gold electrowinning and carbon regeneration that are well proven in the industry.

The process plant design has been based on a nominal capacity of 5.2 Mtpa. Initial plant feed will consist of the soft oxide and upper transition ore types only processed through a grinding and CIL circuit designed for the softer ores. For these materials crushing is not required and the material has low grind energy requirements.

After the third year, plant feed will include more competent lower transition and fresh ores and these will be crushed and ground through a separate crushing and grinding circuit feeding a leach circuit designed to provide the additional leach residence time required by these ores. The partially leached sulphide slurry will then be combined with the oxide feed to the CIL circuit displacing approximately 2.2 Mtpa of the lower grade oxide ores which will be processed later in the mine life.

The oxide plant will be designed for a 5.2 Mtpa throughput in order to accommodate the initial years of mine schedule where only oxide and upper transition ore will be processed. The sulphide plant portion will be added after the third year to allow the treatment of lower transition and fresh ore. The sulphide plant will process 2.2 Mtpa of sulphide ore, with oxide throughput reduced to 3.0 Mtpa to maintain an overall plant throughput of 5.2 Mtpa.

As the oxide and upper transition material is fine grained generally friable and essentially free of quartz the oxide circuit does not include a crusher. The trucks transporting the oxide plant feed will drive across a static grizzly while rear-dumping the ore. The grizzly will be kept clear, as necessary, by a front-end loader. The saprolitic ore will be broken further by chains on the ROM bin discharge apron feeder and will be fed, by conveyor, into a ball mill for slurrying and grinding the small coarse fraction. Similar flowsheets are successfully used elsewhere in West Africa and South and Central America for fine and friable saprolitic material.

The treatment plant design incorporates the following process unit operations:

Oxide Plant

- ROM ore fed through a static grizzly to a surge bin.
- Apron feeder and conveyor feed to the milling circuit.
- A single stage ball mill, in closed circuit with hydrocyclones, to produce a P_{80} grind size of 125 µm.
- A hydrocyclone pack with overflow slurry density of 40% w/w solids for direct feed to the leach tanks.
- A leaching circuit with one leach and seven CIL tanks to achieve the required 24 hours of residence time for optimum leach recovery.
- A tailings thickener for cyanide and water recovery.
- Loaded carbon acid wash and pressure Zadra elution circuit with gold electrowinning and recovery to doré.
- Carbon regeneration kiln to remove organic foulants from the carbon and reactivate the adsorption sites on the activated carbon.

Sulphide Plant

- Primary crushing with a jaw crusher to produce a P_{80} of 105 mm.
- Mill feed surge bin that overflows to an approximately 4,900 t stockpile to provide 18 hours of surge capacity.
- The grinding circuit is a SSAG type, which consists of a closed-circuit single stage SAG mill with pebble recycling to produce a final P_{80} of 75 µm. Provision has been made to install a pebble crusher in the future should, additional throughput capacity be desired.
- A hydrocyclone pack with an overflow slurry density of 25% w/w solids to maintain efficient particle size separation.
- A pre-leach thickener to increase leach slurry density, which in turns also minimizes leach tank volume and reduce overall reagent consumptions.
- A leach circuit with a pre-oxygenation tank followed by three leach tanks to provide 24 hours of residence time for optimum recovery. Partially leached slurry is pumped to the oxide plant for further processing in its CIL circuit providing an overall leach duration of 48 hours for the sulphide ores.

Process block flow diagrams depicting the unit operations incorporated in the oxide, sulphide and combined circuits are presented in Figures 1.6, 1.7 and 1.8. Plan and isometric views of the process plant are provided in Figures 1.9 and 1.10.

Figure 1.6 Overall Process Flow Diagram for Oxide Circuit

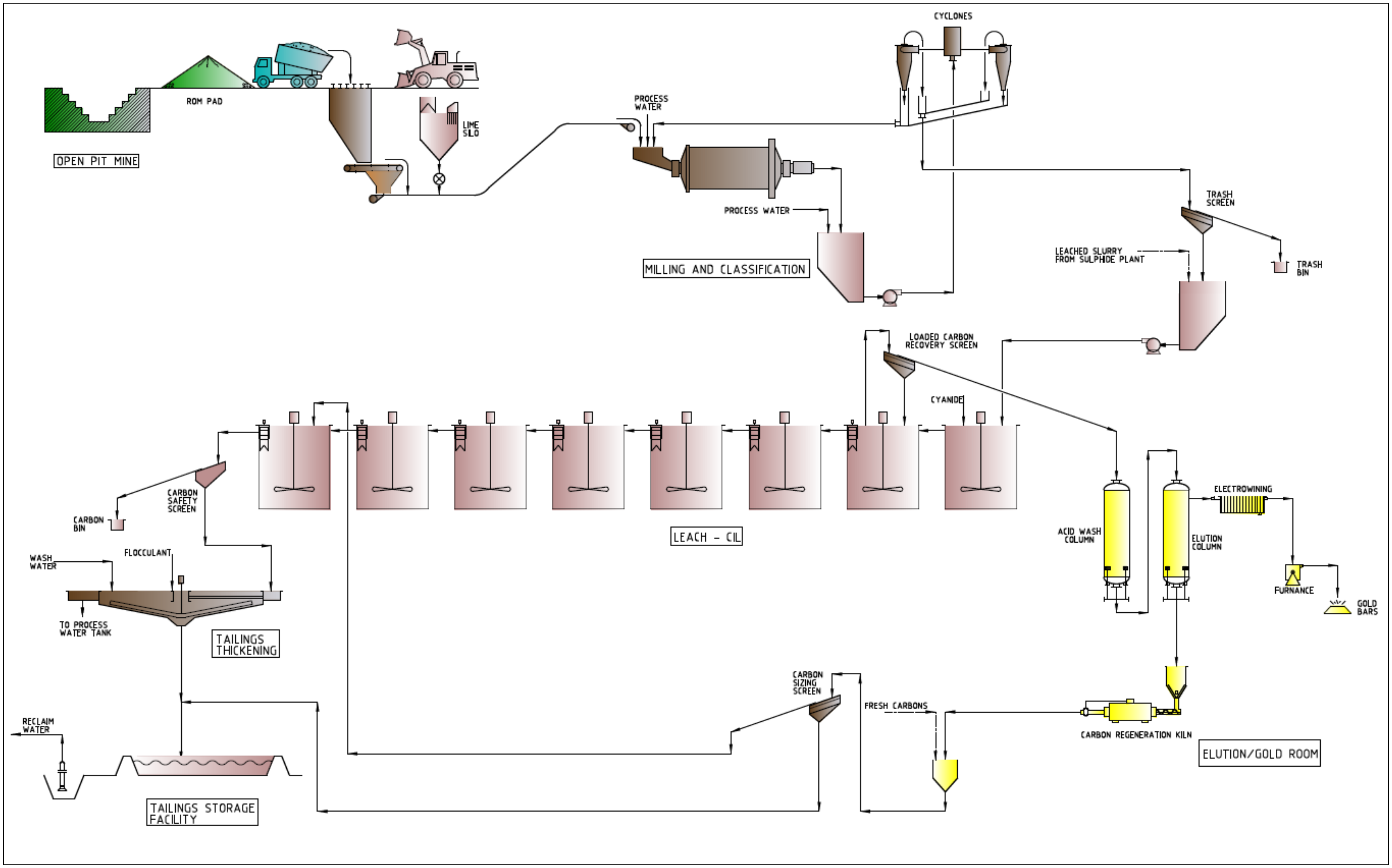


Figure 1.7 Overall Process Flow Diagram for Sulphide Circuit

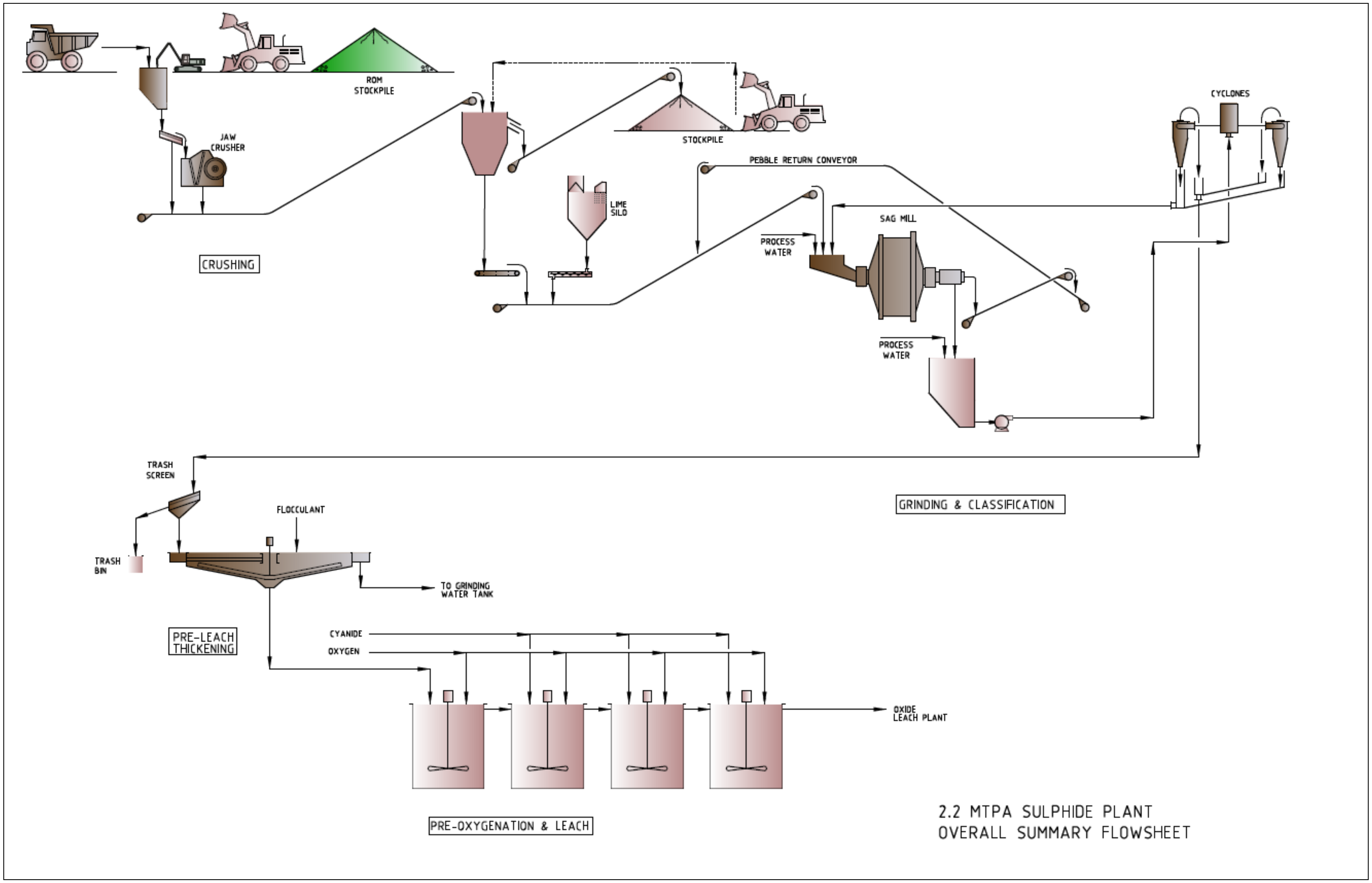


Figure 1.8 Process Flow Diagram for Combined Circuits

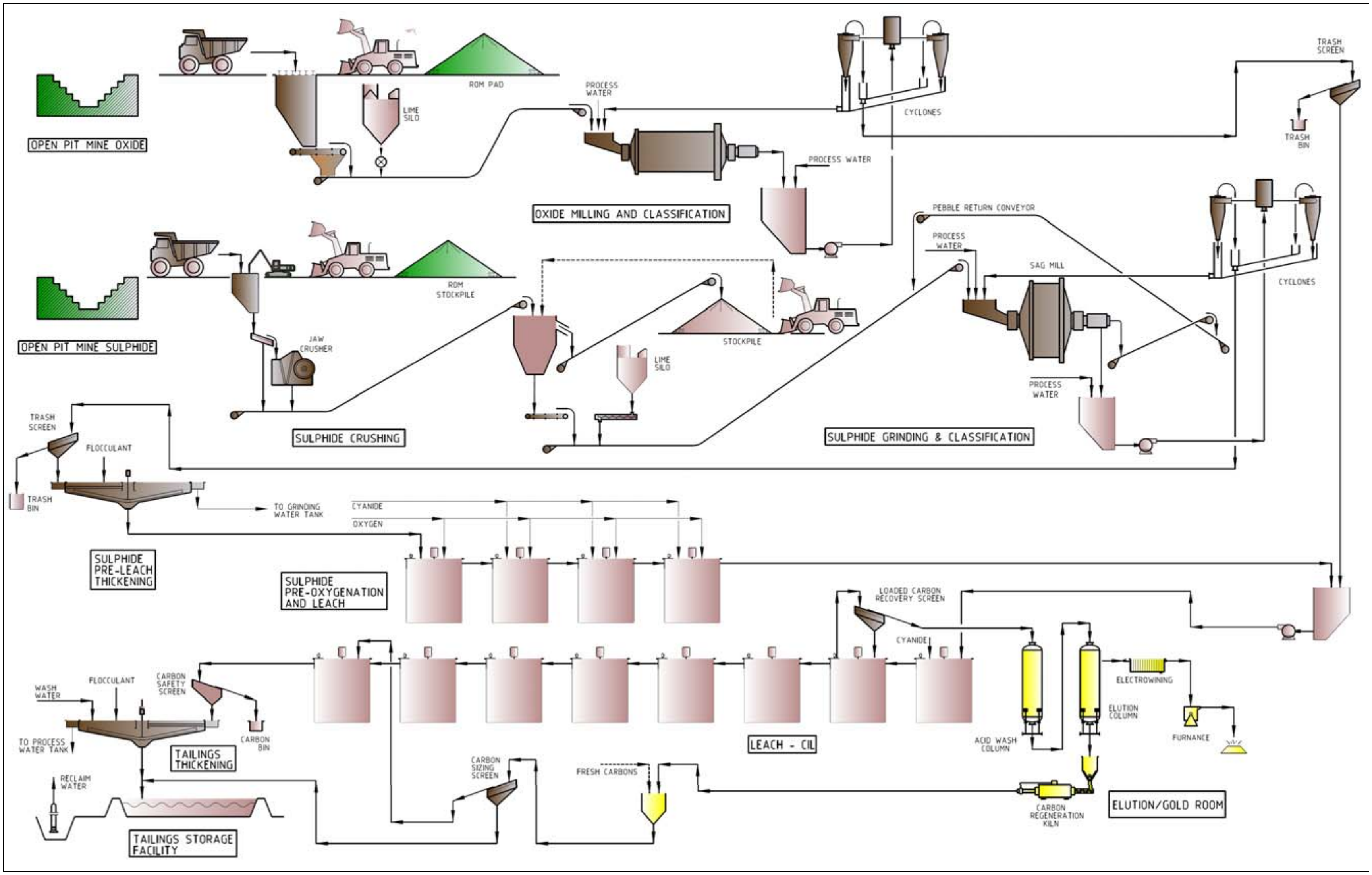


Figure 1.9 Combined Circuits Plan View

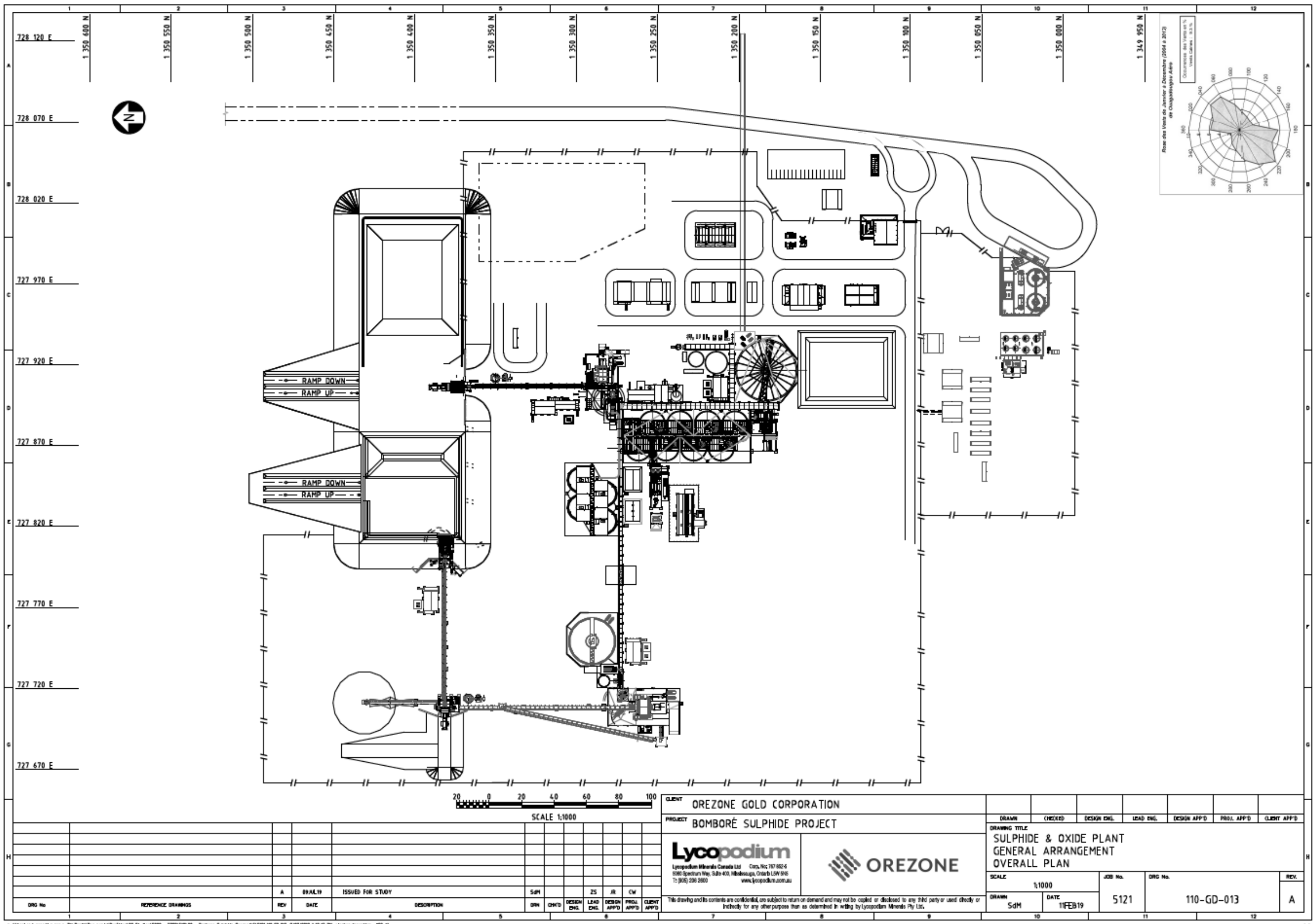
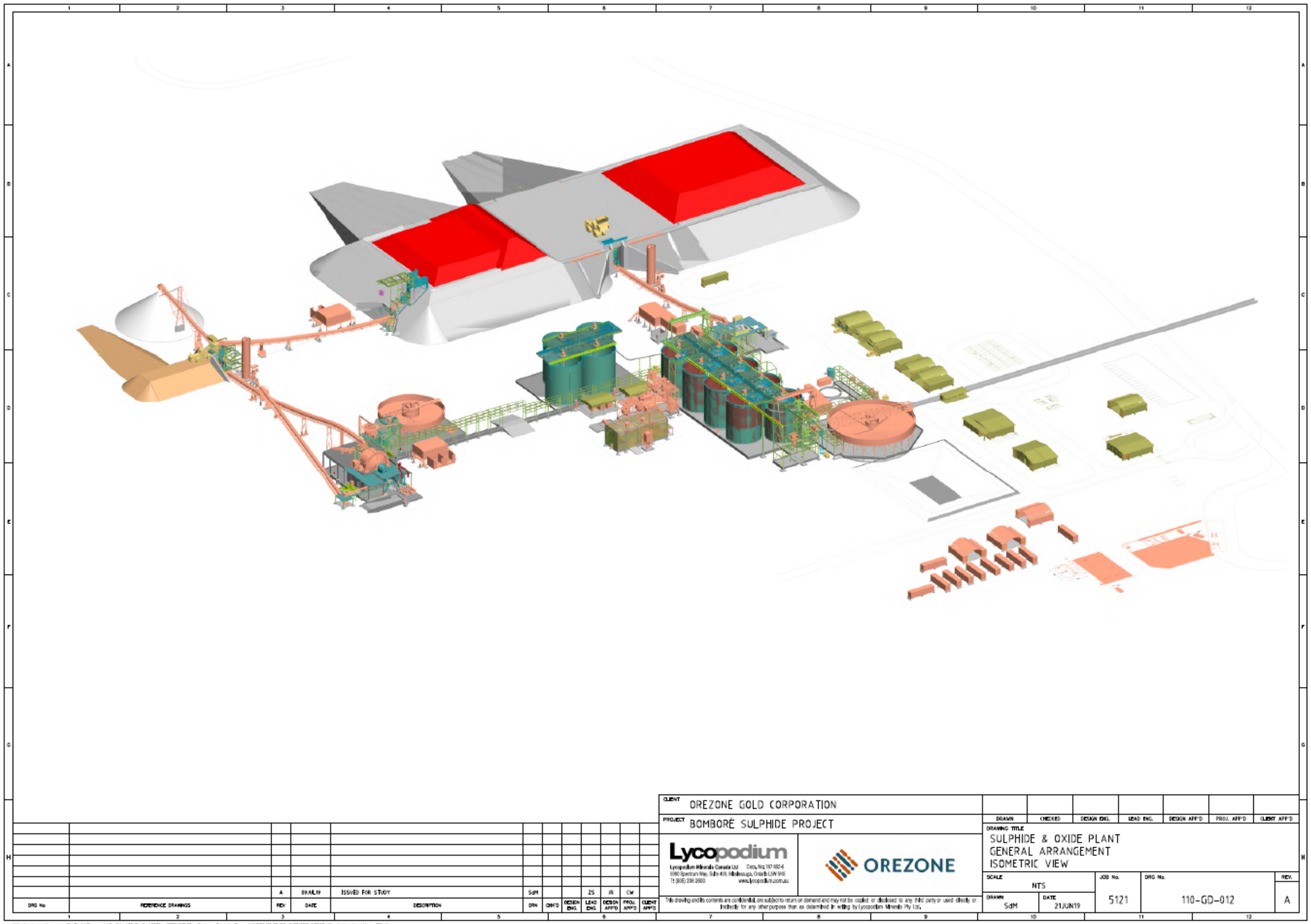


Figure 1.10 Combined Circuits Isometric View



The process plant is designed to operate with zero discharge of process solutions to the environment. To ensure compliance the plant includes a lined event pond designed to contain any foreseeable spillage event. The event pond, combined with the bunded concrete areas within the plant perimeter, is designed to contain the run-off from a one in a hundred-year storm event occurring simultaneously with the catastrophic failure of the largest slurry-containing vessel within the plant site. Material accumulating in the event pond will be returned periodically to the tailings thickener circuit.

To the greatest extent possible the process plant will re-use process water recovered from the tailings thickener and TSF to meet the process plant requirements. Raw water will only be used for applications where water quality with low dissolved solids is required and as make-up in the process water circuit.

The general control philosophy for the plant will be one with a moderate level of automation and central 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.

1.10 Services and Infrastructure

Overall site layouts at the completion of construction and prior to project decommissioning are provided in Figures 1.11 and 1.12 below.

The overall site major facilities include mine open pits, process plant, TSF, mine services, fuel storage and distribution, waste dumps, and access road, camp and relocation areas. Power is provided by an on-site power plant. The site will be fenced to clearly delineate the mine area and deter access by unauthorized persons and prevent grazing animal access.

Figure 1.11 Project Site – Plant Commissioning

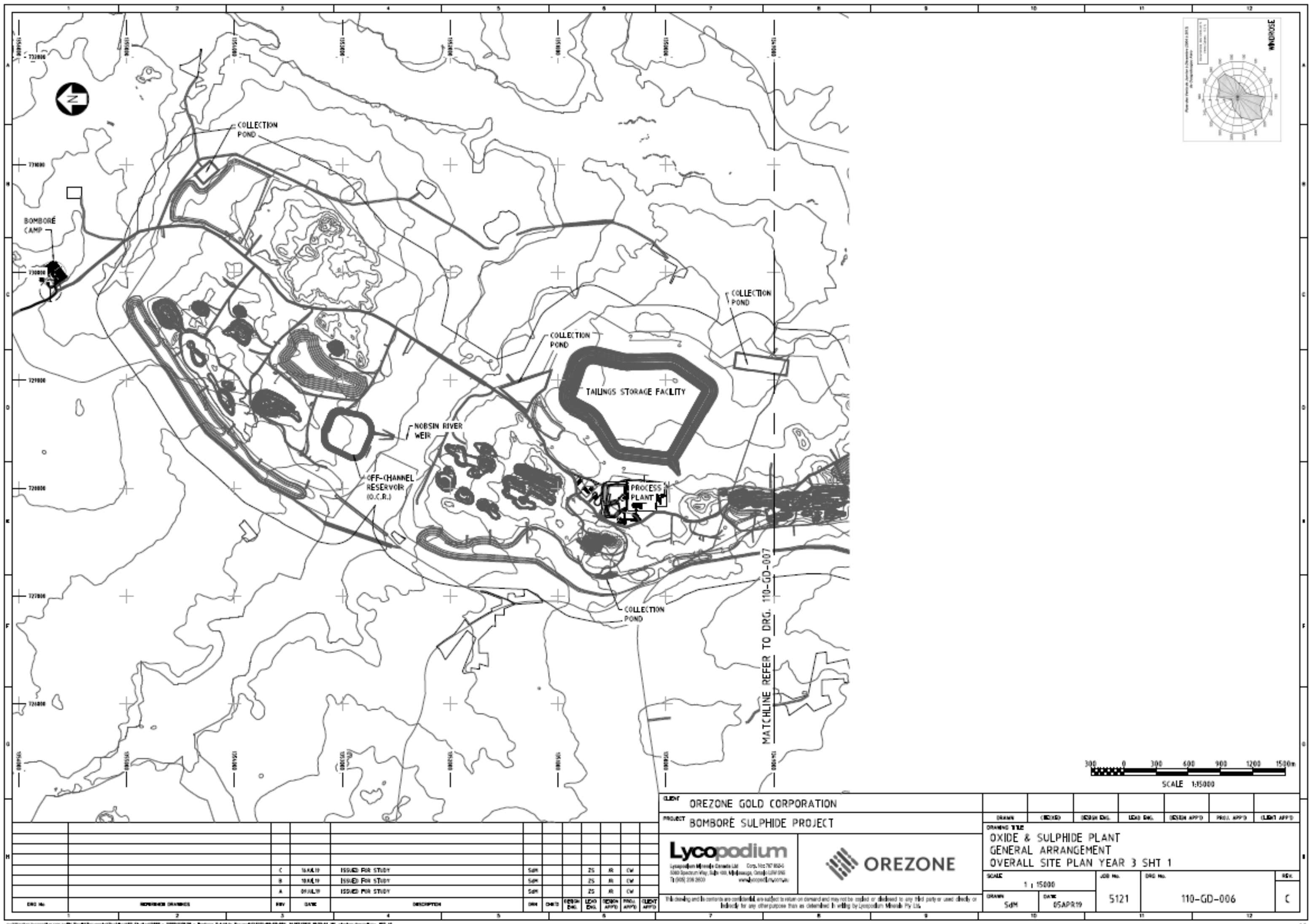
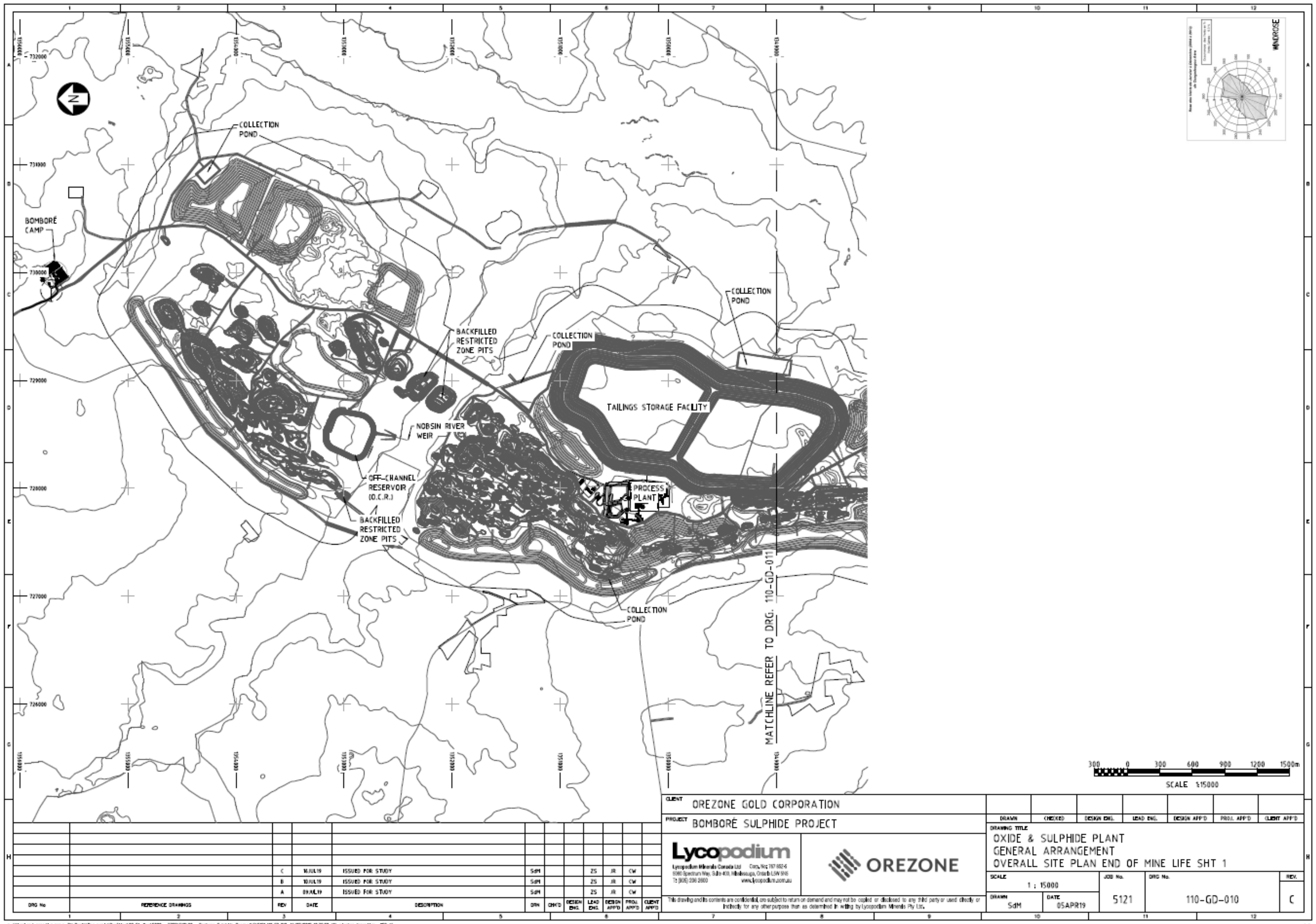
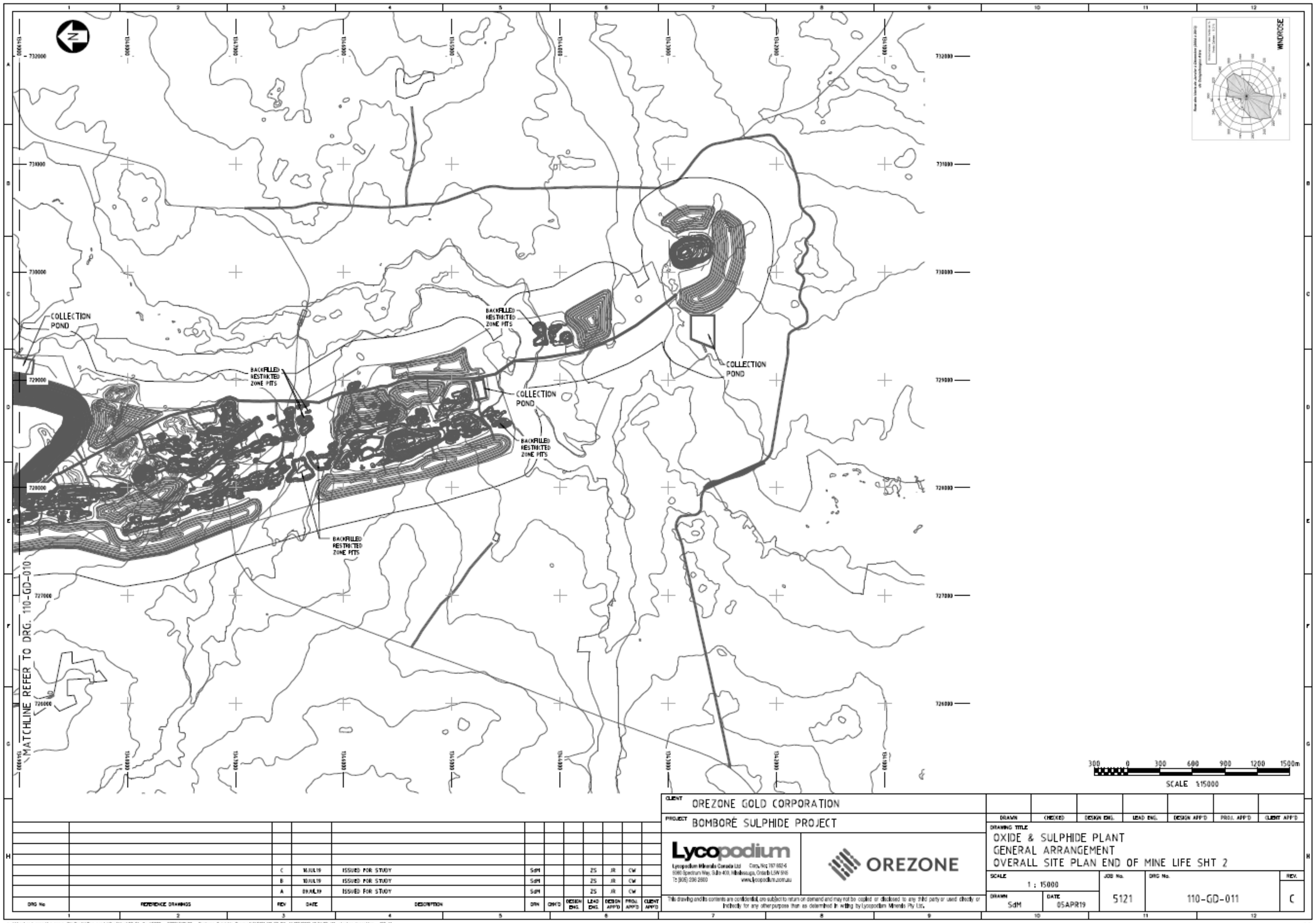


Figure 1.12 **Project Site – Prior to Decommissioning**




COLLECTION POND
BACKFILLED RESTRICTED ZONE PITS
COLLECTION POND
BACKFILLED RESTRICTED ZONE PITS
COLLECTION POND
BACKFILLED RESTRICTED ZONE PITS
COLLECTION POND
BACKFILLED RESTRICTED ZONE PITS
MATCHLINE REFER TO DRG. 110-GD-010
WINDROSE
SCALE 1:15000
OREZONE GOLD CORPORATION
BOMBORÉ SULPHIDE PROJECT
Lycopodium
OREZONE
OXIDE & SULPHIDE PLANT
GENERAL ARRANGEMENT
OVERALL SITE PLAN END OF MINE LIFE SHT 2
1 : 15000
SdM
05APR19
5121
110-GD-011
C
ISSUED FOR STUDY
ISSUED FOR STUDY
ISSUED FOR STUDY

The existing camp at the northern end of the Project area provides accommodation and office facilities for the exploration teams and Orezone support personnel currently at site and has been expanded to accommodate 116 with room for additional accommodation blocks. The camp will accommodate owners and EPC/EPCM engineer's staff during construction and non-local staff during operations. With many administration functions centred in the office in Ouagadougou there will be a minimal requirement for site office space. This will be met by the existing office block at the camp.

The Project site is within a thirty-minute drive from the regional town of Mogtédo, with a population of more than 15,000. The town is developing rapidly with many substantial multi story concrete block buildings established or under construction. Most of the semi-skilled and unskilled labour required for project development and operations will be sourced from Mogtédo and surrounding villages. As the town has the capacity to provide rented rooms and leased accommodation contractors will make their own accommodation arrangements with local businesses. Contractors will also make arrangements for bussing their employees to and from site and for providing a midday meal.

Power and water will be supplied to an area north of the plant site and adjacent to the diesel fuel facility, where the mining contractor will establish the mine service area. The mine service area will provide offices, meals and ablution facilities for the contractor personnel plus workshop/warehouse facilities for servicing the mining fleet.

Power will be provided by a site power station operating under a 'build, own, operate' (BOO) contract arrangement with an independent power provider (IPP). A containerized HFO power station will be provided and considered the most 'fit for purpose' with adequate operating flexibility at a low over-the-fence power cost. The power station will be an 'n+2' configuration (n units running with two on standby) with sufficient installed power to meet the power demand surge when starting the mill.

The electrical system is based on 11 kV distribution and 415 V working voltage. 11 kV overhead power lines will distribute power across the site, stepped down at point of use with pole top transformers, kiosks or conventional transformers and MCCs as appropriate.

During the initial oxide treatment phase the annual average electrical load on site is estimated to be 6.6 MW with a peak demand of 8.6 MW. Annual power consumed is estimated to be 54.9 GWh. The sulphide treatment plant will add a further 8.42 MW of annual average electrical load (9.27 MW peak). When treating sulphides, however, the oxide load will reduce as the oxide circuit will be operating at a lower throughput. The combined annual average electrical load is estimated to be 14.18 MW. The average annual energy consumption with both the oxide and sulphide circuits in operation is estimated to be 124.2 GWh.

Site security is based on concentric lines of fencing/access control. The entire project area will be enclosed within a patrolled agricultural type stock fence line to prevent animal access and discourage casual entry by unauthorized persons. The main point of entry will be located where the main access road enters the site. This point of entry will be provided with a gate and manned security post. Security personnel contracted to Orezone will be supplemented by an armed detachment from the National security forces.

Fuel will be provided by a local supplier and based on a BOO contract. The permanent, supplier operated, fuel depot will have a minimum storage capacity of 14 days of HFO and diesel. With the proximity of the Project to Ouagadougou this is considered an adequate site fuel reserve. During construction a temporary fuel depot will be established using ‘bullet’ type tanks leased from the fuel supplier. The temporary depot will be within a bund constructed in accordance with appropriate international standards to contain fuel spills and will have an oil/water separation system for draining rainwater.

Tailings from the Project process plant will be disposed of in a fully lined TSF that will be stage-developed at a site immediately east of the process plant. The tailings will be conveyed as a slurry and placed hydraulically into the facility in a controlled manner from a series of strategically positioned drop bar pipes around much of the perimeter to build a consolidated and stable deposit. Bleed water will be recovered from a supernatant pond on the surface of the tailings via a decant structure and continuously recycled to the process plant for re-use. The TSF has been designed to a high standard for security, safety, stability and environmental protection. Canadian Dam Safety (CDA) Guidelines have been followed for dam safety, and the principles of the Mining Association of Canada’s (MAC) guidelines for successful overall tailings management have also been followed where applicable.

In general, the operational water management strategy is to utilize water captured within the mine limits to the maximum practical extent in an efficient manner. This includes significant water storage, recirculation, and reuse efforts. Collection and retention of rainfall-run-off that comes into contact with the stockpiles, waste dumps, and TSF dam will be largely achieved by constructing diversion channels and collection ponds. The collected water will be pumped to the process plant and/or TSF ponds for use in the process.

Raw (i.e., fresh) water from the Nobsin River will be harvested during a portion of each wet season and stored in the Off-Channel Reservoir (OCR) for year-round use. The amount of water that will be harvested each year will be a minor portion of the Nobsin River streamflow and will not negatively impact downstream users. The OCR will serve as the main raw water supply source for the Project. Specifically, it will supply water to the process plant, for dust control, and TSF dam construction water (i.e., moisture conditioning of the dam fill) demands.

1.11 Environmental and Social Impact

The approach developed by Orezone throughout the various environmental and social studies that have been conducted since 2009, especially in the context of the Environmental and Social Impact Assessment (ESIA), emphasized stakeholder concerns and integrated the environmental and social aspects into the initial stages of the Project design. This approach maximized the Project’s integration into the environment and has minimized its negative impacts, thus increasing the environmental and social acceptability of the Project. In addition, this approach allowed better consideration of the social aspects arising from the resettlement of households that will be required due to an eventual mining project.

The legal framework with respect to environmental and social aspects related to economic activities is supported by many laws and decrees, including:

- Environmental Code.
- Mining Code.
- Forest Code.
- Public Health Code.
- General Local Authorities Code.
- Act on Rural Land Tenure.
- Act on Agrarian and Land Reorganization.
- Law on Water Management.
- Act on Pastoralism.

There are three types of mining permits according to the Mining Code***:***

- Exploration Permit.
- Industrial Operating Permit.
- Operating Permit for Semi Mechanized Mining.

The application for an Industrial Operating permit requires an ESIA that must first be accepted by MEEVCC. The ESIA must be supported by a Feasibility Study (FS) and must include a Resettlement Action Plan (RAP) that has been accepted by all stakeholders if the project requires the expropriation of land held by any resident.

In 2016, Orezone received the Industrial Operating Permit following the delivery and acceptance by the authorities of the ESIA and RAP.

In February 2019, Orezone signed the mining convention with the State of Burkina Faso. The purpose of the mining convention is to clarify the rights and obligations of the parties and to guarantee Orezone stability, including taxation and foreign exchange regulation. The mining convention is not a substitute for the law but specifies the provisions of the law. It is valid for the initial duration of the operating license and is thereafter renewable for one or more periods of five years at the request of Orezone.

1.11.1 Environmental and Social Management Plan

The Environmental and Social Management Plan (ESMP) presents all the environmental and social management measures to be implemented as part of the Project including all the operational aspects. The ESMP covers all project phases and covers the avoiding, minimizing, enhancing, or compensating of the various anticipated negative impacts by reducing them to an acceptable level for all stakeholders.

The ESMP identifies the objectives to comply with the regulations in Burkina Faso and international best practices in the mining sector. The ESMP also includes environmental monitoring programs and environmental and social follow-up, providing the basis for assessing the effectiveness of management measures to be implemented by Orezone. The ESMP includes several measures to strengthen the capacity of the stakeholders concerned by the application of environmental and social management measures.

Management measures are to be implemented at the earliest stages of the construction phase. Some measures will last throughout the operations at the mine site and others will last beyond the closure and rehabilitation phase of the Project.

Some measures implemented during previous project phases concerning soil, surface water, groundwater, ambient noise, population and social cohesion, economy, and infrastructure, etc. will be maintained during the operational phase. Several additional measures will include the following:

- Monitoring of the mine tailings site in compliance with the applicable regulations and requirements.
- Management of waste rock dumps and progressive re-vegetation to minimize wind erosion.
- Management of water, hazardous materials, wastes, traffic, maintenance of vehicles, etc.
- Mining will be carried out according to best practices and with specific attention to occupational health and safety.

Finally, various management measures are planned for the closure phase and include the following:

- Dismantling of infrastructure and facilities, except for structures that will be kept in place and handled over to the local authorities without compromising the integrity and security of places and people.
- Site rehabilitation and re-vegetation.
- Restoration of livelihood conditions for neighbouring populations and workers.

Access roads, power lines and other infrastructures built for mining will be left in place, as necessary, for use by communities at the end of mine life. Restricted areas may be defined within the permit to protect the environment, the natural habitat, archaeological sites or public interest infrastructures.

A monitoring program will be implemented during the construction phase and will be conducted by Orezone on an ongoing basis. The program will ensure compliance with the commitments agreed to as part of the ESIA and environmental obligations, as well as compliance with the proposed management measures and with laws, regulations and other environmental considerations included in the contractors' technical specifications. These measures to implement will be included in the contractors' technical specifications according to their respective activities.

The main elements planned as part of the Project's follow-up monitoring activities include:

- Surface and ground water quality.
- Ambient air quality.
- Ambient noise.
- Status of the flora and effectiveness of re-vegetation.
- Fauna.
- Local and regional economy.
- Gender.
- Social cohesion.

Regarding water quality, the monitoring will determine if arsenic is leaching from the weathered mining wastes and if it is present in the process water. The geochemical studies conducted to date suggest that arsenic leaching will be minimal. Additional geochemical characterization will be performed at early construction phase to refine the existing geochemical model.

1.11.2 Resettlement Action Plan (RAP)

The resettlement of many people (about 731 households or about 5,095 people) from seven traditional villages, as well as two artisan gold processing sites (about 1,360 households or about 3,100 people) and the expropriation of a large area of agricultural land (about 656 ha) represents a complex activity that will require an immediate and important focused effort. The processing infrastructure is in the northern area of the Project where about 60% of the gold resources are located. This area will have to be cleared prior to the start of any major construction activities. This will require the initial (Stage 1) resettlement of approximately 410 households from traditional villages and the expropriation of approximately 915 households from the Sanam Yaar artisanal gold processing site. The subsequent resettlement (Stage 2) of approximately 250 farming households and the expropriation of 450 households from the Kagtanga artisanal gold processing site, all from the southern area of the Project, could occur after the initial Phase 1 resettlement as this area will not be immediately affected by the mine construction.

Orezone has successfully completed the expropriation and the settlement of the compensations to the households from the Sanam Yaar and Kagtanga artisan gold processing sites and construction of the Phase 1 resettlement sites is in progress.

1.11.3 Closure and Reclamation

The closure, decommissioning and reclamation costs of the Project of US$15.5M (before TSF related closure costs of US$2.4M) was included in the financial analysis for these closure activities related to the environmental and social aspects.

The Closure and Rehabilitation Plan includes work to be conducted from the closure of the mine, at the end of operation activities, as well as progressive rehabilitation work.

The goal is to return the site to a satisfactory state as quickly as possible in terms of:

- Reducing the risks for health and safety.
- Controlling erosion.
- Limiting maintenance and monitoring.
- Developing a compatible profile with the future uses of the site, primarily for the plant site.

The main objectives of the Closure and Rehabilitation Plan include restoring ecosystems and take-over and recovery of land uses. This plan includes:

- Dismantling and removal of plant equipment, machinery and infrastructure (except for structures that will be kept in place and handed over to the local authority without compromising the integrity and security of places and people).
- Progressive rehabilitation to allow rapid recovery of the vegetation cover and the early recovery of the ecosystem.
- Sustainability of rehabilitation work and control of water and wind erosion.
- Take-over and recovery of land uses.
- Maximization of material and equipment recovery.
- Site rehabilitation as part of a participatory approach involving interested communities.
- Implementation of a post-closure monitoring program.

In addition, a waste rock dump development program will be implemented and will notably include the development of agricultural plots. All structures that can be used by communities will be maintained, except for all facilities that may constitute a risk to people or the environment.

1.12 Capital and Operating Costs

1.12.1 Capital Costs

The overall study 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 Technical Report.

The Project Capital Costs in Table 1.8 exclude process operating costs associated with plant operations prior to achieving commercial production on October 1, 2021. The table also excludes the value of gold produced in that period and costs such as bullion transport and refining costs and government royalties associated with this gold production and sales. These additional capitalized expenses and the pre-commercial production gold revenue are addressed in the Project Economic Model.

Table 1.8 Project Capital Costs to 1 October 2021 (US$, 2Q 2019, ±15%)

Project Capital Area	US$ M
Construction In-directs	9.9
Treatment Plant	38.7
Reagents & Plant Services	12.8
Mining infrastructure	0.8
Site Infrastructure	21.3
Management Costs (EPCM)	11.2
Resettlement Action Plan	20.8
Owner's Costs[1]	26.1
Subtotal	**141.7**
Contingency	11.3
Subtotal	**153.0**
Mine costs (2020/2021)	23.9
Total	**176.9**

[1]*Excludes $0.9M in opening stock of consumables reclassified to working capital in the economic analysis.*

The capital cost estimate includes:

- Owner's costs (excluding the RAP expenditure) and other costs during the period.
- RAP expenditure.
- Process facilities.
- Mining infrastructure.
- Site infrastructure.
- Stage 1 of the TSF.
- Initial surface water management facilities.

- Installation costs, EPCM costs and contractor distributable costs.
- Site earthworks and site roads and tracks.
- Project contingency.

Exclusions include the following:

- Project sunk costs (including the site access road upgrade, camp upgrade and RAP costs classified as sunk costs).
- Import duties and taxes on the basis that the Project will be exempt.
- Sulphide plant expansion costs.
- Escalation.

The capital cost of the sulphide expansion is shown in Table 1.9 below.

Table 1.9 Sulphide Capital Costs (US$, 2Q 2019, ±15%)

	US$ M
Construction directs and In-directs	42.7
Management Costs (EPCM)	6.4
Owner's Costs	8.9
Subtotal	**58.0**
Contingency	5.2
Total	**63.2**

[1]*Excludes $1.4M in opening stock of consumables reclassified to working capital in the economic analysis.*

The Sustaining Capital Costs estimate for all areas is summarized in Table 1.10

Table 1.10 Sustaining Capital (US$, 2Q 2019, ±15%)

	Sustaining Capital Costs	Sustaining Total Cost (US$)	Year 2021	Year 2022	Year 2023	Year 2024	Year 2025	Year 2026	Year 2027	Year 2028	Year 2029	Year 2030	Year 2031	Year 2032	Year 2033
	TSF Stage		2	3	4	5	6	7	8	8	9	9	10	10	10
	Infrastructure														
1	Second Stage Tails Pump	239,115		239,115											
2	High Pressure Gland Water Pump	72,762		72,762											
3	TSF	52,833,643	3,681,409	4,940,917	7,461,905	2,816,560	3,404,527	6,570,155	2,742,044	3,894,966	3,834,844	5,226,089	3,395,254	4,864,974	
4	TSF Pipeline and Valves	5,139,674	371,602	428,124	1,556,539	342,849	279,767	642,796	252,497	252,497	244,431	244,431	262,070	262,070	
5	Surface Water Management	1,389,503	567,051	438,701			178,701		68,350		68,350		68,350		
	Mining														
6	Pit Dewatering Capital Costs	3,414,644	1,286,677	1,089,328	171,917	133,578	133,823				10,502	223,156		266,485	99,178
7	Surface Haul Road	1,124,151	300,950	473,439	222,834	71,489	23,145	22,899	6,017					3,378	
8	Main Access Road	530,086	436,891	41,326	51,869										
	G&A														
9	General & Admin items (vehicles, etc.)	1,500,000	100,000	200,000	100,000	200,000	100,000	200,000	100,000	200,000	100,000	100,000	100,000		
	Total	**66,243,578**	**6,744,579**	**7,923,712**	**9,565,063**	**3,564,477**	**4,119,963**	**7,435,850**	**3,168,908**	**4,347,463**	**4,258,127**	**5,793,676**	**3,825,674**	**5,396,907**	**99,178**

Closure and Salvage Costs are summarized in Table 1.11.

Table 1.11 Closure and Salvage Costs

Salvage and Closure Costs	Closure Year (US$M)
Salvage value (end of mine life)	
General Dismantling Cost	2.36
Salvage Value of Mechanical Equipment	-7.94
Closure Costs	
TSF Closure	2.37
Environmental/Social Management Plan/Rehabilitation	15.51
Total	**12.30**

1.12.2 Operating Costs

The Project operating cost estimate is built-up from three components:

- The mine operating costs developed by AMC.
- The process plant operating costs developed by Lycopodium.
- The general and administration (G&A) operating costs developed by Orezone and Lycopodium.

The estimated life-of-mine operating cost per tonne of ore treated and per ounce of gold produced is summarized in Table 1.12.

Table 1.12 Life-of-Mine Operating Costs per Tonne and per Gold Ounce (US$, 2Q 2019)

Cost Components	Total Cost ($M)	$/Tonne Processed	$/oz Au
Mining	386.3	5.51	242
Processing	456.9	6.52	286
G&A	139.4	1.99	87
Refining & Bullion Transport	2.4	0.03	1
Government Royalties & Dev Tax	103.9	1.48	65
Total Cash Cost	**1,089.0**	**15.53**	**681**

1.13 Annual and Life-of-Mine Production

Life of mine (LOM) ore milled will be 70.1 Mt over a 14-year period treating predominantly run of mine ore followed by the treatment of low-grade stockpiles once the pits are exhausted.

Mill feed grade will be significantly higher in the early years as lower grade run-of-mine material will be stockpiled for processing at the end of mine life. Annual tonnes mined, ore milled tonnes and grade and gold production are shown in Table 1.13.

Table 1.13 Annual and LOM Production

Year	Total ore tonnes processed (Mt)	Gold grade (g/t)	Recoveries (%)	Gold Production ('000 oz)
Pre-prod.	1.21	1.02	92.3	37
1	5.19	1.03	92.3	159
2	5.2	0.91	91.2	139
3	5.2	0.97	88.7	144
4	5.2	1.01	88.7	150
5	5.2	0.96	87.2	140
6	5.2	0.89	85.0	126
7	5.2	0.88	86.0	126
8	5.2	0.85	85.4	122
9	5.2	0.85	85.3	122
10	5.2	0.78	85.8	112
11	5.2	0.62	85.8	89
12	5.2	0.50	83.9	70
13	5.2	0.40	80.1	54
14	1.3	0.37	78.7	12
Life of Mine	**70.1**	**0.81**	**87.2**	**1600**

1.14 Economic Analysis

An economic assessment of the Project has been conducted using a pre and after-tax cash flow model prepared by Lycopodium on behalf of Orezone. 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 received from Orezone.

The cash flow model reports:

- All costs in real USD exclusive of escalation or inflation.
- A net present value (NPV) at a 5% discount rate.
- An internal rate-of-return (IRR) based on pre and post-tax net cash flows.
- Payback.

The Project life of mine production summary and cash flow model outcomes based on a gold price of $1,300/oz are summarized in Tables 1.14, 1.15 and 1.16 below.

Table 1.14 Production Summary

	Value
Ore processed	70.1 Mt
Total tonnes mined	236.2 Mt
Average head grade	0.81 g/t Au
Contained gold in material	1.8 Moz
Total gold produced	1.6 Moz
Average gold recovery	87.2%
Production life (processing)	13+ years
Nominal annual processing rate	5.2 Mtpa

Table 1.15 Net Profit after Tax Summary (LOM Summary)

	$M	$/Ore t Processed	$/oz Au
Revenue (99.93% payable)	**$2,078**	**$29.64**	**$1,299**
Mine Operating Cost	$386.3	$5.51	$241.5
Processing Cost	$456.9	$6.52	$285.7
G&A Cost	$139.4	$1.99	$87.2
Refining & Transport Costs	$2.40	$0.03	$1.5
Government Royalties	103.9	$1.48	$65.0
Total Cash Cost	**$1,089**	**$15.53**	**$680.8**
EBITDA	**$997.5**	**$14.23**	**$623.6**
Initial Capital	$153.0	$2.18	$95.6
Expansion Capital	$63.2	$0.90	$39.5
Sustaining Capital	$66.2	$0.94	$41.4
Rehabilitation & Closure (net of salvage)	$12.3	$0.18	$7.7
Total Capital Costs	**$294.7**	**$4.20**	**$184.3**
Gross Profit before tax	**$694.3**	**$9.90**	**$434.0**
Corporate Tax Payable	$187.2	$2.67	$117.0
Net Profit after tax	**$507.1**	**$7.23**	**$317.0**

Table 1.16 Financial Summary

	Value
Revenue from gold (99.93% payable)	$2,078M
Adjusted Operating Costs (AOC)	$681/oz Au
Initial Capital	$153M
Expansion Capital	$63.2M
Sustaining capital	$66.2M
Closure costs/salvage	$12.3M
Pre-tax economics:	
IRR	61.9%
NPV (5%)	$513M
Payback	1.5 Years
After-tax economics:	
IRR	43.8%
NPV (5%)	$361M
Payback	2.5 Years

1.14.1 Project Upfront Capital Costs

The Total Upfront Costs is reproduced in Table 1.17.

Table 1.17 Total Upfront Costs

	$ M
Process Plant	$51.4
Infrastructure	$21.3
Mining (Haul Roads & Pit Dewatering)	$0.8
Construction In-directs	$9.9
EPCM	$11.2
Resettlement Action Plan	$20.8
Owner's Costs	$26.1
Subtotal	**$141.7**
Contingency	$11.3
Total Initial Construction Costs	**$153.0**
Working Capital	$24.9
Pre-production Operating Costs	$8.4
Total Upfront Costs Before Sales	**$186.3**
Pre-production Gold Sales	-$47.6
Total Upfront Costs	**$138.7**

The Total Upfront Costs represent the project capital estimate plus capitalized costs incurred to achieve commercial production (on October 2021) less the value of gold recovered during the pre-commercial production period (June to September 2021 inclusive).

1.14.2 Sulphide Expansion Capital Costs

The Total Project Expansion Capital Costs are shown in Table 1.18.

Table 1.18 Total Project Expansion Capital Costs

	$ M
Process Plant	$36.2
Infrastructure	$1.1
Construction In-directs	$5.4
EPCM	$6.4
Resettlement Action Plan	$3.7
Owner's Costs	$5.2
Subtotal	**$58.0**
Contingency[1]	$5.2
Total Capital Costs	**$63.2**

[1]Excludes $1.4M in opening stock of consumables reclassified to working capital in the economic analysis.

The Total Project Expansion Capital Costs represents the capital estimate plus capitalized costs incurred to install production facilities for the sulphide ore. There is no pre-commercial production period as the plant will be in operation.

1.14.3 Sensitivity Analysis

The Project value was assessed by undertaking sensitivity analyses on gold price, gold recoveries, operating costs and capital costs. The Project is most sensitive to changes in gold price and then operating costs. The results of all sensitivity analyses are presented in Figures 1.13 to 1.16.

Figure 1.13 NPV Sensitivity Analysis (Pre-Tax)

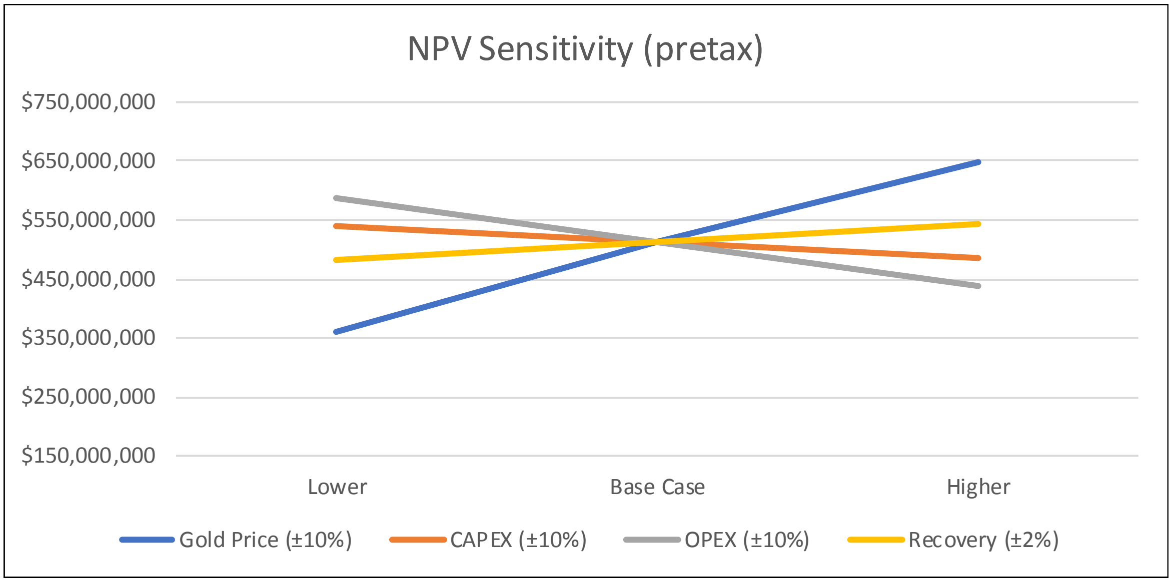


Figure 1.14 NPV Sensitivity Analysis (After tax)

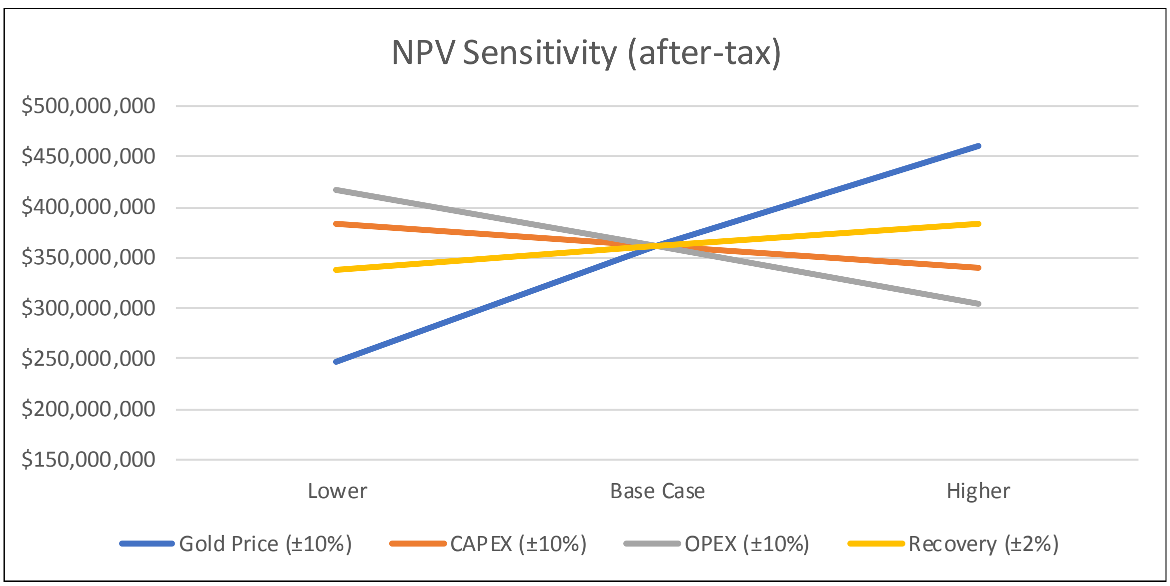


Figure 1.15 IRR Sensitivity Analysis (Pre-tax)

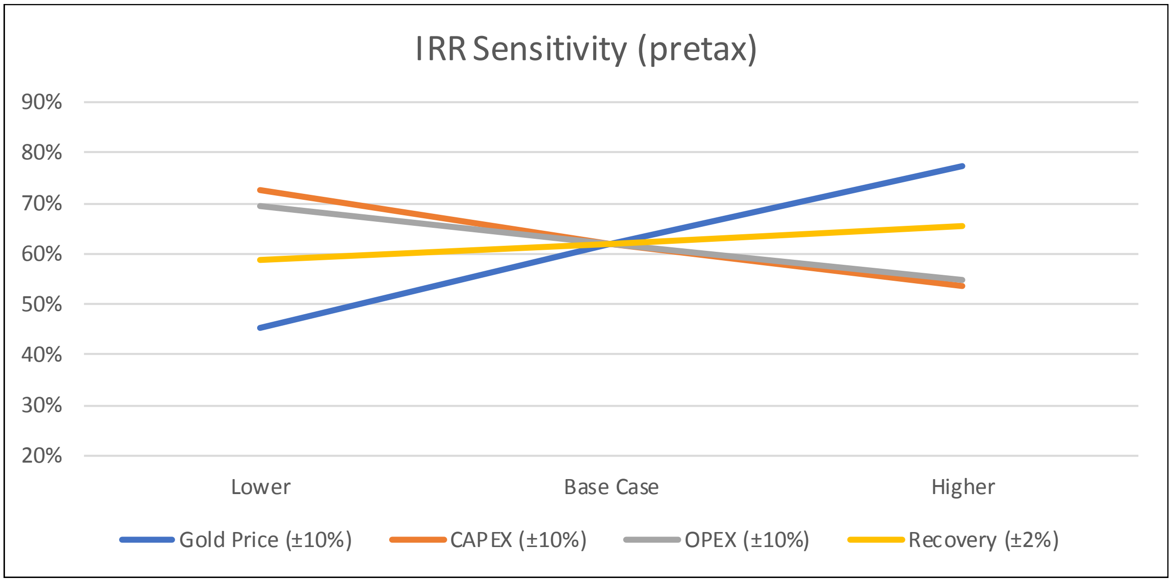
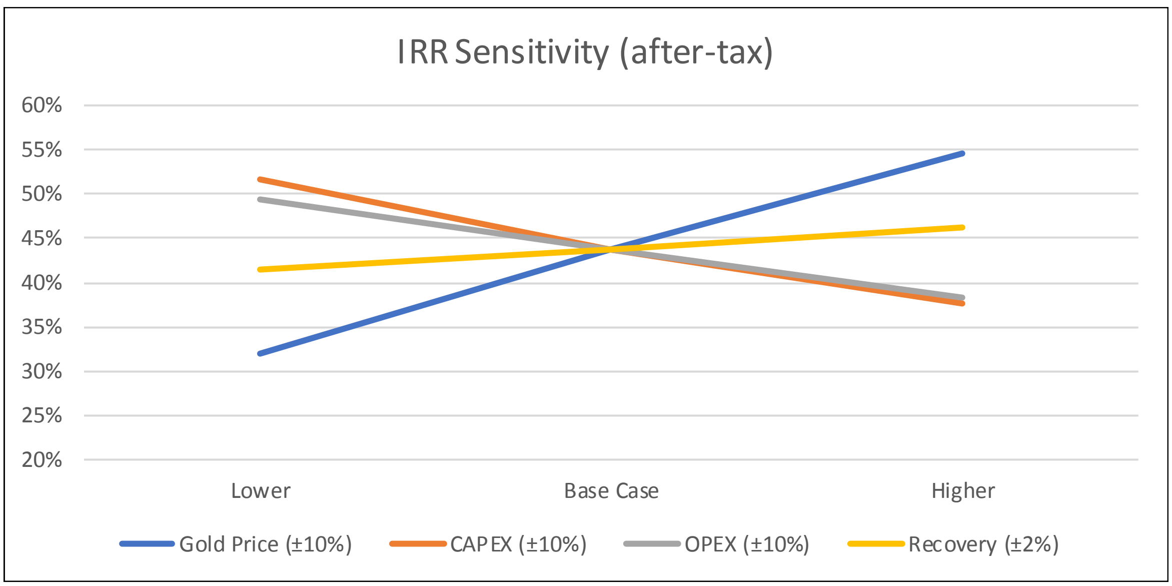


Figure 1.16 IRR Sensitivity Analysis (After tax)



1.15 Conclusions and Recommendations

Based on the work undertaken, as summarized in this Technical Report, and the conclusions listed below from the individual Qualified Persons, the Bomboré Project is a viable development opportunity centred around the initial mining and processing of the oxide and upper transition zones of the mineralization material on the Bomboré tenements followed by the supplemental mining and processing of higher grade lower transition and sulphide material after the staged sulphide expansion to the processing plant.

Risks, when considered within the context of an established and growing mining industry in Burkina Faso, are known and manageable.

It is recommended that Orezone commence implementation of the Project in line with the preliminary implementation plan and schedule developed during the FS, thus committing to the capital expenditure presented in Section 21.

Initial work will include:

- Completing execution of Phase 1 of the RAP (in progress).
- Appointment of a lead EPCM or EPC Engineer.
- Further development of the FS schedule and budgets into detailed control tools for executing the project (in progress).
- Additional metallurgical testwork on the sulphides and Lower Transition to determine optimal recoveries.
- Finalization of Front End Engineering and Design across the Project scope and commencement of detailed design.

2.0 INTRODUCTION

2.1 Terms of Reference and Purpose of this Report

This Technical Report was compiled by Lycopodium for Orezone from contributions from Qualified Persons as set out in Table 2.1 to summarize the results of the July 2019 Feasibility Study of the Bomboré Gold Project. This Technical Report expands on the Feasibility Study and Technical Report issued in 2018 (Feasibility Study of the Bomboré Gold Project, Burkina Faso, 23 August 2018) to incorporate lower transition and fresh rock material into the mine plan and mill feed. This Technical Report was prepared in compliance with the disclosure requirements of NI 43-101 and in accordance with the requirements of Form 43-101 F1.

The Property is comprised of a block of contiguous permits totalling 15,029 ha located in Ganzourgou Province, Burkina Faso, approximately 85 km east of the capital city of Ouagadougou. The Property is easily accessible from Ouagadougou via the paved national highway N4.

AMC, RPA, KP and Lycopodium provided input to this Technical 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 Technical Report. The QPs meet the requirement of independence as defined in NI 43-101.

The Project was the subject of a 2011 Preliminary Economic Assessment (PEA), a 2013 Mineral Resource estimate update, a 2014 Preliminary Economic Assessment, a 2015 FS, a 2016 Mineral Resource update, a 2017 Mineral Resource update and the 2018 FS. In consideration of the geological re-interpretation of the mineralized domains coupled with restrictions on the grade modelling of the low-grade domains, as described in a press release dated August 22, 2016, a new Mineral Resource estimate was prepared with an effective date of September 7, 2016 (the "2016 Mineral Resource"). The 2016 Mineral Resource estimate was subsequently updated, with the results disclosed in a press release dated January 10, 2017. The effective date of the resource update was assigned January 5, 2017.

Table 2.1 Persons Who Prepared this Technical Report

Qualified Persons responsible for the preparation of this Technical Report						
Qualified person	**Position**	**Employer**	**Independent of Orezone**	**Date of last site visit**	**Professional Designation**	**Report Sections**
Manochehr Oliazadeh	Manager Process	Lycopodium Minerals Canada Ltd	Yes	N/A	P.Eng	Compiling section 1 from other sections, 2, 3.1, 3.3, 3.4, 4 to 6, 13, 17, 18 (except 18.12, 18.13), 19 to 24, 25.4 to 25.6, 26.4 to 26.6 and 27.
Alan Turner	Principal Mining Engineer	AMC Consultants	Yes	28 January to 3 February 2018	CEng	3.2, 15, 16, 25.2, 26.2.
Tudorel Ciuculescu	Senior Geologist	Roscoe Postle Associates (RPA)	Yes	10-13 October 2014	P.Geo.	7 to 12, and co-author 14, 25.1, 26.1.
José Texidor Carlsson	Senior Geologist	Roscoe Postle Associates (RPA)	Yes	N/A	P.Geo.	Co-author of 14, 25.1, 26.1.
Thomas Kerr	Senior Executive Project Engineer	Knight Piésold Consulting (KP)	Yes	29 January to 2 February 2018	P.Eng.	18.12, 18.13, 25.3, 26.3.

2.2 Site Visits

Manochehr Oliazadeh did not visit the site.

Alan Turner visited the site between January 28 and February 3, 2018 to assess existing infrastructure, proposed infrastructure locations and mining areas including outcrops, diamond drill core and geological interpretation procedures.

Tudorel Ciuculescu visited the site from October 10 – 13, 2014. While on site, Mr. Ciuculescu visited the chip and core sample handling and processing facility as well as the core and sample storage facility. Several chip and core samples were collected to confirm the presence of gold mineralization.

Jose Texidor Carlsson did not visit the site.

Thomas Kerr visited the site from January 29 to February 2, 2018 to conduct walkover inspections of the areas planned for the TSF, OCR and various site drainage facilities. Mr. Kerr also viewed core from select drill holes at the site that were considered representative of foundation conditions for the TSF.

2.3 Abbreviations

Abbreviation	Meaning
%	Percent/percentage
µm	Micrometre (micron)
AAVV	l'Autorité pour l'Aménagement des Vallées de la Volta
AFD	Agence Française de Développement
AMD	Acid mine drainage
ARD	Acid rock drainage
AVV	Aménagement des Vallées des Volta
BFT	Bomboré First Target
BLEG	Bulk Leach Extractable Gold
BLK	Blank (blind to the preparation laboratory)
BOO	Build-Own-Operate
BSZ	Bomboré Shear Zone
BUMIGEB	Bureau des Mines et de la Géologie du Burkina
BUNEE	Bureau National des Évaluations Environnementales
BWi	Bond ball mill work index
BV	Bed volumes
CaO	Calcium oxide (lime or quicklime)
CDA	Canadian Dam Safety
CIL	Carbon in Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	Carbon in Pulp
cm	Centimetre
CM	Certified Reference Material
CMB	Chambre des Mines du Burkina Faso
CNM	Commission Nationale des Mines
CNT	Commission Nationale de la Transition
COTEVE	Comité technique sur les Évaluations Environnementales
CoV	Coefficient of variation
CRM	Certified Reference Material
Cu	Copper
CWi	Bond low-energy impact
d	day
DFGS	Differential Global Positioning System
DFS	Definitive Feasibility Study
DGMG	Direction Générale des Mines et de la Géologie
DGPE	Direction Générale de Préservation de l'Environnement
DGPS	Differential Global Positioning System

DNEF	Direction Nationale des Eaux et Forêts
EIA	Environmental Impact Assessment
EPCM	Engineering Procurement Construction Management
ERP	Emergency Response Plan
ESIA	Environmental and Social Impact Assessment
ESMP	Environmental Social Management Plan
FA	Fire Assay
FD	Field Duplicate (blind to preparation laboratory)
Fe	Iron
FEED	Front End Engineering Design
FEL	Front end loader
FS	Feasibility Study
g	Gram
g Ag/t	Grams of silver per tonne
g Au/t	Grams of gold per tonne
G&A	General and Administration
g/t	Grams per tonne
GARD	Global Acid Rock Drainage
GMC	Générale de Mines et de Carrières
GWh	Gigawatt hours
H_2SO_4	Sulphuric acid
ha	Hectare
HCl	Hydrochloric acid
HCN	Hydrogen cyanide
HDPE	High density poly-ethylene
HFO	Heavy fuel oil
Hg	Mercury
HGO	High Grade Oxide
HGS	High Grade Sulphide
HL	Heap leach
HQ	63.5 mm diameter core drill tube
Hr or h	Hours
ICP	Inductively couple plasma
ID2	Inverse distance squared
ID3	Inverse distance cubed
IHC	In-house referenced material
IP	Induced polarization
IPAQ	Hewlett-Packard brand name for data logger device
IPP	Independent power provider
IRR	Internal rate of return

kg	Kilogram
kg/t	Kilograms per tonne
km	Kilometre
kPag	Kilopascal Gauge
kW	Kilowatt
L	Litre
LAPD	Lab-Aware Pulp Duplicate (known to the analytical laboratory)
LAQE	Laboratoire d'Analyse de la Qualité de l'Environnement
LOM	Life of mine
LW	LeachWELL
m	Metre
M&I	Measured and Indicated
m^2	Metres squared/square metre
m^3	Metres cubed/cubic metre
m^3/h	Cubic metres per hour
MAC	Mining Association of Canada
MC	Master Composite
MMC	Ministère des Mines et des carrière
MCC	Motor control centre
ME	Ministère de l'énergie
MEDD	Ministry of the Environment and Durable Development
MEEVCC	Ministère de l'Environnement de l'Économie Verte et du Changement Climatique
MEMC	Ministère de l'Énergie des Mines et des Carrières
MGO	Medium Grade Oxide Ores
MGS	Medium Grade Sulphide Ores
Mins	minutes
ML	Leachable metals
MLCM	Million Loose Cubic Metres
mm	Millimetre
MMC	Ministère des Mines et des carrières
MRE	Mineral Resource estimate
mS	Micro-Siemen
Mt	Million tonnes
Mtpa	Million tonnes per annum
MPa	Megapascal
MW	Megawatt
NaCN	Sodium cyanide
NaOH	Sodium hydroxide/Caustic soda
NC	Non certified reference material
Ni	Nickel

NN	Nearest neighbour
NSR	Net smelter royalty
OCR	Off-Channel Reservoir
OIT	Operator interface terminals
OK	Ordinary kriging
oz	Ounce
PCS	Process control system
PD	Pulp Duplication (blind to analytical laboratory)
PEA	Preliminary Economic Assessment
PFR-G	Plan foncier rural du Ganzourgou
PFS	Pre-feasibility Study
pH	Measure of acidity/basicity
PLC	Programmable logic controller
pmp	Probable maximum precipitation
pmf	Possible maximum flood
ppb	Parts per billion
ppm	Parts per million
PQ	85 mm diameter core drill tube
QA/QC	Quality Assurance/Quality Control
QEM-RMS	Method of determining mineralogy of a sample using an electron microscope
RAB	Rotary air blast
RAP	Resettlement Action Plan
RC	Reverse Circulation
rpm	Revolutions per minute
ROM	Run of mine
RQD	Rock Quality Designation
RSD	Rotary sample divider
RWi	Bond rod mill work index
SCADA	Supervisory control and data acquisition
sec	Seconds
SMBS	Sodium metabisulphite
SMU	Selective mining unit
SO_2	Sulphur dioxide
STD	Standard (blind to the preparation laboratory)
t	Tonne
Te	Tellurium
ToR	Terms of Reference
TSF	Tailings storage facility
TVA	Taxe sur la Valeur Ajoutée (Value Added Tax)
UCS	Unconfined compressive strength

U/F	Underflow
UPS	Un-interrupted power supply
US$	United States Dollar
UTM	Universal Transverse Mercator
V	Volt
VLF-EM	Very low frequency – electromagnetic
VMP	Vibrating wire piezometers
WAD	Weak acid dissociable
WAF	West African Resources
WRD	Waste rock dumps
w/o	Without
w/w	Weight/weight
yr	Year
XFR	X-Ray fluorescence

3.0 RELIANCE ON OTHER EXPERTS

3.1 Legal Standing of Tenements

Regarding the legal standing of the Project's tenements the authors of this Report relied on a legal opinion provided by Mr. Bobson Coulibaly of Yanogo Bobson Avocats Lawyers, Rue 30.81, ZAD, Ouagadougou, Burkina Faso, dated March 5, 2019 indicating that fees and taxes have been paid, permits are in good standing and OBSA has the authority to run mining production activities.

3.2 Pit Slope Design Recommendations for Saprolite

In undertaking the design of the open pits in the saprolitic orebodies AMC relied on the interpretation of geotechnical data and resulting recommendations for slope angles and bench configurations contained in the technical memorandum 'Pit Slope Design Recommendations for Saprolite, Bomboré Mine, Burkina Faso – Revision 1' dated 12 June 2018 and authored by George Lightwood, Senior Engineer, of Golder Associates, Reno, Nevada.

AMC also relied on the interpretation of geotechnical data and recommendations for transitional and fresh rock material contained in the technical report 'Bomboré Project Feasibility Level Pit Slope Design Report' dated April 2013, Golder Associates. AMC used the inter-ramp angle and bench face angle from Golder to determine the berm width for the transitional and fresh rock domains based on bench heights (vertical distance between berms) of 6m and 12m.

The content and recommendations contained within the technical memorandum and report were reviewed and believed to be reasonable and appropriate for the purposes of the FS.

3.3 Environment and Permitting

Lycopodium has relied on the knowledge and expertise of David Ramel of Antea®Group, France, to provide the content of Section 20 and information relating to the environmental impact and permitting status of the Bomboré Project. David is registered with the International Association of Hydrogeologists (French chapter, CFH registration 641). David visited site in June 2019 and is familiar with the region and subject matter.

3.4 Tax

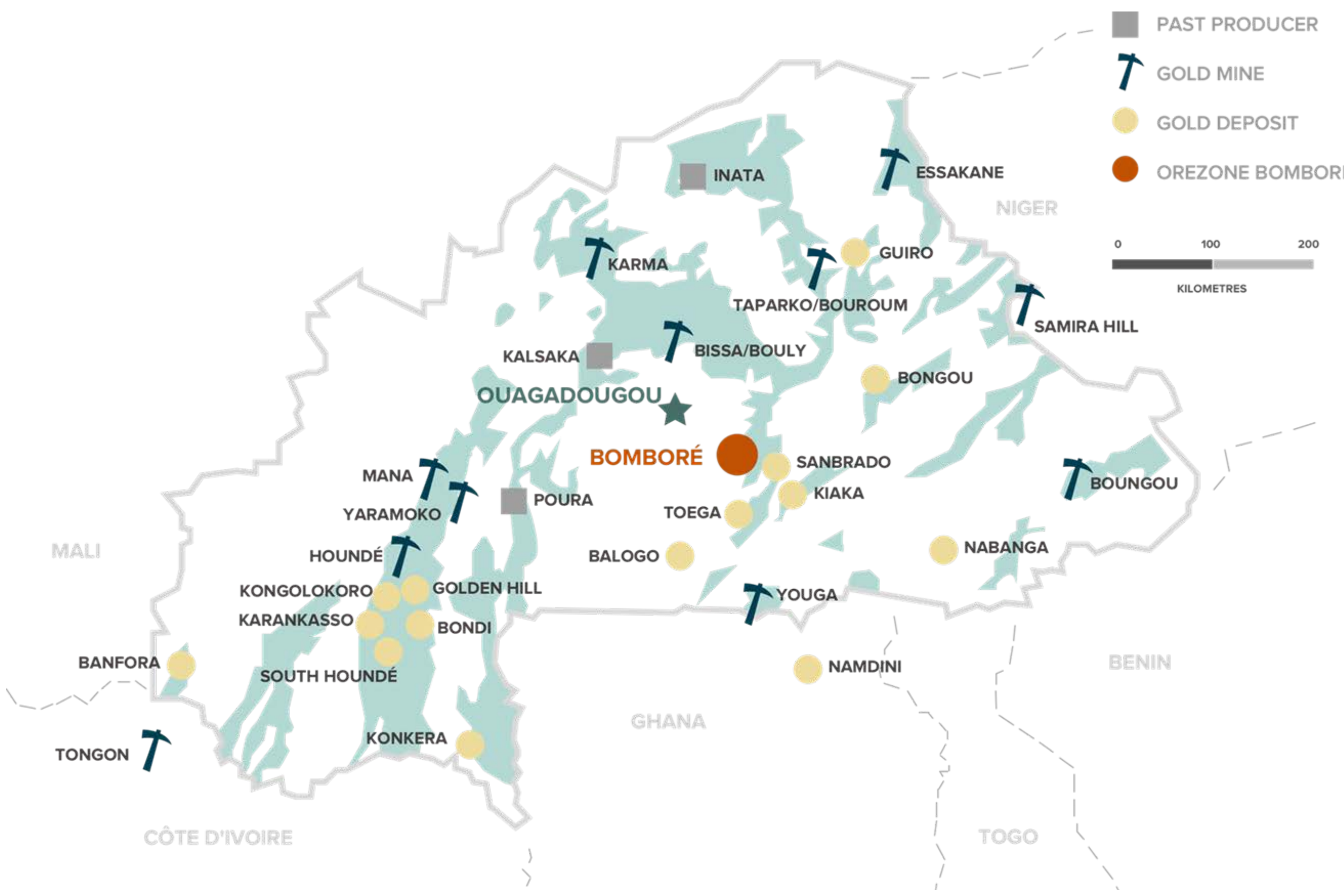
Lycopodium has relied on Orezone and Orezone's tax advisor for tax advice where developing the cash flow model.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Property is comprised of a block of contiguous permits totalling 15,029 ha located in Ganzourgou Province, Burkina Faso, approximately 85 km east of the capital city of Ouagadougou (Figure 4.1). The Property is easily accessible from Ouagadougou via the paved, national highway N4.

Figure 4.1 Property Location



The UTM co-ordinates for the approximate centre of the Property are 1,348,800mN, 728,100mE (Zone 30, Clarke 1880 ellipsoid, Adindan datum). The geographic co-ordinates for the approximate centroid of the currently defined Bomboré gold deposit are 12°12’N Latitude and 0°12’W Longitude.

4.2 Land Tenure

The Property covers an area of 15,029 ha and consists of one Industrial Operating Permit (the Bomboré Mining Permit) of 2,500 ha, surrounded by four Mining Exploration Permits: The Toéyoko Exploration Permit of 4,669 ha, the Bomboré II Exploration Permit of 1,815 ha, the Bomboré III Exploration Permit of 4,810 ha and the Bomboré IV Exploration Permit of 1,235 ha.

Mining permits are granted by Decree of the Council of Ministers and exploration permits are granted by order of the MEMC of Burkina Faso. The Government of Burkina Faso retains a 10% free carried interest in a mining company holding a mining permit. The government's free carried interest cannot be diluted.

Exploration permits are issued for an initial three-year term as of the date of issuance and may be renewed for a maximum of two consecutive three-year terms according to the Mining Act.

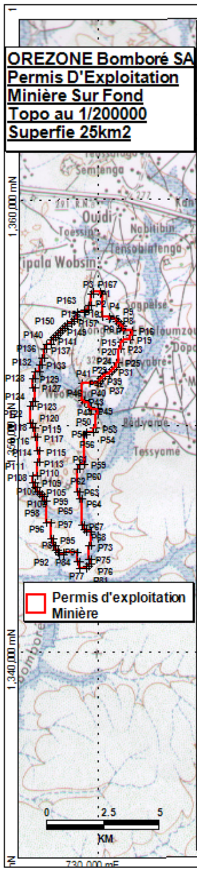
Exceptional extensions of up to three additional years have been granted for several permits in recent years.

The land tenure information presented herein is derived from copies of the Decree and Orders by which the Property permits were granted.

The boundaries of each permit are defined by corner posts positioned according to geographic coordinates (UTM Clarke 1880 ellipsoid, Adindan datum, Zone 30) as indicated in Tables 4.1 to Table 4.5. The boundaries of mining permits must be physically marked on the ground and legally surveyed within six months of its issuance.

The boundaries of exploration permits are not subject to this requirement.

Table 4.1 Bomboré Mining Permit Boundaries



Sommets	X	Y	Sommet	X	Y	Sommet	X	Y	Sommets	X	Y	Sommet	X	Y
P1	729820	1355687	P35	730320	1351987	P69	729420	1345187	P103	727520	1346787	P137	727620	1353187
P2	730120	1355687	P36	730220	1351987	P70	729520	1345187	P104	727420	1346787	P138	727720	1353187
P3	730120	1355587	P37	730220	1351887	P71	729520	1345087	P105	727420	1346887	P139	727720	1353387
P4	730220	1355587	P38	730120	1351887	P72	729720	1345087	P106	727320	1346887	P140	727920	1353387
P5	730220	1354587	P39	730120	1351787	P73	729720	1344487	P107	727320	1347087	P141	727920	1353587
P6	730520	1354587	P40	730020	1351787	P74	729620	1344487	P108	727220	1347087	P142	728020	1353587
P7	730520	1354487	P41	730020	1351687	P75	729620	1343887	P109	727220	1347287	P143	728020	1353687
P8	730820	1354487	P42	729320	1351687	P76	729720	1343887	P110	727120	1347287	P144	728120	1353687
P9	730820	1354387	P43	729320	1350887	P77	729720	1343687	P111	727120	1347587	P145	728123	1353783
P10	730920	1354387	P44	729420	1350887	P78	729620	1343687	P112	727320	1347587	P146	728220	1353787
P11	730920	1354287	P45	729420	1350787	P79	729620	1343587	P113	727320	1348187	P147	728220	1353887
P12	731020	1354287	P46	729620	1350787	P80	729520	1343587	P114	727420	1348187	P148	728320	1353887
P13	731020	1354187	P47	729620	1350587	P81	729520	1343487	P115	727420	1348687	P149	728320	1353987
P14	731120	1354187	P48	729720	1350587	P82	729220	1343487	P116	727320	1348687	P150	728420	1353987
P15	731120	1353987	P49	729720	1350487	P83	729220	1343787	P117	727320	1349287	P151	728420	1354087
P16	731520	1353987	P50	730020	1350487	P84	729120	1343787	P118	727220	1349287	P152	728520	1354087
P17	731520	1353787	P51	730020	1350287	P85	729120	1344187	P119	727220	1349687	P153	728520	1354187
P18	731420	1353787	P52	729920	1350287	P86	728520	1344187	P120	727120	1349687	P154	728620	1354187
P19	731420	1353587	P53	729920	1349687	P87	728520	1344087	P121	727120	1349887	P155	728620	1354287
P20	731120	1353587	P54	729820	1349687	P88	728320	1344087	P122	727020	1349887	P156	728820	1354287
P21	731120	1353487	P55	729820	1349487	P89	728320	1344187	P123	727020	1350687	P157	728820	1354387
P22	731020	1353487	P56	729520	1349487	P90	728220	1344187	P124	727120	1350687	P158	728920	1354387
P23	731020	1353187	P57	729520	1349387	P91	728220	1344287	P125	727120	1350887	P159	728920	1354487
P24	730920	1353187	P58	729420	1349387	P92	728120	1344287	P126	727220	1350887	P160	729120	1354487
P25	730920	1352587	P59	729420	1348087	P93	728120	1344487	P127	727220	1351487	P161	729120	1354787
P26	730820	1352587	P60	729220	1348087	P94	728020	1344487	P128	727120	1351487	P162	729420	1354787
P27	730820	1352387	P61	729220	1347887	P95	728020	1344787	P129	727120	1351887	P163	729420	1354887
P28	730720	1352387	P62	729120	1347887	P96	727920	1344787	P130	727220	1351887	P164	729620	1354887
P29	730720	1352287	P63	729120	1346887	P97	727920	1345487	P131	727220	1352187	P165	729620	1355087
P30	730620	1352287	P64	729220	1346887	P98	727720	1345487	P132	727420	1352187	P166	729720	1355087
P31	730620	1352187	P65	729220	1346587	P99	727720	1346487	P133	727420	1352487	P167	729720	1355587
P32	730520	1352187	P66	729320	1346587	P100	727620	1346487	P134	727520	1352487	P168	729820	1355587
P33	730520	1352087	P67	729320	1345387	P101	727620	1346687	P135	727520	1352787			
P34	730320	1352087	P68	729420	1345387	P102	727520	1346687	P136	727620	1352787			

Source: RPA NI 43-101 Technical Report, 31 October 2016

Table 4.2 Toéyoko Permit Boundaries

Corner	Easting	Northing
A	724,197	1,349,248
B	724,860	1,349,248
C	724,860	1,349,742
D	725,450	1,349,742
E	725,450	1,342,800
F	733,000	1,342,800
G	733,000	1,338,297
H	727,739	1,338,297
I	727,739	1,339,930
J	726,323	1,339,930
K	726,323	1,338,590
L	723,820	1,338,590
M	723,820	1,342,457
N	724,197	1,342,457

Note: UTM Projection: Clarke 1880, Adindan datum, Zone 30 North

Source: Orezone Inc. s.a.r.l.

Table 4.3 Bomboré II Permit Boundaries

Corner	Easting	Northing
A	729,200	1,359,200
B	733,000	1,359,200
C	733,000	1,353,987
D	731,120	1,353,987
E	731,120	1,354,187
F	731,020	1,354,187
G	731,020	1,354,287
H	730,920	1,354,287
I	730,920	1,354,387
J	730,820	1,354,387
K	730,820	1,354,487
L	730,520	1,354,487
M	730,520	1,354,587
N	730,220	1,354,587
O	730,220	1,355,587
P	730,120	1,355,587
Q	730,120	1,355,687
R	729,820	1,355,687
S	729,820	1,355,587
T	729,720	1,355,587
U	729,720	1,355,087
V	729,620	1,355,087
W	729,620	1,354,887
X	729,420	1,354,887
Y	729,420	1,354,787
Z	729,200	1,354,787

Note. UTM Projection: Clarke 1880, Adindan datum, Zone 30 North

Source: Orezone Inc. s.a.r.l.

Table 4.4 Bomboré III Permit Boundaries

Corner	Easting	Northing
A	725,450	1,353,400
B	727,550	1,353,400
C	727,550	1,353,887
D	728,220	1,353,887
E	728,220	1,353,787
F	728,120	1,353,787
G	728,120	1,353,687
H	728,020	1,353,687
I	728,020	1,353,587
J	727,920	1,353,587
K	727,920	1,353,387
L	727,720	1,353,387
M	727,720	1,353,187
N	727,620	1,353,187
O	727,620	1,352,787
P	727,520	1,352,787
Q	727,520	1,352,487
R	727,420	1,352,487

Corner	Easting	Northing
S	727,420	1,352,187
T	727,220	1,352,187
U	727,220	1,351,887
V	727,120	1,351,887
W	727,120	1,351,487
X	727,220	1,351,487
Y	727,220	1,350,887
Z	727,120	1,350,887
AA	727,120	1,350,687
AB	727,020	1,350,687
AC	727,020	1,349,887
AD	727,120	1,349,887
AE	727,120	1,349,687
AF	727,220	1,349,687
AG	727,220	1,349,287
AH	727,320	1,349,287
AI	727,320	1,348,687
AJ	727,420	1,348,687
AK	727,420	1,348,187
AL	727,320	1,348,187
AM	727,320	1,347,587
AN	727,120	1,347,587
AO	727,120	1,347,287
AP	727,220	1,347,287
AQ	727,220	1,347,087
AR	727,320	1,347,087
AS	727,320	1,346,887
AT	727,420	1,346,887
AU	727,420	1,346,787
AV	727,520	1,346,787
AW	727,520	1,346,687
AX	727,620	1,346,687
AY	727,620	1,346,487
AZ	727,720	1,346,487
BA	727,720	1,345,487
BB	727,920	1,345,487
BC	727,920	1,344,787
BD	728,020	1,344,787
BE	728,020	1,344,487
BF	728,120	1,344,487
BG	728,120	1,344,287
BH	728,220	1,344,287
BI	728,220	1,344,187
BJ	728,320	1,344,187
BK	728,320	1,344,087
BL	728,520	1,344,087
BM	728,520	1,344,187
BN	729,120	1,344,187
BO	729,120	1,343,787
BP	729,220	1,343,787
BQ	729,220	1,343,487
BR	729,520	1,343,487
BS	729,520	1,343,587
BT	729,620	1,343,587

Corner	Easting	Northing
BU	729,620	1,343,687
BV	729,720	1,343,687
BW	729,720	1,343,887
BX	729,620	1,343,887
BY	729,620	1,344,487
BZ	729,720	1,344,487
CA	729,720	1,345,087
CB	729,520	1,345,087
CC	729,520	1,345,187
CD	729,420	1,345,187
CE	729,420	1,345,387
CF	729,320	1,345,387
CG	729,320	1,346,587
CH	729,220	1,346,587
CI	729,220	1,346,887
CJ	729,120	1,346,887
CK	729,120	1,347,887
CL	729,220	1,347,887
CM	729,220	1,348,087
CN	729,420	1,348,087
CO	729,420	1,349,387
CP	729,520	1,349,387
CQ	729,520	1,349,487
CR	733,000	1,349,487
CS	733,000	1,342,800
CT	725,450	1,342,800

Note. UTM Projection: Clarke 1880, Adindan datum, Zone 30 North

Source: Orezone Inc. s.a.r.l.

Table 4.5 Bomboré IV Permit Boundaries

Corner	Easting	Northing
P1	731,520	1,353,987
P2	733,000	1,353,987
P3	733,000	1,349,487
P4	729,820	1,349,487
P5	729,820	1,349,687
P6	729,920	1,349,687
P7	729,920	1,350,287
P8	730,020	1,350,287
P9	730,020	1,350,487
P10	729,720	1,350,487
P11	729,720	1,350,587
P12	729,620	1,350,587
P13	729,620	1,350,787
P14	729,420	1,350,787
P15	729,420	1,350,887
P16	729,320	1,350,887
P17	729,320	1,351,687
P18	730,020	1,351,687
P19	730,020	1,351,787
P20	730,120	1,351,787
P21	730,120	1,351,887
P22	730,220	1,351,887
P23	730,220	1,351,987
P24	730,320	1,351,987
P25	730,320	1,352,087
P26	730,520	1,352,087
P27	730,520	1,352,187
P28	730,620	1,352,187
P29	730,620	1,352,287
P30	730,720	1,352,287
P31	730,720	1,352,387
P32	730,820	1,352,387
P33	730,820	1,352,587
P34	730,920	1,352,587
P35	730,920	1,353,187
P36	731,020	1,353,187
P37	731,020	1,353,487
P38	731,120	1,353,487
P39	731,120	1,353,587
P40	731,420	1,353,587
P41	731,420	1,353,787
P42	731,520	1,353,787

Note. UTM Projection: Clarke 1880, Adindan datum, Zone 30 North

Source: Orezone Inc. s.a.r.l.

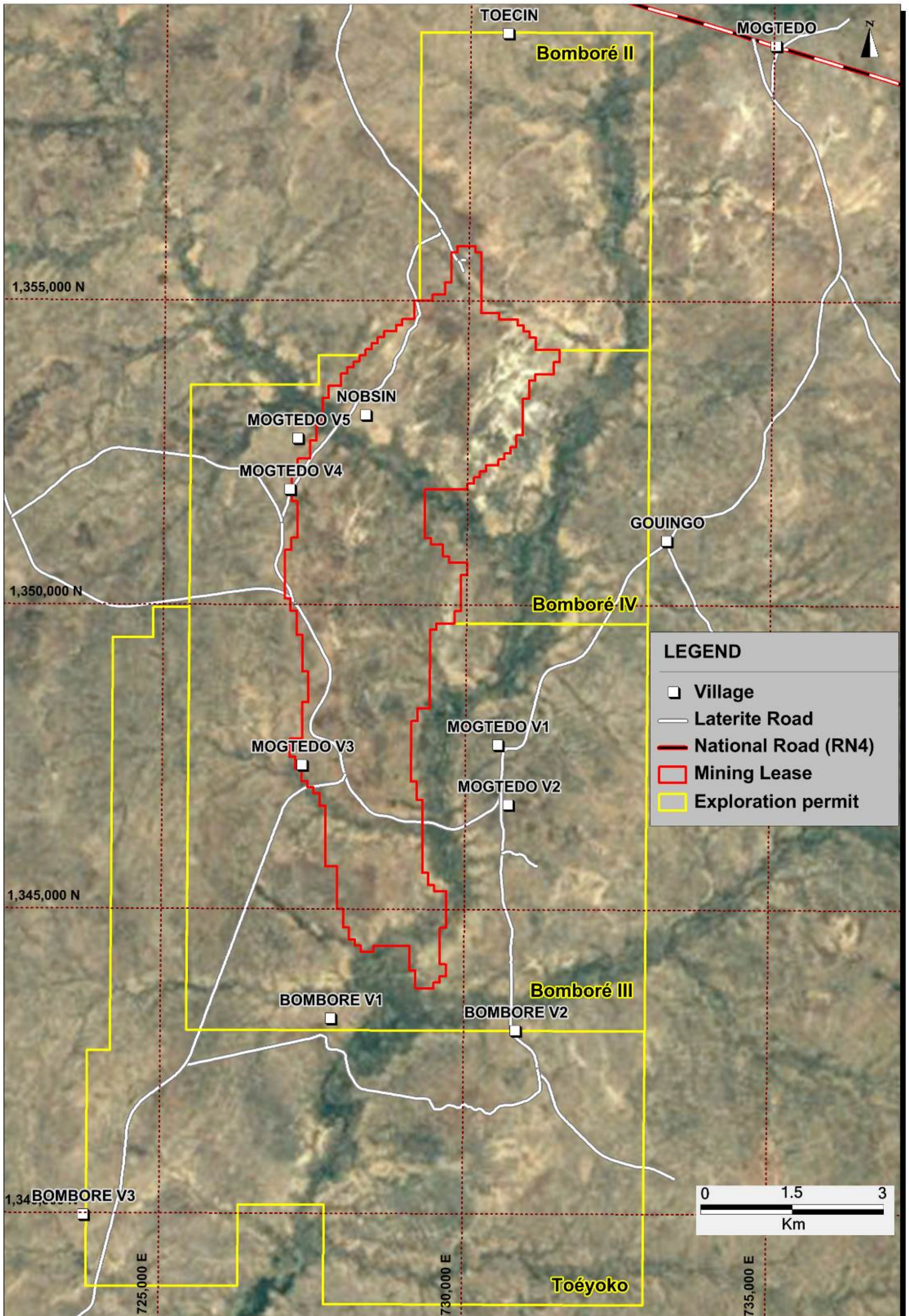
The Bomboré Mining Permit is registered in the name of OBSA, a 90%-owned subsidiary of OSARL, itself a 100%-owned subsidiary of OINC, which is 100% owned by Orezone. The Bomboré Mining Permit was granted to OBSA by way of Decree No. 2016-1266/PRES/PM/MEMC/MINEFID/MEEVCC dated 30 December 2016 and is valid for an initial tenure of 10.7 years but can be extended if the mine life is extended beyond what was initially applied for.

The Toéyoko Exploration Permit is currently registered in the name of OSARL. It was granted to OINC in July 2011 and is valid until July 13, 2020 when it will be renewable for one last exceptional three-year additional term.

The Bomboré II, Bomboré III and Bomboré IV Exploration Permits are registered in the name of OSARL. They were granted to OSARL on January 17, 2017 and are valid until January 17, 2020 when they will be renewable for the first of three possible three-year additional terms.

The Mineral Resources reported in this Report are essentially located within the Bomboré Mining Permit (Figure 4.2), with one small deposit on the Toéyoko Exploration Permit (P17S) and one small deposit on the Bomboré III Exploration Permit (P17N).

Figure 4.2 Bomboré Tenements



File name : Carte des permis_2018-07-17_Ang.wor
OREZONE ; Datum : WGS 84 UTM Zone 30N

4.3 Underlying Agreements

The current Property was originally covered by a prospecting authorization covering 605,800 ha, granted to Générale de Mines et de Carrières (GMC) in 1989. In January 1994, following changes in the Mining Act in 1993, a modified exploration permit covering 210,800 ha was issued to GMC.

Channel Mining (Barbados) Company, Ltd. (Channel) entered into an option agreement with GMC in 1994 giving it a 90% working interest in the exploration permit, leaving GMC with a 10% carried interest. In the summer of 1997, GMC converted its 10% interest into Channel shares.

A sub-option agreement reached with Solomon Resources Limited (Solomon) allowed Channel to secure financing for further exploration. By the end of 1997, Solomon had earned a 45% interest, leaving Channel with a 45% interest in the permit, assuming a 10% free-carried interest owned by the Government if the project were to be developed. The exploration permit was renewed in early 1998.

In 1999, Placer Dome (Africa) Inc. (Placer) reached an agreement to earn a 20% interest in the exploration permit but never fulfilled the conditions to earn in. In July 2001, the exploration permit was reduced to 150,000 ha upon renewal.

Orezone's rights to the Property arise from an initial option agreement signed in 2002 by Orezone's predecessor Orezone Resources Inc. (ORINC) with Channel and Solomon granting ORINC the right to earn a 50% interest in the Project. In 2004, the original Bomboré exploration permit expired and a new Bomboré I exploration permit covering 25,000 ha was granted to Société Orezone, a subsidiary of ORINC, by the MEMC on February 17, 2004. ORINC earned its 50% interest in the Project by issuing 150,000 common shares, making a C$40,000 payment, and spending C$2 million on exploration before January 17, 2007. The Bomboré I exploration permit was renewed on May 14, 2007 and reduced to 10,450 ha.

On September 3, 2008, ORINC announced that it had purchased the remaining interest in the Bomboré I exploration permit from Channel and Solomon in consideration of one million common shares of ORINC (ORINC press release dated September 3, 2008).

On February 25, 2009, ORINC and IAMGOLD Corporation (IAMGOLD) announced that IAMGOLD had acquired ORINC pursuant to a plan of arrangement under the Canada Business Corporations Act. As part of the plan of arrangement, a new exploration company, Orezone Gold Corporation, was incorporated and acquired certain assets and liabilities of ORINC, including the Bomboré I exploration permit. There was no further relationship with IAMGOLD after the transaction closed other than IAMGOLD becoming a shareholder of Orezone Gold Corporation.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Property is located in Ganzourgou Province, Burkina Faso, approximately 85 km east of the capital city of Ouagadougou. The Property is easily accessible from Ouagadougou via paved national highway N4. The paved road as far as the turn off to the mine site has a single lane in each direction and is in excellent condition. Orezone, has recently upgraded the 5 km unsealed road from N4 to the accommodation camp at the north of the Property, including installation of culverts to allow all weather access. This road is also a single lane in each direction.

All of the Property is accessible in the dry season. Access in the wet season can be restricted by the flooding of local watercourses but, as part of the project development, permanent bridges will be installed providing year-round access for light and heavy vehicles.

Although Burkina Faso is landlocked, road and rail links to the West African coast are well established and currently support mining and other activities in Burkina Faso and its regional neighbours. Good road links exist to the ports of Lome in Togo, Tema and Takoradi in Ghana and road and rail links to Abidjan in the Ivory Coast. Lome in particular has been used as the port of entry for recent mine development in the region and Ghana is a source of consumables for the mining industry including lime and cyanide.

The International Airport in Ouagadougou is serviced on a regular basis by several international and regional carriers including Air France, Brussels Airlines, Royal Air Maroc and Turkish Airlines. Regular air cargo services are available and the airport also accepts charter cargo flights. A new international airport is planned for development to the northeast of Ouagadougou.

5.2 Climate

The local climate consists of dry and wet seasons. It is common for rain to occur from April through October, however, the highest concentration of rainfall events occurs between late June and late September. On average, approximately 800 mm of rainfall occurs annually, typically in short bursts of heavy rain. Construction and mining operations can be scheduled year-round, with short delays during heavy rainfall events expected.

Temperatures range from a low of about 10°C in December and January to highs of about 43°C in March and April with average daily temperatures in the range of 23° to 33°. Between the end of the wet season and March the north-easterly trade winds bring dust down from the Sahara (the Harmattan) resulting in reduced visibility.

5.3 Local Infrastructure and Resources

Ouagadougou is a typical inland West African city with limited heavy industry but with an established network of companies and suppliers servicing the regional mining industry. Several large regional contracting companies maintain a presence in the country and are equipped with, or can readily mobilize, the resources necessary to construct and support mine development.

The project site is within 15 km of the regional town of Mogtédo, with a population greater than 15,000. The town is developing rapidly with many substantial multi-storey concrete block buildings established or under construction.

Most of the semi-skilled and unskilled labour required for project development will be sourced from Mogtédo and surrounding villages. As the town has the capacity to provide rented rooms and leased accommodation for the contractor's skilled workforce, the contractors will be required to make their own accommodation arrangements with local businesses.

The national power grid does not service the Bomboré site. Power will be provided by an on-site power station using HFO. Hydrocarbons are readily available from depots in Ouagadougou and three major regional fuel distributors have expressed interest in establishing a satellite depot on site to provide diesel, HFO and lubricants to the Project.

Bore water in the area is suitable for treatment for human consumption. Process water will be harvested from the seasonal rivers and stored for use in the dry season (refer Section 18).

The Wayen quarry adjacent to N4 and just 14 km from the project site has recently been reopened with a new aggregate crushing plant installed. Aggregate from this quarry was previously tested and found suitable for construction purposes.

5.4 Physiography

The topography of the property is generally flat with low hills, in the order of 30 m to 50 m in elevation. The land surface consists of outcrop, sub-crop, and hard ferruginous lateritic cap rock that form a gently southwesterly-sloping plateau.

The seasonal Bomboré River crosses the Project area along a north-northeast south-southwest course and its tributaries follow northeast and northwest directions. The Bomboré River is a tributary of the Nakanbé River. The drainage pattern is rectangular–dendritic, reflecting late fracture systems trending north-south, east-west, and northwest-southeast and the predominantly north-northeast trend of the stratigraphic units.

Vegetation in uncultivated areas comprises mostly savannah woodlands, with dense bush growing only near streams and rivers. Farmers cultivate staple crops such as millet, rice, sorghum, maize corn, and cash crops, such as cotton, sesame and groundnuts. Deforestation is widespread over the permit area. Wildlife is mostly restricted to small game and birds, but snakes are common, and a few monkey sightings have been reported. The south-west corner of the property lies approximately 11 km away from the classified forest of the Nakanbé River (Volta Blanche or White Volta River). As it flows southwards toward Ghana, the Nakanbé marks the border of this protected area.

Figure 5.1 shows typical landscapes during the dry and wet seasons.

Figure 5.1 Dry and Wet Season Landscapes

Source: *Orezone*

6.0 HISTORY

Information in this section is updated from the Lycopodium Technical Report dated August 23, 2018 which was in turn compiled from prior reports issued over the previous to thirty-year duration of exploration activities at the Project.

6.1 Exploration History

6.1.1 Exploration by GMC 1989-1994

Initial exploration work at Bomboré was conducted by GMC during the period 1989 to 1994. Exploration activities included soil geochemical surveys, trenching, rock sampling, and data collection from the artisanal miners.

6.1.2 Exploration by Channel 1994-2000

During 1994 to 1995, Channel commenced exploration activities with mapping and sampling of outcrops and orpaillage sites, soil geochemistry, ground magnetic, and electromagnetic (VLF-EM) surveying. RC drilling returned significant gold intersections on the P16, P17, P12 (Siga), P8, P9, and KT zones on the Bomboré First Target (BFT), which was identified as the major exploration target on the permit.

Further exploration was conducted by Channel in 1996 in preparation for a drill program. During this time, additional geological mapping and rock and soil sampling were carried out complemented by airborne geophysical surveying and satellite data acquisition.

Subsequent soil and termite-mound sampling, trenching, geological mapping and rock sampling and some RC drilling were conducted on the BFT. Outside this area, the target definition work consisted of geological mapping, and rock and soil sampling over five grids (Sabse, Sogdin, Zabre, Ziga, Boudry-Tanguin and Meguet). Soil sampling was also performed over the southwestern and southeastern parts of the permit (Nakanbé and Mankarga). These areas lay outside the current Bomboré I permit area.

Channel completed the first internal resource estimation over the BFT in 1997. To define the main mineralized axes within the BFT, a systematic drilling program was initiated. RAB drill fences were completed across the entire width of the BFT, which helped to define and extend the main axes of mineralization and detect new zones. Preliminary metallurgical and mineralogical tests were conducted on RAB and RC drill samples. A petrographic study was performed by Lakefield Research, Ltd. on polished thin sections cut from 21 samples.

In 1998, ten core boreholes confirmed the width and grade of previous drill intersections. The drilling data were used to estimate mineral resources for the Project as documented in a report prepared by Channel (Guérard and Learn, 1998). In 2000, the focus of exploration conducted with Placer was on RC and RAB drilling into the P8/P9, Maga, and CFU zones and in the Mankarga Target Area.

6.2 Previous Mineral Resource Estimates

Mineral resources were previously evaluated in 1997, 1998, 2008, 2010 and 2012. Mineral resources were first estimated by Channel in 1997 and 1998. These historical resource estimates were prepared prior to the development of NI 43-101 and the results from these estimates should not be relied upon. All previous mineral resource statements have been superseded by the Mineral Resource Statement presented in Section 14.

In 2007, ORINC commissioned Met-Chem Canada Inc. (“Met-Chem”) to prepare an initial mineral resource statement for the BGP. This mineral resource estimate considered drilling information to March 2007 and is documented in a Technical Report prepared by Met-Chem and dated February 28, 2008.

In June 2008, ORINC commissioned SRK to audit an updated mineral resource model prepared by Orezone for the BGP. This mineral resource statement considered drilling information to May 2008 and is documented in a Technical Report prepared by SRK and dated November 26, 2008.

In July 2010, Orezone again commissioned SRK to audit an updated mineral resource model prepared by Orezone for the BGP. This mineral resource estimate considered drilling information to July 2010 and is documented in a Technical Report prepared by SRK and dated November 29, 2010.

In March 2012, Orezone commissioned SRK to update the mineral resource model for the BGP with assistance of Orezone for the geological and domain modelling. This mineral resource estimate considered drilling information to June 2012 and is documented in a Technical Report prepared by SRK and dated October 11, 2012. This report also included technical information and economic parameters used in an earlier PEA completed by GMSI in August 2011.

SRK completed a further resource model in 2013 which is presented in the KCA Technical Report dated April 28, 2015.

In consideration of additional drilling data and a geological re-interpretation of the mineralized domains, coupled with restrictions on the grade modelling of the low-grade domains, RPA completed a new mineral resource estimate for the Project as reported in the Technical Report of October 31, 2016. This was further refined to produce the January 5, 2017 mineral resource estimate presented in the August 23 2018 Technical Report.

An update, by RPA, of the January 5, 2017 mineral resource estimate, incorporating previously excluded “Restricted Zones” and all drilling completed to that date on the high-grade P17S deposit, is presented in Section 14 of this Technical Report and was used for the 2019 mineral reserve prepared by AMC.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The geology of Burkina Faso is dominated by the Proterozoic Baoulé-Mossi Domain, which corresponds to the eastern portion of the West African Craton. Neoproterozoic to Paleozoic sedimentary rocks cover the west and southeast of the country (inset of Figure 7.1).

The Baoulé-Mossi Eburnean orogenic domain contains Birimian (Lower Proterozoic) volcano-sedimentary units arranged in elongated belts and relics of the Archean basement. The belts generally trend north-northeast but form arcuate belts to the north of Ouagadougou. They are bounded by older granite gneiss terrains and have been intruded by syn- to late tectonic granite bodies.

The Birimian Supergroup has been divided into a Lower sequence comprised of wacke, argillite and volcaniclastic rock, and an Upper sequence of basalt with interflow sedimentary rock. Post-Eburnean marine and continental sedimentary rocks unconformably overlie the Lower Proterozoic sequences. The Birimian formations have been affected by three tectono-metamorphic phases with up to greenschist facies metamorphism.

The Project lies in a small northeast-trending belt located to the west of the major Tiébélé-Dori-Markoye Shear Zone that sub-divides the country into domains characterized by different structural patterns. Lithological and structural elements of the Project area are illustrated in Figures 7.1 and 7.2.

Figure 7.1 Regional Geology and Gold Deposits

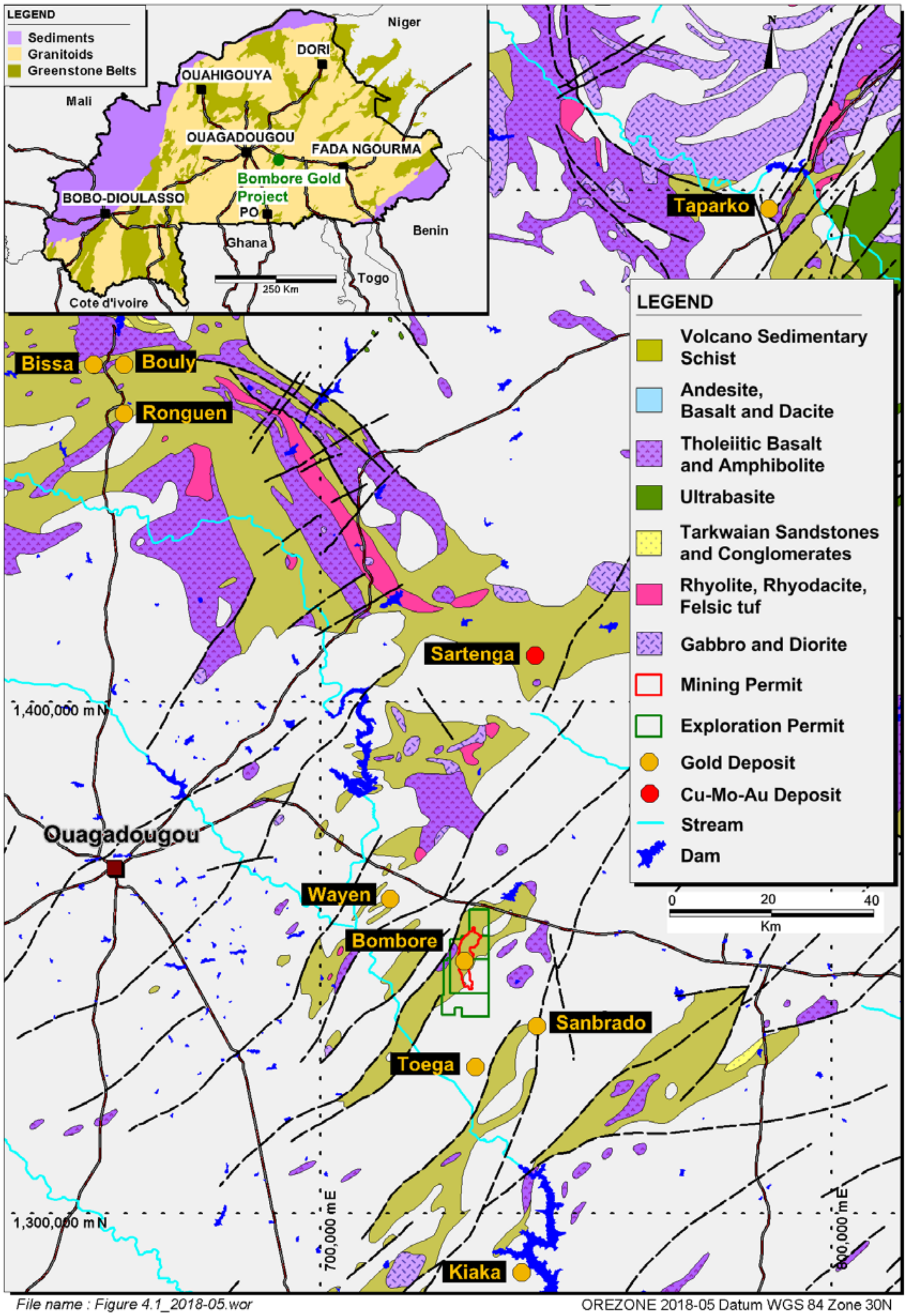
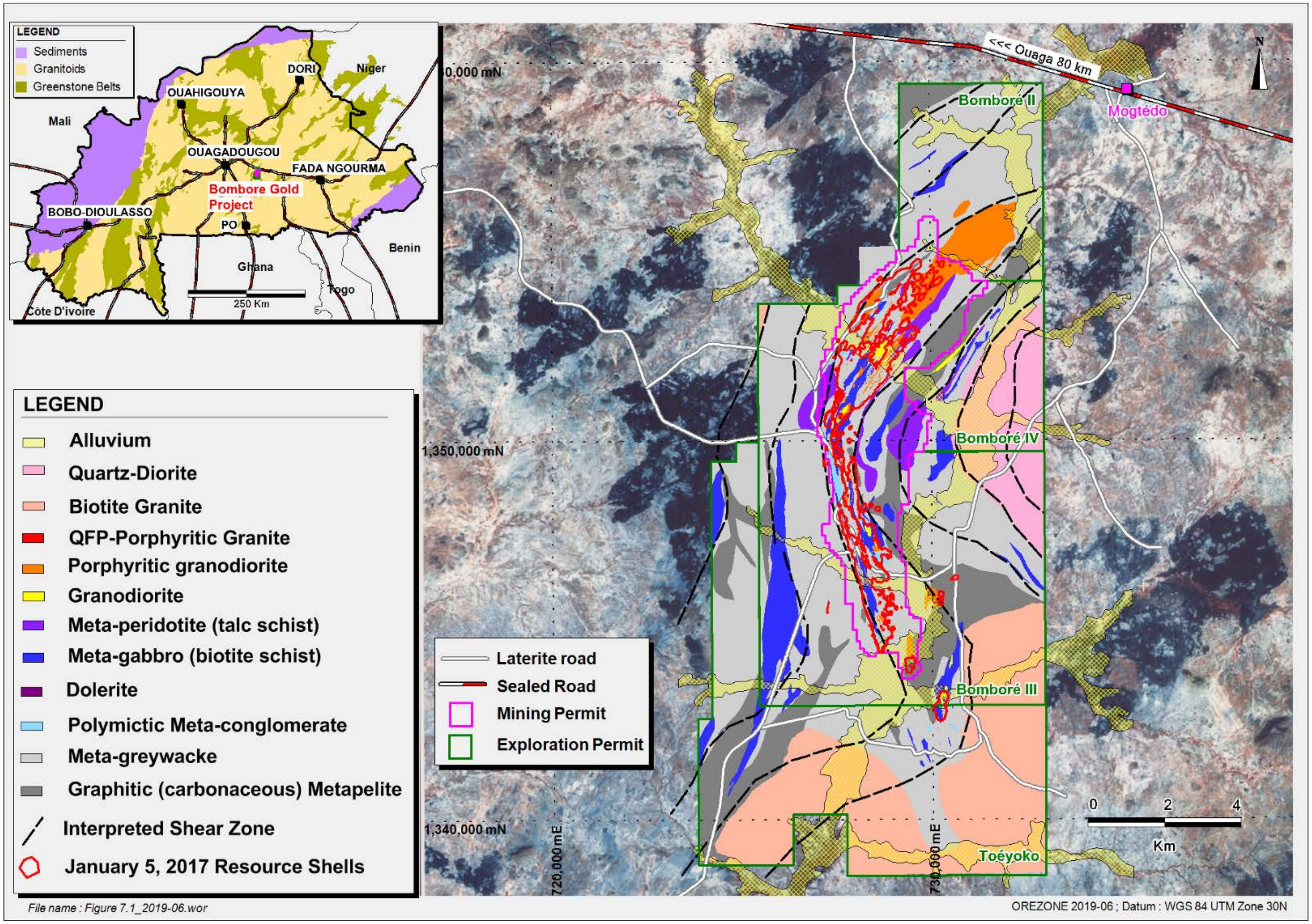


Figure 7.2 Local Geology



7.2 Property Geology

The Project covers part of a northeast-southwest trending greenstone belt extending for 50 km from the southwest corner to the village of Meguet in the northeast. The permit area is underlain mainly by a meta-sedimentary flysch-type sequence dominated by meta-sandstones with subordinate carbonaceous meta-pelites and polymictic meta-conglomerates. This metasedimentary sequence is intruded by early meta-gabbroic and ultramafic intrusives and then syntectonic granodioritic intrusives. Late-tectonic quartz-feldspar porphyries occur as dikes and larger bodies within the greenstone belt. Large biotite granite intrusives are present on the Property to the west and to the south of the greenstone belt that is also moulded on a large quartz diorite intrusive located along the eastern limit of the Project. A syenitic intrusion referred to as the Petite Suisse is exposed in the west portion of the Property.

The BSZ is a major, one to three-kilometre thick structure that contains the Bomboré gold mineralization and represents the dominant structural feature of the area. The Bomboré gold mineralization trend is defined by a gold-in-soil anomaly exceeding 0.1 g/t Au (Figures 7.3 and 7.4), as well as by the presence of numerous gold showings and *orpaillage* (artisanal miners) sites. The Bomboré anomaly measures 14 km in length, is several hundreds of metres in width, and occurs within the BSZ. Figures 7.3 and 7.4 illustrate the main mineralized areas.

Surface weathering has affected the rocks to an average depth of 35 m to 50 m but can be as deep as 100 m on the P8/P9 and CFU hanging wall and as shallow as 5 m to 10 m in the P17 area.

Figure 7.3 Location of Drill Hole Collars, Gold in Soil Anomalies and Outline of Conceptual Pit Shells

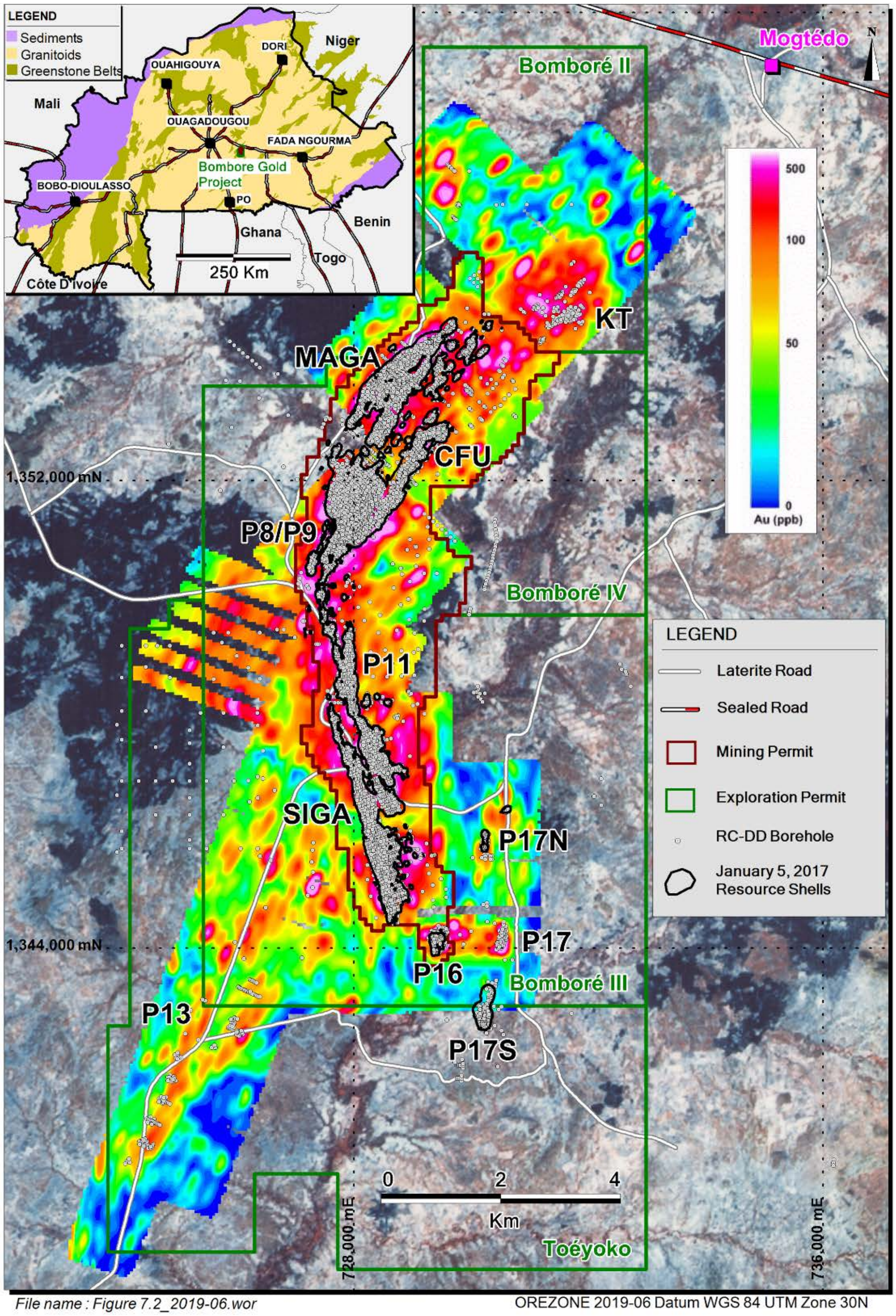
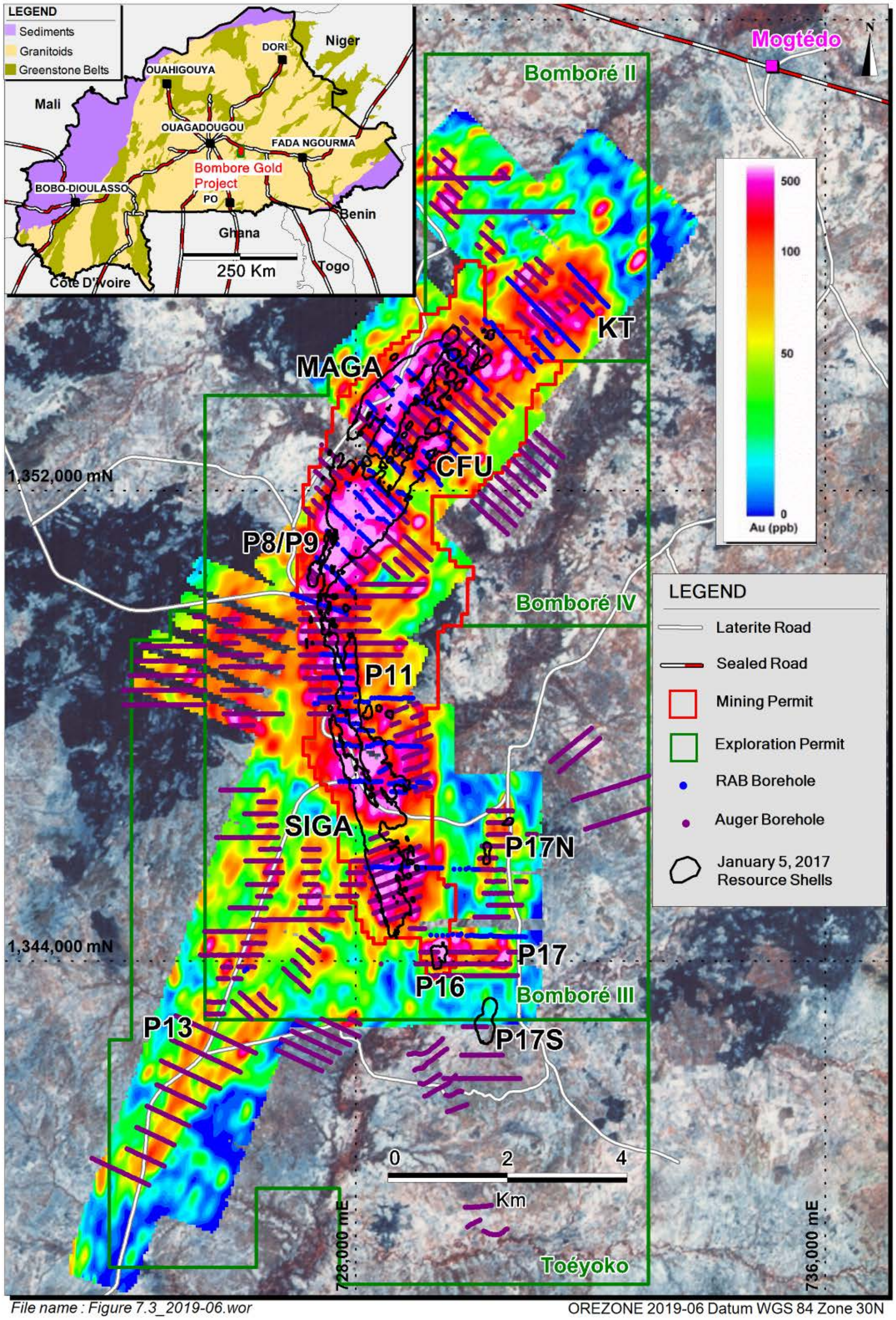


Figure 7.4 Location of Auger and RAB Drill Hole Collars, Gold in Soils Results, and Outline of Conceptual Pit Shells



7.2.1 Lithologies

Several lithological units have been recognized by Orezone on the Property, in surface outcrops, drill core, and RC chips. The current geological model integrates new information derived from additional drilling, petrographic examinations, and the systematic field X-ray fluorescence (XRF) analyses of all samples. The main lithological units are described below, from the oldest to youngest based on the current understanding of the Property litho-stratigraphic history. Representative major litho-types, as seen in the drill core, are shown in Figures 7.5 to 7.8. Plan maps showing the distributions of the lithologies of the Maga – P8/P9 and Siga – P11 areas are shown in Figures 7.9 and 7.10, respectively.

S4: Meta-Pelitic Rock

This unit consists of a sequence of laminated to finely bedded dark grey graphitic or carbonaceous meta-argillite and grey to grey-greenish meta-siltstone and fine meta-sandstone. This sequence can be up to 500 m thick, with a lateral extent greater than ten kilometres. It is the second most common unit within the greenstone sequence and it is interpreted to be the oldest volcano-sedimentary unit recognized in the area. The map pattern suggests that this unit forms regional tight folds with major closures located to the southwest on the Toéyoko Exploration Permit, to the north in the Maga area and to the south in the P16 area. Although primary sedimentary structures and textures can be locally preserved, they are generally overprinted by the regional deformation and metamorphism.

S3: Meta-Sandstone Rock

This unit is the most important within the portion of the greenstone belt underlying the Property and is interpreted to be overlying the meta-argillite unit. It occurs as a sequence up to 1 km thick dominated by greyish meta-sandstones interbedded with carbonaceous dark grey lamina. Although primary sedimentary structures and textures can be locally preserved, they are generally overprinted by the regional deformation and metamorphism. In thin section, the meta-sandstone beds consist of a quartz-sericite-biotite±graphite (carbonaceous matter) schist with a lepidoblastic to granoblastic texture. The main fabric is a pressure solution cleavage on which primary lithological contacts and early veinlets are typically transposed.

S1: Polymictic Meta-Conglomerate Rock

This unit occurs as elongated lenses adjacent to the meta-sandstone unit. The lenses are typically less than 100 m thick but can display a kilometric lateral extent. They consist of poorly sorted polymictic meta-conglomerate and conglomeratic lithic meta-sandstone. The lithic clasts consist of meta-sandstone, chert, carbonaceous-graphitic meta-argillite, granite, and quartz, predominantly as granules, pebbles, and cobbles set in a matrix of meta-sandstone. In thin section, the sandy matrix consists mostly of chlorite, sericite, quartz, and calcite with a lepidoblastic to granoblastic texture. The abundant chlorite and carbonate in this unit seems to represent a retrogressive assemblage overprinting the regional metamorphic assemblage; it is responsible for the greenish colour of this unit below the weathering profile.

I3: Mafic Intrusive Rock

This unit intrudes the metasedimentary sequence where it is generally para-conformable to the regional pressure-solution cleavage. The meta-gabbro is characterized by heterogeneous strain, locally with a massive or brecciated texture but in most instances strongly deformed (MI3 sub-unit) with a mylonitic foliation that can be crenulated or micro-folded. Where least-altered and least-deformed, it is greenish and fine to medium grained. It is composed of idiomorphic plagioclases and interstitial pyroxenes, with subordinate hornblende and biotite. In the P11 and Siga areas, gabbroic intrusives may contain millimetric blue quartz phenocrysts. This unit is commonly metasomatized and strongly overprinted by a ductile deformation event that has transformed the meta-gabbro into a quartz-biotite-actinolite-albite-calcite-ankerite±pyrite±pyrrhotite schist that is a major host of the Bomboré gold mineralization; this unit displays a characteristic brownish colour below the weathering profile.

I4: Ultramafic Intrusive Rock

Ultramafic intrusive units are present essentially in the northern portion of the Property. Least deformed and least altered occurrences outside of the BSZ consist of massive meta-peridotite where primary olivine and pyroxene are largely retrograded to an assemblage of talc, asbestos, chlorite, and carbonate pseudomorphs. Talc schists host gold mineralization in the CFU area.

IIC: Pre- to syn-tectonic Micro-Porphyritic Granodiorite Intrusives

Within the BSZ, a fine grained micro-porphyritic granodiorite occurs as narrow dikes and larger elongated intrusions, mostly on the hanging wall of the main P8/P9, P11 and Siga East deposits, but also as the main mineralized unit of the P17 and P17S deposits. This unit seems to be mostly intruding the meta-gabbro unit, and forms with the metasediments and the meta-gabbro a sequence that has been folded prior to or synchronously with the main gold deformation and mineralization event. Most of the rare occurrences of visible gold within the BSZ are associated with this unit.

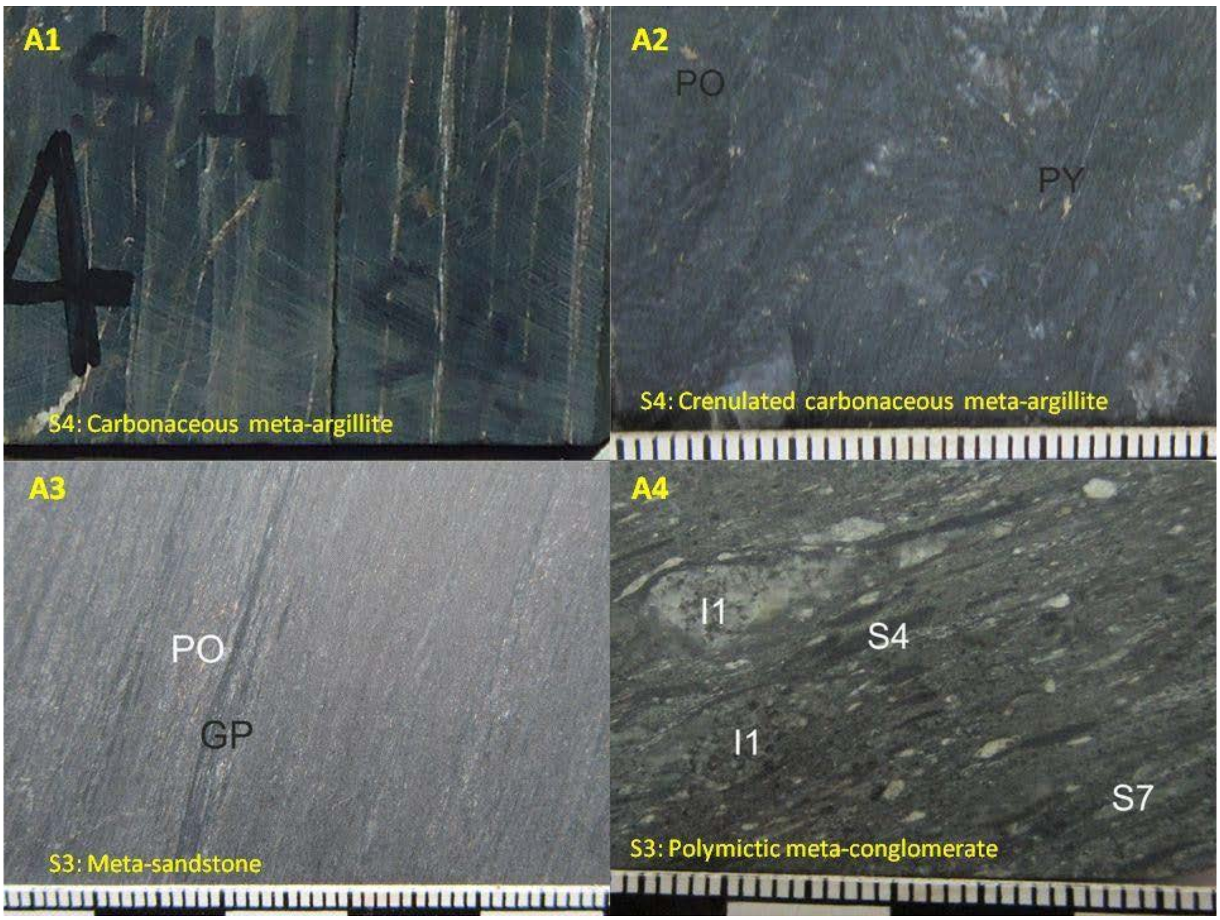
I2: Syn-tectonic Porphyritic Granodiorite Intrusives

Within the BSZ, a porphyritic granodioritic intrusive characterized by abundant zoned plagioclase phenocrysts up to 12 mm set in a groundmass of fine-grained quartz-biotite-sericite commonly occurs as narrow (1 m to 100 m) dikes typically at a low counter-clockwise angle to the pre-existing lithological units and fabrics, but also as larger elongated intrusions in the Maga and KT areas. They are syn-tectonic and pre- to syn-gold mineralization but are less deformed and less well mineralized than the older units that they are intruding. The sheared and mineralized porphyritic granodiorite is often difficult to distinguish from the sheared and deformed meta-sandstone even in core boreholes.

I1: Late Quartz Feldspar Porphyry Granite Dikes

Within the BSZ, late pale grey fine-grained granitic dikes characterized by abundant corroded quartz and plagioclase-albite phenocrysts set in a microlitic and sericitic ground mass occur as narrow metric (typically one to three metres wide) dikes, mostly in the Maga and P8/P9 area. They are post-tectonic and post-gold mineralization.

Figure 7.5 Photographs of the Meta-sedimentary Units



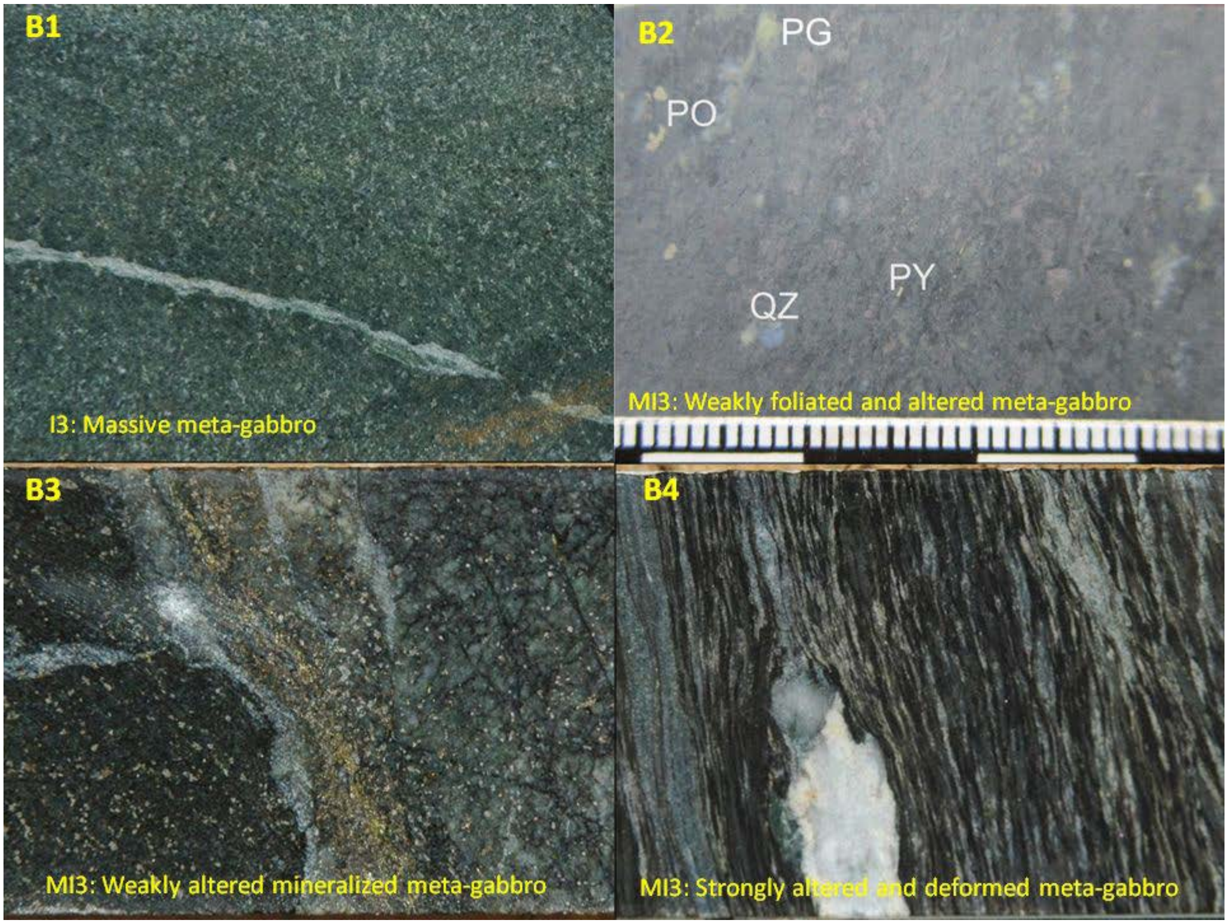
A1: Carbonaceous meta-argillite.

A2: Crenulated carbonaceous meta-argillite. P16, BBD0012, 40.79 m.

A3: Meta-sandstone. Disseminated pyrrhotite (PO) and graphite (GP) laminae. P16, BBD0214, 146.0 m.

A4: Polymictic meta-conglomerate. Lithic clasts of meta-argillite (S4), chert (S7) and felsic intrusive (I1). P8/P9, BBD0069, 75.65 m.

Figure 7.6 Photographs of the Meta-gabbro Units



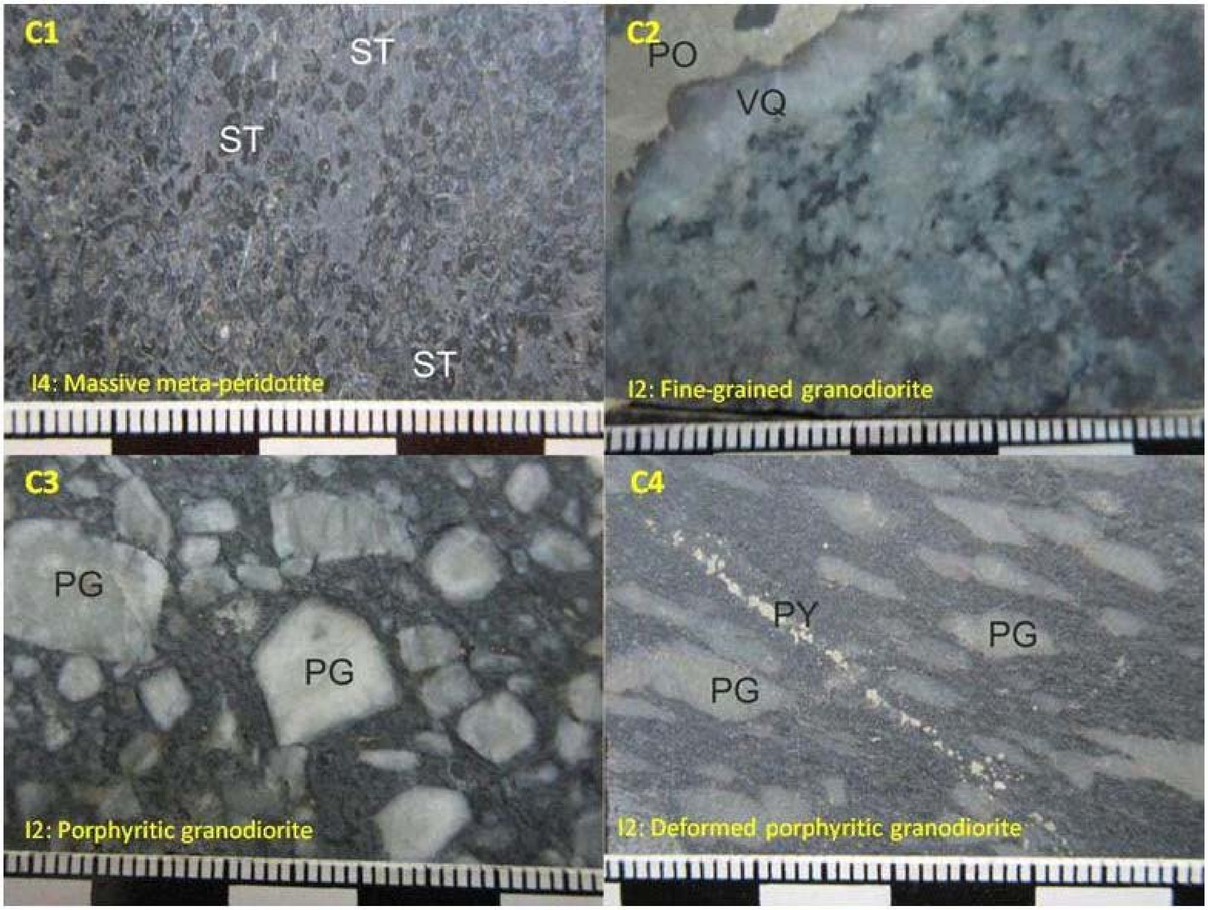
B1: Massive meta-gabbro. P11, BDD0029.

B2: Weakly foliated and altered meta-gabbro with blue quartz (QZ) and altered plagioclase (PG) phenocrysts. Disseminated pyrite (PY) and pyrrhotite (PO). Siga W, BBD0013, 120.29 m.

B3: Weakly altered mineralized meta-gabbro. P11, BBD0029.

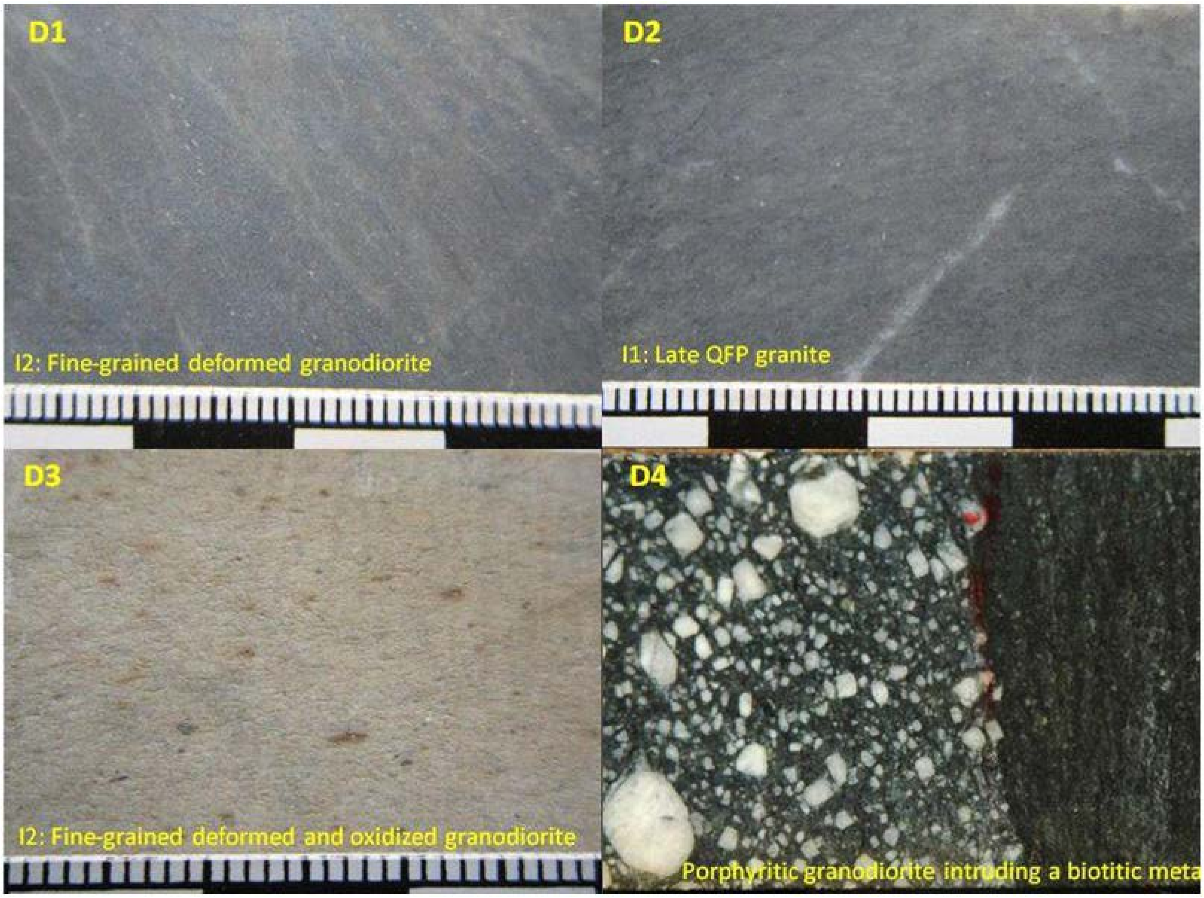
B4: Strongly altered biotitic meta-gabbro. P11, BBD0029.

Figure 7.7 Photographs of the Peridotite and Granodiorite Units



C1: Massive meta-peridotite with serpentine (ST) olivine pseudomorphs. Outcrop AE14, P11 east area.
C2: Fine-grained granodiorite cut by a quartz (VQ) and pyrrhotite (PO) vein. P8/P9, BBD0491, 120.18 m.
C3: Porphyritic granodiorite with large zoned plagioclase (PG) phenocrysts. P8/P9, BBD0612, 53.8 m.
C4: Deformed porphyritic granodiorite. P8/P9, BBD0529, 133.55 m.

Figure 7.8 Photographs of the Granodiorite and Granite Units



D1: Fine-grained deformed granodiorite. Plagioclase (PG) phenocrysts are altered and deformed. Pyrite (PY) veinlet. Maga, BBD0130, 83.0m.

D2: Late quartz feldspar porphyry (QFP) granite. P8/P9, BBD0039, 89.3 m.

D3: Fine-grained weathered granodiorite. Maga, BBD0055, 34.25 m.

D4: Late QFP granite intruding a deformed biotitic meta-gabbro. P11, BBD0029.

Figure 7.9 Geology of the Northern Area Showing Collar Location of Exploration Boreholes

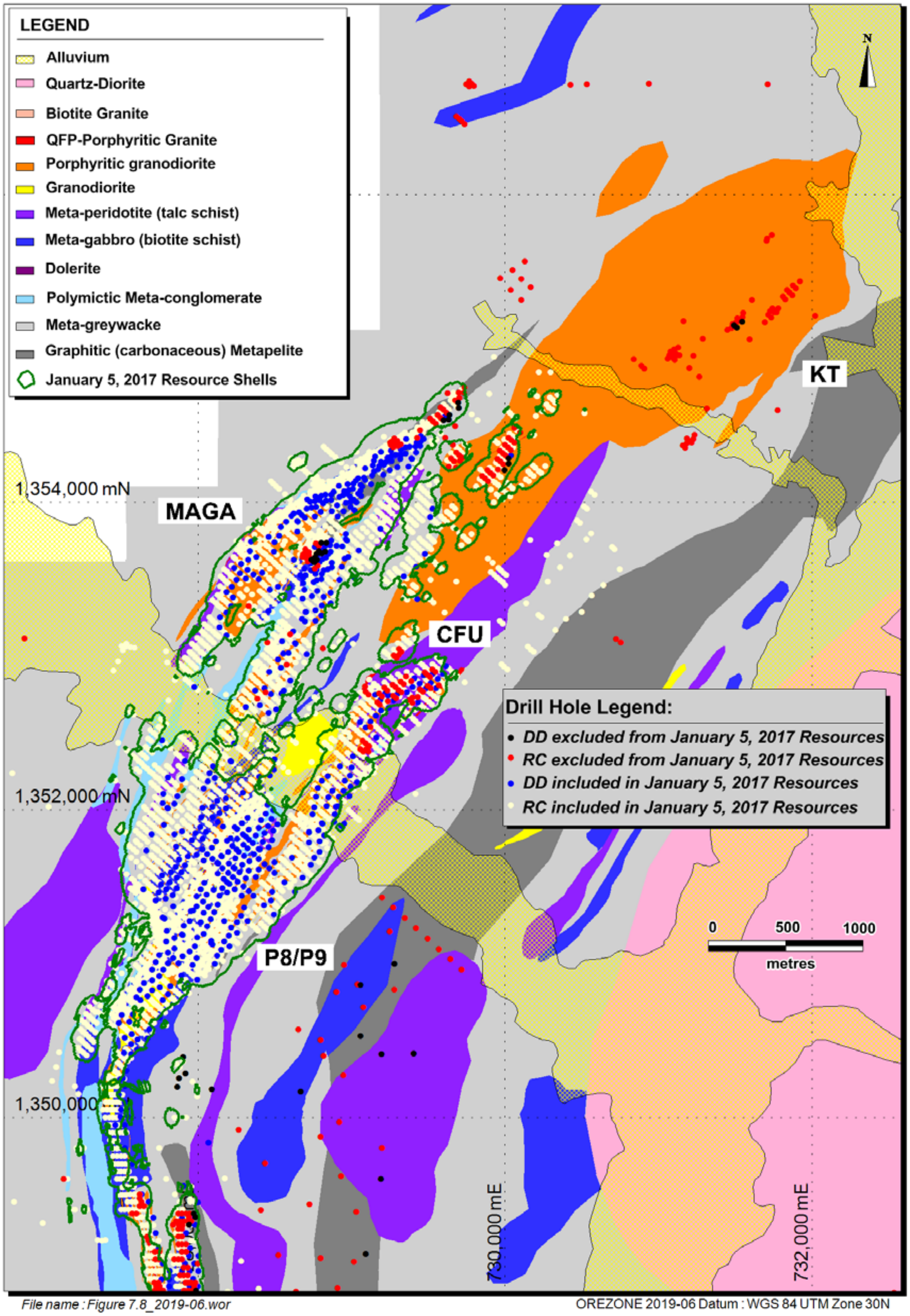
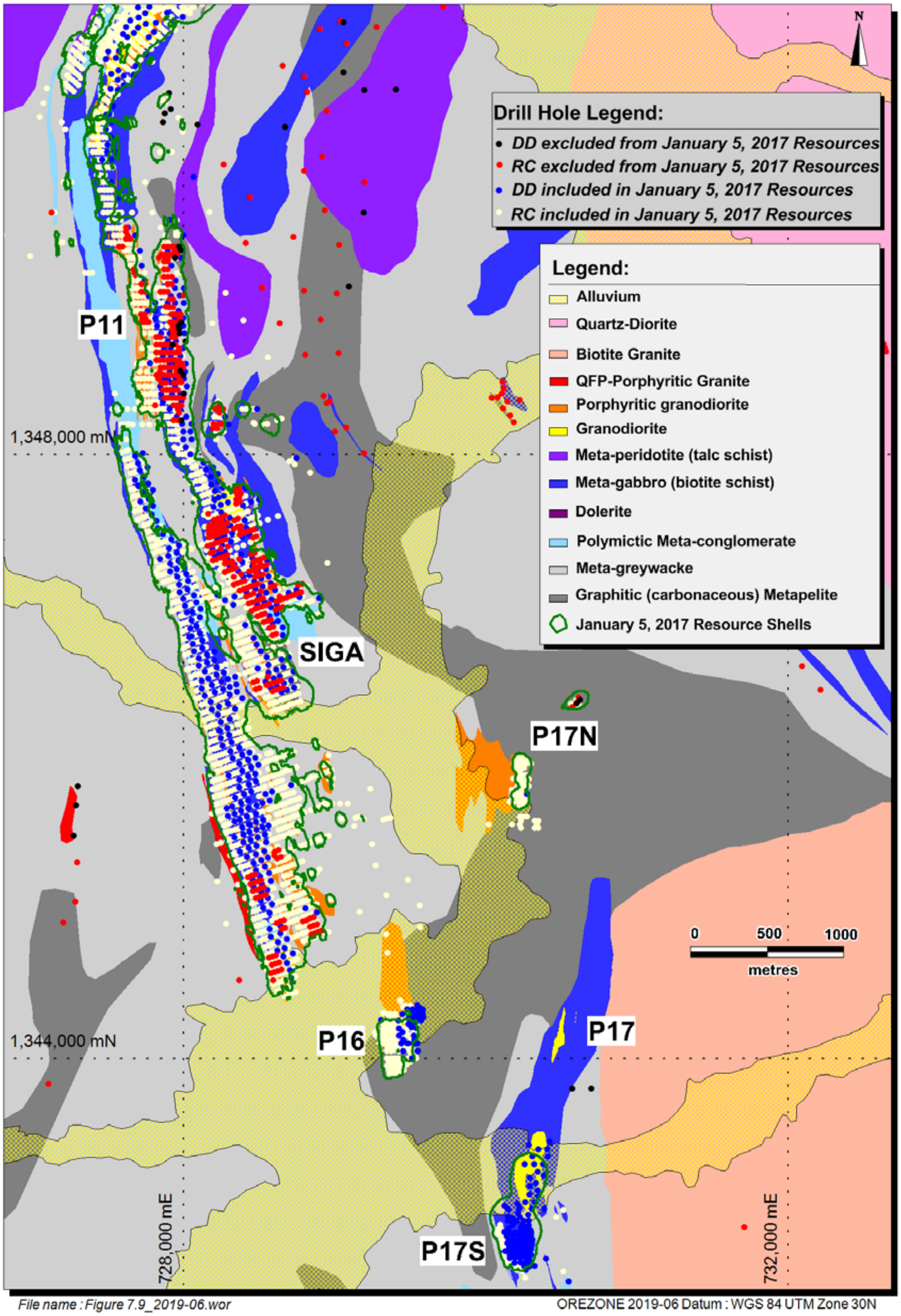


Figure 7.10 Geology of the Southern Area Showing Collar Location of Exploration Boreholes



7.2.2 Structural Geology

The gold mineralization on the Property is hosted in the BSZ, a major north-northwest to north-northeast trending structure. This shear zone has an arcuate shape and extends over tens of kilometres beyond the limits of the Property. It is interpreted as a secondary structure to the Tiébélé-Dori-Markoye Fault, a regional north-northeast trending sinistral fault that represents a major discontinuity in the Birimian rocks, across which regions of contrasting structural styles are juxtaposed.

The BSZ is visible on the aerial photos and exhibits a strong signature on the geophysical (magnetic and induced polarization) maps. The curvature of this shear zone is interpreted to be caused by the moulding of the greenstone belt on the late quartz-diorite intrusive that is located along the eastern margin of the Property.

The BSZ is oriented 040° in the northern portion of the Property and 340° in the southern portion of the Property (Figure 7.11). Most of the syn- to late-tectonic dikes are located within the BSZ, together with the gold mineralized schists and barren quartz vein arrays. The dip of the main foliation and the main lithological contacts is approximately 65º towards the southeast in the northern portion of the Property (Maga, CFU, and P8/P9 deposits), although it steepens in the Maga footwall area to approximately 75º and is approximately 55º towards the northeast in the southern portion of the Property (P11, Siga East, and Siga West-Siga South deposits). The main foliation is oriented 360° to 010° and is sub-vertical in the southeast area where the small satellite deposits (P16, P17N and P17) are located. At P17S, the main foliation is dipping about 55º towards the east-northeast and is parallel to the axial planes of W-shaped folds defined by the sequence of metasandstone-metagabbro-metagranodiorite.

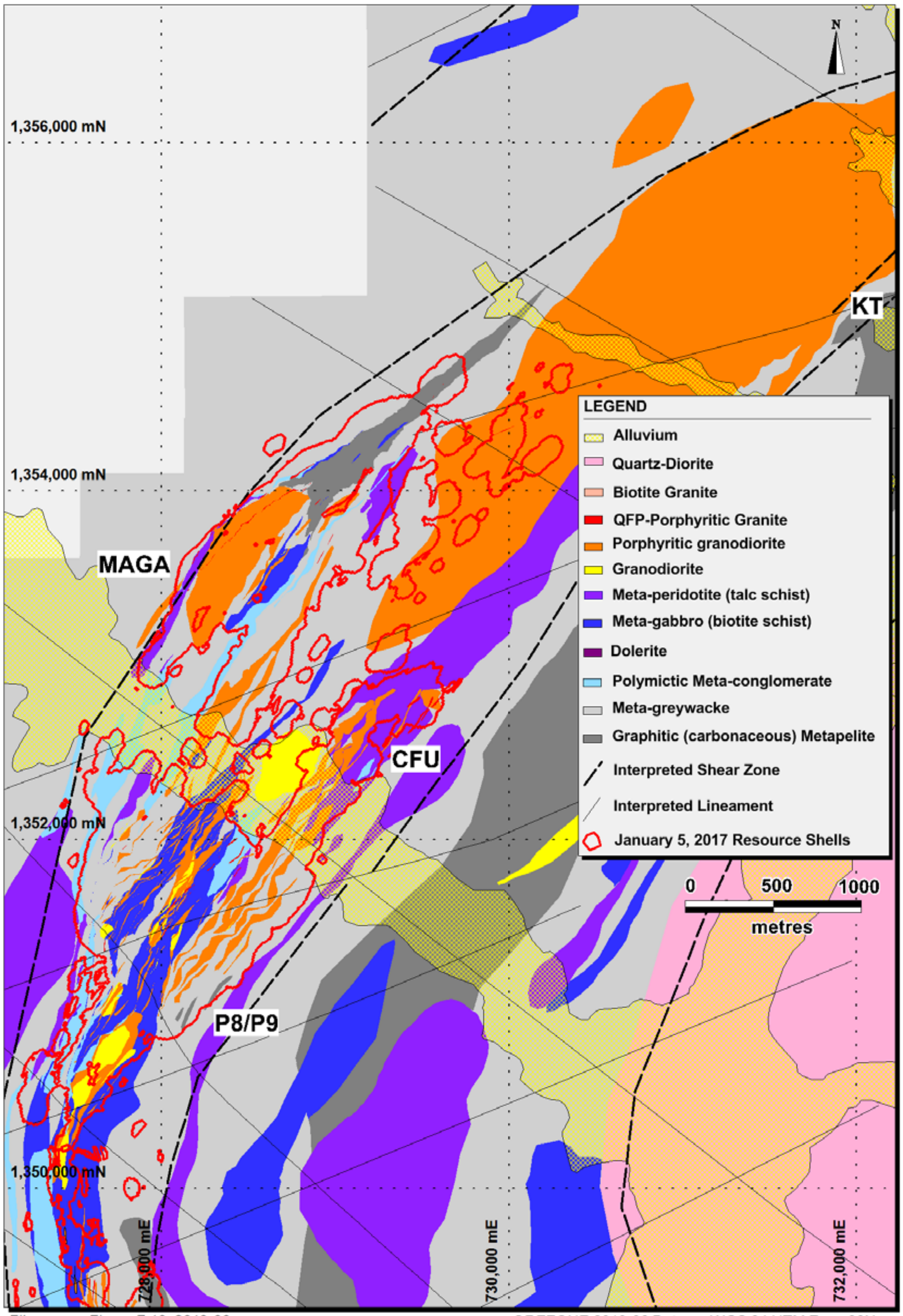
A north-south fault system is visible on satellite imagery, as well as an east-northeast and a west-northwest system. In addition, breaks in the magnetic data and apparent displacements in the mineralization support the presence of the systems oriented at 070° and 110° (Figures 7.11 and 7.12) responding to a late faulting event. Some of the gold mineralization appears to have been remobilized along the latter orientations. Fractures and quartz veins (in place auriferous) oriented roughly east-west are also noted on outcrops and in trenches.

Observations in surface outcrops and borehole geophysical data suggest that rock units within the BSZ exhibit brittle-ductile behaviour with folds, transposition, and an anastomosing pattern typical of a shear zone environment. The presence of brittle structures, such as breccias and quartz veins, have also been recorded in the drill logs and surface maps. In the P11 and Siga areas where the S_0-S_1 fabric is oriented north-northwest to north, a northeast secondary cleavage has been observed in several localities, suggesting that the dominant northeast trending regional foliation might be overprinting earlier fabrics (S_0 and S_1).

In outcrop, the foliation fabric contains a stretching lineation plunging moderately (45° to 55°) to the north. The current geological model, well constrained by oriented core observations, clearly shows that several units, including the auriferous meta-gabbro units, plunge moderately (45° to 55°) to the north-northeast, but recent work suggests that shallower plunges (20° to 35°) are typical of the fold hinges, boudins and mineralized zones in the Maga, P11, Siga East, P16 and P17S areas.

All rock units within the shear zone have been affected by a heterogeneous (brittle) - ductile strain, except the late QFP granitic dikes and some late dolerite dikes

Figure 7.11 **Geology of the Northern Area Showing Major Shear Zones and Lineaments**



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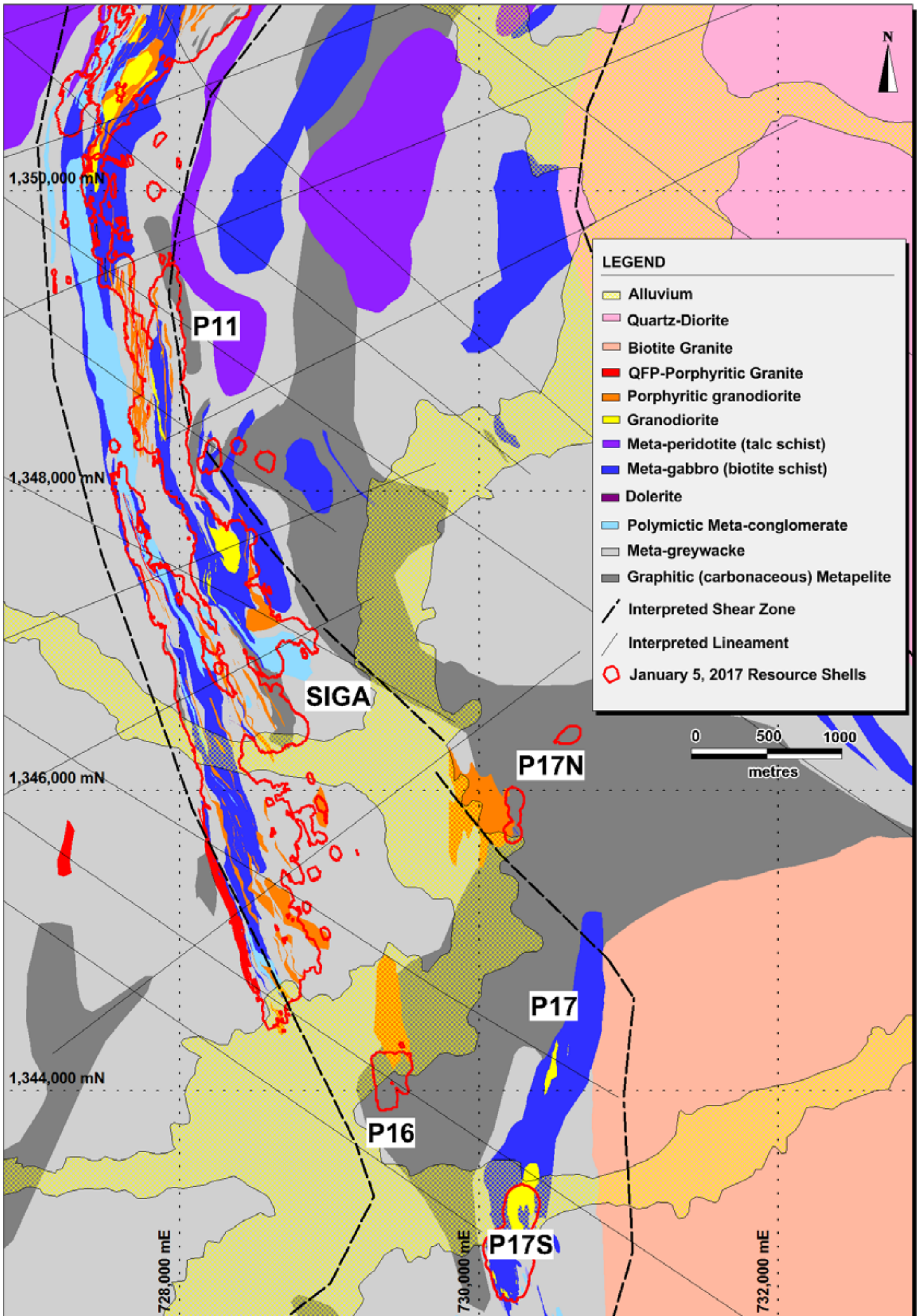
OREZONE 2019-06 Datum : WGS 84 UTM Zone 30N

Figure 7.12 Geology of the Southern Area Showing Major Shear Zones and Lineaments



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OREZONE 2019-06 Datum WGS 84 UTM Zone 30N

7.3 Mineralization

The Bomboré gold deposit occurs within the regional BSZ, a major north to northeast trending structure considered as a subsidiary to the Tiébélé-Dori-Markoye Fault. Eleven separate auriferous zones have been delineated by drilling within the 14 km segment of the BSZ located within the Property. The gold deposits were discovered by tracing gold-in-soil anomalies (Figure 7.2) to bedrock by drilling. The auriferous zones are defined by geographic coordinates in Table 7.1.

The gold mineralization in the Property area is associated with metasomatic replacement zones where the mineral assemblage of the protolith is replaced by an assemblage of silica, carbonate (calcite and ankerite), albite, sericite, biotite and sulphides developed within strain portions of the BSZ. The BSZ is also characterized by the presence of arrays of structurally controlled quartz veins and veinlets that representant about 1 to 2% of the volume of the BSZ. Most quartz veins are oriented sub-parallel to the foliation and exhibit strong strain, however, the presence of relatively unstrained quartz veins and breccia in drill core attest the protracted history of vein formation and deformation. Late west-trending veins crosscutting the main foliation fabric are also observed. Locally, there is evidence suggesting that gold mineralization was remobilized into northeast and southeast dilation zones associated with late faults.

Table 7.1 Location of the Bomboré Gold Zones

Bomboré Gold Zones	Easting*		Northing*	
	Minimum	Maximum	Minimum	Maximum
Kiin Tanga (“KT”)	731,300	731,900	1,354,500	1,355,100
Maga	727,900	730,500	1,352,600	1,354,800
Colline de Fusille (“CFU”)	728,600	729,700	1,352,000	1,353,200
P8/P9	727,000	729,100	1,349,800	1,352,600
P11	727,000	728,200	1,348,000	1,350,100
Siga East (“SE”)	727,800	729,000	1,346,200	1,348,000
Siga West-Siga South (“SW-SS”)	727,500	729,000	1,344,400	1,348,200
P16	728,200	729,700	1,343,800	1,344,300
P17	730,400	730,800	1,343,900	1,344,400
P17N	730,100	730,800	1,345,600	1,346,500
P17S	730,000	730,300	1,342,600	1,342,900

** UTM Projection – WGS84 datum, Zone 30 North.*

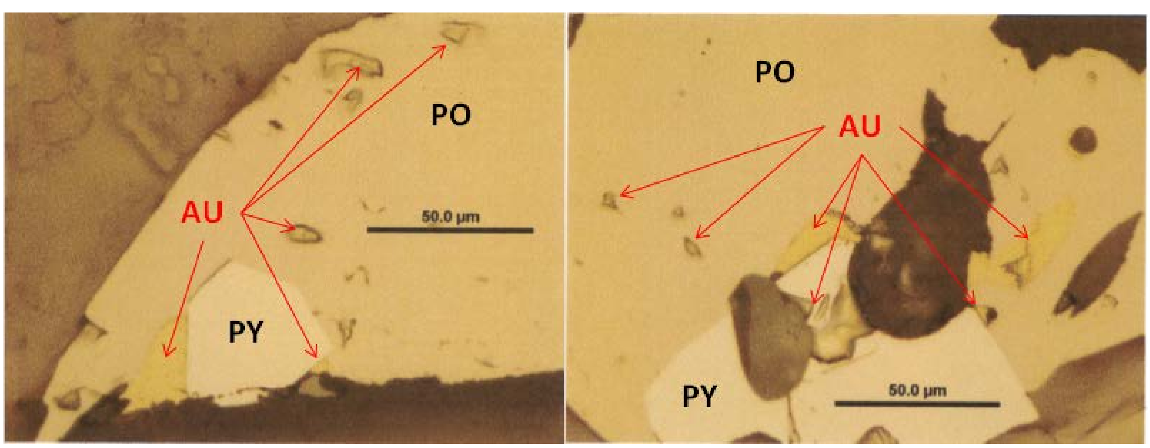
The quartz associated with the gold mineralization is milky white to smoky, locally vitreous and may contain tourmaline. The widths of the veins range from two centimetres to 1 m, with an average of ten centimetres. The near surface gold mineralization with grades of up to 0.2 g/t Au is pervasive regardless of quartz veining and is associated with fine disseminated sulphides, predominantly pyrite. With the exception of some late smoky quartz veinlets where visible gold has been observed in a few instances, quartz vein material within the auriferous zones is typically white and barren, as demonstrated by the gold deportment and metallurgical test work completed on the Project.

Generally, the gold occurs as fine grain electrum (< 10 µm) but can be visible in outcrop. Artisanal mining over the 1990-2016 period attests to the existence of coarser gold locally. Gold occurs as free gold and is mainly associated with pyrite, pyrrhotite, chalcopyrite, and arsenopyrite. Most sulphides occur as disseminations and fine stringers sub-parallel to the foliation fabric suggesting development in active shear zone or re-mobilization. Magnetite and graphite are present locally. Although the sulphide content can be as much as 5%, it is on average only 1% to 2% in fresh (i.e., non-weathered) mineralized rocks.

Gold mineralization is most commonly hosted in the biotite schist (meta-gabbro) and its host rocks (typically the meta-sandstones) and the granodiorite dikes that intrude the gabbros, although in Maga north, P16 and P17N areas the meta-argillites are the main host. The syn-tectonic granodioritic intrusives are also mineralized, although to a lesser degree than the biotite schist and the meta-argillites. The meta-conglomerate and meta-peridotite are unfavourable hosts. The meta-gabbro might represent the best chemical trap given its high iron content if gold was transported as a thio-complex, as suggested by the pervasive fine pyritic assemblage that is associated with the gold mineralization in the sulphide zone. Although much of the gold resources defined within the Project area are hosted in the meta-gabbro unit, the deformed granodiorite and its contact zone with the meta-gabbro host is where the better-grade mineralization is concentrated.

Petrographic work on fresh rock samples in 2008 (Schandl 2008a, b and c) revealed that the gold mineralization is predominantly associated with silica and iron carbonate, although sericite is a ubiquitous and often an abundant alteration mineral in a number of the gold-enriched rocks. Gold occurs as electrum, native gold, and gold telluride (calaverite). Small gold grains are included in pyrite, in fractures of pyrite grains (Figure 7.13), and as free gold in the fine-grained quartz-goethite matrix in the weathered zone. The major sulphides are pyrite and pyrrhotite with subordinate amounts of chalcopyrite, covellite, galena, and arsenopyrite. Pyrrhotite and chalcopyrite are found mostly in the biotite schist and arsenopyrite in the metasediments. The gangue of the saprolitic ore consists of an assemblage of quartz, sericite, kaolinite, hematite, and goethite

Figure 7.13 Primary Gold Mineralization (Sulphide Zone): Gold (Au) Occurring as Inclusions in Pyrite (PY) and Pyrrhotite (PO)



At a cut-off grade of approximately 0.2 g/t Au, the gold mineralization exhibits reasonable continuity over a strike length of approximately 10 km. At this cut-off grade, the gold mineralization forms more restricted corridors (500 m to 1,000 m in length and 10 m to 100 m in width) defining anastomosing patterns, parallel and slightly oblique to the general trend of the BSZ.

These higher-grade corridors formed the basis for defining geostatistical domains within each litho-domain considered for resource estimation. One of the benefits of the 2010 to 2013 infill drilling programs was the delineation of higher-grade sub-domains based on a cut-off grade of 0.5 g/t Au with the broader low-grade domains based on a lower cut-off grade of 0.2 g/t Au. The higher-grade sub-domains have a strike length of up to 500 m and a width typically between 5 m and 30 m.

The typical texture of the gold mineralization host rocks in drill core is shown in Figure 7.14.

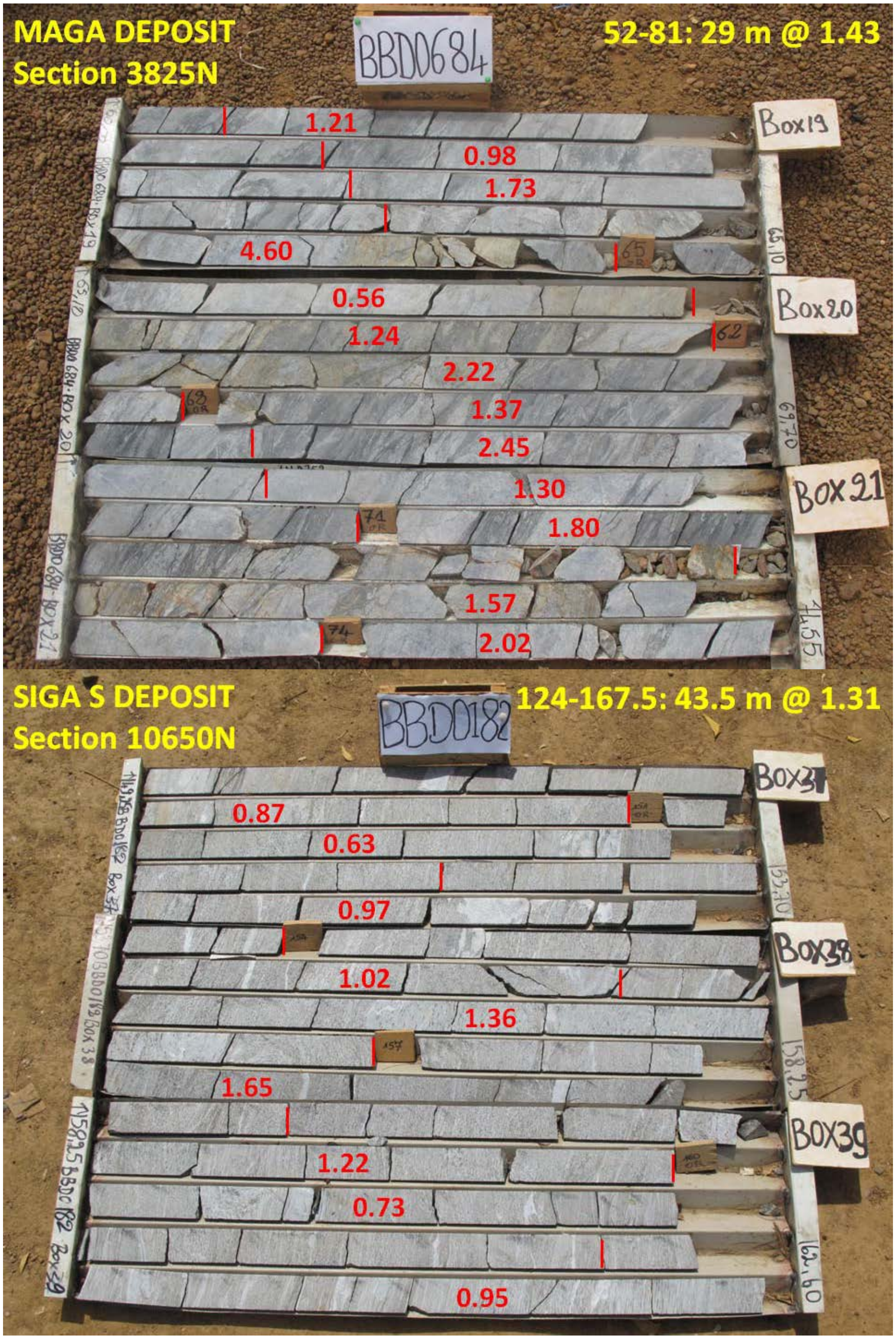
7.3.1 Sequence of Geological Events

Based on the work completed by Orezone since 2008, the following geological history is interpreted for the Property area:

- Sedimentation of a sequence of carbonaceous argillite followed by sandstone and capped by polymictic conglomerates.
- Intrusion of mafic (gabbroic to dioritic) sills and dikes, followed by ultramafic (peridotite) intrusives.
- Regional deformation and prograde metamorphism culminating under greenschist facies, biotite zone conditions, with pre- to syn-tectonic intrusion of fine-grained granodiorite dikes and small intrusion and later sets of syn-tectonic dikes and larger intrusions of porphyritic granodiorite.
- Syn-to late metamorphic albite-calcite-tourmaline-biotite-pyrite metasomatism – the main gold mineralizing event.
- Late-tectonic intrusion of QFP granitic dikes.

Retrograde brittle-ductile deformation – local remobilization of gold in late quartz veins.

Figure 7.14 Typical Texture of the Gold Mineralization in the Core of the Maga Deposit Meta-Argillite (top) and Siga South Deposit Biotite (bottom)



8.0 DEPOSIT TYPES

The following is updated from the Technical Report entitled Preliminary Economic Assessment Bomboré Gold Project Burkina Faso prepared by GMSI and dated January 22, 2014 (Gourde, Gignac and Menard, 2014).

The Bomboré gold deposit is the principal gold mineralization of potential economic significance found to date on the Property. It is located in an area that is principally prospective for orogenic gold deposits. Similar to gold deposits found elsewhere in late Proterozoic Birimian terrains of West Africa, the Bomboré gold deposit exhibits a structural control and hydrothermal activity. It is a large tonnage, low grade system that has similar characteristics to other Birimian gold deposits such as Kiaka in Burkina Faso, Damang, Yamfo-Selwi and Obuasi in Ghana, and the Sadiola deposit in Mali.

Hydrothermal deposits are typically late orogenic deposits and exhibit strong relationship with regional arrays of major shear zones. The gold mineralization is typically associated with a network of quartz veins containing subordinate amounts of carbonate, tourmaline, sulphides, and native gold. In these deposits, the gold is typically free milling. Alternatively, the gold mineralization can also be associated with disseminated sulphides in strongly deformed alteration zones. In the latter case, gold may be free milling but also locked in the sulphide lattice structure and refractory. The Bomboré deposits are essentially stratabound disseminated sulphide bodies preferentially hosted in the meta-gabbro and meta-argillite lithologies, which are interpreted to have acted as preferential gold traps during a syntectonic deformation and metasomatic event due to their chemical and rheological characteristics.

The wet paleoclimate that preceded the current semi-arid climate in Burkina Faso has resulted in extensive surface oxidation of bedrock and a deep weathering profile. Oxidized bedrock can occur up to a vertical depth of 100 m. Gold deposits span both the surface oxide zone and a deeper sulphide zone. In the oxide zone, gold typically occurs in a free milling form but is grind sensitive in the sulphide zone.

9.0 EXPLORATION

The following is taken from the final land tenure report filed by Orezone for the Bomboré I Exploration Permit (Derra and Tamani, 2016).

9.1 Bomboré Exploration Permit

In 2002, ORINC entered into an option agreement with Channel and Solomon and assumed the funding and execution of exploration activities on the 150,000 ha Bomboré exploration permit until its expiry in January 2004. Exploration activities during this period consisted of data compilation and a RC drilling programme (Ackert, 2004). The work is described in detail in a series of Orezone reports (Zongo, 2003a and b; and Marquis, 2003).

9.2 Bomboré I, Toeyoko, Bomboré II, Bomboré III and Bomboré IV Exploration Permits

In February 2004, ORINC was granted the 25,000 ha Bomboré I exploration permit, which covered the most prospective portion of the former 150,000 ha Bomboré exploration permit, i.e. the BFT area. In January 2010, the Bomboré I permit was reduced to 10,450 ha, and in July 2011, ORINC was granted most of the Bomboré I permit area abandoned in 2010 as the 6,300 ha Toeyoko permit. In February 2016, the Bomboré I permit expired. In December 2016, OBSA was granted a 2,500 ha Industrial Mining Permit that covers the reserves identified within the former Bomboré 1 exploration permit, and in January 2017, Orezone Inc. SARL was granted the 1,815 ha Bomboré I permit, the 4,810 ha Bomboré III permit and the 1,235 ha Bomboré IV permit, which together recaptured most of the Bomboré I permit ground outside of the Orezone Bomboré SA Industrial Mining permit. A summary of the activities undertaken at the Property from 2003 to 2018 by Orezone is presented in Table 9.1. Other project development work such as metallurgical testwork, geotechnical drilling and environmental studies is included in the summary.

Table 9.1 Summary of Exploration on the Bomboré Property

Period	Exploration Activities and Studies
2003	• RC drilling: Mankarga grid: 8 boreholes (614 m); P8/P9: 11 boreholes (747 m); Kiin Tanga: 13 boreholes (640 m).
2004	• Compilation work. • Report on the 2003 RC drilling programme.
2005	• 217 RC boreholes (13,829 m) at P8/P9, Maga and Kiin Tanga.
2006	• Establishment of a pair of trigonometric beacons in P8/P9 area and of survey control points from Kiin Tanga to Siga. • Survey of all RC and core boreholes that could be found. • Photogrammetric airborne survey (112 km^3). • 121 RC boreholes (8,770 m) at Maga, P8/P9, P11 and Siga. • Ground gradient induced polarization (IP) survey (153.6 km; 100 m line spacing; 25 m stations) at Maga, P8/P9 and CFU. • 1,450 check assays on the 2005 RC samples.

Period	Exploration Activities and Studies
	• 614 RC composite samples collected for cyanidation metallurgical test work. • 39 core and 17 rock outcrop samples petrographic study.
2007	• Met-Chem Resource Estimate based on RC and core borehole data up to March 2007 – Initiated in August 2007, delivered in February 2008. • Systematic mapping, prospecting, sampling, and gold assaying of outcrops and gold workings. • 57 core boreholes (5,314 m November 2007 to February 2008) mostly within the 2007 resource model core samples assayed for gold and also used for structural measures, multi-element inductively coupled plasma (ICP) orientation study, petrographic study, and petrophysical analyses.
2008	• Systematic mapping, prospecting, sampling, and gold assaying of outcrops and gold workings. • 268 RC boreholes (19,963 m February to April 2008). • Quality assurance/quality control (QA/QC) report, 2007-2008 RC-core programmes. • Cyanidation test work under the supervision of H.C. Osborne and Associates (Commerce City, CO, USA) completed in September 2008. • Petrographic studies by Dr. Schandl (Toronto, ON, Canada) completed between April and September 2008. • 662 multi-element ICP analyses from core samples. • Re-logging of all RC and core boreholes to reconcile the surface mapping, petrography, and ICP data. • SRK Resource Estimate based on RC and core data up to August 2008 – Initiated in June 2008, delivered in November 2008. • Compilation of all historical RC borehole detailed journals to create a penetration rate model. • Academic study of the petrography and structure of the Bomboré 1 deposits by H. Zongo under the supervision of Dr. Lompo from the Université de Ouagadougou – initiated in May 2008. • Check sampling of RC boreholes with poor QA/QC scores (3,211 samples).
2009	• Re-logging of selected RC and core boreholes, revised geological model. • High-resolution (50 m 10 m) resistivity surveys (237 km) over the Maga, P8/P9, P11, and Siga areas in March-April 2009. • Core drilling programme (April to June 2009) including 20 boreholes (4,502 m) to a vertical depth of 175 m in the P8/P9 area on two fences 200 m apart and three PQ boreholes (235 m) for metallurgical sampling. • Commissioning of bench top rotary sample dividers for all pulverized samples. • Validation of all historical core specific gravity determinations, including new more closely spaced determinations. • QA/QC reports on the April-June 2009 core drilling programme and on various check assay programmes. • Metallurgical test work (June to November 2009) by AMMTEC (Perth, WA, Australia) under the supervision of GBM Engineering Consultants Limited (GBM) (Twikenham, Middlesex, UK), including bottle tests on coarse material and milled material, flotation/leaching tests on milled material, gravity concentration tests, and column leaching, AMD, UCS, Bond impact, Bond abrasion, Bond rod mill, Bond ball mill and JK Drop-weight tests on two sets of composite samples representative of the oxide, transitional and fresh Bomboré mineralized material – final AMMTEC report January 2010. • Petrographic and structural study of the Bomboré gold deposits; Ph.D. progress report. • Environmental Baseline Study (April to July 2009) by Bureau d'Études des Géosciences, des Énergies et de l'Environnement (BEGE) – report July 2009.

Period	Exploration Activities and Studies
	• Preliminary Environmental Impact Study (EIA) by BEGE (April to September 2009) – Report September 2009. • Bench-top XRF programme commenced in April 2009 – All available historical core pulp samples (> 10,000) were analyzed for a suite of 35 elements. • Core drilling programme (November-December 2009), five boreholes (3,001 m) drilled to a vertical depth of 300 m in the Maga, P8/P9, and Siga areas. • Several programmes of check assays on RC (n = 130), core (n = 388) and leach tail (n = 914) samples.
2010	• Migration of all historical RC and core data to Datashed database. • QA/QC report on the November-December 2009 core drilling programme. • Mogtédo camp expansion and commissioning of laboratory with two rotary sample dividers (RSDs) for RC samples. • Commissioning of a dedicated laboratory equipped with three bench top RSDs at the Kossodo office. • Check assay programme on SGS Tarkwa and Abilab Ouagadougou RC samples (n = 354) with a soluble gold grade > 1 g/t. • 389-borehole auger drilling programme to investigate the overburden-saprolite interface and saprolite over the Siga South, P11-P8/P9 gap, and other targets (3,055 m). • 617 borehole RC definition and resource expansion drilling programme (42,456 m). • Establishment of four new pairs of trigonometric beacons at Kiin Tanga, Maga, Siga, and Siga South followed by a check survey of all historical survey control stations and corrections of historical collar positions. • Several programmes of check assays on RC (n = 2,236), core (n = 141) and leach tail (n = 12,411) samples. • QA/QC report on the 2010 RC drilling programme and various check assay programmes. • Bench-top XRF analysis of approximately 40,000 samples. • Geological 3D model updated in Gemcom. • SRK Resource Estimate based on RC and core data up to April 2010 – Initiated in July 2010, delivered in October 2010. • 2,374 line-km high-resolution magnetometry and radiometry airborne survey over the Bomboré 1 permit by UTS-Aeroquest in November 2010. • Initiation of a major core drilling definition programme in November 2010 to define the 2010 sulphide resources on a 50 m by 50 m drilling pattern to a vertical depth of about 125 m. The programme was completed in June 2012 and totalled 770 boreholes for 116,795.5 m. • Initiation of the construction of a new base camp for the Project in November 2010. • High-resolution (50 m by 10 m) resistivity surveys (243 km) over the KT, Maga, CFU, P11, Siga South, P16 and P17 areas in December 2010 and January 2011. • Initiation of a major RC drilling programme in February 2011 to define the 2010 oxide resources on a 50 m by 25 m drilling pattern and to test several new targets. The programme was completed in June 2012 and totalled 2,375 boreholes and 135,167 m.
2011	• 2,547 borehole auger drilling programme between February 2011 and July 2011 to investigate the overburden-saprolite interface and the saprolite over several new targets (12,146 m). • Report from AccuMin Minerals Services on the lithostructural controls of the Bomboré gold mineralization. • High-resolution photogram metric base map of the Bomboré permit generated by Photosat in May 2011.

Period	Exploration Activities and Studies
	• PEA by GMSI delivered in June 2011. • Initiation of a detailed baseline environmental and impact study based on the PEA/Carbon-in-Leach (CIL) project. The socio-economic study by Société de Conseil et de Realisation pour la Gestion de l'Environnement (SOCREGE) commenced in May 2011 and the EIA by BEGE commenced in September. • Toéyoko permit granted to Orezone in July 2011. • Commissioning of a weather station at the new Bomboré camp in September 2011. • Initiation of a detailed CIL process metallurgical study in September 2011, using McClelland Laboratories, Inc. (McClelland) under the supervision of Woods Process Services. A suite of 76 samples representative of the various oxide, transition, and sulphide facies of each of the deposits sent to McClelland. Final report delivered in February 2013. • 1,901 line-km high-resolution magnetometry and radiometry airborne survey over the Toéyoko permit by UTS-Aeroquest in October 2011. • Delivery by SOCREGE of the report on the socio-economic study relevant to the 2011 PEA/CIL project. • High-resolution photogram metric base map of the Toéyoko permit generated by Photosat in December 2011.
2012	• Initiation of an environmental testing study in February 2012 on a series of 28 composite waste samples by McClelland. Results received in July 2012. • 598 borehole auger drilling programme during the April-May 2012 period to investigate the overburden-saprolite interface and saprolite over several new targets on the Bomboré 1 permit (2,299 m). • 587 borehole auger drilling programme during the April-May 2012 period to investigate the overburden-saprolite interface and saprolite over several new targets on the Toéyoko permit (2,561 m). • Petrographic and mesoscopic catalogue of photographs of the Bomboré lithologies; Ph.D. progress report. • Report from Economic Geology Consulting on the mineralogy of the master composite samples used by McClelland for the detailed CIL process metallurgical study. • Delivery by BEGE in July 2012 of the report on the environmental baseline study relevant to the 2011 PEA/CIL project. • High-resolution (50 m 10 m) resistivity surveys (41 km) over the P17N area on Bomboré 1 permit in July 2012. • High-resolution (50 m 10 m) resistivity surveys (51 km) over the P17S area on Toéyoko 1 permit in July 2012. • Prospecting, outcrop sampling, and geological mapping on new regional targets on Bomboré 1 permit from March to July 2012 (401 samples). • Prospecting, outcrop sampling, and geological mapping on new regional targets on Toéyoko 1 permit from March to June 2012 (190 samples). • SRK Resource Estimate based on RC and core data up to March 2012 – Initiated in March 2012, delivered in August 2012. • Initiation of a CIL process FS in June 2012 under the direction of GMSI from Brossard, QC, Canada. • Pit slope geotechnical study by Golder from Montreal, Canada, initiated in August 2012. The final report was delivered by Golder in April 2013

Period	Exploration Activities and Studies
	• Core drilling definition programme started in September 2012 to define the 2012 Inferred sulphide resources on a 50 m 50 m drilling pattern to a vertical depth of about 150 m. The first phase of the programme was completed in February 2013 and totalled 121 boreholes for 23,109.5 m. • QA/QC report, RC, core, and auger programmes completed from November 2010 to June 2012. • QA/QC report, RC, core, and check assay programmes completed from June 2012 to September 2012. • RC drilling programme started in September 2012 to define the 2012 Inferred oxide resources on a 50 m 25 m drilling pattern and to test several new targets. The first phase of the programme was completed in April 2013 and totalled 541 boreholes and 32,440 m. • Initiation in October 2012 by Golder from Montreal, Canada, of a geochemical characterization study of waste rock, tailings and potential construction material at the Project. Final report received in December 2013. • Initiation in November 2012 of site investigation by Golder from Montreal, Canada, for a feasibility level geotechnical study of the tailings and water management structures for the Project. Preliminary report delivered by Golder in April 2013. • Initiation in November 2012 of site investigation by Golder from Montreal, Canada, for a feasibility level geotechnical study of the design of foundations at the processing plant site and at the Nobsin and Bomboré bridges for the Project. Preliminary technical memorandum delivered by Golder in May 2013. • Completion of the preliminary scrubbing test work completed by Orezone under the direction of GMSI, targeting the saprolite gold resources. • QA/QC report, RC, core and check assay programmes completed from October 2012 to December 2012. • Initiation in December 2012 of a complementary comminution study with Hazen Research Inc., from Golden, CO, USA, on three granodiorite samples from the weathered zone. Report delivered in February 2013. • Initiation in December 2012 of a complementary comminution study with SGS Canada Inc from Lakefield, ON, Canada, on twenty sulphide samples. Report delivered in May 2013.
2013	• Initiation in January 2013 of an eight-borehole, 280 m saprolite PQ core drilling programme for a scrubber test work programme with Met-Solve Laboratories Inc. (Met-Solve) from Langley, BC, Canada. Report delivered in May 2013. • Delivery by SOCREGE of an interim socio-economic study relevant to the 2011 CIL Definitive Feasibility Study (DFS) project. • Final report of the study of the archaeological artifacts collected by BEGE. • Final report from McClelland on the CIL DFS process metallurgical study. • Initiation in March 2013 of a metallurgical study on pyrrhotitic samples with COREM from Quebec City, QC, Canada. Report delivered in May 2013. • SRK Resource Estimate updated based on RC and core borehole data up to November 2012 for the North and South models and March 2013 for the Southeast model. • Decision to interrupt in June 2013 the CIL process DFS due to adverse economic conditions. Initiation of a review of the 2011 Heap Leach (HL) PEA. • Interim report from BEGE on complementary botanical and archaeological studies completed for the CIL DFS. • Decision in August 2014 to update the 2011 HL PEA, under the supervision of GMSI, and with the support of Kappes, Cassiday & Associates (KCA) from Reno, NV, USA, and Golder from Reno, NV, USA for the process engineering.

Period	Exploration Activities and Studies
	• QA/QC report, RC, core and check assay programmes completed from November 2012 to June 2013. • Preliminary PEA HL facility design delivered by Golder in November 2014. • Final DFS report from Golder in December 2013 on the CIL process geochemical characterization of waste rock, tailings, and potential construction material.
2014	• Release in January 2014 of the findings of the HL PEA completed under the direction of GMSI. • Decision to proceed with an HL DFS in January 2014. • Initiation in January 2014 of the HL metallurgical DFS under the supervision of KCA. This study includes one sample consisting of the coarse fraction scrubbed from the Met-Solve Laboratories Inc. (Met-Solve) 2013 oxide core samples scrubbing programme. Final report delivered in August 2014. • Preliminary design delivered by Golder in February 2014 of the HL facility at a new site retained for the HL DFS. • Initiation of the HL DFS geotechnical study under the supervision of Golder. Field report delivered in July 2014. The field programme executed in February and March 2014 included seven new core boreholes (170 m), eight new pressure metre boreholes (167 m), 51 new RC boreholes (2,167 m; a piezometer was installed in 8 boreholes), and 71 test pits (up to 5 m deep). All the samples were described. Laboratory test work was completed and reported from April to July 2014. All the samples were, if possible, used by Orezone as part of the sterilization programme, i.e., they were described, assayed for gold, and analyzed by XRF (multi-elements Orezone bench top Niton units). • Updated CIL process interim baseline environmental study from BEGE delivered in March 2014. • Report delivered in April 2014 of the audit completed by WSP Canada Inc. (WSP Canada) on the ESIA and RAP work completed by BEGE and SOCREGE since 2011, and on the gaps to fill to complete the HL DFS. • Cyanide leach report delivered in April 2014 by Met-Solve on the fine fraction scrubbed from the 2013 oxide core samples. • Limited core drilling programme (1,114 m) in May 2014 in the CFU and P17S areas. • RC drilling programme (21,383 m) from May to July 2014, essentially infill definition drilling in the north area of the project. • Hiring of KCA to coordinate and deliver the hybrid process DFS. • Complementary DFS comminution test work report delivered by SGS Lakefield Canada in June 2014. • Updated CIL process interim baseline environmental study from BEGE delivered in June 2014. • Updated CIL process interim socio-economic study from SOCREGE delivered in July 2014. • Decision to assess a hybrid process (HL and CIL) for the DFS based on the HL metallurgical study results to reduce the operational risks inherent to the high-cement agglomeration requirement for the saprolite material. • Release in June 2014 of the preliminary conclusions about the hybrid process based on the Met-Solve and KCA test work. Final report on the hybrid process preliminary test work delivered by KCA in November 2014. • Hybrid design trade-off study delivered by Golder in June 2014. • Revised ESIA and RAP terms of reference relevant to the hybrid process submitted to the Ministry of Environment in July 2014. • PFS assessment of the hybrid process facility (tailings storage and heap leach pad) delivered by Golder in August 2014. • Preliminary assessment by WSP Canada in August 2014 of the 2009 and 2014 Bomboré

Period	Exploration Activities and Studies
	metallurgical results relevant to the environmental impacts of the hybrid process DFS. • Preliminary assessment by WSP Canada in September 2014 of the baseline air quality conditions relevant to the environmental impacts of the hybrid process DFS. • Preliminary assessment by WSP Canada in September 2014 of the baseline acoustic conditions relevant to the environmental impacts of the hybrid process DFS. • QA/QC report, RC, core, geotechnical and check assay programmes completed from July 2013 to August 2014. • DFS pit slope recommendations from Golder delivered in November 2014. • Initiation of the hybrid process DFS geotechnical study under the supervision of Golder. The field programme was executed in December 2014 and included 40 new RC holes (1,153 m; a piezometer was installed in one borehole) and 58 test pits (up to 5 m deep). All the samples were described. Laboratory test work was completed and reported from January 2015 to March 2015. All of the samples were, if possible, used by Orezone as part of the sterilization programme, i.e. they were described, assayed for gold, and analyzed by XRF (multi-elements Orezone bench top Niton units).
2015	• DFS Assessment, Hybrid Facility (Tailings Impoundment and Heap Leach Pad), final report delivered by Golder in January 2015. • Progress report delivered by BEGE in January 2015 on the complementary archaeological and ethnographic studies relevant to the hybrid process DFS. • QA/QC report, geotechnical and check assay programmes completed from September 2014 to February 2015. • Various reports delivered by Golder such as the design of the waste rock dumps, plant foundations, bridge foundations, and surface water management infrastructure. • Final revision of the ESIA and RAP terms of reference relevant to the hybrid process submitted to the Ministry of Environment in February 2015.
2016	• A small drill programme completed from November to December 2016, including 3,162 m of RC drilling in the P13 and P17S areas, and 2,806 m of core definition drilling in the P17S area. • A small induced polarization and gravimetry test survey was also completed during the drill programme.
2017	• A metallurgical test work programme on one P17S granodiorite composite sample was initiated in July 2016 and completed in January 2017. The test work included head analysis, QEM-RMS mineralogy, Bond ball mill grindability, gravity separation, whole material and gravity tailing cyanidation, flotation and concentrate cyanidation testwork. • Induced polarization surveys (various configurations) totalling 37, 425 m of profiles were completed by Sagax Afrique on the Toéyoko and Bomboré IV permits in the P17 and P17S deposits area. • Auger drilling programmes totalling 470 boreholes and 1,693 m were completed in January and February 2017 in the MV3, BV1 and BV2 areas before the mandatory 25% surface reduction of the Toéyoko permit. • A small reverse circulation drilling programme of 27 boreholes totalling 985 m was completed in March 2017 in the P17S area on Toéyoko permit to complete the definition of the shallow mineralization and follow up on some auger anomalies. • A core drilling program of 52 boreholes totalling 7,457 m was completed from February to June 2017 in the P17S area on Toéyoko permit to advance the definition of the P17S deposit. • An auger drilling programme totalling 144 boreholes and 528 m and a small RC drilling programme of 17 boreholes totalling 857 m were completed in February and March 2017 on the new Bomboré II permit to follow up on auger anomalies in the vicinity of planned resettlement sites, which were sterilized.

Period	Exploration Activities and Studies
	• A small auger drilling programme totalling 76 boreholes and 339 m was completed in February 2017 on some weak geochemical anomalies within the limits of the new Bomboré III permit. • A core drilling programme of 13 boreholes totalling 3,275 m was completed from April to June 2017 in the P17S northeast extension on the new Bomboré IV permit to advance the definition of the gap between the P17 and P17S deposits. • Petrographic descriptions from a series of 14 granodiorite core samples were completed in July 2017. • A RC drilling programme of 249 boreholes totalling 13,911 m was completed from July to October 2017 in the Siga South, Siga East, P11 and CFU oxide areas on the mining lease to tighten up the drill spacing to 25 m by 25 m and better define discrete better-than-average gold zones present in these areas. • A metallurgical test work program on 28 composite samples from the oxide, upper transition and lower transition zones was initiated in October 2017 and completed in February 2018 by SGS Lakefield, ON, Canada.
2018	• A small drill programme completed from January to March 2018 on the Bomboré II permit, including 1,070 m of RC drilling and 238 m of core drilling in the Kiin Tanga area to follow up on past RAB and RC scout anomalies. • A drill programme completed from January to March 2018 on the Bomboré III permit, including 387 m of RC drilling and 2,240 m of core drilling, with much of the core drilling to further advance the definition of the gap between the P17 and P17S deposits. • A drill programme completed from January to March 2018 on the Toeyoko permit, including 1,447 m of RC drilling on the P13 prospect and 2,439 m of core drilling, with much of the core drilling to further advance the definition of P17S deposit. • A RC drilling programme of 196 boreholes totalling 12193 m was completed from April to June 2018 in the Siga East, P11 and Maga oxide areas on the mining lease to tighten up the drill spacing to 25 m by 25 m and better define discrete better-than-average gold zones present in these areas. • A core drilling programme of 30 boreholes totalling 3,756 m was completed from April to June 2018 in the P11 and Maga areas on the mining lease to better define discrete better-than-average shallow sulphide gold zones present in these areas. • A small RC drilling programme of 219 m completed in December 2018 on the Bomboré IV permit to investigate the stratigraphic profile along a possible bridge location across the Nobsin River. • A small geotechnical programme consisting of 20 RC boreholes totalling 504 m, 10 core holes totalling 357 m, 8 test pits and two seismic profiles was completed under Knight Piésold supervision in November and December 2018 on the mining lease and on the Toéyoko exploration permit. • A metallurgical test work program on 15 composite samples from the lower transition and sulphide zones was initiated in November 2018 and completed in May 2019 by Base Metals Laboratories Ltd., BC, Canada.

10.0 DRILLING

10.1 Drilling Programs

The following is taken from the Orezone Bomboré Operating Mining permit application submitted to the MEMC in May 2015. Subsequent to that report, Orezone has undertaken several drill programs, between late 2016 and December 2018.

Orezone undertook core, RC, and auger drilling on the Property from 2003 to 2018 that supported the geological model used for the Mineral Resource estimate described in Section 14. Auger sample assay results were not used for the Mineral Resource grade estimation, but auger drill data was used to interpret the geological model. The location of the core and RC borehole collars is shown in Figure 10.1. A summary of Orezone drilling from 2003 to 2018 is presented in Table 10.1. Drilling by previous owners is summarized in Section 6.

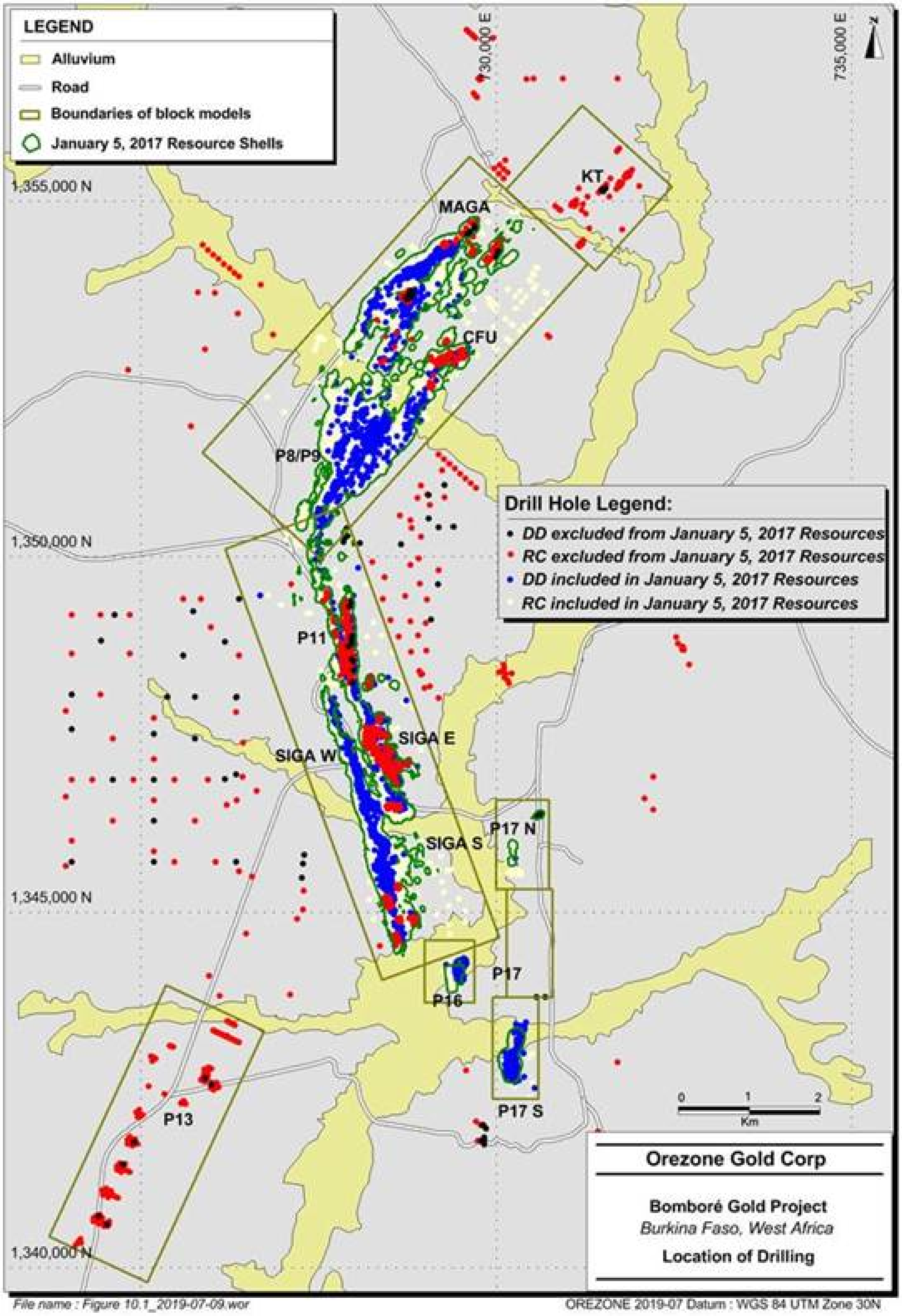
Table 10.1 Summary of Bomboré Project Drilling to December 31, 2018

Company	Years	Drilling Type	Number of Holes Drilled	Length (m)
Channel	1994 to 2000	RC	261	19,500.50
Channel	1997 to 1998	RAB	1,000	34,249.00
Channel	1998	Core	10	1,080.00
Orezone	2003	RC	24	1,387.00
Orezone	2005	RC	217	13,829.00
Orezone	2006	RC	102	7,187.00
Orezone	2007 to 2008	RC	287	21,246.00
Orezone	2007 to 2008	Core	57	5,714.00
Orezone	2009	Core	29	7,738.30
Orezone	2010	Auger	489	3,053.50
Orezone	2010	RC	619	42,456.00
Orezone	2010 to 2013	Core	830	131,091.00
Orezone	2011 to 2013	Auger	3,732	17,004.00
Orezone	2011 to 2013	RC	2,636	152,616.00
Orezone	2012 to 2014	Core	110	11,525.95
Orezone	2012 to 2014	RC	799	44,650.00
Orezone	2016	RC	72	3,162.00
Orezone	2016	Core	27	2,806.00
Orezone	2017	Auger	700	2,558.00
Orezone	2017	RC	293	15,753.00
Orezone	2017	Core	65	10,731.00
Orezone	2018	RC	299	15,820.00
Orezone	2018	Core	74	9,030.00

Company	Years	Drilling Type	Number of Holes Drilled	Length (m)
Sub-total Channel (Included in January 5, 2017 resources)		Auger	0	0
		RAB	745	26,707
		RC	244	17,730
		Core	10	1,080
Sub-total Orezone (Included in January 5, 2017 resources)		Auger	162	1,467
		RAB	0	0
		RC	4,384	268,524
		Core	1,068	167,090
Sub-total Channel (Excluded from January 5, 2017 resources)		Auger	0	0
		RAB	182	6,179
		RC	9	825
		Core	0	0
Sub-total Orezone (excluded from January 5, 2017 resources)		Auger	2,402	11,596
		RAB	0	0
		RC	977	50,095
		Core	104	8,974
Total as of December 31, 2018		Auger	4,921	22,616
		RAB	1,000	34,249
		RC	5,691	340,101
		Core	1,201	179,716

A total of 1,090 (986+104) RC and diamond core holes were excluded from the estimate of Mineral Resources because the data were either located outside the resource area (593 holes totalling 29,134 m) or received after the effective date of the Mineral Resource (497 holes totalling 30,760 m). RPA confirmed that the holes drilled within the resource area confirmed mineralization with similar thicknesses and grades as modelled.

Figure 10.1 Location of Drilling



10.2 Drilling Procedures

10.2.1 Type of Drilling

Orezone chose to drill significantly more RC holes than core holes because of the shallow and weathered nature of the targeted portion of the BSZ. Core drilling was used to define deeper targets within the sulphide zone.

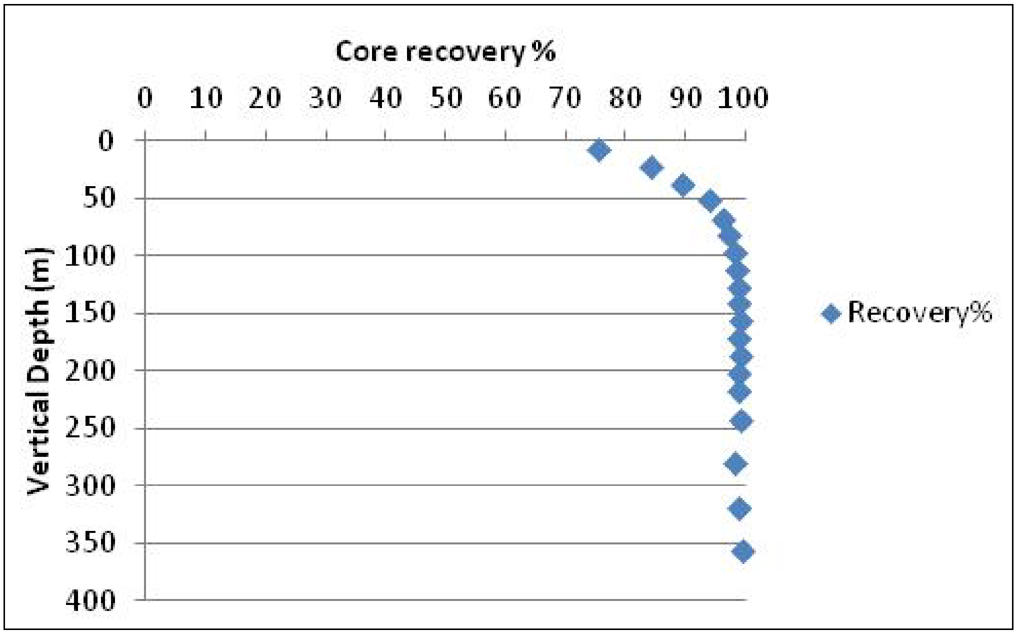
Since 2010, core drilling has been completed by JMS Drilling Inc. (JMS) using up to five Boart Longyear 44 rigs with an HQ core barrel for the weathered zone and an NQ core barrel for the fresh bedrock. Prior to 2012, RC drilling was mostly completed by Boart Longyear using either a CatMax or a DeltaBase RC drilling rig equipped with a 5.25 in. hammer bit. Since February 2012, all drilling has been carried out using Orezone's own Hardab rig operated by JMS until December 2014 and subsequently by Orezone. The Orezone rig was used to complete 46,556 m of drilling from December 2011 to June 2012, 38,055 m from September 2012 to April 2013, 24,703 m in 2014 and 35,096 m from November 2016 to December 2018.

10.2.2 Water Table Elevation and Sample Recovery

The water table is encountered at an average depth of 20 m and is shallower in the southern (Siga) area and deeper in the northern (Maga) area. The RC rigs are equipped with compressors powerful enough to completely flush the borehole between rod additions and during bit advancement. Recovery from the RC boreholes is based on sample weight and has been estimated to average between 83% and 92% in the oxide zone, 91% and 95% in the transition zone, and 84% and 89% in the sulphide zone. Estimates are based on a theoretical volumetric density of 1.83 g/cm^3, 2.35 g/cm^3 and 2.86 g/cm^3, respectively, for each of those weathering zones.

Core recovery, based on detailed geotechnical logs representing approximately 129,000 m of drilling, averages 76% in the top 15 m and increases across the weathering profile to greater than 98% from the top of the sulphide zone (Figure 10.2).

Figure 10.2 Core Recovery by Vertical Depth



10.2.3 Borehole Orientation and Drilling Pattern

The RC and core boreholes were drilled towards the northwest in the northern area (Maga, CFU and P8/P9 deposits), the west in the area of P11, P16, and P17, and the west-southwest in the Siga area. In all areas, the drilling direction is opposed to the dip and orthogonal to the average strike of the lithological units, major fabric, and mineralized envelopes. The plunge of the boreholes at the collar is commonly 50°±5º, intersecting the lithological units, major fabric, and mineralized envelopes at an angle between 65º and 90º.

The oxide resources have been defined along 50 m sections with 25 m between the drill collars. The sulphide resources have been defined along 50 m sections with 50 m between the drill collars. In some areas, such as the Maga North and P8/P9 starter pits, the RC drill collars were drilled on a 25 m by 25 m pattern.

10.2.4 Planning and Borehole Implementation

Drilling programs are planned by the exploration team, under the supervision of the Senior Vice President of Exploration and the Exploration Manager. A handheld GPS with a precision of ±5 m is used by a technician to locate and prepare drilling pads. The borehole collars are spotted in the field and pegged using a Differential Global Positioning System (DGPS) that is set to achieve a sub-metre accuracy.

Once drilled, the casing is first surveyed using both DGPS then a Total Station instrument coupled with DGPS accessing the national network of CORS DGPS stations now installed in Burkina Faso. This system provides accurate coordinates over the entire Project area. The DGPS accuracy is validated on a known control station at the beginning and end of each work shift.

Upon completion, a 3 m PVC pipe is inserted in RC holes and a 6 m PVC pipe is inserted in core holes. The top of the pipe is capped by a concrete beacon on which the Hole-ID, the final depth, and the date of completion are recorded.

10.2.5 Borehole Trajectory

An Orezone crew conducts downhole deviation surveys in open RC boreholes after drilling is completed, rods have been pulled and the rig moved, but before the PVC casing is capped. Readings are taken at 25 m increments starting at six metres below the collar. The reading at a depth of six metres is used to control the quality of the drill collar alignment.

Readings in core boreholes are taken once or twice a day with the instrument positioned six metres ahead of the drill string to avoid magnetic interference. If the distance between successive tests exceeds 30 m, rods are removed to take additional readings and to maintain on average 25 m between successive readings.

The path of the Orezone boreholes was surveyed using a Reflex Instrument that measures several parameters, including the plunge of the borehole and the three components of the magnetic field. It relies on a compass to read the azimuth. The azimuth angles are validated against the measured intensity of magnetic field, and an accelerometer reading to ensure the compass was stable when measurements were taken. The magnetic azimuth is converted to a geographic azimuth using the declination applicable at the time of the survey.

The borehole path of RC boreholes at the Property typically steepens up with depth, contrary to core holes that have the opposite behaviour, but both types of boreholes deviate to the right of the collar azimuth. Borehole deviation is not a critical issue because more than 75% of the boreholes are shorter than 70 m and 93% of the boreholes do not exceed a depth of 90 m. The Reflex Instrument occasionally produces incorrect results. Spurious readings can be filtered out and the deviation path can be interpolated. In RPA’s opinion, the survey method applied by Orezone conforms to industry best practice.

10.2.6 Description of RC Cuttings

RC holes are sampled at 1 m intervals by collecting 100% of the material reporting to the cyclone. Small samples of screened and washed chips from each 1 m run are saved in labelled plastic boxes (chip boxes). A log of RC chips is done at the drill site to monitor the drill advance and extend the borehole if necessary as the Orezone objective is to reach the top of the sulphide zone. Before a hole is stopped, although the large majority of RC holes reach a minimal depth of 50 m even if the top of the sulphide zone is significantly shallower.

The bags containing the RC chips are transported by truck from the drill site to Orezone's storage area at the Property where they are weighed to estimate recovery, and magnetic susceptibility is measured. The description of the RC cuttings is recorded as logs in Microsoft Excel spreadsheets.

Significant gold assay results are added on the chip box cover once they become available. Chip boxes have all been photographed (Figure 10.3) to facilitate the validation of the description during the 3D geological modelling.

The RC material remaining after sampling is saved in plastic bags at the Property storage area (Figure 10.4). Once the QA/QC check-assaying program is completed, barren sample rejects from multi-metre zones are discarded but all samples from mineralized or geochemically anomalous (> 0.1 g/t Au) zones are kept. Some sample bags damaged by UV light despite of the protective sheet are also discarded.

Figure 10.3 Chip Box Photograph

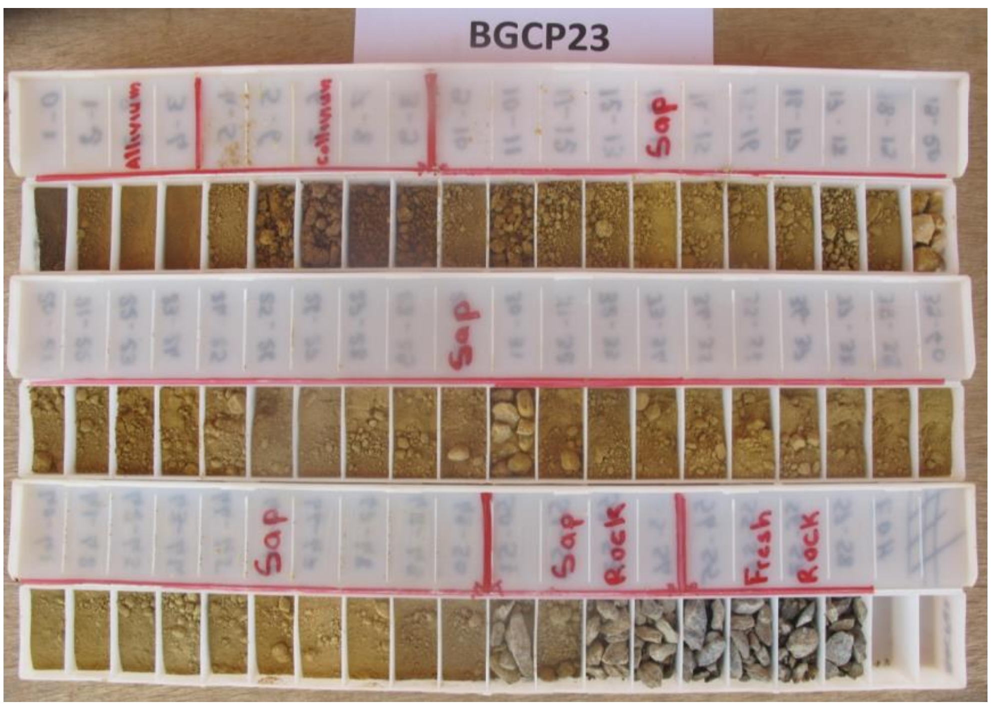


Figure 10.4 RC Sample Storage at the Bomboré Main Camp

Orezone uses a well-designed procedure for logging the RC samples and the subsequent integration of this information into the exploration database. All field measurements, geological logging (lithological, structural, mineralization, and alteration features) and sampling parameters of the RC boreholes are captured directly in fixed forms with menus on a Microsoft Excel platform loaded in handheld computers. Logs are checked daily by the project geologist for completeness and accuracy. The validation of the field descriptions and measurements involves the samplers, technicians, junior geologists, and the project geologist before the data are sent to Ouagadougou for further validation by the GIS and database team responsible for importing the data in Datashed, and the senior geologist who is responsible for the 3D geological modelling. The GIS team and the senior geologist use MapInfo and GEMS to validate the geology of each hole and model the geology of each deposit. Over the years, chips from previous RC boreholes drilled by Orezone have been re-logged to record additional standardized information and ascertain consistency between the descriptions of a large number of geologists.

10.2.7 Description of drill core

Orezone uses procedure for drill core similar to the RC program, but with the addition of measurements and descriptions specific to drill core, i.e., density measurements, a geotechnical description (Rock Quality Designation (RQD), joint/fracture analyses, material type and rock strength), and the goniometric measurements of structural elements. Core boxes are photographed before and after sampling (Figure 10.5).

Figure 10.5 Core Boxes Photographs, Before and After Sampling

10.2.8 RPA Comments

In the opinion of RPA, the drilling procedures employed by Orezone conform to industry best practice and the resultant drilling pattern is sufficient to interpret the geometry and the boundaries of the gold mineralization with confidence. Qualified personnel under the direct supervision of appropriately qualified geologists conducted all drilling sampling.

RPA is not aware of any drilling, sampling, or recovery factors that could materially affect the Mineral Resource estimate.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following is generally summarized from the Orezone Bomboré Operating Mining permit application submitted to the MEMC in May 2015; and updated to include the 2017and 2018 drill programs.

11.1 Sampling and Analysis by Orezone 2003 to 2015

11.1.1 2003 RC

For the 2003 RC drilling program, samples were collected at the rig cyclone at 1 m intervals. Each sample was logged, then split with a riffle splitter and recombined in a two-metre composite sample that was submitted to Abilab Afrique de l'Ouest s.a.r.l. from Bamako, Mali, to analyse gold by the fire assay (FA) method. The series of 1,115 samples that were submitted included approximately 1% of duplicate and triplicate samples and two certified reference material (CRM) samples.

11.1.2 2005 to 2007 RC Programs

The protocols used by Orezone for the sampling and the QA/QC, and the protocols used by the sample preparation laboratory and analytical laboratories for the RC drilling programs completed in 2005, 2006, and 2007 are summarized in Table 11.1. Most sample preparation was completed at Abilab-Ouagadougou, SGS-Tarkwa, Ghana, and SGS-Siguiri, Guinea; and a minor amount at SGS-Ouagadougou.

Table 11.1 Summary of the Sampling Protocol from 2005 to 2007

PROTOCOLS
SAMPLING AND SAMPLE PREPARATION
• 100% of the material reporting to the RC rig cyclone is collected at 1 m intervals during the drilling advance in a 500 mm by 800 mm polypropylene bag.
• Each bag is identified with a black marker by the Hole-ID and the depth interval of the sample and is assigned a unique sample number from a multi-stubs sample book.
• No material other than the blow-back material is abandoned at the drilling site.
• The samples are transported immediately to the Orezone camp where the sample splitting and detail logging take place.
• A four to five-kilogram fraction is collected from each one metre sample using a riffle splitter, after sun-baking the rare damp or wet samples. The riffle splitter is cleaned with a brush or a rag after each sample.
• The rest of the sample is kept under an open shed, in heaps covered by a tarp or black plastic sheeting, over a compacted laterite base that is protecting the heap of bags from the run-off waters.
• The five-kilogram samples are packaged in lots of eight samples in 100 kg rice bags and sent in large lots to the QPS sample preparation facility in Ouagadougou. Each shipment includes specific instructions to the preparation laboratory regarding the insertion of QA/QC samples (see QA/QC below).
• At QPS, each sample is entirely dried, pulverized with a Keegor mill and divided in two halves of about two kilograms with a riffle splitter.
• The samples are then packaged in lots of 20 in 100 kg rice bags and sent in large lots to the analytical laboratory.

PROTOCOLS
QA/QC • A second riffle splitter fraction of 5 kg is collected by Orezone and identified as a field sample duplicate (code FD) in Orezone sampling log. • FD samples are collected according to a predetermined random list and represent 2% of the stream of samples submitted to the analytical laboratory. The FD follows immediately the five-kilogram sample from the same parent sample and is blind to the preparation laboratory. • A duplicate sample of the pulverized material is collected at the preparation laboratory and identified as a pulp duplicate (code PD) in the sampling log; an empty bag with its sample number stub was inserted by Orezone in the stream of samples submitted to the preparation laboratory, and the list of PD samples to be collected by QPS is submitted by Orezone to QPS for each lot of samples. • PD samples are collected according to a predetermined random list and represent 4% of the stream of samples submitted to the analytical laboratory. The PD sample follows immediately the two kg sample from the same parent sample and is blind to the preparation laboratory. • Non-certified reference material samples are inserted by QPS according to instructions submitted by Orezone to QPS for each lot of samples; an empty bag with its sample number stub was inserted by Orezone in the stream of samples submitted to the preparation laboratory. • Non-certified reference material samples are inserted according to a predetermined random list and represent 4% of the stream of samples submitted to the analytical laboratory, i.e., 2% of blank material and 2% of mineralized material. • The reference material is made of bulk samples of several hundred kilograms collected on Orezone exploration projects: the gold grade thus varies from batch to batch and the reference material can therefore be used to detect sample mix-up or cross-contamination issues but cannot be used to monitor the accuracy of the analytical results. Each bulk sample was submitted separately to QPS, totally dried, pulverized, and divided in two-kilogram fractions for subsequent insertion in the stream of prepared samples, according to Orezone instructions. The reference material to be inserted is selected by Orezone to blend (colour-wise) into the sequence as inconspicuously as possible for the analytical laboratory.
ANALYTICAL WORK • In 2005, the gold content of each two-kilogram samples was analysed by bottle-roll cyanidation (BLEG) after 24 hours of rolling. • If the soluble gold grade is equal to or greater than 0.5 g/t, the leach residues are neutralized, dried and prepared for a gold analysis by fire assay on a 50 g aliquot and atomic absorption spectrometry finish (AAS) finish; the leach residue assay is reported together with the bottle roll assays by some laboratories or separately by others. • In 2006 and 2007, the gold content of each two-kilogram samples was analysed by bottle-roll cyanidation (LeachWELL) after 12 hours of rolling. • If the soluble gold grade is equal to or greater than 0.5 g/t, the leach residues are neutralized, dried and prepared for a gold analysis by fire assay on a 50 g aliquot and AAS finish; the leach residue assays are reported together with the bottle roll assays; the leach residue assay is reported together with the bottle roll assays by some laboratories or separately by others.

11.1.3 2008 Drilling Program

The sampling, analytical, and QA/QC protocols used during the 2008 RC drilling program are summarized in Table 11.2.

Table 11.2 Summary Sampling, Analytical and QA/QC Protocols Used During the 2008 RC Drilling Program

PROTOCOLS
SAMPLING AND SAMPLE PREPARATION

SAMPLING AND SAMPLE PREPARATION

- 100% of the material reporting to the RC rig cyclone is collected at 1 m intervals during the drilling advance in a 500 mm by 800 mm polypropylene bag.
- Each bag is identified with a black marker by the Hole-ID and the depth interval of the sample and is assigned a unique sample number from a multi-stubs sample book.
- No material other than the blow-back material is abandoned at the drilling site.
- The samples are transported immediately to the Orezone camp where the sample splitting and detail logging take place.
- A 2.5 kg fraction is collected from each one metre sample using a riffle splitter in several stages, after sun-baking the rare damp or wet samples. The riffle splitter is cleaned with a brush or a rag after each sample.
- The rest of the sample is kept under an open shed, in heaps covered by a tarp or black plastic sheeting, over a compacted laterite base that is protecting the heap of bags from the run-off waters.
- The 2.5 kg samples are packaged in lots of 15 samples in 100 kg rice bags and sent in large lots to the Orezone Kossodo facility in Ouagadougou. Each shipment includes specific instructions regarding the insertion of QA/QC samples (see QA/QC below).
- The Orezone Kossodo team inserts QA/QC samples in the stream of field samples that are then submitted in lots of about 200 samples to a commercial sample preparation laboratory in Ouagadougou.
- At the commercial sample preparation laboratory, each sample is entirely dried, pulverized with a Keegor mill, and then returned to the Orezone Kossodo team.

QA/QC

- At Bomboré, a second riffle splitter fraction of 2.5 kg is collected by Orezone and identified as a field sample duplicate (code FD) in Orezone sampling log.
- FD samples are collected according to a predetermined random list and represent 2% of the stream of samples submitted to the analytical laboratory. The FD follows immediately the 2.5 kg sample from the same parent sample and is blind to the preparation laboratory.
- The Orezone Kossodo team receives the field sample delivery, validates the list of samples, monitors their respective weight and inserts reference material samples (blank or mineralized) according to the instructions received from the Bomboré team and based on the Orezone predetermined random list.
- Reference material samples represent 4% of the stream of samples submitted to the analytical laboratory, i.e., 2% blanks and 2% standards.
- The Orezone Kossodo team retrieves the batch of samples prepared by the commercial sample preparation laboratory, validates the list of samples, monitors their respective weight and produces a one-kilogram split sample with a riffle splitter.
- Riffle splitter pulp duplicate samples (code PD) are collected according to a predetermined random list and represent 4% of the stream of samples submitted to the analytical laboratory. The PD follows immediately the 2.5 kg sample from the same parent sample and is blind to the preparation laboratory.
- Riffle splitter pulp duplicate samples (code LAPD) are collected so that the analytical laboratory has 10% of duplicate samples for internal QA/QC purpose.
- The one-kilogram samples are packaged in lots of 20 samples in 100 kg rice bags by the Orezone Kossodo team and then submitted in lots of about 200 samples to a commercial analytical laboratory in Ouagadougou.
- Orezone is recording the weight of the field sample, the weight of each fraction generated by riffle splitting before the

PROTOCOLS
sample preparation, the weight recorded by the sample preparation laboratory, the weight of each fraction generated by riffle splitting after the sample preparation and the weight recorded by the analytical laboratory.
• These weights are compared so as to detect (i) any losses during the transfer from the Bomboré project to Kossodo, or between Kossodo and the sample preparation laboratory or the analytical laboratory, (ii) excessive losses at the sample preparation stage, (iii) sample mix-up at any stage, or (iv) in complete use of the sample submitted by the analytical laboratory.
ANALYTICAL WORK
• From October 2007, one-kilogram samples have been analyzed for gold by bottle-roll LeachWELL cyanidation for ten hours.
• If the leachable gold grade is greater than or equal to 0.5 g/t for any given sample, the commercial analytical laboratory is instructed to neutralize, dry, and pulverize this sample with a LM2 shatter box before assaying for gold a 50 g split by FA with an AAS finish.

11.1.4 2010 to Present RC Drilling Programs

The sampling, analytical, and QA/QC protocols used since 2010 on RC drilling programs are summarized in Table 11.3.

Table 11.3 Summary Sampling, Analytical and QA/QC Protocols Used in RC Drilling Programs Since 2010

PROTOCOLS
SAMPLING AND SAMPLE PREPARATION
• 100% of the material reporting to the RC rig cyclone is collected at 1 m intervals during the drilling advance in a 500 mm by 800 mm polypropylene bag.
• Each bag is identified with a black marker by the Hole-ID and the depth interval of the sample and is assigned a unique sample number from a multi-stubs sample book.
• No material other than the blow-back material is abandoned at the drilling site.
• The samples are transported immediately to the Orezone sampling facility where the sample splitting and detail logging take place.
• A ±2.1 kg fraction is collected from each one metre sample using RSD in at most two stages, after sun-baking the rare damp or wet samples. The RSD is cleaned with a brush or a rag and with compressed air after each sample.
• The rest of the sample is kept under an open shed, in heaps covered by a tarp or black plastic sheeting, over a compacted laterite base that is protecting the heap of bags from the run-off waters.
• The 2.1 kg samples are packaged in lots of 15 samples in 100 kg rice bags and sent in large lots to the Orezone Kossodo facility in Ouagadougou. Each shipment includes specific instructions regarding the insertion of QA/QC samples (see QA/QC below).
• The Orezone Kossodo team inserts QA/QC samples in the stream of field samples that are then submitted in lots of approximately 200 samples to a commercial sample preparation laboratory in Ouagadougou.
• During the period from June 2012 to July 2014, most of the samples were submitted to the Bomboré sample preparation laboratory operated by SGS and in this case the QA/QC samples were inserted at Bomboré by the Orezone Bomboré team.
• At the commercial sample preparation laboratory, each sample is entirely dried, pulverized with a Keegor mill, and then

PROTOCOLS
returned to the Orezone Kossodo team. • The samples prepared at the Bomboré sample preparation laboratory were pulverized with a LM2 shatter box.
QA/QC
• At Bomboré, a second RSD fraction of ±2.1 kg is collected by Orezone and identified as a field sample duplicate (code FD) in the Orezone sampling log. • FD samples are collected according to a predetermined random list and represent 2% of the stream of samples submitted to the analytical laboratory. The FD follows immediately the ±2.1 kg sample from the same parent sample and is blind to the preparation laboratory. • The Orezone team receives the field sample delivery, validates the list of samples, monitors their respective weight, and inserts reference material samples (blank or mineralized) according to the instructions received from the Bomboré team and based on Orezone predetermined random list. • Reference material samples represent 4% of the stream of samples submitted to the analytical laboratory, i.e., 2% blanks and 2% standards. • The Orezone Kossodo team retrieves the batch of samples prepared by the commercial sample preparation laboratory, validates the list of samples, monitors their respective weight and produces a 1 kg split sample with a riffle splitter. • RSD duplicate pulp samples (code PD) are collected according to a predetermined random list and represent 4% of the stream of samples submitted to the analytical laboratory. The PD follows immediately the 2.5 kg sample from the same parent sample and is blind to the preparation laboratory. • Riffle splitter pulp duplicate samples (code LAPD) are collected so that the analytical laboratory has 10% of duplicate samples for internal QA/QC purpose. • The one-kilogram samples are packaged in lots of 20 samples in 100 kg rice bags by the Orezone Kossodo team and then submitted in lots of about 200 samples to a commercial analytical laboratory in Ouagadougou. • Orezone is recording the weight of the field sample, the weight of each fraction generated by RSD splitting before the sample preparation, the weight recorded by the sample preparation laboratory, the weight of each fraction generated by RSD after the sample preparation, and the weight recorded by the analytical laboratory. • These weights are compared so as to detect (i) any losses during the transfer from the Bomboré project to Kossodo, or between Kossodo and the sample preparation laboratory or the analytical laboratory, (ii) excessive losses at the sample preparation stage, (iii) sample mix-up at any stage, or (iv) the incomplete use of the sample submitted by the analytical laboratory. • From 2011, in addition to the sieve tests completed and reported by the sample preparation laboratory, Orezone conducted independent sieve tests on about 5% of the prepared samples.
ANALYTICAL WORK
• One-kilogram samples have been analyzed for gold by bottle-roll LeachWELL cyanidation for ten hours. • If the leachable gold grade is greater than or equal to 0.2 g/t for any given sample, the commercial analytical laboratory is instructed to neutralize, dry, and pulverize this sample with a LM2 shatter box before assaying for gold a 50g split by FA with an AAS finish.

August 2010 to March 2013

From August 2010 to March 2013, Orezone used several commercial laboratories for sample preparation, analysis, and QA/QC, including from May 2012 a preparation facility built at Bomboré and operated by SGS. Summary statistics of the sample preparation activities by laboratory and type of samples are presented in Table 11.4.

Table 11.4 Summary of Sample Preparation Activities by Laboratory and Type of Sample for the Period from August 1, 2010 to March 15, 2013

Sample Type	ALS	SO	BIGS	ACT	SGS-B	Total
Primary Samples						
Rock (outcrop)		396		174	19	589
Core (definition)	11,962	46,767	33,660	6,993	28,260	127,642
Core (geotechnical)					123	123
RC (geotechnical)					1,856	1,856
Trench (geotechnical)					151	151
Metallurgy					172	172
Auger	3,458	2,074	2,065			7,597
RC (definition)	39,635	60,446	27,039	1,507	51,211	179,838
Check Samples						
Core (definition)					60	60
Auger					16	16
RC (definition)	717	4,946	109	124	521	6,417
Bottle-roll Cyanidation Leach Residue Samples						
Core (definition)			33,578			33,578
Core (geotechnical)			1			1
RC (geotechnical)			43			43
Metallurgy			2,530			2,530
RC (definition)			54,236			54,236
Core (definition check samples)			298			298
RC (definition check samples)			3,906			3,906
Umpire Samples						
Core (definition)						0
RC (definition)	353					353
RC (definition ≥ 5 g/t Au)					34	34
Total	**56,125**	**114,629**	**157,465**	**8,798**	**82,423**	**419,440**

ALS ABILAB Burkina s.a.r.l., a subsidiary company of the ALS Group in Ouagadougou
SO SGS Burkina Faso SA, a subsidiary company of the SGS Group in Ouagadougou
BIGS BIGS Global Burkina s.a.r.l. in Ouagadougou
ACT ACTLABS Burkina Faso s.a.r.l., a subsidiary company of the ACTLABS Group in Ouagadougou
SGS-B SGS Burkina Faso SA, a subsidiary company of the SGS Group operating the Orezone Bomboré sample preparation facility

Summary statistics of analytical services by laboratory for the period 2010 to 2013 are presented in Table 11.5.

Table 11.5 Summary of Analytical Activities by Laboratory and Type of Sample for the Period from August 1, 2010 to March 15, 2013

Sample Type	LeachWELL 1kg		FA-AAS 50g			FA-GRAV 50g
	SGS	BIGS	ALS	SGS	Total	ALS
Primary Samples						
Rock (outcrop)		611			0	
Core (definition)		120,897	14,180		14,180	
Core (geotechnical)		132			0	
RC (geotechnical)		1,894			0	
Trench (geotechnical)		158			0	
Metallurgy (LW on FA ≥0.4 g/t Au)		2,492			0	
Metallurgy (scrubber testwork)		124		144	144	
Auger		7,834			0	
RC (definition)		185,361			0	
Check Samples						
Core (definition)		741	821		821	
Auger		16			0	
RC (definition)		7 085			0	
Bottle-roll Cyanidation Leach Residue Samples (Primary Samples)						
Core (definition)			5,012	28,498	33,510	
Core (geotechnical)				4	4	
RC (geotechnical)				22	22	
Metallurgy			958	1,803	2,761	
RC (definition)			21,779	36,645	58,424	
RC (definition) – 2010 Program			1,030	224	1,254	
Bottle-roll Cyanidation Leach Residue Samples (Check Samples)						
Core (definition)			12	264	276	
RC (definition)			3,394	773	4,167	
Check Assays Leach Residue Samples						
Metallurgy (LW on FA ≥0.4 g/t Au)				4	4	
Core (definition)			112	1,522	1,634	
RC (definition)			117	1,779	1,896	
RC (definition) – Check Assays				34	34	
Umpire Assays						
Core (definition)	2,100			331	331	
RC (definition)	4,763				0	
Core (definition ≥ 5 g/t Au)					0	475
RC (definition ≥ 5 g/t Au)					0	744
Total	**6,863**	**327,345**	**47,415**	**72,047**	**119,462**	**1,219**

LeachWELL 1kg *Bottle-roll cyanidation with LeachWELL on a one-kilogram sample over ten hours*

FA-AAS 50g	*Fire assay and atomic absorption spectrophotometry.*
FA-GRAV 50g	*Fire assay and gravimetric finish.*
ALS	*ABILAB Burkina s.a.r.l., a subsidiary company of the ALS Group in Ouagadougou.*
SO	*SGS Burkina Faso SA, a subsidiary company of the SGS Group in Ouagadougou.*
BIGS	*Global Burkina s.a.r.l. in Ouagadougou.*

March 2013 to January 2015

From March 2013 to January 2015, Orezone used several commercial laboratories for the sample preparation, analysis, and QA/QC, including a preparation facility built at Bomboré and operated by SGS. Summary statistics of the sample preparation activities by laboratory and type of samples are presented in Table 11.6.

Table 11.6 Summary of Sample Preparation Activities by Laboratory and Type of Sample for the Period from March 16, 2013 to January 27, 2015

Sample Type	**SO**	**BIGS**	**SGS-B**	**Total**
Drill Core	56		1,071	1,127
Geotech Core	29			29
Geotech RC	3,559			3,559
Geotech Pits	296			296
RC	6		27,158	27,164
Geotech RC Check Assays	46			46
RC Check Assays	77		284	361
Core Leach Residue		279		279
Geotech DD Leach Residue		2		2
Geotech RC Leach Residue		521		521
Geotech Pits Leach Residue		29		29
RC Leach Residue		8,237		8,237
Leach Residue Check Assays		226		226
Core Umpire	30			30
RC Umpire	380			380
RC Umpire FA GRAV			76	76
Total	**4,479**	**9,294**	**28,589**	**42,362**

Summary statistics of analytical services by laboratory for the period 2013 to 2015 are presented in Table 11.7.

Table 11.7 Summary of Analytical Activities by Laboratory, Type of Analytical Services and Type of Sample for the Period from March 16, 2013 to January 27, 2015

Sample Type	LeachWELL 1kg		FA-AAS 50g		FA-GRAV 50g	
	SGS	**BIGS**	**ALS**	**SGS**	**ALS**	**SGS**
Core	2	1,185				2
Geotech Core		30				
Geotech RC		3,670				
Geotech Pits		312				
RC	12	27,993				
Core Check Assays		57				
RC Check Assays		368			0	
Geotech RC Check Assays		48				
Geotech Pits Check Assays		12				
Core Leach Residue				371		
Geotech DD Leach Residue				3		
Geotech RC Leach Residue				494		
Geotech Pits Leach Residue				34		
RC Leach Residue				8,677		
Core Leach Residue Check Assay				28		
RC Leach Residue Check Assay				116		
Geotech RC Leach Residue Check Assay				8		
Core Umpire	221					
RC Umpire	1,002					
Geotech RC Umpire	114					
Core Umpire FA GRAV					56	20
RC Umpire FA GRAV					106	80
Geotech RC Umpire FA GRAV					4	6
Core Leach Residue Umpire Assay			35			
Core Leach Residue Umpire Assay			419			
Total	**1,351**	**33,675**	**454**	**9,733**	**166**	**106**

11.1.5 Certifications and Independence

The accreditations of each of the laboratories used on the Property are listed below:

- ABILAB Burkina s.a.r.l., a subsidiary company of the ALS Group, is not accredited, however its hub laboratory located in South Africa is accredited with ISO/IEC 17025:2005.
- SGS Burkina Faso SA, a subsidiary company of the SGS Group, is accredited in accordance to ISO/IEC 17025:2005 (announced July 15, 2015).

- BIGS Global Burkina s.a.r.l. is not accredited.

- ACTLABS Burkina Faso s.a.r.l., a subsidiary company of the ACTLABS Group, is accredited in accordance to ISO 9001:2008 (original issue date February 24, 2014).

Orezone and RPA are independent of ACT, ALS, BIGS, SGS-B, and SO.

11.2 Specific Gravity Data

The specific gravity database, used to estimate Mineral Resources, includes 91,499 records generated by Orezone from measurements on core from 1,073 boreholes. Measurements were conducted on site using the water displacement method. Generally, a single piece of core, 10 cm to 15 cm in length, is selected and measured in each core box prior to core splitting for assaying. Wax coating or a plastic film wrap were applied whenever necessary. Specific gravity data were subsequently classified by rock and material type.

Since samples are not fully dried prior to the water displacement test, water present in core samples could potentially overestimate the specific gravity measurement, especially in the oxide and transition zones. To estimate this moisture content, Orezone weighed samples at the sample preparation laboratory before and after drying. The average loss of moisture in the oxide core samples is 5.7%, in the transition core samples 2.8%, and in the fresh core samples 0.2%. As such Orezone applied a reduction factor of 5.5% and 2.5% to individual samples in the oxide and transition zones, respectively, for the block model density calculations.

Orezone has observed that the specific gravity increases with depth through the weathering profile and is then fairly homogeneous within the fresh zone for a given lithology. During the resource estimation process, specific gravity values were interpolated into the block model by Orezone using an OK estimator to generate a realistic model of the tonnage within the Bomboré deposits.

Specific gravity data by lithology and material type, excluding the new P17 core drilling data, is presented in Table 11.8.

Table 11.8 Specific Gravity Data by Lithology and Material Type

Lithology	Specific Gravity				
	Count	Average	SD	Min	Max
Upper Oxide Zone (Ox_U)					
Regolith	655	1.88	0.18	1.29	3.00
Late granitic intrusives	57	1.72	0.20	1.38	2.22
Granodiorite Zr-rich	99	1.81	0.16	1.43	2.22
Porphyritic Granodiorite	1,383	1.75	0.18	1.28	2.85
Dolerite	1	2.17		2.17	2.17
Meta-gabbro (type II) Cr-rich	139	1.78	0.19	1.33	2.77
Gabbro-Diorite	5	1.79	0.08	1.67	1.87
Meta-gabbro (type I) Cr-poor	1,135	1.82	0.18	1.35	3.04
Meta-peridotite	253	1.74	0.17	1.39	2.49
Meta-conglomerate	250	1.87	0.18	1.42	2.52
Meta-sandstone	1,937	1.79	0.19	1.33	2.91
Meta-argillite	917	1.77	0.15	1.26	2.56
Lower Oxide Zone (Ox_L)					
Late granitic intrusives	41	1.83	0.17	1.54	2.41
Granodiorite Zr-rich	91	1.89	0.21	1.37	2.36
Porphyritic Granodiorite	1,356	1.87	0.20	1.30	2.57
Dolerite	2	1.83	0.01	1.83	1.83
Meta-gabbro (type II) Cr-rich	176	1.97	0.20	1.60	2.70
Gabbro-Diorite	17	1.93	0.15	1.59	2.25
Meta-gabbro (type I) Cr-poor	1,279	1.94	0.18	1.28	2.81
Meta-peridotite	309	1.83	0.16	1.46	2.37
Meta-conglomerate	244	1.98	0.20	1.43	2.75
Meta-sandstone	1,828	1.92	0.21	1.26	2.95
Meta-argillite	1,005	1.92	0.16	1.49	2.76
Upper Transition Zone (TR_U)					
Late granitic intrusives	43	2.41	0.20	1.59	2.76
Granodiorite Zr-rich	87	2.34	0.20	1.79	2.66
Porphyritic Granodiorite	1,170	2.29	0.16	1.51	2.69
Meta-gabbro (type II) Cr-rich	178	2.42	0.24	1.93	3.08
Gabbro-Diorite	24	2.30	0.25	1.78	2.68
Meta-gabbro (type I) Cr-poor	932	2.32	0.20	1.68	2.85
Meta-peridotite	117	2.24	0.26	1.81	2.90
Meta-conglomerate	273	2.39	0.19	1.99	2.80
Meta-sandstone	1,261	2.30	0.15	1.66	2.77
Meta-argillite	643	2.27	0.13	1.83	2.65
Lower Transition Zone (TR_L)					

Lithology	Specific Gravity				
	Count	Average	SD	Min	Max
Late granitic intrusives	42	2.43	0.30	1.57	2.72
Granodiorite Zr-rich	126	2.47	0.18	1.87	2.76
Porphyritic Granodiorite	1,149	2.41	0.16	1.63	2.83
Meta-gabbro (type II) Cr-rich	129	2.50	0.22	1.82	2.96
Gabbro-Diorite	39	2.40	0.27	1.83	2.83
Meta-gabbro (type I) Cr-poor	962	2.46	0.20	1.74	3.02
Meta-peridotite	126	2.24	0.28	1.55	2.94
Meta-conglomerate	213	2.47	0.16	1.96	2.86
Meta-sandstone	1,454	2.44	0.17	1.66	3.11
Meta-argillite	685	2.45	0.14	1.67	2.90
Upper Sulphide Zone (Fr_U)					
Late granitic intrusives	176	2.69	0.06	2.44	2.91
Granodiorite Zr-rich	766	2.76	0.06	2.30	3.15
Porphyritic Granodiorite	4,036	2.74	0.07	2.16	3.22
Dolerite	2	3.02	0.17	2.90	3.15
Meta-granite	6	2.68	0.02	2.66	2.70
Meta-gabbro (type II) Cr-rich	784	2.97	0.11	2.26	3.69
Gabbro-Diorite	579	2.95	0.15	2.17	3.25
Meta-gabbro (type I) Cr-poor	4,349	2.93	0.13	2.23	3.84
Meta-peridotite	401	2.80	0.13	1.84	3.06
Meta-conglomerate	1,231	2.81	0.10	2.28	3.11
Meta-sandstone	4,032	2.76	0.07	1.98	3.32
Undifferentiated sediments (P17S)	2	2.99	0.11	2.91	3.06
Meta-argillite	1,366	2.72	0.06	2.38	2.99
Lower Sulphide Zone (Fr_L)					
Late granitic intrusives	727	2.71	0.07	2.48	3.17
Granodiorite Zr-rich	2,182	2.77	0.04	2.50	3.16
Porphyritic Granodiorite	8,563	2.76	0.06	1.94	3.25
Dolerite	6	2.95	0.08	2.88	3.11
Meta-granite	15	2.71	0.05	2.66	2.83
Meta-gabbro (type II) Cr-rich	1,242	2.97	0.10	2.63	3.27
Gabbro-Diorite	1,049	2.91	0.12	2.61	3.32
Meta-gabbro (type I) Cr-poor	11,906	2.94	0.12	2.50	3.47
Meta-peridotite	1,231	2.82	0.08	2.39	3.20
Meta-conglomerate	2,489	2.82	0.07	2.60	3.34
Meta-sandstone	11,509	2.78	0.06	2.34	3.43
Undifferentiated sediments (P17S)	143	2.89	0.10	2.71	3.19

11.3 Quality Assurance and Quality Control

Quality control measures are typically set in place to ensure the reliability and trustworthiness of exploration data. These measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management, and database integrity. Appropriate documentation of quality control measures and regular analysis of quality control data are important as a safeguard for project data and forms the basis for the quality assurance program implemented during exploration.

Analytical control measures typically involve internal and external laboratory control measures implemented to monitor the precision and accuracy of the sampling, sample preparation and assaying. They are also important to prevent sample mix-up and to monitor the voluntary or inadvertent contamination of samples.

Assaying protocols typically involve regularly duplicating and replicating assays and inserting quality control samples to monitor the reliability of assaying results throughout the sampling and assaying process. Check assaying is normally performed as an additional test of the reliability of assaying results; it generally involves re-assaying a set number of sample pulps at a secondary umpire laboratory.

Detailed reviews of analytical quality control measures implemented since December 2012 (and March 2013 for the P16 deposit) are provided in (Maes, October 2014) and (Maes, February 2015). Detailed reviews of analytical quality control measures implemented prior to December 2012 (and prior to March 2013 for the P16 deposit) are provided by Met-Chem (Buro and Saucier, 2008) and SRK (Cole and El-Rassi, 2008; Cole and El-Rassi, 2010; Cole et al., 2012; Gourde, 2014; Defilippi et al., 2015).

Historical sampling, analytical, and QA/QC protocols used on the Property are summarized in Tables 11.9 and 11.10.

Table 11.9 Summary of the Sampling, Analytical and QA/QC Protocols Used on the RC Programs Since 1994

Period	Program (m)	Division	QA/QC Samples					Analyses (Method/Weight)
			FD	LAPD	PD	BLK	STD	
1994 to 2000	19,501	Riffle?	No	No	No	No	No	FA 30g
2003	1,387	Riffle	1%	0%	0%	No	0,2%	FA 50g
2005	13,829	Riffle	2%	8%	4%	2% NC	2% NC	BLEG 2kg
2006 to 2007	8,770	Riffle	2%	9%	4%	2% NC	2% NC	LW 2kg
2008	19,663	Riffle	2%	10%	4%	2% IHC	2% IHC	LW 1kg
2010	42,456	RSD	4%	10%	4%	3% IHC	4% IHC	LW 1kg
2011 to 2014	197,266	RSD	2%	5%	3%	2% IHC	3% IHC	LW 1kg
2016 to 2018	21,819	RSD	2%	5%	3%	2% IHC	3% IHC	LW 1kg

RSD: Rotary Sample Divider
FD: Field Duplicate, blind to the preparation laboratory
PD: Pulp Duplicate, blind to the analytical laboratory
LAPD: Lab-Aware Pulp Duplicate, known to the analytical laboratory
BLK: Blank, blind to the preparation laboratory
NC: Non-certified reference material
IHC: In-house referenced material
FA: Fire Assay, with AAS finish
BLEG: Bulk Leach Extractable Gold
LW: LeachWELL

STD: Standard; blind to the preparation laboratory

In all instances, the percentage is based on the total stream of samples submitted to the analytical laboratory, i.e., the primary samples plus the FD, PD, BLK, and STD, but excluding the LAPD.

Table 11.10 Summary of the Sampling, Analytical and QA/QC Protocols Used on the Core Drilling Programs Since 1999

Period	Program (m)	Division	QA/QC Samples					Analyses (Method and weight)
			CD	LAPD	PD	BLK	STD	
1998	1,080	Saw	No	No	No	No	No	FA 50g
2007 to2008	5,714	Saw	2%	No	4%	2% CM	2% CM	FA 50g
2009	7,738	Saw; RSD	2%-0%	9-10%	4%	2%-3% IHC	2%-6% IHC	LW 1kg; FA 50g
2010 to 2011	73,025	Saw; RSD	No	5%	3%-5%	2% IHC	3% IHC	LW 1kg; FA 50g
2012 to 2014	65,795	Saw; RSD	No	5%	3%	2% IHC	3% IHC	LW 1kg
2016 to 2018	18,454	Saw; RSD	No	5%	5%	2% IHC	3% IHC	LW 1kg

RSD: Rotary Sample Divider
CD: Crush Duplicate, blind to the analytical laboratory
PD: Pulp Duplicate, blind to the analytical laboratory
LAPD: Lab-Aware Pulp Duplicate, known to the analytical laboratory
BLK: Blank, blind to the preparation laboratory
STD: Standard; blind to the preparation laboratory
CM : Certified Reference Material
NC: Non certified reference material
IHC: In-house reference material
FA: Fire Assay, with AAS finish
BLEG: Bulk Leach Extractable Gold
LW: LeachWELL

In all instances, the percentage is based on the total stream of samples submitted to the analytical laboratory, i.e., the primary samples plus the CD, PD, BLK, and STD, but excluding the LAPD.

During this period, Orezone relied partly on the internal analytical quality control measures implemented by ALS, BIGS, and SGS. In addition, Orezone implemented external analytical control measures on all RC, diamond drill hole, and trench sampling consisting of using control samples as well as duplicate sampling in all sample batches submitted for assaying.

Commercial CRMs, including standards and blanks were used on core samples, and core and RC tail samples analyzed by FA by SGS and ALS. In-house blanks and standards were used on RC and core samples analyzed by LeachWELL by BIGS and SGS. Field duplicates were used on RC samples analyzed by LeachWELL by BIGS and SGS. Pulp duplicates were used on all sampling including tails and were run by all laboratories. Lab-aware pulp duplicates were used on RC and core samples analyzed by LeachWELL by BIGS.

The type and location of the control samples in the sample stream has been determined on the basis of randomly generated numbers.

Orezone also did check assaying on RC and core samples analyzed by LeachWELL by BIGS and on core samples analyzed by FA by SGS, at a secondary umpire laboratory, SGS and ALS, respectively.

11.3.1 LeachWELL QA/QC

Since October 2007, Orezone introduced a procedure of internal certification for the reference material inserted in the stream of one-kilogram samples analyzed by the LeachWELL method. The in-house certified material is made of barren saprolite “spiked” with CRM. This method allows the insertion of blind QA/QC samples in the stream of samples and the monitoring of the accuracy of the analytical results; it was possible and economically affordable to implement this new procedure given the presence of three reliable commercial analytical laboratories in Ouagadougou from 2007.

Orezone has been using two different sources of barren oxidized material for the preparation of in-house CRM, saprolite of coarse-grained granite and lateritized coarse-grained granite. The material is collected in a quarry in batches of approximately 100 kg, crushed, dried and split in two-kilogram bags: to be accepted as a blank batch, five of the 50 two-kilogram samples are submitted to BIGS Global Burkina s.a.r.l. (BIGS). In Ouagadougou, for sample preparation and LeachWELL analysis of the gold content, and all samples must return a gold analysis less than or equal to the detection limit (i.e., 1 ppb or less) for the batch to be accepted.

Once a batch of blank material is accepted, it can be used as a blank in the stream of samples to be prepared, or it can be used as a blank base that will be spiked, with CRM. Using various certified materials and various proportions of barren blank and CRM, Orezone can create reference material with a theoretical gold grade within the range normally expected for the samples of a given exploration project. For Bomboré, Orezone focused on a range of grades between 0.2 g/t Au and 1.5 /t Au as approximately 97% of the mineralized samples. In the Project’s assay database display a grade less than or equal to 1.5 g/t Au, with a lower cut-off grade of 0.2 g/t Au used to define the mineralized envelopes of economic significance.

The list of CRM used for the preparation of the in-house standards since 2007 is presented in Table 11.11.

Table 11.11 Specification of Control Samples Used to Produce In-House Standards Used for LeachWELL by Orezone from October 2007 to January 2015

Certified Reference Material	Source	Certified Grade (Au ppm)	
		Average	Standard Deviation
Amis23	Amiso	3.57	0.33
Amis43	Amiso	1.65	0.85
HiSilK2*	Rocklabs Ltd	3.474	0.087
HiSilP1*	Rocklabs Ltd	12.05	0.33
OREAS 62c	ORE	8.79	0.01
OREAS 67a*	ORE	2.24	0.01
OxC102	Rocklabs Ltd	0.207	0.054
OxJ47	Rocklabs Ltd	2.384	0.048
OxK48	Rocklabs Ltd	3.557	0.042
OxQ75	Rocklabs Ltd	50.03	0.48
SF30	Rocklabs Ltd	0.81	0.021
SG56*	Rocklabs Ltd	1.027	0.21
SH24	Rocklabs Ltd	1.32	0.043
SH55*	Rocklabs Ltd	1.344	0.045
SL34	Rocklabs Ltd	5.893	0.14
SL46	Rocklabs Ltd	5.867	0.17
SL51	Rocklabs Ltd	5.909	0.023
SL61*	Rocklabs Ltd	5.931	0.177
SN26	Rocklabs Ltd	8.543	0.175
SN38	Rocklabs Ltd	8.573	0.018
SN50	Rocklabs Ltd	8.685	0.021
SQ36	Rocklabs Ltd	30.04	0.02
SQ48*	Rocklabs Ltd	30.25	0.17

** Used during the December 2012 to January 2015 period*

The list of CRM used for all the primary samples analyzed by FA is presented in Table 11.12.

Table 11.12 Specifications of Control Samples Used for Fire Assay Analysis of Primary Samples from October 2007 to January 2015

Certified Reference Material	Type	Source	Certified Grade (Au ppm)		Number of Samples Used
			Average	Standard Deviation	
Amis43	Certified Standard	Amiso	1.650	0.085	4
BLK10	Certified Blank	Rocklabs Ltd	<0.002	-	33
BLK12	Certified Blank	Rocklabs Ltd	<0.002	-	182
BLK13	Certified Blank	Rocklabs Ltd	<0.002	-	93
BLK24	Certified Blank	Rocklabs Ltd	<0.002	-	195
BLK31	Certified Blank	Rocklabs Ltd	<0.002	-	14
BLK35	Certified Blank	Rocklabs Ltd	<0.002	-	81
BLK44*	Certified Blank	Rocklabs Ltd	<0.002	-	54 (39)
HiSilK2*	Certified Standard	Rocklabs Ltd	3.474	0.087	(1)
HiSilP1*	Certified Standard	Rocklabs Ltd	12.050	0.330	35 (27)
OREAS 15h	Certified Standard	ORE	1.019	0.025	1
OREAS 65a	Certified Standard	ORE	0.520	0.017	1
OREASBLK	Certified Blank	ORE	<0.024	-	12
OxA71	Certified Standard	Rocklabs Ltd	0.085	0.006	8
OxC88	Certified Standard	Rocklabs Ltd	0.203	0.010	116
OxD73	Certified Standard	Rocklabs Ltd	0.416	0.013	22
OxE86	Certified Standard	Rocklabs Ltd	0.613	0.021	40
OxG83	Certified Standard	Rocklabs Ltd	1.002	0.027	113
OxJ68	Certified Standard	Rocklabs Ltd	2.342	0.064	15
SE58	Certified Standard	Rocklabs Ltd	0.607	0.019	67
SG56	Certified Standard	Rocklabs Ltd	1.027	0.033	1
SH41	Certified Standard	Rocklabs Ltd	1.320	0.041	71
SH55	Certified Standard	Rocklabs Ltd	1.375	0.045	2
SL46	Certified Standard	Rocklabs Ltd	5.867	0.170	8
SL61*	Certified Standard	Rocklabs Ltd	5.931	0.177	26 (22)
SN26	Certified Standard	Rocklabs Ltd	8.543	0.175	5
SQ48*	Certified Standard	Rocklabs Ltd	30.250	0.510	20 (14)

** Used during the December 2012 to January 2015 period*

The list of CRM used for all the leach residue samples analyzed by FA is presented in Table 11.13.

Table 11.13 Specifications of Control Samples Used for Fire Assay on LeachWELL Residues from October 2007 to January 2015

Certified Reference Material	Type	Source	Certified Grade (Au g/t)		Number of Samples Used
			Average	Standard Deviation	
BLK12	Certified Blank	Rocklabs Ltd	<0.002	-	58
BLK13	Certified Blank	Rocklabs Ltd	<0.002	-	295
BLK24	Certified Blank	Rocklabs Ltd	<0.002	-	205
BLK31	Certified Blank	Rocklabs Ltd	<0.002	-	168
BLK35	Certified Blank	Rocklabs Ltd	<0.002	-	218
BLK44*	Certified Blank	Rocklabs Ltd	<0.002	-	872 (306)
BLK50*	Certified Blank	Rocklabs Ltd	<0.002	-	74 (38)
BLK51*	Certified Blank	Rocklabs Ltd	<0.002	-	55 (13)
BLK9	Certified Blank	Rocklabs Ltd	<0.002	-	13
BLKORZ	Certified Blank	ORE	<0.002	-	128
OREAS 15f	Certified Standard	ORE	0.334	0.016	146
OREAS 15h	Certified Standard	ORE	1.019	0.025	67
OREAS 65a	Certified Standard	ORE	0.520	0.017	100
OREASBLK	Certified Blank	ORE	0.024	-	784
OxA71	Certified Standard	Rocklabs Ltd	0.085	0.006	538
OxA89*	Certified Standard	Rocklabs Ltd	0.084	0.008	140 (111)
OxC102	Certified Standard	Rocklabs Ltd	0.207	0.011	48
OxC109*	Certified Standard	Rocklabs Ltd	0.201	0.008	199 (159)
OxC72	Certified Standard	Rocklabs Ltd	0.200	0.012	207
OxC88	Certified Standard	Rocklabs Ltd	0.203	0.010	277
OxD73	Certified Standard	Rocklabs Ltd	0.416	0.013	2
OxD87	Certified Standard	Rocklabs Ltd	0.417	0.013	24
OxE101*	Certified Standard	Rocklabs Ltd	0.607	0.016	218 (110)
OxE106*	Certified Standard	Rocklabs Ltd	0.606	0.013	(30)
OxE86	Certified Standard	Rocklabs Ltd	0.613	0.021	245
OxF65	Certified Standard	Rocklabs Ltd	0.760	0.036	95
OxG83	Certified Standard	Rocklabs Ltd	1.002	0.027	286
OxJ47	Certified Standard	Rocklabs Ltd	2.384	0.048	87
OXJ47	Certified Standard	Rocklabs Ltd	2.365	0.059	54
OxJ68	Certified Standard	Rocklabs Ltd	2.342	0.064	69
OxK48	Certified Standard	Rocklabs Ltd	3.557	0.042	46
OxK48	Certified Standard	Rocklabs Ltd	3.460	0.093	47
OXL51	Certified Standard	Rocklabs Ltd	5.850	0.123	13
SE44	Certified Standard	Rocklabs Ltd	0.590	0.028	202

Certified Reference Material	Type	Source	Certified Grade (Au g/t)		Number of Samples Used
			Average	Standard Deviation	
SE58	Certified Standard	Rocklabs Ltd	0.607	0.019	328
SF23	Certified Standard	Rocklabs Ltd	0.831	0.027	21
SF30	Certified Standard	Rocklabs Ltd	0.832	0.021	51
SF30	Certified Standard	Rocklabs Ltd	0.810	0.031	140
SG56*	Certified Standard	Rocklabs Ltd	1.027	0.033	282 (43)
SG66*	Certified Standard	Rocklabs Ltd	1.086	0.032	(48)
SH24	Certified Standard	Rocklabs Ltd	1.326	0.043	6
SH41	Certified Standard	Rocklabs Ltd	1.320	0.041	310
SH55*	Certified Standard	Rocklabs Ltd	1.375	0.045	138 (34)
SL34	Certified Standard	Rocklabs Ltd	5.893	0.140	13
SL34	Certified Standard	Rocklabs Ltd	5.770	0.140	30
SL46	Certified Standard	Rocklabs Ltd	5.867	0.170	2

** Used during the December 2012 to January 2015 period*

11.4 RPA Comments

In RPA's opinion, the sampling preparation, security, and analytical procedures used by Orezone are consistent with, and often exceed, generally accepted industry best practices and are, therefore, adequate for use in the estimation of Mineral Resources.

In RPA's opinion, the QA/QC program as designed and implemented by Orezone is adequate and the assay results within the database are suitable for use in a Mineral Resource estimate.

12.0 DATA VERIFICATION

Orezone uses a quality management system with procedures at all stages of work from exploration through to resource estimation. RPA reviewed these procedures and results RPA also conducted independent checks during a site visit, a series of digital queries, and checks of laboratory certificates. RPA is of the opinion that the work complies with industry standards and the drill hole data are adequate for the purposes of Mineral Resource estimation.

12.1 Orezone Database Verification Procedures

All field data are captured in digital files and subsequently downloaded to network linked computers. The database is checked for input errors at different stages, from the field office to the Ouagadougou office. The data is imported by a team of geologists and technicians in a master database that is managed by a geologist who is responsible for the quality control and sampling protocols. Field and assay data are transferred into the master database in Ouagadougou with additional auditing performed by a geologist, a resource geologist, the Exploration Manager, and the Senior Vice President, Exploration. Data is stored in a Datashed database.

Data in the master database can easily be tracked by project number, date, and activities. Orezone tracks the sample stream by recording the project, job, certificate numbers, and date at different stages of the sampling stream. Paper maps and original signed laboratory certificates are saved and the old files are archived. The integrity of the database is protected by restricted access, transfer of data using the VLOOKUP function, Excel filters and control formulas, GEMS validations, Datashed validation, and visual examination.

Sample shipments and assay deliveries were routinely monitored as produced by the preparation and assaying laboratories. Assay results and quality control data produced by the various laboratories are inspected visually and analyzed using various bias and precision charts. At the end of each drilling program, Orezone produces quality control reports summarizing protocol and the quality control.

12.2 Site Visit and Core Review

RPA visited the property most recently from October 10 to 13, 2014. All the aspects and procedures of the exploration program, from drill set-up through to sample shipment, were reviewed. Drill core and chips from typical holes were reviewed and compared to digital logs and on vertical cross sections, following the lithological description and interpretation, and the gold mineralization. Several RC and core independent samples collected by RPA during the site visit confirmed the presence of gold and show good grade correlation with original samples.

Table 12.1 Assay Check Samples

Type	Prospect	HoleID	Sample	From (m)	To (m)	Au (g/t)	Check Sample	Check Au (g/t)
RC	P8/P9	BBC2232	1003628	42	43	0.77	196062	0.76
RC	Siga S	BBC3338	1155309	32	33	3.425	196063	3.48
RC	P8/P9	BBC3480	1171086	33	34	0.815	196065	0.71
RC	P8/P9	BBC2669	1068512	27	28	1.32	196066	1.36
core	Siga E	BBD0450	1048667	182	183	1.248	196068	1.26
core	Maga	BBD0665	1053153	133	134	0.57	196069	0.62

12.3 Assay Table Review

RPA received 5,553 assay certificates in Excel format from Orezone. These included assays for samples collected from 2005 to 2015, covering the entire Project, as well as the QA/QC material. Assay certificates of cyanide leach and tails were grouped and assembled in Excel, then matched to a combined North and South database. Data from 1,410 cyanide leach certificates and 525 tails certificates were used to match in excess of 40,000 leach and 10,000 tails assays in the unified database. No significant discrepancies were identified.

12.4 Basic Database Verification Tests

RPA reviewed the resource database using a number of basic database and resource model checks including:

- Visual review for collar location above or below topographic surface and for drill hole traces with unreasonable directions.
- Basic database queries and sorting checks for from/to errors, unreasonable assay interval length, and missing and duplicate sample numbers.
- Plotting and querying of the relationship between fire assays (total gold) and LeachWELL results.

RPA considers the resource database reliable and appropriate to prepare a Mineral Resource estimate.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Extensive testwork programs have been carried out at different laboratories for the Bomboré Project with the first test program started in 1997 and the latest completed in 2019. The test programs were conducted on drill core composites, RC cuttings, and RAB drill samples considered representative of the ore deposit at the time of each test program. A summary list of the programs is included in Table 13.1. A map illustrating the metallurgical sampling locations is shown in Figure 13.1 and Figure 13.2.

Table 13.1 Summary of Testwork Programs

Program	Leachwell Recoveries	Head Analysis	Variability	Cyanidation	Gravity	Flotation	Carbon-in-Leach (CIL)	Carbon Adsorption & Equilib.	Column Leach (HL)	Comminution	Scrubbing	Gold Deportment	Petrography	Thickening / Rheology	Neutralization	Lime Demand	Acid Mine Drainage
SGS / ITS 1997			✓	✓									✓				
Osborne 2008			✓	✓													
AMMTEC 2009		✓	✓	✓	✓	✓	✓		✓	✓							✓
McClelland 2012*		✓	✓	✓	✓	✓	✓			✓			✓	✓	✓		
Phillips 2012										✓							
OREZONE Scrubbing 2012			✓	✓							✓	✓					
Met-Solve 2013											✓	✓					
SGS Lakefield 2013										✓							
COREM 2013				✓						✓			✓				
Met-Solve 2014				✓			✓				✓						
Consolidated Database 2013	✓																
Kappes 2014		✓	✓	✓			✓		✓		✓	✓		✓	✓		
SGS Lakefield 2014										✓							
SGS Lakefield 2016				✓	✓	✓				✓			✓				
SGS Lakefield 2017/2018			✓	✓						✓						✓	
Outotec 2018														✓			
Base Metallurgical Lab 2019		✓	✓	✓						✓			✓	✓			
SGS Lakefield 2019								✓									

****Includes Pocock report in appendix*

The following sections summarize selected testwork results that are pertinent to the Project. Additional details of each program are provided in, individual reports.

Figure 13.1 Metallurgical Sampling Locations (Oxide Samples)

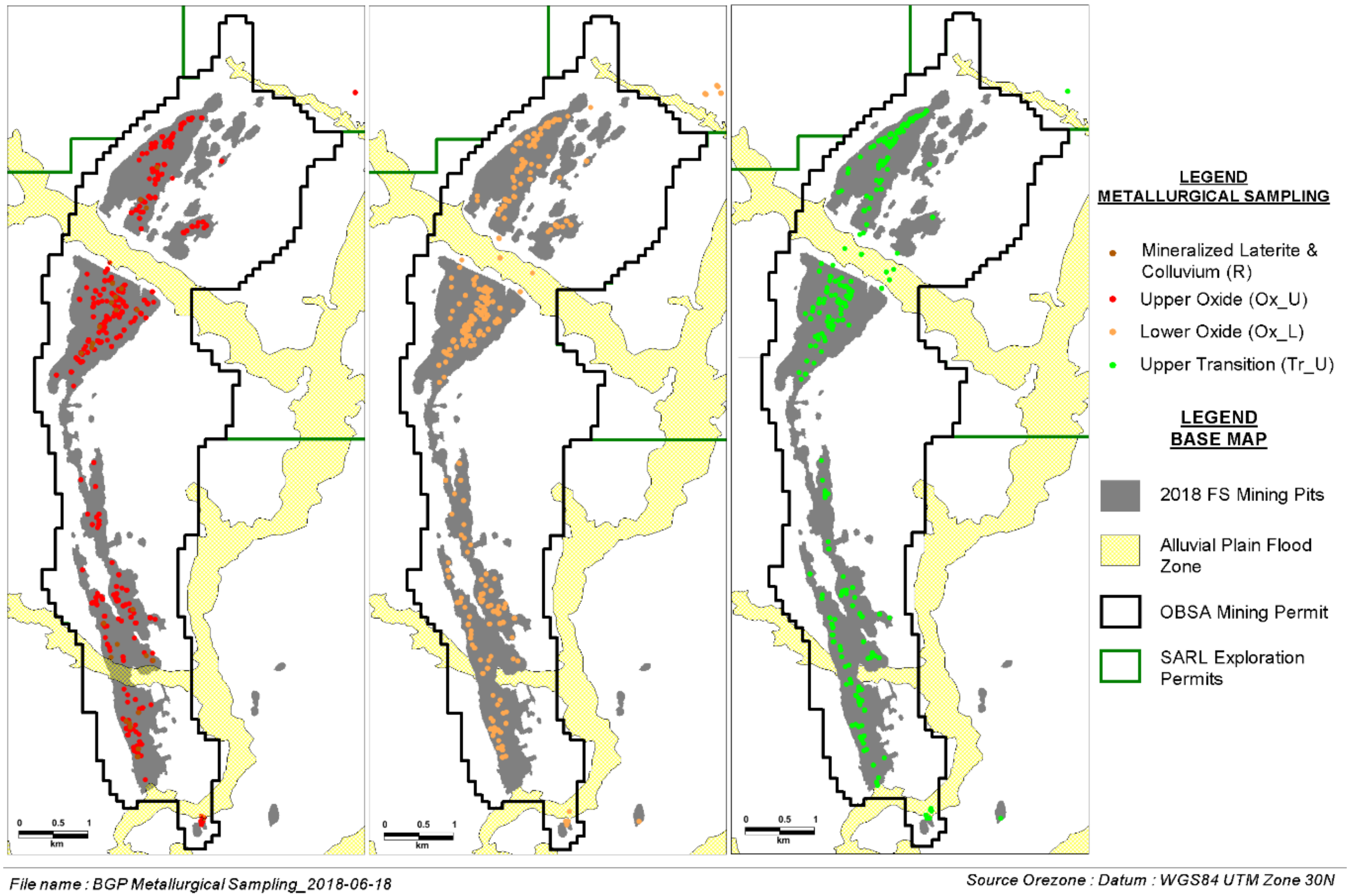
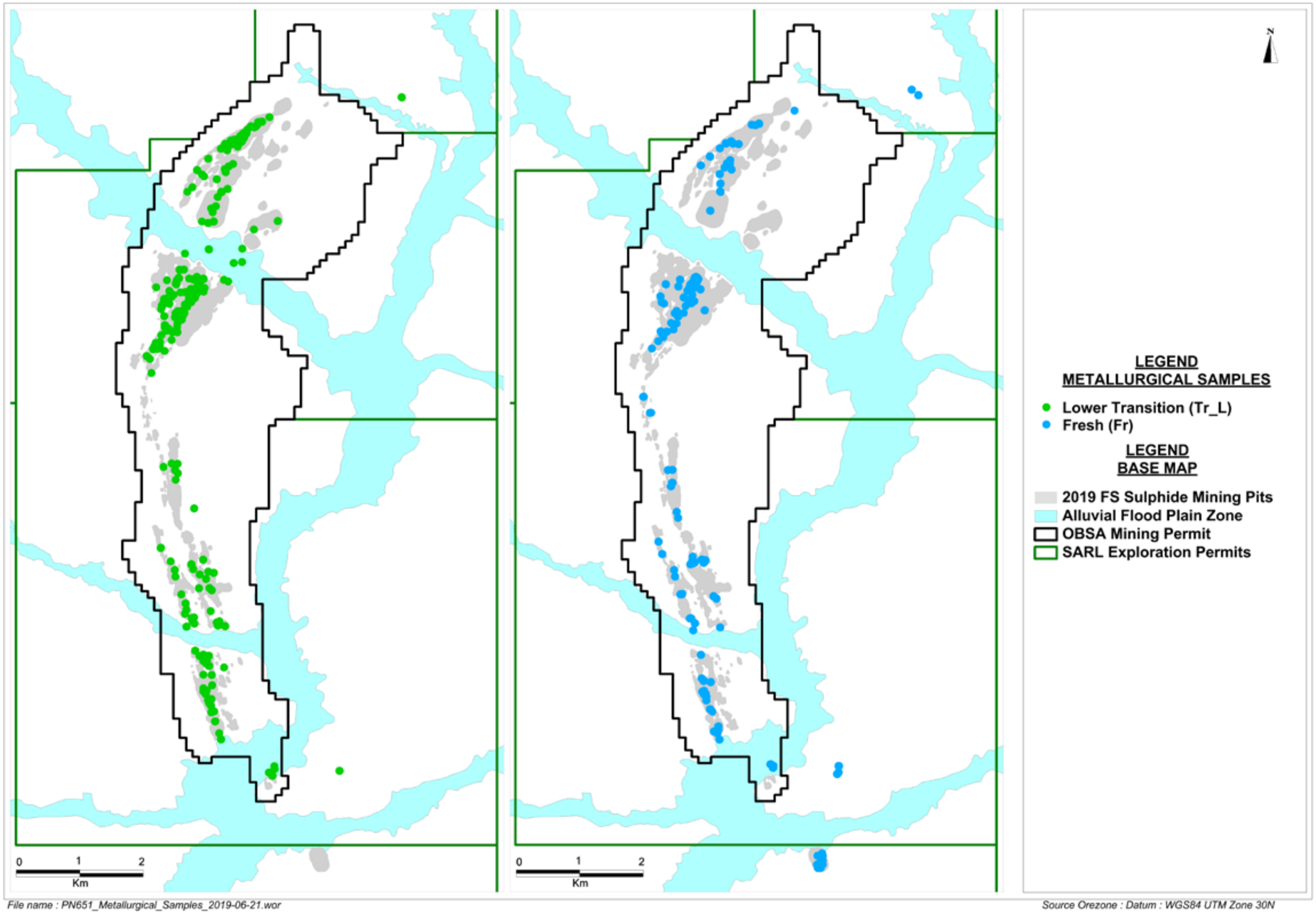


Figure 13.2 Metallurgical Sampling Locations (Sulphide Samples)



13.2 Historical Testwork Programs

13.2.1 SGS/ITS 1997

Ninety-one (91) samples from the Siga, Maga and P8/P9 mineralized zones were collected for the 1997 testwork program. Of these, 67 samples were from RC drilling, which were used in bottle roll tests at SGS (Ghana). The remaining 24 samples were from RAB (rotary air blasting) drillings that were used at the ITS Laboratories (Burkina Faso). Mineralogy analyses were also conducted on 10 diamond drill core samples at the SGS Lakefield Research facility in Canada.

Highlights from this program are as follows:

SGS (Ghana)

- Of the 67 RC samples, 57 were oxide and transition ore types, and 10 were sulphides.
- Gold extraction by cyanide leaching ranged from 67% to 99%, with an average of 92% for the RC oxide and transition samples, and 87% for the sulphide samples.

ITS Laboratories (Burkina Faso)

- All 24 RAB drill samples were oxide material.
- Gold extractions by cyanide leaching ranged from 63% to 95%, with an average of 85%.

SGS Lakefield Research (Canada)

- Of the 10 diamond drill samples, 8 were oxides and 2 were sulphides.
- The occurrences of gold in the oxide samples fell under the category of a) liberated gold, b) liberated gold with discontinuous rims and attachment of goethite ± silicates, and c) gold attached to, or encapsulated in, gangue minerals.
- One sulphide sample showed 37% locked gold in pyrite, and the other sulphide sample showed no visible gold.
- The grain size of gold particles ranged from 0.5 µm to 800 µm, with ~85% less than 30 µm in diameter.

13.2.2 Osborne 2008 Testwork Program

One hundred and eighty-four (184) composite RC samples from the Siga, Maga, Maga S, P11, KT, P8/P9 and CFU mineralized zones were used in this program to conduct three streams of testwork. The tests involved bottle roll bulk leach extractable gold (BLEG) tests conducted at Alibab (Ouagadougou) and fire assay (FA) conducted at Alibab (Bamako).

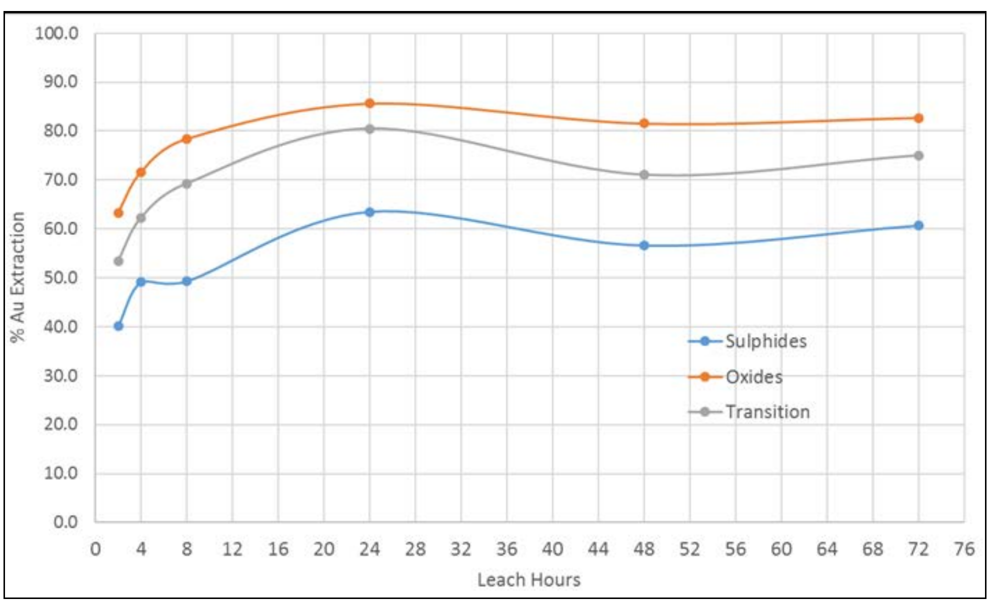
Descriptions of each testwork stream are as follows:

- Stream 1 – Samples were pulverized to 75 µm, subjected to 24-hour bottle roll BLEG, and FA on leach residue at 24 hours.
- Stream 2 – Samples were pulverized to 106 µm, subjected to 24-hour bottle roll BLEG, and FA on leach residue at 24 hours. The +2 mm fractions were subjected to FA for coarse gold detection.
- Stream 3 – Samples were pulverized to 75 µm, subjected to 72-hour bottle roll BLEG, and FA on leach residue at 2, 4, 8, 24, 48 and 72 hours.

The highlights from the leach test results are as follows:

- According to Osborne, after eliminating one sample, gold extraction at 24 hours averaged 93% for oxide, 92% for transition, and 83% for sulphide samples.
- However, based on Lycopodium's observation (see Figure 13.2) developed from raw testing data supplied by Orezone (not included with the available Osborne reports), gold extraction at 24 hours averaged 85% for oxide, 80% for transition, and 63% for sulphide samples.
- Leaching was substantially complete after 24 hours for all samples as seen in Figure 13.3.

Figure 13.3 Osborne Average Leach Curves



13.2.3 AMMTEC 2009 Testwork Program

The 2009 AMMTEC program was a scoping-level metallurgical testwork program supervised by GBM. The program included head analysis, acid mine drainage (AMD) analysis, comminution testwork, heap leach amenability via column leach tests, CIL cyanidation, grind optimization, and flotation testwork. The samples provided to AMMTEC were PQ and HQ drill cores taken from the fresh rock, transition, and oxide ore zones.

The following composites were formed:

Comminution

- AM01 (fresh rock), AM02 (transition), and AM03 (oxide).

Extraction & Acid Mine Drainage Analysis

- AM11 (fresh rock) from P8/P9 – Maga area, and AM12 (fresh rock) from Siga area.
- AM21 (transition), and AM31 (oxide), all excluding metasediments (sandy and clay-rich portions).
- AM22 (transition), and AM32 (oxide), all are metasediments.

Selected head analysis results are presented in Table 13.2.

Table 13.2 AMMTEC Head Analysis

Sample ID	AM11	AM12	AM21	AM22	AM31	AM32
Description	Fresh Rock - P8P9-Maga area	Fresh Rock - Siga area	Transition excluding metasediments	Transition metasediments	Oxide excluding metasediments	Oxide metasediments
Expected g/t Au	0.90	0.90	0.62	0.59	0.61	0.65
Head assays						
Au (g/t)	1.09	0.56	0.61	0.59	0.63, 0.74	1.52, 0.80
Ag (g/t)	< 2	< 2	< 2	< 2	< 2	< 2
As (ppm)	606	1,495	187	986	172	657
C_{Total} (%)	1.01%	0.58%	0.16%	0.22%	0.04%	0.21%
C_{Org} (%)	< 0.03	< 0.03	< 0.03%	0.24%	0.04%	0.22%
CO_3 (%)	5.05%	2.90%	0.80%	< 0.10%	0.00%	< 0.05%
Cu (ppm)	257	31	215	57	224	58
Ni (ppm)	24	21	26	52	32	56
Pb (ppm)	10	22	43	15	36	41
Zn (ppm)	67	164	83	75	75	119

Organic carbon content is considered low for fresh rock and oxide and transition samples that excluded metasediments, therefore, preg-robbing effect is not anticipated during leaching. For the samples that contained oxide and transition metasediments organic carbon was present in a sufficient amount for Osborne to consider these as having preg-robbing potential at the time; however, this was negated by later testwork.

Copper content is present in relatively low amounts in all the samples; therefore, excess cyanide consumption is not expected from the production of copper-cyanide complexes in solution.

Silver content is low and therefore silver extraction is not addressed in this program.

Acid mine drainage results are presented in Table 13.3. As part of this test, specific gravities were also determined.

Table 13.3 AMMTEC Acid Mine Drainage and Specific Gravity Results

Sample ID	**AM11**	**AM12**	**AM21**	**AM22**	**AM31**	**AM32**
Description	Fresh Rock - P8P9-Maga area	Fresh Rock - Siga area	Transition excluding meta-sediments	Transition meta-sediments	Oxide excluding meta-sediments	Oxide meta-sediments
Specific gravity	2.85	2.82	2.79	2.71	2.82	2.65
Acid Neutralizing Capacity (kg H_2SO_4/t)	97.3	75.9	23.6	10.0	8.2	79.5
Net Acid Generation (kg H_2SO_4/t)	-10.3	-9.4	-10.4	-6.1	-7.4	-5.3
Total Acid Production Potential (kg H_2SO_4/t)	98.5	54.6	6.7	1.8	3.7	0.9
Net Acid Production Potential (kg H_2SO_4/t)	1.3	-21.3	-16.9	-8.2	-4.6	-7.0
pH	6.27	4.76	6.17	3.92	3.99	3.9
Conductivity (mS/cm)	1.502	1.137	0.26	0.200	0.179	0.18

The results showed that all samples tested are capable of neutralizing any acidic solutions formed from oxidation of sulphides, therefore, acid mine drainage is not expected to be problematic at Bomboré.

The comminution results are presented in Table 13.4.

Table 13.4 AMMTEC Comminution Results

Sample ID	AM01		AM02		AM03
Description	Fresh Rock		Transition		Oxide
Crushing Work Index (kWh/tonne) Average	12.2		4.6		n/a
Minimum	5.7		2.6		n/a
Maximum	28.1		6.6		n/a
Bond Work Rod Mill Work Index (kWh/tonne)	19.5		7.8		5.4
Bond Work Ball Mill Work Index (kWh/tonne)	16.8		8.2		1.9
Indicative Energy Consumption (kWh/tonne)	3.78		2.67		n/a
JK Drop Weight AG/SAG Parameters A	61.6		56.7		n/a
b	0.54		7.18		n/a
t_a	7.18		3.73		n/a
Abrasion Index (g)	0.4064		0.0383		0.0051
Unconfined Compressive Strength (MPa)	148.87	Cataclasis	9.36	Shear	n/a
	69.47	Shear	9.26		
	41.19		16.78		
	74.12		3.71		
	60.96		6.04		

The results indicated that fresh ore is hard, while transition and oxide ores are soft. When the fresh rock A x b parameters are compared to other samples in the JKTech database, only about 20% are harder. Fresh ore is moderately abrasive, while transition ore is mildly abrasive, and oxide ore is non-abrasive.

Table 13.5, Table 13.6 and Table 13.7 provide high-level summaries of the leach extraction and flotation results for fresh, transition and oxide ores, respectively.

Table 13.5 AMMTEC Fresh Rock Leach Extraction & Flotation Results

Sample ID	AM11				AM12			
Description	Fresh Rock - P8P9-Maga area				Fresh Rock - Siga area			
Coarse Bottle Roll @ 4 Crush Sizes (mm)	<25	<19	<12.5	<4	<25	<19	<12.5	<4
Gold Extraction (%) @6 Days	13.46	15.92	19.43	38.22	28.42	17.85	35.25	43.84
NaCN Consumption (kg/t) @6 Days	0.35	0.35	0.32	0.48	0.36	0.40	0.43	0.51
CIL Cyanidation @ 4 P_{80} Grind Sizes (µm)	150	106	75	53	150	106	75	53
Gold Extraction (%) @8 Hours	65.21	71.35	72.67	73.56	83.2	88.12	91.87	88.46
NaCN Consumption (kg/t) @8 Hours	1.26	1.39	1.33	1.38	1.28	1.18	0.95	1.35
Flotation Testwork @ 4 P_{80} Grind Sizes (µm)	150	106	75	53	150	106	75	53
Flotation Gold Extraction (%)	92.54	87.79	91.82	92.35	80.74	80.55	87.11	85.04
Mass Pull (%)	12.62	11.71	10.53	11.36	9.05	8.16	7.31	7.38
Gold Extraction w Flot Conc Leaching (%)	71.7				75.8			
Grav Conc/Grav Tails CIL, All @ P_{80} 53 µm								
Gold Extraction (%) @48 Hours	63.68				83.65			
NaCN Consumption (kg/t) @48 Hours	1.66				1.76			

Table 13.6 AMMTEC Transition Samples Leach Extraction & Flotation Results

Sample ID	**AM21**				**AM22**			
Description	Transition excluding metasediments				Transition metasediments			
Coarse Bottle Roll @ 4 Crush Sizes (mm)	<25	<19	<12.5	<4	<25	<19	<12.5	<4
Gold Extraction (%) @6 Days	69.14	72.51	71.26	68.57	79.19	72.43	69.29	82.17
NaCN Consumption (kg/t) @6 Days	0.89	0.97	0.97	1.04	0.71	0.92	0.96	1.04
CIL Cyanidation @ 4 P_{80} Grind Sizes (µm)	150	106	75	53	150	106	75	53
Gold Extraction (%) @8 Hours	89.48	90.8	90.94	90.93	92.56	91.63	91.66	94.23
NaCN Consumption (kg/t) @8 Hours	1.75	1.84	1.76	1.85	1.69	1.74	1.57	1.70
Flotation Testwork, All @ P_{80} 75µm								
Flotation Gold Extraction (%)	69.91				69.68			
Mass Pull (%)	8.09				16.71			
Grav Conc/Grav Tails CIL, All @ P_{80} 5 3µm								
Gold Extraction (%) @48 Hours	84.73				89.66			
NaCN Consumption (kg/t) @48 Hours	2.98				2.75			
Column Leach, @ <15mm, After 44 Days								
Gold Extraction (%)	77.33				87.52			
NaCN Consumption (kg/t)	0.39				0.54			
Column Diameter (mm)	190				190			
Column Height (mm)	1244				1291			
Final Slumpage (%)	9.3				11			
Pellet Quality After 44 Days	Fair				Fair			
Leach Tails Grade (g/t)	0.139				0.07			
Portland Cement Addition (kg/t)	3				3			

Table 13.7 AMMTEC Oxide Samples Leach Extraction & Flotation Results

Sample ID	**AM31**				**AM32**			
Description	Oxide excluding metasediments				Oxide metasediments			
Coarse Crush Bottle Roll @ 4 Sizes (mm)	<25	<19	<12.5	<4	<25	<19	<12.5	<4
Gold Extraction (%) @6 Days	82.45	81	78.66	81.17	84.16	84.77	83.58	86.15
NaCN consumption (kg/t) @6 Days	0.99	1.04	0.99	1.14	0.82	0.79	0.94	1.07
CIL Cyanidation @4 P_{80} Grind Sizes (µm)	150	106	75	53	150	106	75	53
Gold Extraction (%) @8 Hours	87.35	90.2	90.96	92.35	91.47	92.6	92.6	94.83
NaCN consumption (kg/t) @8 Hours	1.74	1.59	1.73	1.65	1.72	1.74	1.68	1.78
Flotation Testwork, All @ P_{80} 75 µm								
Flotation Gold Extraction (%)	65.01				67.22			
Mass Pull (%)	11.64				21.22			
Grav Conc/Grav Tails CIL, All@P_{80} 53 µm								
Gold Extraction (%) @48 Hours	85.55				79.02			
NaCN consumption (kg/t) @48 Hours	2.53				3.27			
Column Leach, @ <25mm, After 44 Days								
Gold Extraction (%)	82.65				80.35			
NaCN Consumption (kg/t)	0.14				0.32			
Column Diameter (mm)	190				190			
Column Height (mm)	1433				1391			
Final Slumpage (%)	3.6				8.5			
Pellet Quality After 44 Days	Excellent				Excellent			
Leach Tails Grade (g/t)	0.12				0.16			
Portland Cement Addition (kg/t)	9				9			

A high-level summary of the AMMTEC leach and flotation results is as follows:

- Coarse crush cyanidation results indicated that <19 mm crush size would be sufficient for heap leaching transition ore and <25 mm for the oxide ore. Coarse crush cyanidation results for the fresh rock showed low gold extraction even at the finest crush size of <4 mm, therefore, column leach tests were not conducted for the fresh rock samples, and only conducted for the transition and oxide ores.

- Results from the column leach tests predicted that heap leaching could provide a gold recovery of approximately 80% for both transition and oxide ores.

- CIL results (at finer grind sizes) indicated that gold recovery could improve by 10% to 12% when compared to heap leaching.

- Fresh rock samples contained sulphide sulphur in reasonable quantity, therefore, good flotation recoveries were achieved for these, however, the concentrate grades were not high enough to be considered a saleable product and further treatment would be required.

- Ultra fine grinding (P_{80} of 10 µm) of the flotation concentrate followed by cyanidation was conducted to determine the overall combined recovery. The results showed lower recoveries than simply conducting whole ore CIL for both fresh rock samples.

- Results from gravity concentration followed by gravity tails cyanidation could not be properly assessed because in every one of the cases, the overall recovery was poorer than that of the whole ore CIL. It was expected that at the very least, the recovery would match that of the CIL. AMMTEC considered these results anomalous.

- It was recommended at the conclusion of this program that heap leaching should be the approach for transition and oxide ores, and the fresh rock ore should be treated by milling and leaching.

13.2.4 McClelland 2012 Testwork Program

The 2012 McClelland program was a feasibility-level metallurgical testwork program supervised by Mr. J. Woods, a metallurgical consultant from WSP. The testwork included ore variability composite testing, comminution testing, CIL/CIP, residue characterization, and waste rock testing. Gravity concentration and bulk flotation tests were conducted at a scoping-level. A total of 76 drill core samples were submitted for a detailed head analysis. Four composite samples were then formed from 33 oxide samples and 33 sulphide samples, then stage-crushed to <12.5 mm, and categorized by ore type and grade. High and medium grade oxide ores were denoted as HGO and MGO, and high and medium grade sulphide ores were denoted as HGS and MGS.

Selected head analysis results for the four composites are presented in Table 13.8.

Table 13.8 McClelland Head Analysis

Analysis		HGO	MGO	HGS	MGS
Fire Assay	g/t Au	2.38	0.57	1.81	0.74
Met Screen Assay	g/t Au	2.62	0.61	1.70	0.85
ICP Scans	g/t Ag	1.0	2.0	1.0	1.0
	Duplicate g/t Ag	1.2	1.54	1.09	1.78
	Al %	8.05	7.87	6.06	6.39
	g/t As	715	251	1,810	1,065
	g/t Cu	120.5	174.5	84.5	75.0
	Fe %	5.55	6.49	6.23	6.17
	g/t Hg	0.02	0.03	0.02	0.03
	g/t Ni	50.7	54.9	27.2	55.8
	g/t Te	1.22	2.02	1.03	1.10
	C (Total) %	0.11	0.07	1.04	1.09
	C (Organic) %	0.11	0.02	0.07	0.05
	C (Inorganic) %	<0.05	<0.05	0.93	0.98
	S (total) %	0.04	0.16	1.88	2.26
	S (Sulphate) %	0.01	0.12	0.01	<0.01
	S (Sulphide) %	0.04	0.03	1.71	2.16

There were notable levels of aluminium (6% to 8%) in the samples. Mercury content was low, indicating that a mercury removal system need not be considered in the plant design. Copper was also present at low levels, indicating that excess cyanide consumption is not to be expected. There were notable levels of sulphides in the HGS and MGS samples as expected. The total organic carbon was also low (0.2% to 0.11%), indicating preg-robbing should not be expected.

Previous head analysis conducted by Osborne suggested the possibility of preg-robbing, therefore, a preg-robbing analysis was conducted in this program using a spiked (Au-AA31) and cyanide shake (Au-AA31a) test to determine the Preg Rob factor. A factor that approaches zero or negative value indicates low likelihood of preg-robbing. The analysis showed that the Bomboré oxide and sulphide ores are unlikely to encounter preg-robbing issues. Refer to Table 13.9 for more details.

Table 13.9 McClelland Preg-Robbing Analysis

Sample	TOC %	g/t Au					% Preg Rob Factor
		Au-AA31	Au-AA31a	Difference	Spike Value	Difference	
HGO	0.11	5.55	1.97	3.58	3.43	-0.15	-4.4
MGO	0.02	4.08	0.48	3.60	3.43	-0.17	-5.0
HGS	0.07	4.87	1.16	3.71	3.43	-0.28	-8.2
MGS	0.05	3.94	0.54	3.40	3.43	0.03	0.9

Scoping level gravity concentration tests were conducted at <850 µm feed size. The results showed that oxide ore did not respond well to gravity concentration treatment at this feed size, while the sulphide ore responded better but not particularly well either. Table 13.10 provides a summary of the results.

Table 13.10 McClelland Gravity Concentration Test Results

Sample	Mass Pull %		Grade, g/t Au				Gold Extraction %	
	Cleaner Conc.	Rougher Conc.	Cleaner Conc.	Rougher Conc.	Rougher Tails	Calc'd Head	Cleaner Conc.	Rougher Conc.
HGO	0.25	1.29	75.42	26.84	1.29	1.62	11.6	21.3
MGO	0.35	1.19	15.71	6.35	0.54	0.54	9.0	12.4
HGS	0.42	1.52	123.66	41.31	1.50	2.11	24.6	29.9
MGS	0.45	1.62	80.25	25.30	0.66	1.06	34.1	38.7

Microscopic examinations of each gravity concentrate showed presence of particulate gold in HGO, MGO and HGS composites, with size ranging from 0.16 mm to 0.71 mm.

Scoping level bulk sulphide flotation tests were conducted at P_{80} of 75 µm feed size. The results showed that the oxide ore did not respond well to conventional bulk sulphide flotation treatment at this feed size, while the sulphide ore responded well under the same conditions with gold extractions of 92.5% and 89.4% for the HGS and MGS, respectively. Table 13.11 provides a summary of the results.

Table 13.11 McClelland Flotation Concentration Test Results

Sample	Mass Pull %		Grade, g/t Au				Gold Extraction %	
	Cleaner Conc.	Rougher Conc.	Cleaner Conc.	Rougher Conc.	Rougher Tails	Calc'd Head	Cleaner Conc.	Rougher Conc.
HGO	6.27	21.51	15.40	5.77	0.62	1.73	55.9	71.8
MGO	1.35	8.16	12.58	3.66	0.39	0.66	25.9	45.5
HGS	6.07	9.68	26.98	17.38	0.15	1.82	90.1	92.5
MGS	9.92	23.43	7.25	3.32	0.12	0.87	82.7	89.4

Microscopic examinations of each flotation cleaner concentrate showed no presence of particulate gold.

Grind size optimization tests using bottle roll leaching by cyanidation were conducted at three P_{100}'s of 212 µm, 106 µm and 53 µm for the oxide ore, and five P_{80}'s - 150 µm, 106 µm, 75 µm, 53 µm, and 45 µm for the sulphide ore. A summary of the results is presented in Table 13.12.

Table 13.12 McClelland Summary of Grind Optimization Tests

Ore Type	Grind Size	72-hr Gold Extraction %	g/t Au			Reagents kg/t	
			Extracted	Tails	Calc'd Head	NaCN	Lime Added
Oxide (avg. of HGO & MGO)	P_{100}-212 µm	93	1.29	0.09	1.37	0.12	3.1
	P_{100}-106 µm	93.0	1.19	0.08	1.27	0.08	3.1
	P_{100}-53 µm	91.3	1.13	0.08	1.21	0.28	3.0
Sulphides (avg. of HGS & MGS)	P_{80}-150 µm	75.3	1.06	0.32	1.38	0.47	1.0
	P_{80}-106 µm	78.2	1.05	0.29	1.34	0.39	1.0
	P_{80}-75 µm	80.2	1.12	0.27	1.39	0.52	1.1
	P_{80}-53 µm	84.1	1.22	0.23	1.44	0.76	1.3
	P_{80}-45 µm	84.4	1.28	0.24	1.52	2.36	1.7

The results showed that both oxide and sulphide composites were readily amenable to direct cyanidation at all feed sizes. The oxide composites had consistently good gold extraction for all three feed sizes, while the sulphide composites appeared to have small improvement with decreasing feed size. Cyanide consumptions were low in all cases except for the P_{80} of 45 µm case (HGS) for sulphides where the consumption was significantly higher. The P_{80} of 45 µm case for the HGS composite was also the only test conducted with mechanical agitation. Cyanide consumption did appear to increase slightly with decreasing feed size. Lime consumptions were moderate for the oxide composites and low for the sulphide composites.

The leach kinetics for all the composites were considered rapid and gold extraction showed signs of plateauing after the 24-hour mark. The gold leach profiles are illustrated in Figure 13.4 to Figure 13.7.

Figure 13.4 McClelland Gold Leach Profile for High Grade Oxide Composite (HGO)

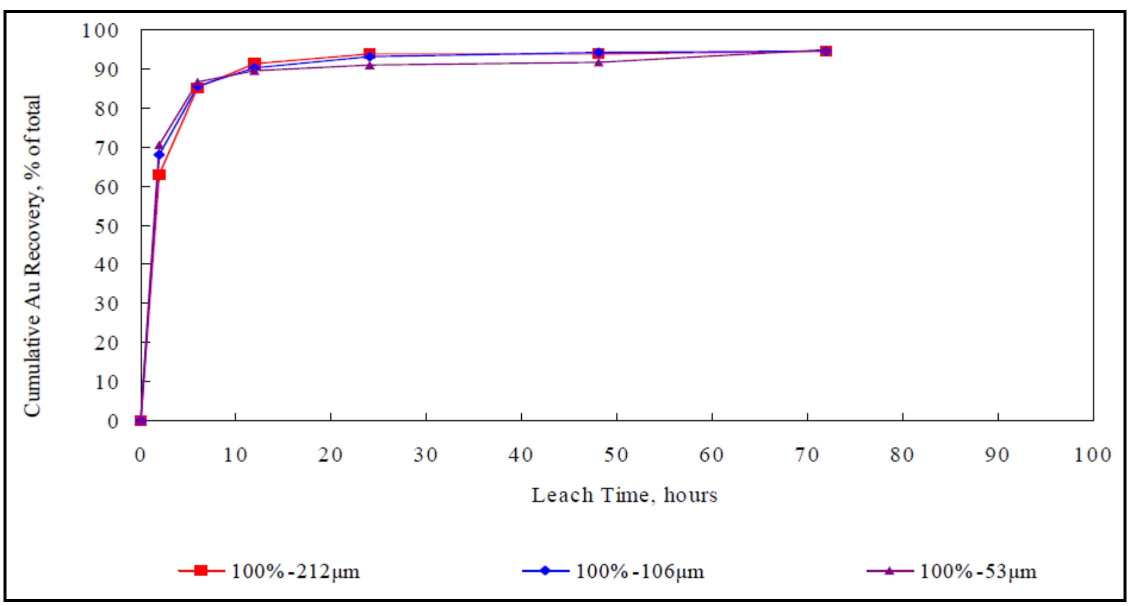


Figure 13.5 McClelland Gold Leach Profile for Medium Grade Oxide Composite (MGO)

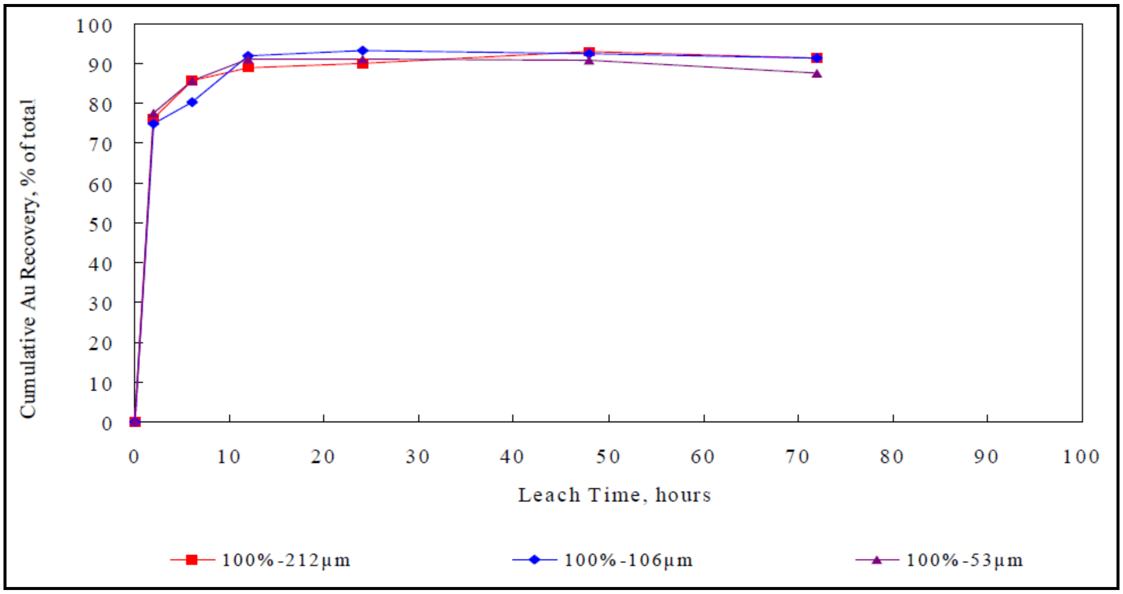


Figure 13.6 McClelland Gold Leach Profiles for High Grade Sulphide Composite (HGS)

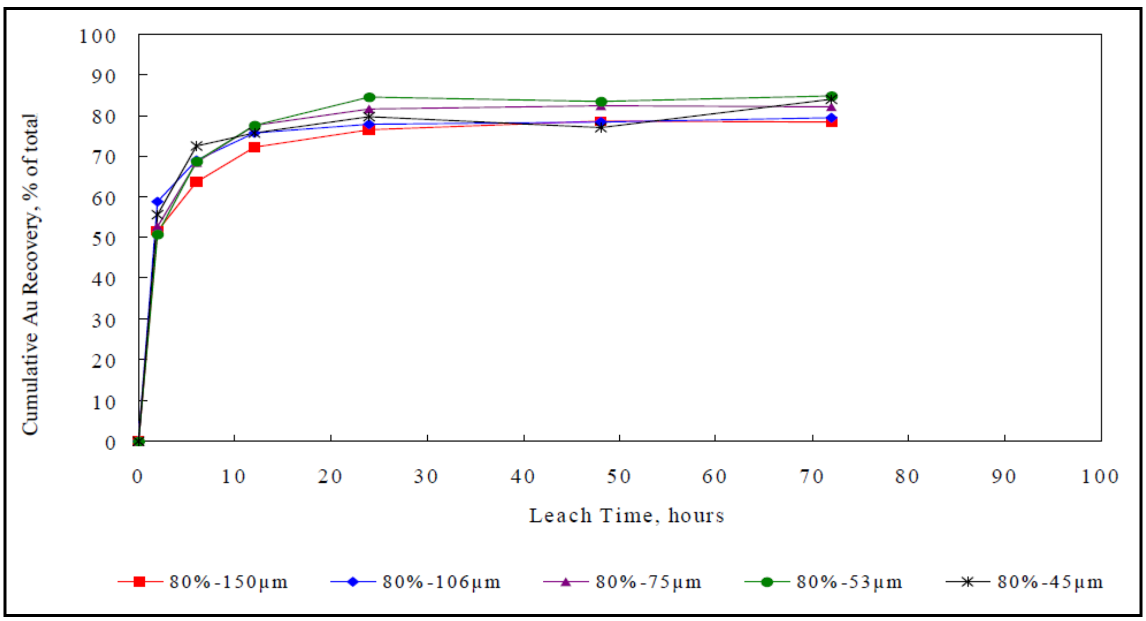
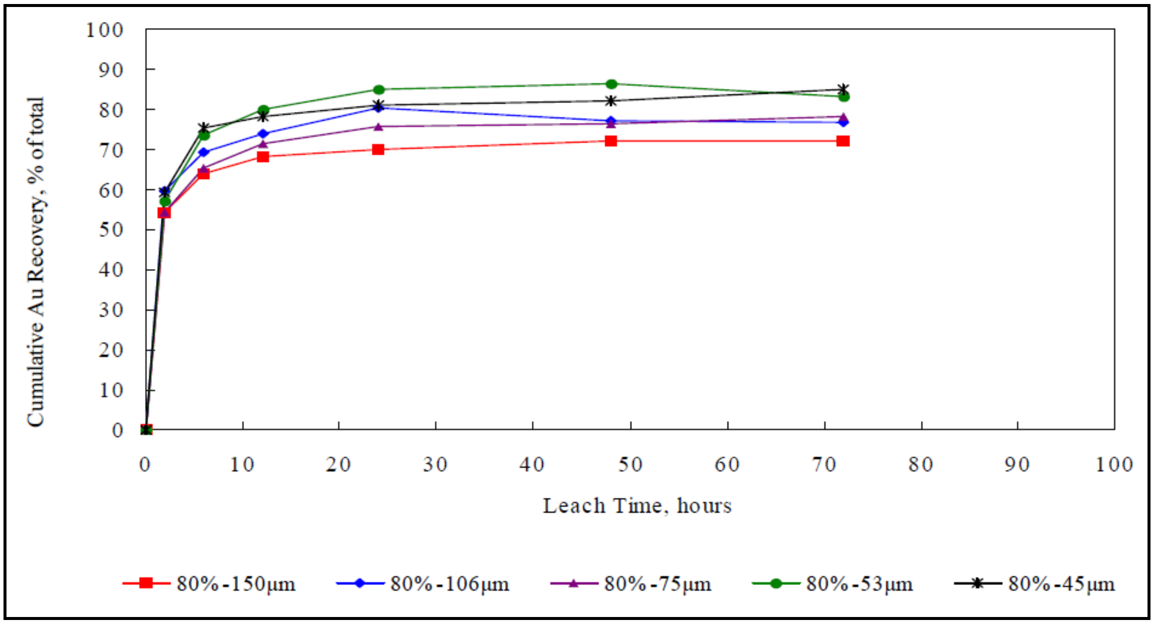


Figure 13.7 McClelland Gold Leach Profiles for Medium Grade Sulphide Composite (MGS)



Additional bottle roll tests were conducted on all the composites to optimize the cyanide dosing rates. The oxide composites were tested at three grind sizes (-212 µm, -106 µm, and -53 µm) and four NaCN concentrations (0.5, 1.0, 1.5, and 2.0 g/L). The sulphide composites were tested at two grind sizes (-75 µm and 53 µm) and at the same four NaCN concentrations as the oxides. The results are summarized in Table 13.13.

Table 13.13 McClelland Grind Size and Cyanide Concentration Optimization Tests

Ore Type	Grind Size	g/L NaCN	No. of Tests	Gold Extraction %	g/t Au			Reagents, kg/t	
					Extracted	Tail	Calc'd Head	NaCN	Lime Added[2]
Oxide	P_{100}-212 µm	0.5	2	93.4	1.18	0.07	1.25	<0.07	3.4
		1.0	4	93.7	1.26	0.07	1.33	0.15	3.2
		1.5	2	94.7	1.24	0.06	1.29	0.17	3.3
		2.0	2	91.8	1.23	0.08	1.31	0.21	3.2
Oxide	P_{100}-150 µm	0.5	2	94.1	1.20	0.06	1.26	0.09	3.5
		1.0	2	89.6	1.10	0.09	1.19	0.16	3.6
		1.5	2	94.7	1.23	0.06	1.29	0.19	3.6
		2.0	2	94.0	1.18	0.07	1.24	0.18	3.4
*Oxide	P_{100}-106 µm	1.0	2	93.0	1.19	0.08	1.27	0.08	3.1
Oxide	P_{100}-75 µm	0.5	2	94.1	1.13	0.07	1.20	0.09	4.0
		1.0	2	94.4	1.11	0.06	1.16	0.09	4.0
		1.5	2	95.1	1.09	0.05	1.14	0.17	3.7
		2.0	2	95.5	1.14	0.05	1.18	0.16	3.6
Oxide[1]	P_{100}-53 µm	1.0	2	94.1	1.13	0.07	1.20	0.09	4.0

Ore Type	Grind Size	g/L NaCN	No. of Tests	Gold Extraction %	g/t Au			Reagents, kg/t	
					Extracted	Tail	Calc'd Head	NaCN	Lime Added[2]
Sulphide[1]	P_{80}-150µm	1.0	2	75.3	1.06	0.32	1.38	0.47	1.0
Sulphide[1]	P_{80}-106µm	1.0	2	78.2	1.05	0.29	1.34	0.39	1.0
Sulphide	P_{100}-75µm	0.5	2	80.0	1.20	0.28	1.48	0.23	1.4
		1.0	4	79.6	1.10	0.27	1.37	0.5	1.1
		1.5	2	83.4	1.41	0.28	1.68	1.13	0.9
		2.0	2	79.4	1.11	0.27	1.38	1.88	0.8
Sulphide	P_{100}-53µm	0.5	2	82.3	1.16	0.25	1.40	0.47	1.5
		1.0	4	83.3	1.19	0.23	1.42	1.28	1.4
		1.5	2	80.8	1.14	0.25	1.39	1.52	1.0
		2.0	2	81.0	1.13	0.25	1.38	2.62	0.8
Sulphide[1]	P_{80}-45µm	1.0	2	84.4	1.28	0.24	1.52	2.36	1.7

[1] Results from Table 13.12 were also included for the sake of completeness and comparison.
[2] Testwork report does not specify lime purity or whether CaO or $Ca(OH)_2$

The results showed similar gold extraction percentage for oxide composites at the four cyanide concentrations, with an average of 94% gold extraction overall. It is expected that gold recoveries will not increase significantly with decreasing feed size or with increasing cyanide concentration. It was recommended that the near optimum conditions for oxide ore type samples were -150 µm feed size and 0.5 g/L NaCN.

For the sulphide composites, the results showed varying gold extraction percentages at the four cyanide concentrations, but no significant improvement beyond 1.0 g/L NaCN. Average cyanide and lime consumptions for the sulphide samples increased with decreasing feed size. It was recommended that the near optimum conditions for sulphide ore type samples were P_{80} of 53 µm feed size and 1.0 g/L NaCN.

The combined gravity concentration and gravity tails cyanidation results are presented in Table 13.14.

Table 13.14 McClelland Combined Gravity Concentration/Gravity Tails Cyanidation Tests

Sample	Feed Size	Gold Extraction %					Gold Grade, g/t Au					Reagents kg/t	
		Grav. Conc.	CN Leach	Combined	CN Tails	Total	Grav. Conc.	CN Leach	Tail	Calc'd Head	Assayed Head	NaCN	Lime Added
HGS	F_{80}-75 µm	30.9	54.2	85.1	14.9	100	0.61	1.07	0.30	1.98	1.76	0.30	1.1
HGS	F_{80}-53 µm	22.3	63.5	85.8	14.2	100	0.41	1.16	0.26	1.83	1.76	0.52	1.3
MGS	F_{80}-75 µm	14.0	69.5	83.5	16.5	100	0.10	0.52	0.12	0.74	0.80	0.34	1.1
MGS	F_{80}-53 µm	18.2	68.4	86.6	13.4	100	0.14	0.53	0.10	0.77	0.80	0.17	1.1
HGO	F_{100}-212 µm	6.2	89.7	95.9	4.1	100	0.10	1.5	0.07	1.67	2.50	0.04	2.9
HGO	F_{100}-150 µm	12.0	84.9	96.9	3.1	100	0.22	1.54	0.06	1.82	2.50	0.04	2.9
HGO	F_{100}-75 µm	12.8	82.8	95.6	4.4	100	0.21	1.38	0.07	1.66	2.50	0.07	3.1
MGO	F_{100}-212 µm	7.7	84.1	91.8	8.2	100	0.05	0.54	0.05	0.64	0.59	0.18	3.2
MGO	F_{100}-150 µm	10.2	82.6	92.8	7.2	100	0.07	0.55	0.05	0.67	0.59	0.07	4.4
MGO	F_{100}-75 µm	3.2	90.0	93.2	6.8	100	0.02	0.58	0.04	0.64	0.59	0.07	3.6

The oxide composites responded very well to combined gravity concentration treatment followed by gravity tails cyanidation at all three feed sizes evaluated, -212 µm, -150 µm, and -75 µm. Although, when compared to whole ore cyanidation tests at the same feed sizes, no significant improvement to the overall gold extraction was observed for the oxide composites.

The sulphide composites responded moderately well to the combined gravity concentration treatment followed by gravity tails cyanidation at the two feed sizes evaluated, P_{80} of 75 µm and P_{80} of 53 µm. When compared to whole ore cyanidation at the same feed sizes, the sulphide composites had relatively higher overall gold extraction between 2% to 5%. The results suggest a potential benefit to using gravity concentration treatment prior to a leaching circuit for the Bomboré sulphide ore.

Variability testwork for a milling/cyanidation process was conducted on 33 oxide, 5 transition, and 33 sulphide samples under the selected leach conditions deemed as near optimum. A summary of the results is presented in Table 13.15.

Table 13.15 McClelland Variability Test Summary for Milling/Cyanidation

Test Count	Gold Extraction %			Head Grade g/t Au			NaCN Consumption kg/t			Lime added kg/t		
	Avg.	Min.	Max.	Avg.	Min.	Max.	Avg.	Min.	Max.	Avg.	Min.	Max.
Oxide Ore Type, P_{80}-150 µm, 0.5 g/L NaCN												
33	90.4	77.1	97.3	1.25	0.38	7.15	0.21	0.07	0.43	2.2	0.7	5.6
Transition Ore Type, P_{80}-150 µm, 0.5 g/L NaCN												
5	91.5	87.5	95.7	1.02	0.66	2.01	0.10	<0.07	0.13	2.1	1	4.1
Transition Ore Type, P_{80}-53 µm, 1 g/L NaCN												
5	92.9	88.9	95.8	1.06	0.69	1.84	0.1	<0.07	0.1	3	1.2	5.2
Sulphide Ore Type, P_{80}-53 µm, 1 g/L NaCN												
33	82.9	39.9	96.4	1.44	0.39	4.22	1.14	0.3	5.11	0.8	0.6	1.0

The results showed that both oxide and transition variability samples were readily amenable to whole ore milling and cyanidation treatment, with average gold extraction of 90.4% and 92.2%, respectively. Most of the sulphide samples were also amenable to whole ore milling and cyanidation with an average gold extraction of 82.9%. Only one sulphide sample (1027558) performed poorly with a gold extraction of 39.9%. It was also observed that the gold leach rates were rapid with the extraction essentially complete in 24 hours. Only one transition and four sulphide samples out of the 71 samples exhibited slower leach rates. Cyanide consumptions were low for oxide and transition samples (<0.07 kg/t to 0.43 kg/t) but varied from low to high for the sulphide samples (0.3 kg/t to 5.11 kg/t). Lime additions varied substantially for the oxide and transition samples (0.7 to 5.6 kg/t) and were consistently low for the sulphide samples (0.6 kg/t to 1 kg/t).

CIL and CIP carbon adsorption capacity tests were also conducted and the results are presented in Figure 13.8 to Figure 13.11. However, no conclusions for carbon adsorption kinetics or equilibrium isotherms were possible from these results as these tests were conducted at too long of a period for determining the kinetics and too short of a period for determining the equilibrium isotherms.

Figure 13.8 McClelland Carbon-In-Leach Adsorption Capacity for Oxides

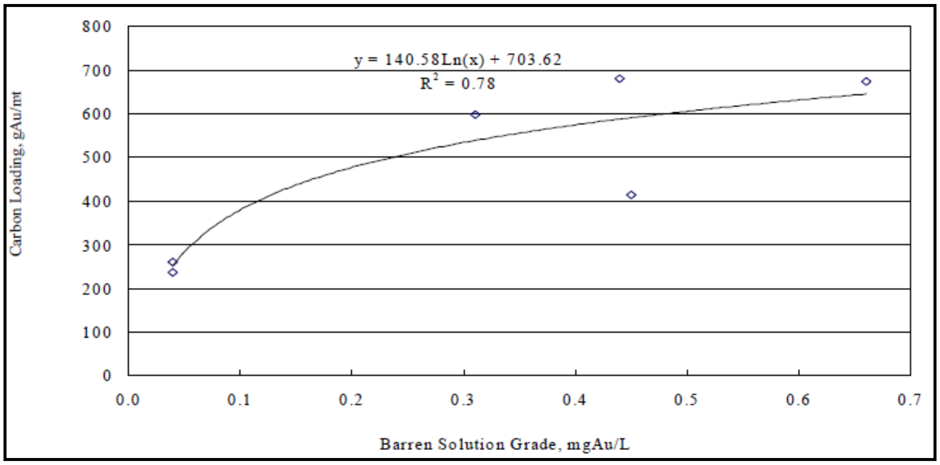


Figure 13.9 McClelland Carbon-In-Leach Adsorption Capacity for Sulphides

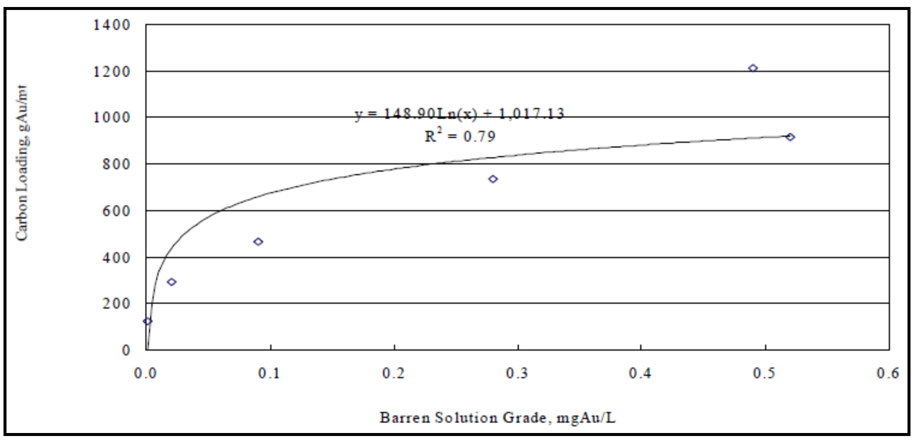


Figure 13.10 McClelland Carbon-In-Pulp Adsorption Capacity for Oxides

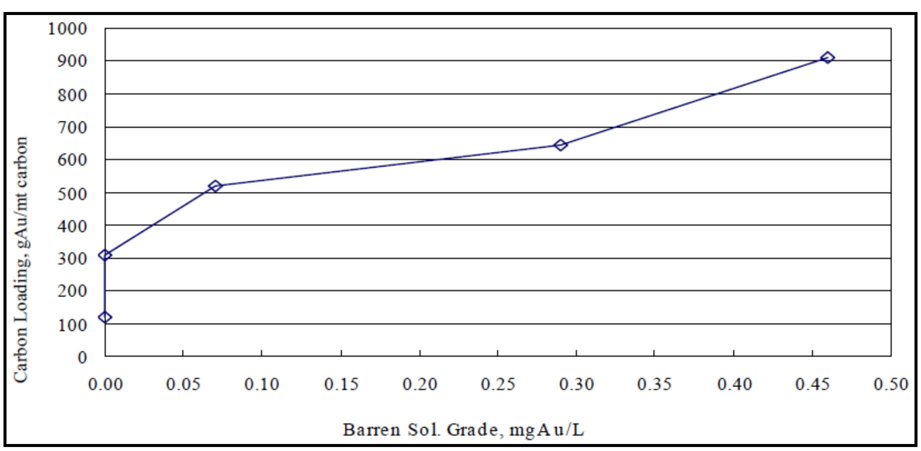
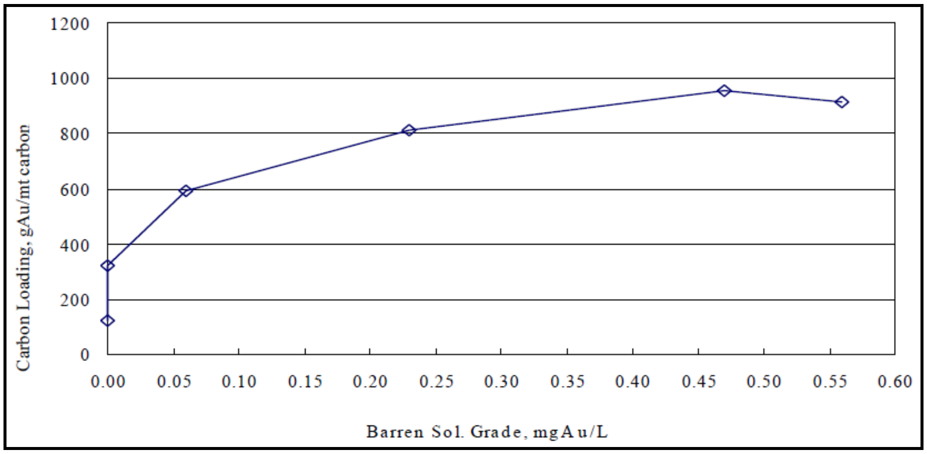


Figure 13.11 McClelland Carbon-In-Pulp Adsorption Capacity for Sulphides



Standard cyanide destruction testing was conducted on both oxide and sulphide slurries using SO_2/Air treatment with SMBS used as the source of SO_2. A summary of the results is shown in Table 13.16.

Table 13.16 McClelland SO_2/Air Treatments Results for P_{80} of 150 µm Feeds

Sample	Stream	Retention Minutes	Run #	Sol'n Analyses, mg/L		Molar Ratio, SO_2/CN		g/g CN_{WAD} in Feed			g SO_2/ g CN_{WAD} Treated	kg/t		SMBS Utilization Efficiency %
				Slurry pH	CN_{WAD}	Target	Actual	SMBS	SO_2	$CuSO_4$		SMBS	Lime	
Oxide Master	Feed	95.1	1st	10.2	273*									
	Effluent			8.5	0.024	3:1	2.3:1	6.86	5.57	0.58	5.57	3.51	8.93	44.2
	Feed	95.1	2nd	8.5	238									
	Effluent				0.036	3:1	2.7:1	8.12	6.59	0.69	6.60	3.51	5.95	37.3
	Feed	95.1	3rd	8.6	178									
	Effluent				0.034	3:1	4.3:1	13.08	10.63	1.11	10.63	3.51	5.36	23.2
Sulphide Master	Feed	95.1	1st	10.6	200									
	Effluent			8.5	0.28	3:1	4.9:1	14.87	12.08	0.83	12.09	5.31	8.12	20.4
	Feed	95.1	2nd	8.6	158									
	Effluent				0.42	3:1	6.0:1	18.3	14.86	1.03	14.9	5.31	14.88	16.5

**Calculated based on dilution of final preg WAD concentration*

Pocock Industrial conducted solids-liquids separation tests and flocculant screening as part of the McClelland program for both the oxide and sulphide slurries. The results are summarized in Table 13.17 and Table 13.18.

Table 13.17 McClelland Program – Pocock Conventional Thickener Recommended Design Parameters

Material Tested		Flocculant			Min. Unit Area at Specified Feed Solids Conc. & U/F Density (m^2/t/d) (3)					Max. U/F Solids Conc. (%)
		Type	Dose (g/t)	Conc. (g/L) (2)	10% Feed[1]	15% Feed[1]	20% Feed[1]	25% Feed[1]	30% Feed[1]	
Oxide	CY-142 Treated Slurry	Hychem AF303	25-30	0.1-0.2	0.262	0.311	--	--	--	52%
	CY-149 Treated Slurry	Hychem AF303	20-25	0.1-0.2	--	--	0.213	0.245	--	66%
Sulphide	CY-150 Leach Residue	Hychem AF303	50-60	0.1-0.2	0.191	0.283	0.237	--	--	54%
	CY-151 Leach Residue (w/o lime)	Hychem AF303	65-75	0.1-0.2	--	--	--	0.205	0.249	66%

Notes:
(1) Recommended thickener feed solids concentration range by weight.
(2) Recommended flocculant concentration prior to contact with the pulp.
(3) Unit Area includes a 1.25 scale-up factor. The range of unit areas provided corresponds to the range of feed solids concentration and underflow densities shown. Typically, conventional thickener sizing of less than 0.125 m^2/t/d is impractical due to rise rate limitations in full-scale industrially sized equipment.

Table 13.18 McClelland Program – Pocock High Rate Thickener Recommended Design Parameters

Material Tested		Tested Feed Solids % (1)	Flocculant			Design Basis Net Feed Loading ($m^3/m^2/h$) (5)	Predicted O/F TSS Conc. Range (mg/L) (6)	Predicted U/F Density (7)
			Type	Dose (g/t) (3)	Conc. (g/L) (4)			
Oxide	CY-142 Treated Slurry	11.9	Hychem AF303	50-55	0.1-0.2	4	150-250	52%
	CY-149 Treated Slurry	25.3	Hychem AF303	20-25	0.1-0.2	2.54	198-376	66%
	CY-150 Leach Residue	13.8	Hychem AF303	40-45	0.1-0.2	3.97	150-250	54%
Sulphide	CY-151 Leach Residue (w/o lime)	24.2	Hychem AF303	70 - 80	0.1-0.2	--	339 - 1010	66%
	CY-151 Leach Residue (with lime)	18.1	Hychem AF303	30 - 35	0.1-0.2	3.26	150 - 250	64-66%

Notes:
(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) Recommended Hychem flocculant type. Flocculants from other manufacturers with similar specifications would also serve.
(3) Recommended flocculant dose in grams per metric tonne (g/t).
(4) Recommended flocculant concentration prior to contact with the pulp.
(5) Recommended design basis (not feed loading rate) in m^3 of feed slurry per hour per square metre of thickener area (m^3/m^2 h). This basis can be used to calculate the required thickener area based on the volumetric feed rate at the design solids concentration. The feed loading rates shown correspond to the feed solids concentrations shown in the table. Since hydraulic design bases are specified independent of solids tonnage, an operable feed solids concentration range is required to properly specify a thickener design 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 litre as measured using a 0.45 m septum.
(7) Maximum underflow solids concentration recommended based on viscosity considerations and experience with similar materials.

Comminution testwork in this program consisted of abrasion index tests on <12.5 mm crushed size and ball mill grindability tests at 150 µm closed screens. The results are presented in Table 13.19.

Table 13.19 McClelland Abrasion Index and Ball Mill Grindability Tests

Sample	Abrasion Index	BWi (kWh/t)
HGS	0.1035	15.22
MGS	0.0746	15
MGO	0.0666	5.71
HGO	Not Tested	4.13

Conclusions from the McClelland program were as follows:

- Gravity concentration prior to cyanidation could benefit gold extraction for sulphide composites.
- Oxide composites did not respond well to flotation treatment, but sulphide composites did.
- The optimum feed size for whole ore cyanidation of oxide composites was -212 µm to 150 µm.
- The optimum feed size for whole ore cyanidation of sulphide composites was a P_{80} of 53 µm.
- The optimum cyanide concentration for leaching of oxide composites was 0.5 g/L NaCN.
- The optimum cyanide concentration for leaching of sulphide composites was 0.5 to 1.0 g/L NaCN.
- Gold extraction rates generally were rapid and essentially completed in 24 hours for most samples.
- Almost all variability samples were amenable to whole ore cyanidation at optimized conditions.
- Conventional SO_2/air tails slurry treatment was effective in decreasing WAD CN to acceptable levels.

13.2.5 Phillips 2012 Testwork Program

Phillips Enterprises performed further comminution testwork on granodiorite, the most competent of all the lithologies, which represents about 2% of the oxide and 2.4% of the transition reserves. The results are understood to be a conservative indication of the work indices of these two ore types, as all other lithologies are expected to be softer. Three samples were submitted from PQ hole BBD075, at depths of 29.0 m to 32.5 m (oxide), 32.5 m to 37 m (bottom of the oxide profile) and 37.0 m to 40.0 m (transition zone). The crushing work index test and ball mill work index test results are presented in Table 13.20.

Table 13.20 Phillips Crushing Work Index and Ball Mill Grindability Tests

Sample	CWi (kWh/t)	BWi (kWh/t)
BBD075 (29.0-32.5m)	2.23	7.08
BBD075 (32.5-37.0m)	2.76	7.49
BBD075 (37.0-40.0m)	3.51	10.73

13.2.6 Orezone 2012 Scrubbing Testwork Program

Orezone conducted scrubbing testwork to study the grain size distribution and gold deportment of oxide ore from different ore zones. On average, 62% of the oxides were 100 µm, and 83% of the oxides were- 200 µm. The average LeachWell gold extraction was 92%. The results are summarized in Table 13.21.

Table 13.21 Orezone Summary of Cumulative Grain Size Distribution and Gold Deportment

Prospect	Fraction	Fraction Weight, Dry kg	Fraction Dry %Passing	Fraction Dry Cum% Passing	LeachWell Avg. g/t Au	Tails Avg. g/t Au	LeachWell + Tails Combined g/t Au	Gold Extraction %
Maga	+6.3mm	18.72	15.1%	100.0%	0.829	0.058	0.887	93%
	+.200mm	30.93	24.9%	84.9%	0.618	0.020	0.638	97%
	+.100mm	5.53	4.4%	60.0%	0.667	0.047	0.714	93%
	-.100mm	69.05	55.6%	55.6%	0.617	0.042	0.659	94%
CFU	+6.3mm	2.97	16.2%	100.0%	1.407	0.161	1.568	90%
	+.200mm	3.31	18.1%	83.8%	1.009	0.044	1.053	96%
	+.100mm	0.30	1.6%	65.7%	Not Tested			
	-.100mm	11.73	64.1%	64.1%	0.349	0.045	0.394	89%
P8P9	+6.3mm	30.69	20.1%	100.0%	2.533	0.505	3.037	83%
	+.200mm	31.66	20.8%	79.9%	1.069	0.063	1.131	94%
	+.100mm	8.00	5.2%	59.1%	1.408	0.071	1.479	95%
	-.100mm	82.18	53.9%	53.9%	0.715	0.062	0.777	92%
P11	+6.3mm	2.56	9.2%	100.0%	2.703	0.091	2.794	97%
	+.200mm	5.64	20.3%	90.8%	2.438	0.087	2.525	97%
	+.100mm	1.62	5.8%	70.5%	Not Tested			
	-.100mm	17.94	64.6%	64.6%	1.118	0.061	1.179	95%
Siga E	+6.3mm	12.61	19.4%	100.0%	1.283	0.034	1.317	97%
	+.200mm	12.01	18.5%	80.6%	1.422	0.034	1.456	98%
	+.100mm	4.73	7.3%	62.0%	Not Tested			
	-.100mm	35.51	54.8%	54.8%	0.487	0.024	0.511	95%
Siga W-S	+6.3mm	14.89	18.6%	100.0%	0.429	0.019	0.448	96%
	+.200mm	15.26	19.1%	81.4%	0.868	0.057	0.925	94%
	+.100mm	8.51	10.7%	62.3%	0.625	0.056	0.681	92%
	-.100mm	41.22	51.6%	51.6%	0.569	0.051	0.620	92%
P16	+6.3mm	2.13	9.6%	100.0%	0.732	0.050	0.782	94%
	+.200mm	4.32	19.4%	90.4%	1.453	0.047	1.500	97%
	+.100mm	2.23	10.0%	71.0%	Not Tested			
	-.100mm	13.53	60.9%	60.9%	0.429	0.030	0.459	93%
ALL	+6.3mm	84.56	17.3%	100.0%	1.521	0.215	1.735	88%
	+.200mm	103.14	21.1%	82.7%	1.034	0.046	1.080	96%
	+.100mm	30.92	6.3%	61.7%	0.812	0.059	0.870	93%
	-.100mm	271.15	55.4%	55.4%	0.633	0.048	0.681	93%

13.2.7 SGS 2013 Testwork Program

Twenty-six (26) sulphide composite samples of various lithologies were submitted to SGS for comminution testing. A statistical analysis of the results is presented in Table 13.22.

Table 13.22 SGS 2013 Comminution Test Statistics for Sulphides

Statistics	JK Values		Rel. Density**		Work Indices (kWh/t)		
	A x b (SMC)	DWi kWh/m^3	CWi [50]	SMC [21]	CWi	RWi	BWi
Results Available	26	26	4	26	4	5	26
Average	41.5	7.6	2.72	2.81	10.7	15.5	15.2
Standard Deviation	13.2	2.8	0.26	0.11	2.8	1.0	2.0
Rel. S.D. (%)	32	37	10	4	26	6	13
Min	61.7	4.4	2.48	2.7	6.8	14.0	10.8
10th Percentile	57.0	4.8	-	2.72	-	-	12.8
25th Percentile	51.6	5.4	-	2.74	-	-	14.0
Median	45.7	6.2	2.65	2.77	11.4	15.7	15.3
75th Percentile	27.3	10.1	-	2.82	-	-	16.5
90th Percentile	24.9	11.3	-	2.97	-	-	17.5
Max	24.0	13.0	3.09	3.18	13.3	16.3	19.0

** Min and Max refer to Softest and Hardest for the tests.*
*** The density values were measured by a water displacement technique. The number in parentheses refers to the geometrical mean (expressed in mm) of the rocks used for the measurement.*

The SMC 'A x b' values ranged from moderately soft to very hard. The other comminution indices, namely the CWi, RWi and BWi, respectively, demonstrated less variability and were placed in the soft to moderately hard range of the SGS database. Based on the bimodal ore hardness observed in the coarse size fraction, it was recommended by SGS to evaluate the test results with respect to a mine plan and by depth in order to better understand the ore hardness variability within the deposit.

13.2.8 COREM 2013 Testwork Program

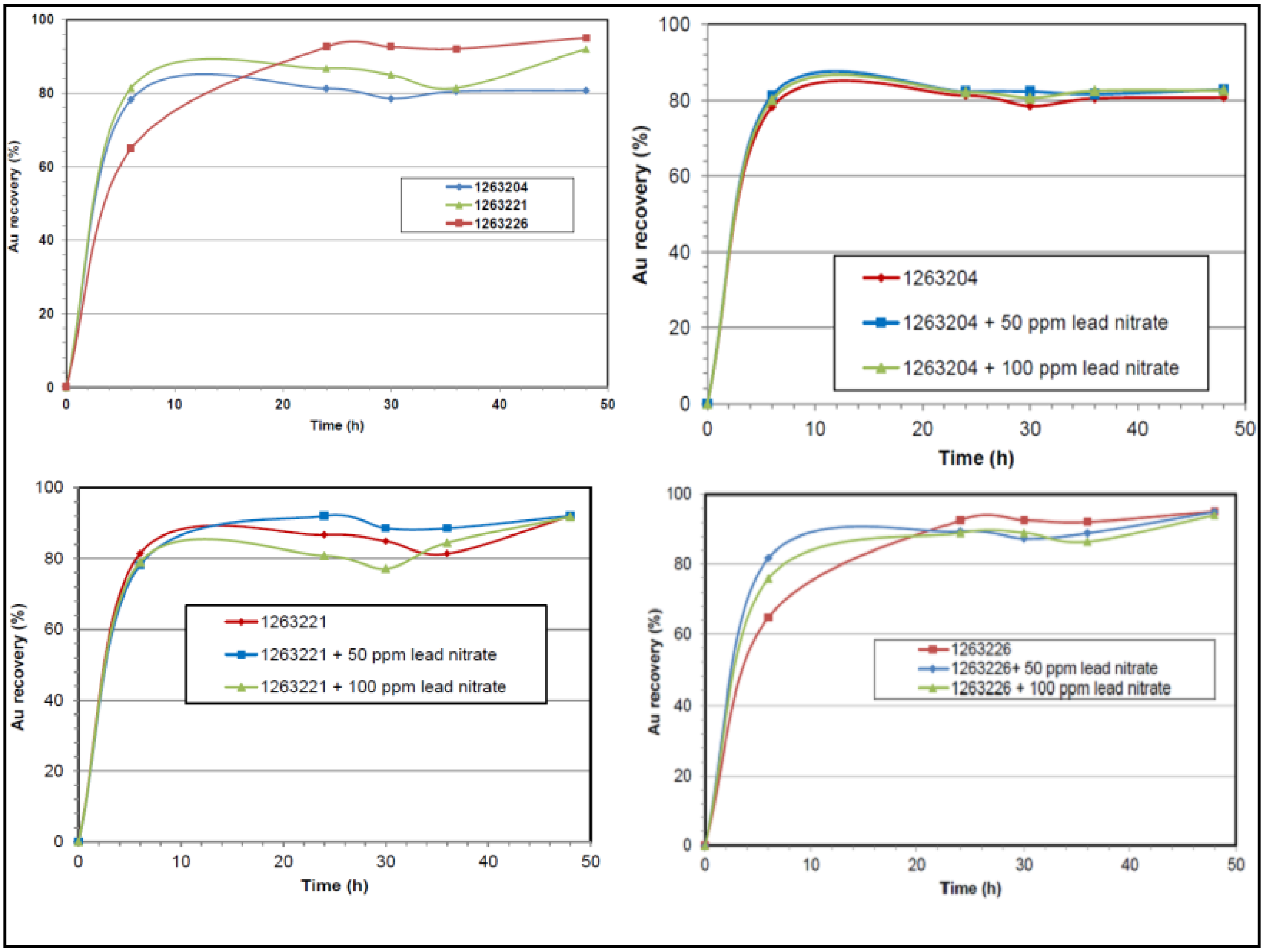
Five (5) of the 26 samples from SGS program were sent to COREM for further testing to assess the impact of lead nitrate on cyanidation of pyrrhotite-rich samples. Three of the five samples were determined to be pyrrhotite-rich, hence, chemical analyses for sulphur and arsenic were performed to estimate the levels of sulphide mineral phases for each sample. The results are shown in Table 13.23.

Table 13.23 COREM Mineralogical Analysis of High Pyrrhotite Samples

Analysis		Sample #1263204 (49 min. grind)	Sample #1263221 (49 min. grind)	Sample #1263226 (49 min. grind)
Chem.	As (%)	0.17	<0.01	0.26
	S (%)	1.20	2.53	1.17
Modal	Pyrite (%)	0.20	4.60	Not Detected
	Pyrrhotite (%)	2.90	0.20	2.85
	Arsenopyrite (%)	0.40	<0.02	0.57

The leach curves for the pyrrhotite-rich samples are presented in Figure 13.12.

Figure 13.12 COREM Leach Curves for Pyrrhotite-Rich Samples and Impact of Lead Nitrate

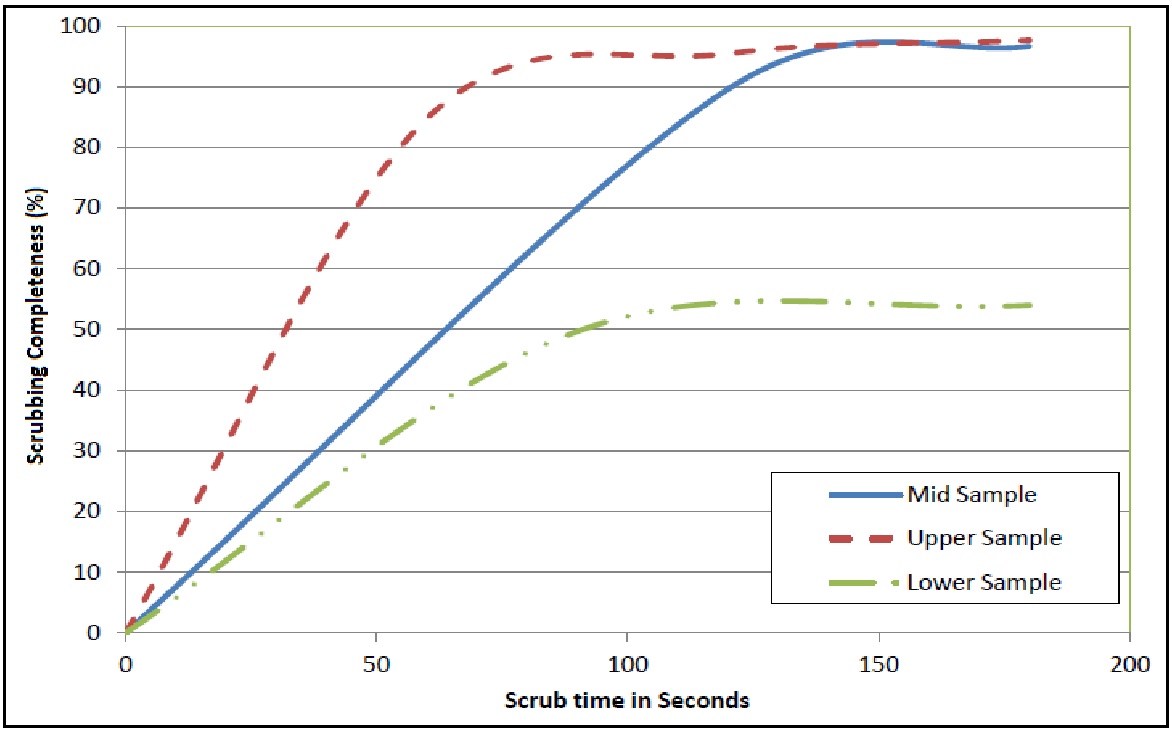


A comparative analysis of gold dissolution profiles for samples containing pyrrhotite showed that the pyrrhotite content of the tested samples was not the only factor controlling the gold recovery. Sample #1263226 showed the highest gold recovery (92.6%) despite its high content of pyrrhotite (2.85%). The addition of 50 ppm lead nitrate enhanced the overall gold extraction for sample #1263204 by about 2%. The use of lead nitrate significantly improved the gold extraction for sample #1263221 from 88.6% to 92% after 24 hours of cyanidation, however, no improvement after 48 hours. The testwork for sample #1263221 may not be as reliable because the results fluctuated significantly throughout the leach period. Some improvements on gold extraction were observed for sample #1263226, but only during the first 6 hours of cyanidation. Increasing lead nitrate concentration to 100 ppm was counterproductive for all the tested samples.

13.2.9 Met-Solve 2013/2014 Testwork Program

Oxide samples were submitted to Met-Solve for scrubbing testwork. The estimated scrubbing completeness for <75 mm size at 50% pulp density as a function of time for different ore zones (upper, middle and lower) is presented in Figure 13.13.

Figure 13.13 2013 Met-Solve Scrubbing Test - Scrubbing Completeness % versus Time



For scrubbing to be complete, a minimum of 95% completeness is required. As seen in Figure 13.13, the upper and middle oxide samples met this requirement, however, the lower oxide sample failed to scrub satisfactorily.

Note that material from the middle zone initially did not scrub successfully at 50% and 33% pulp density and it was only after pre-shredding the material to <75 mm that the scrubbing test was successful. Cyanide leach tests were also conducted on the screened undersize at -150 µm from the upper, middle and lower ore zones. The leach curves at different cyanide concentrations are shown in Figure 13.14. The cyanide and lime consumptions are shown in Table 13.24.

Figure 13.14 Met-Solve Leach Curves at Different Cyanide Concentrations

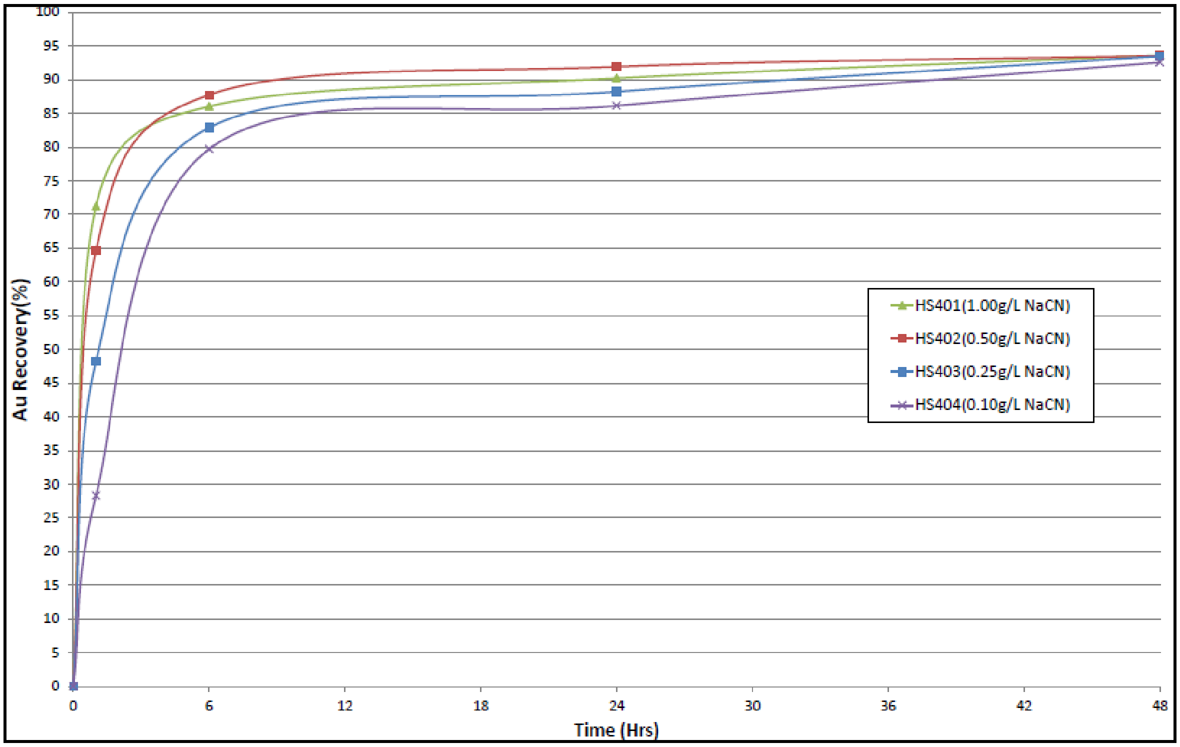


Table 13.24 Met-Solve Cyanide Leach Results on Screened U/S (-150 µm)

Test No.	Pulp Density %	g/L NaCN Conc.	Au Extraction %				Residue Assay (g/t Au)	Head Grade		Reagents Req'd (kg/t)	
			1 hr	6 hrs	24 hrs	48 hrs		Calc'd	Direct	NaCN Consumed	CaO Added
HS401	33	1	71	86	90	94	0.05	0.83	0.76	0.4	1.31
HS402	33	0.5	65	88	92	94	0.06	0.85		0.33	1.74
HS403	33	0.25	48	83	88	93	0.05	0.77		0.08	1.76
HS404	33	0.1	28	80	86	93	0.06	0.8		0.04	1.59

The cyanidation results showed that it was possible to achieve 92.6% gold extraction at cyanide concentration of 0.1 g/L NaCN, hence yielding NaCN consumption of only 0.04 kg/t for this sample. Increasing the cyanide concentration improved leach kinetics within the first 6 hours only but had minimal overall gold extraction improvement.

13.2.10 KCA 2014 Heap Leach (HL) Test Program

Scrubbed material from Met-Solve was shipped to the KCA laboratory in 2014 where a global composite was formed and used for further metallurgical testing. KCA also received 26 drums of oxide and transition samples, with eight used to form four composites, and the other 18 to form 12 separate composites. This program was developed to complete heap leach testwork on 17 samples as shown in Table 13.25.

Table 13.25 KCA Generated Composites for Heap Leach Test Program

KCA Sample No.	Met ID	Testwork	Lithology	Type	Position	Grade Bin	Crush Size, mm
70405	Met-Solve	Agglom./Column	Saprolite	Oxide	U+L	AVG	25
70401	BHK-01	Compacted Permeability	S3+I2	Oxide	Upper, Lower	LG+AVG	25
70402	BHK-02	Compacted Permeability	MI3	Oxide	Upper, Lower	LG+AVG	25
70403	BHK-03	Compacted Permeability	S3+I2+MI3	Transition	Upper	LG+AVG	12.5
70404	BHK-04	Compacted Permeability	S3+I2+MI4	Transition	Lower	LG+AVG	12.5
70406	BHK-05	Column Leach	S3+I2	Oxide	Upper	LG	25
70407	BHK-06	Column Leach	S3+I2	Oxide	Upper	AVG	25
70408	BHK-07	Column Leach	S3+I2	Oxide	Lower	LG	25
70409	BHK-08	Column Leach	S3+I2	Oxide	Lower	AVG	25
70410	BHK-09	Column Leach	MI3	Oxide	U+L	LG	25
70411	BHK-10	Column Leach	MI3	Oxide	U+L	AVG	25
70412 A	BHK-11	Column Leach	S3+I2	Transition	Upper	LG+AVG	25
70412 B		Column Leach	S3+I2	Transition	Upper	LG+AVG	12.5
70413 A	BHK-12	Column Leach	S3+I2	Transition	Lower	LG+AVG	25
70413 B		Column Leach	S3+I2	Transition	Lower	LG+AVG	12.5
70414 A	BHK-13	Column Leach	MI3	Transition	Upper	LG+AVG	25
70414 B		Column Leach	MI3	Transition	Upper	LG+AVG	12.5
70415 A	BHK-14	Column Leach	MI3	Transition	Lower	LG+AVG	25
70415 B		Column Leach	MI3	Transition	Lower	LG+AVG	12.5
70416 A	BHK-15	Column Leach	S4	Transition	Upper	LG+AVG	25
70416 B		Column Leach	S4	Transition	Upper	LG+AVG	12.5
70417 A	BHK-16	Column Leach	S4	Transition	Lower	LG+AVG	25
70417 B		Column Leach	S4	Transition	Lower	LG+AVG	12.5

Head analyses were conducted on all samples except for KCA Sample No's. 70401 to 70404. Table 13.26 provides a summary of the gold and silver head assays.

Table 13.26 KCA HL Program – Summary of Gold & Silver Head Analyses

KCA Sample No.	Description	Material Type	Average Assay, g/t Au	Average Assay, g/t Ag	Wt. Avg. Screen Head g/t Au	Wt. Avg. Screen Head g/t Ag
70405	Met-Solve	Oxide	--	--	1.487	0.40
70406	BHK-05	Oxide	0.467	0.51	0.432	0.76
70407	BHK-06	Oxide	0.922	0.41	0.825	0.55
70408	BHK-07	Oxide	0.531	0.62	0.610	0.64
70409	BHK-08	Oxide	1.826	0.51	1.682	0.57
70410	BHK-09	Oxide	0.463	0.79	0.412	1.09
70411	BHK-10	Oxide	1.308	0.51	1.265	0.60
70412 A	BHK-11	Transition	--	--	0.962	0.44
70412 B	BHK-11	Transition	0.685	0.31	0.689	0.39
70413 A	BHK-12	Transition	--	--	0.937	0.47
70413 B	BHK-12	Transition	1.030	0.41	0.955	0.43
70414 A	BHK-13	Transition	--	--	0.790	0.55
70414 B	BHK-13	Transition	0.855	0.51	0.670	0.55
70415 A	BHK-14	Transition	--	--	1.134	0.65
70415 B	BHK-14	Transition	1.190	0.62	1.656	0.63
70416 A	BHK-15	Transition	--	--	1.114	0.61
70416 B	BHK-15	Transition	1.426	0.62	1.070	0.58
70417 A	BHK-16	Transition	--	--	1.144	0.57
70417 B	BHK-16	Transition	1.123	0.41	1.396	0.54

Agglomeration and compacted permeability testwork was conducted on the Met-Solve global composite and the other four composites, BHK-01 to BHK-04, to assess the permeability of material under different cement agglomeration levels. The results are summarized in Table 13.27, and Figure 13.15 to Figure 13.16.

Table 13.27 KCA HL Program – Summary of Primary Compacted Permeability Testwork

KCA Sample No.	KCA Test No.	Description	Phase	Crush Size, P_{100}, mm	Cement Added, kg/t	Effective Height, m	Flow Rate, L/h/m^2	% Slump	Pass/Fail
70401	70421 A	BHK-01	Primary	25	16	40	6	0	Fail
70401	70421 B	BHK-01	Primary	25	16	8	837	4	Pass
70401	70426 A	BHK-01	Primary	25	20	16	1,345	2	Pass
70401	70426 B	BHK-01	Primary	25	26	16	3,264	2	Pass
70402	70422 A	BHK-02	Primary	25	16	40	0	0	Fail
70402	70422 B	BHK-02	Primary	25	16	8	931	3	Pass
70402	70427 A	BHK-02	Primary	25	20	16	97	3	Fail
70402	70427 B	BHK-02	Primary	25	26	16	2,180	2	Pass
70403	70423 A	BHK-03	Primary	12.5	0	16	1,343	1	Pass
70403	70428 A	BHK-03	Primary	12.5	4	16	4,191	3	Pass
70404	70424 A	BHK-04	Primary	12.5	0	16	5,810	2	Pass
70404	70429 A	BHK-04	Primary	12.5	2	16	7,016	3	Pass
70405	70420	Met-Solve	Primary	25	0	40	88	2	Fail
70405	70425	Met-Solve	Primary	25	4	40	158	0	Pass
70405	70430 A	Met-Solve	Primary	25	4	8	1,184	1	Pass

Figure 13.15 KCA HL Program – Compacted Permeability Results for BHK-01 to BHK-04

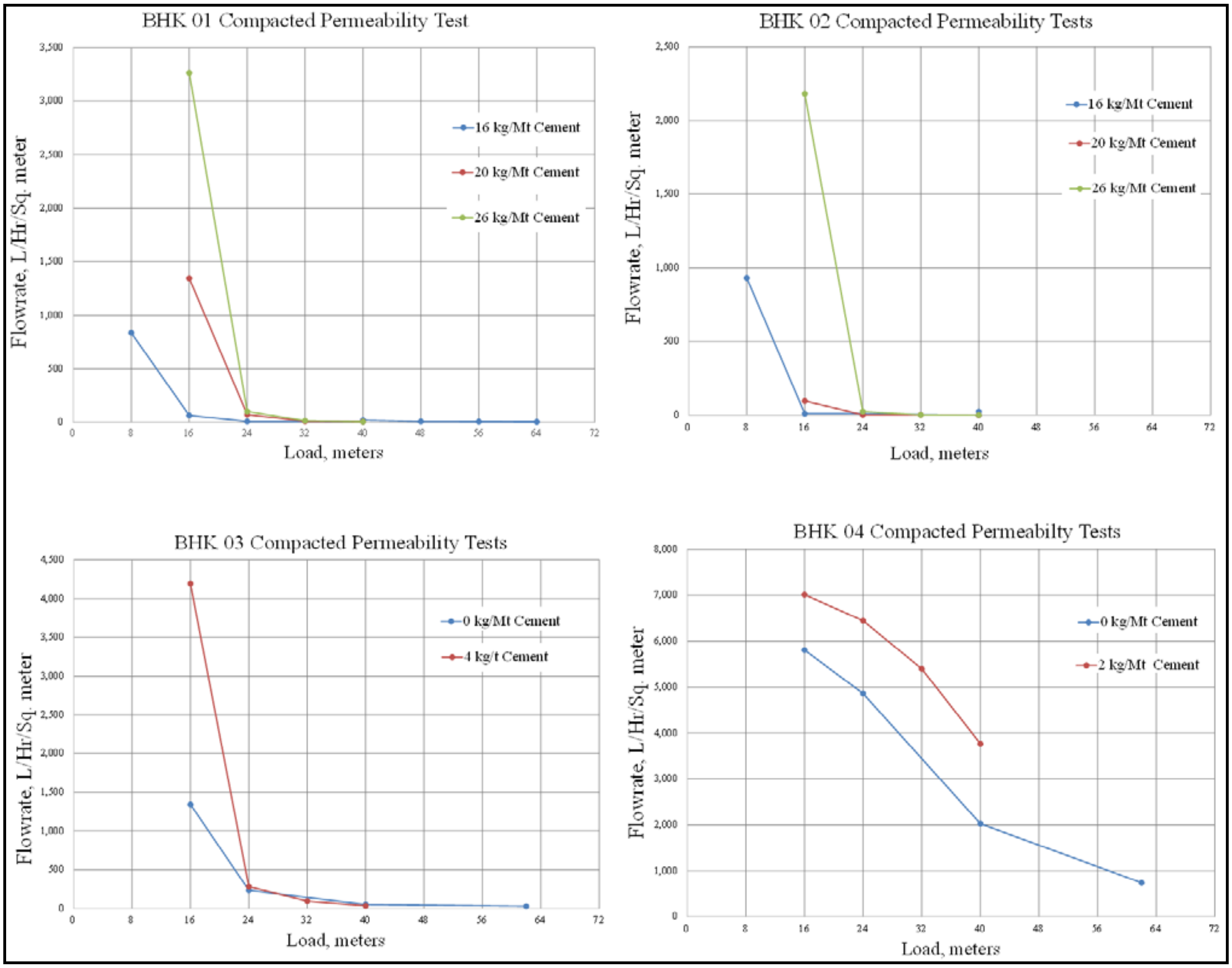
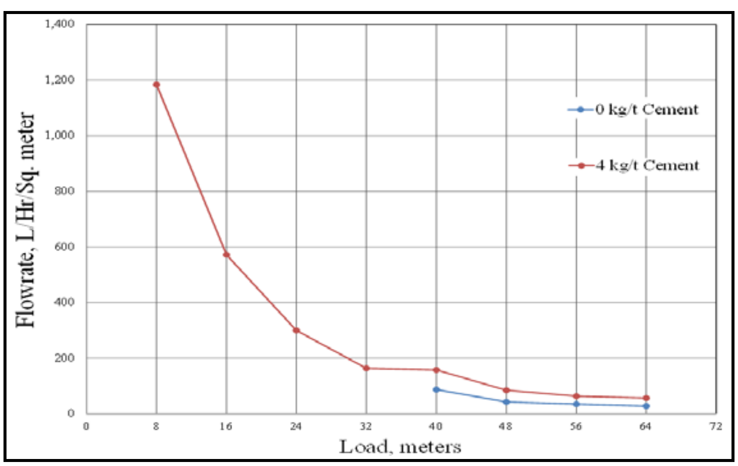


Figure 13.16 KCA HL Program – Compacted Permeability Results for Met-Solve Composite



The results showed all except four tests passing the agglomeration and compacted permeability criteria as set out by KCA. However, a large amount of cement was required to maintain the heap higher than 16 m for BHK-01 to BHK-04 tests. The Met-Solve material, which was previously scrubbed, required only a small amount of cement (4 kg/t) for obtaining an acceptable maximum heap height of 64 m.

Column leaching was also conducted for the Met-Solve composite and the other remaining samples. A summary of the results is shown in Table 13.28.

Table 13.28 KCA HL Program – Summary of Column Testwork - Gold

KCA Sample No.	Description	Oxidation State	Crush Size, mm	Calculated Head, g/t Au	Extracted, % Au	Calculated Tail P_{80}, mm	Days of Leach	Consumption NaCN, kg/t	Addition Cement, kg/t
70405	Met-Solve	Oxide	25	1.43	88%	11.6	75[1]	1.33	4.01
70406	BHK-05	Oxide	25	0.456	87%	1.13	44	0.34	19.64
70407	BHK-06	Oxide	25	0.834	93%	0.352	44	0.32	19.59
70408	BHK-07	Oxide	25	0.627	84%	5.05	44	0.31	19.60
70409	BHK-08	Oxide	25	1.540	86%	10.8	44	0.24	18.94
70410	BHK-09	Oxide	25	0.384	88%	1.47	44	0.38	25.32
70411	BHK-10	Oxide	25	1.173	86%	8.93	44	0.37	25.50
70412 A	BHK-11	Transition	25	0.930	71%	17.7	45	0.76	4.02
70412 B			12.5	0.761	78%	8.82	45	0.81	4.01
70413 A	BHK-12	Transition	25	0.873	77%	18.0	44	0.90	2.99
70413 B			12.5	0.812	82%	9.27	44	0.96	3.01
70414 A	BHK-13	Transition	25	0.786	74%	14.7	44	0.94	4.00
70414 B			12.5	0.750	83%	7.09	44	0.71	3.99
70415 A	BHK-14	Transition	25	1.255	78%	17.3	44	0.63	2.99
70415 B			12.5	1.618	81%	8.45	44	0.65	3.00
70416 A	BHK-15	Transition	25	1.207	68%	15.4	44	0.95	4.03
70416 B			12.5	1.023	87%	7.97	44	0.76	4.02
70417 A	BHK-16	Transition	25	1.110	69%	19.1	44	0.65	3.01
70417 B			12.5	1.326	75%	9.34	44	0.62	2.99

Note 1: Days 61 through 75 were water wash days.

Conclusions presented by KCA for the HL program are as follows:

- Gold extraction for oxide samples ranged from 84% to 93% with cyanide consumption ranging from 0.24 kg to 0.38 kg/t NaCN for BHK-05 to BHK-10, and 1.33 kg/t NaCN for the scrubbed material (Met-Solve).
- Gold extraction for transition samples (BHK-11 to BHK-16) ranged from 68% to 87% with cyanide consumptions ranging from 0.62 kg to 0.96 kg/t NaCN.
- Based on industry experience with clean ore, containing low cyanide soluble metals, cyanide consumptions for production heaps tend to be one third of the laboratory values. Ores at Bomboré appear to be clean with low amount of copper, hence fitting well with this assumption.

- Overall, gold extractions are acceptable; however, agglomeration of the finer ore portion requires high cement addition and hence high cost. This type of ore may best benefit from a combined process where the scrubbed oversize is heap leached and the scrubbed undersize is processed via a CIL circuit.

13.2.11 KCA 2014 Preliminary Hybrid Test Program

A preliminary hybrid program combining scrubbing, heap leaching and CIL was explored for Bomboré ore in this 2014 KCA program. The samples included the scrubbed material from Met-Solve designated as KCA Sample No. 70488 (global lithology composite), and its reject portions which were used to form a weighted composite designated as KCA Sample No. 71115.

The head analyses for the two samples are shown in Table 13.29.

Table 13.29 KCA Preliminary Hybrid Program – Head Analyses

KCA Sample No.	Description	Calc'd P_{80}, mm	Weighted Avg. Head Assay, g/t Au	Weighted Avg. Head Assay, g/t Ag
70488	Global composite	4.5	0.761	0.91
71115	Weighted Scrubbed Composite	11	0.901	0.88

Direct bottle roll leach tests were conducted on portions of the scrubbed material that were screened at 0.106 mm. Twelve (12) portions of the undersized scrubbed material (1,000 g each) were also subjected to CIL bottle roll tests. The results are summarized in Table 13.30 to Table 13.32.

Table 13.30 KCA Preliminary Hybrid Program – Summary of Direct Bottle Roll Tests

KCA Sample No.	Description	Screen Size, mm	Weight Distribution, %	Calculated Head, g/t Au	Au Extracted, %	Calculated Head, g/t Ag	Ag Extracted, %	Leach Time, hours	Consumption kg/t NaCN	Addition $Ca(OH)_2$, kg/t
71105 A	<0.212 mm	-0.212 +0.106	5.5%	1.069	74%	0.61	79%	24	0.01	1.75
71105 B	<0.212 mm	-0.106	94.5%	0.544	88%	0.54	72%	24	<0.01	2.50
	Overall Wt. Average			0.573	87%	0.54	72%			

Table 13.31 KCA Preliminary Hybrid Program – Summary of CIL Tests – Gold Results

KCA Sample No.	Description	Head Average, g/t Au	Calculated Head, g/t Au	Au Extracted, %	Leach Time, hours	Consumption kg/t NaCN	Addition $Ca(OH)_2$, kg/t
70490 A	<0.212 mm	0.668	0.570	53%	2	0.08	1.00
70490 A	<0.212 mm	0.668	0.596	75%	4	0.08	1.50
70490 A	<0.212 mm	0.668	0.671	74%	6	0.06	2.00
70490 A	<0.212 mm	0.668	0.534	82%	6	0.16	2.00
70490 A	<0.212 mm	0.668	0.695	75%	8	0.09	2.50
70490 A	<0.212 mm	0.668	0.693	76%	10	0.09	2.50
70490 A	<0.212 mm	0.668	0.606	87%	12	0.09	2.50
70490 A	<0.212 mm	0.668	0.545	84%	12	0.12	2.50
70490 A	<0.212 mm	0.668	0.629	86%	16	0.12	2.50
70490 A	<0.212 mm	0.668	0.638	85%	20	0.12	2.50
70490 A	<0.212 mm	0.668	0.624	87%	24	0.12	2.50
70490 A	<0.212 mm	0.668	0.594	87%	24	0.16	2.50

Table 13.32 KCA Preliminary Hybrid Program – Summary of CIL Tests – Silver Results

KCA Sample No.	Description	Head Average, g/t Ag	Calculated Head, g/t Ag	Ag Extracted, %	Leach Time, hours	Consumption kg/t NaCN	Addition $Ca(OH)_2$, kg/t
70490 A	<0.212 mm	0.79	1.09	29%	2	0.08	1.00
70490 A	<0.212 mm	0.79	1.06	39%	4	0.08	1.50
70490 A	<0.212 mm	0.79	1.04	38%	6	0.06	2.00
70490 A	<0.212 mm	0.79	0.93	59%	6	0.16	2.00
70490 A	<0.212 mm	0.79	0.94	42%	8	0.09	2.50
70490 A	<0.212 mm	0.79	0.95	44%	10	0.09	2.50
70490 A	<0.212 mm	0.79	0.80	49%	12	0.09	2.50
70490 A	<0.212 mm	0.79	0.84	66%	12	0.12	2.50
70490 A	<0.212 mm	0.79	0.92	55%	16	0.12	2.50
70490 A	<0.212 mm	0.79	0.83	63%	20	0.12	2.50
70490 A	<0.212 mm	0.79	0.74	58%	24	0.12	2.50
70490 A	<0.212 mm	0.79	0.94	78%	24	0.16	2.50

Fifteen (15) additional portions were later subjected to more CIL tests to generate slurry for Pocock Industrial Inc. (Pocock) to conduct solid/liquid separation testwork. The Pocock flocculant screening results and thickener sizing parameters are presented in Table 13.33 and Table 13.34.

Table 13.33 KCA Preliminary Hybrid Program – Pocock Flocculant Screening and Selection

Material	pH	Temp (°C)	Initial Solids Concentration of Slurry Tested	Minimum Effective Dose Range (g/t)	Flocculant Concentration Used (g/L)(1)	Flocculant Selected
Scrubbed Composite	8.25	20	10%	30-35	0.1	Hychem AF 304(2)
Leached Composite	10.42	20	10%	20-25	0.1	Hychem AF 304(2)

Notes:

(1) Flocculant solution concentration prior to contact with the pulp.

(2) Product selected (Hychem AF 304 is a medium to high molecular weight, 15% charge density, anionic polyacrylamide. Products meeting the dame description may also serve.

Table 13.34 KCA Preliminary Hybrid Program – Pocock Thickener Sizing Parameters

Sample	Floc. Type	Temp (C)	Floc Dose (g/t)	Floc Conc (g/L)	Max Thickener Feed Solids (%)	Min. Unit Area for Conv. Thick. Sizing ($m^2/t/d$)	Hydraulic Rate for High Rate Thick. Sizing ($m^3/m/h$)	Estimated U'Flow Density for Standard Thickener (%)	Thickener Type Recommended
Scrubbed Composite	Hychem AF 304	20	30-35	0.1-0.2	10%-15% (Conv. Type)	0.360-0.390 0.375 (Avg.)	---	48%-52%	Standard Conventional Type or High Rate
Leached Composite	Hychem AF 304	20	20-25	0.1-0.2	10%-15% (Conv. Type) 10%-12.5% (High Rate)	0.330-0.345 0.338 (Avg.)	4.00-4.80 4.40 (Avg.)	48%-52%	Standard Conventional Type or High Rate

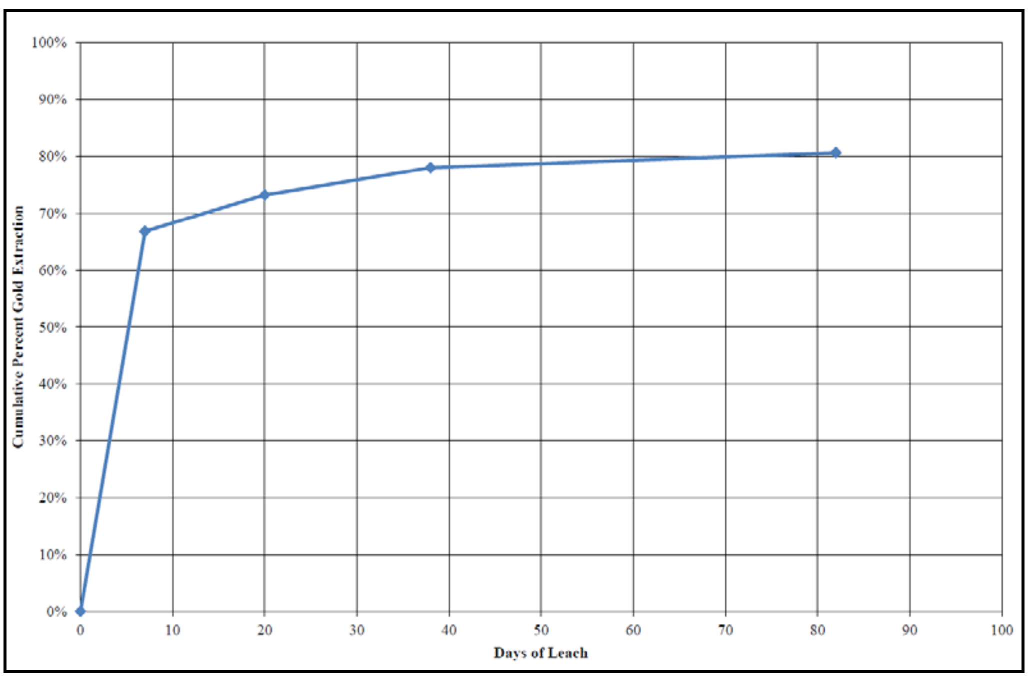
Finally, 10 additional portions were subjected to CIL tests to generate material for Golder Associates to conduct hydrometer testwork. The results indicated no other exceedance other than arsenic at 0.029 ppm in decant solution and 0.012 ppm in filtered solution against a drinking water guideline of 0.010 ppm (WHO & EPA).

The oversized scrubbed material designated as KCA Sample No. 70489 A, at P_{100} of 25 mm and >0.212 mm, were used for column leach testwork. The results are presented in Table 13.35 and Figure 13.17.

Table 13.35 KCA Preliminary Hybrid Program – Column Leaching for Oversized Scrubbed Material

KCA Sample No.	Calculated Head, g/t au	Extracted, Au %	Calculated Head, g Ag/t	Extracted, Ag %	Calculated Tail p80 Size, mm	Days of Leach	Consumption kg/tNaCN	Addition $Ca(OH)_2$, kg/t
70489 A	1.083	81%	0.88	40%	13.9	82	0.99	3.94

Figure 13.17 KCA Preliminary Hybrid Program – Oversized Scrubbed Material Column Leach Curve



Copper and mercury content in column leach solutions were analyzed and reported low, indicating that excess reagent consumption from cyanide soluble species is unlikely.

In summary, the overall gold and silver extraction from a hybrid process is estimated to be 85% and 65%, respectively. Refer to Table 13.36.

Table 13.36 KCA Preliminary Hybrid Program – Summary of Overall Gold & Silver Extraction

	Mass Split %	Au Extraction %	Ag Extraction %
Column Test	35.5%	81%	40%
CIL Test	64.5%	87%	78%
Total	100%	85%	65%

13.2.12 KCA 2014 Feasibility Study (FS) Hybrid Test Program

Results from the previous hybrid program were encouraging; hence, KCA conducted another hybrid program to meet a feasibility study requirement. KCA was provided with new core material that was representative of the resource's spatiality and grade for each ore type.

The scope of work for the program included the following:

- Sample characterization analyses:
  - Wet head screen analysis and scrubbing program.
- Agglomeration and compacted permeability tests:
  - Eight +212 µm samples, wet head screen analysis, physical characterization, agglomeration testwork, and compacted permeability testwork.
- Column leach tests:
  - Initial head assay characterization, wet head screen analysis with assays by size fraction, column leach testwork (-25 mm), and detox and environmental characterization.
- Direct bottle roll leach and CIL on undersized scrubber product:
  - Eight <212 µm samples, direct bottle roll and CIL leach tests at variable leach times, solid/liquid separation testwork for leached and scrubbed composites.

This KCA Hybrid FS program was extensive, and therefore, only high-level summaries of the results are provided. Refer to KCA Report No. KCA0140096_BOM03_02 for the complete results.

Head screen analysis and gold assay of the composite samples used in this program are shown in Table 13.37.

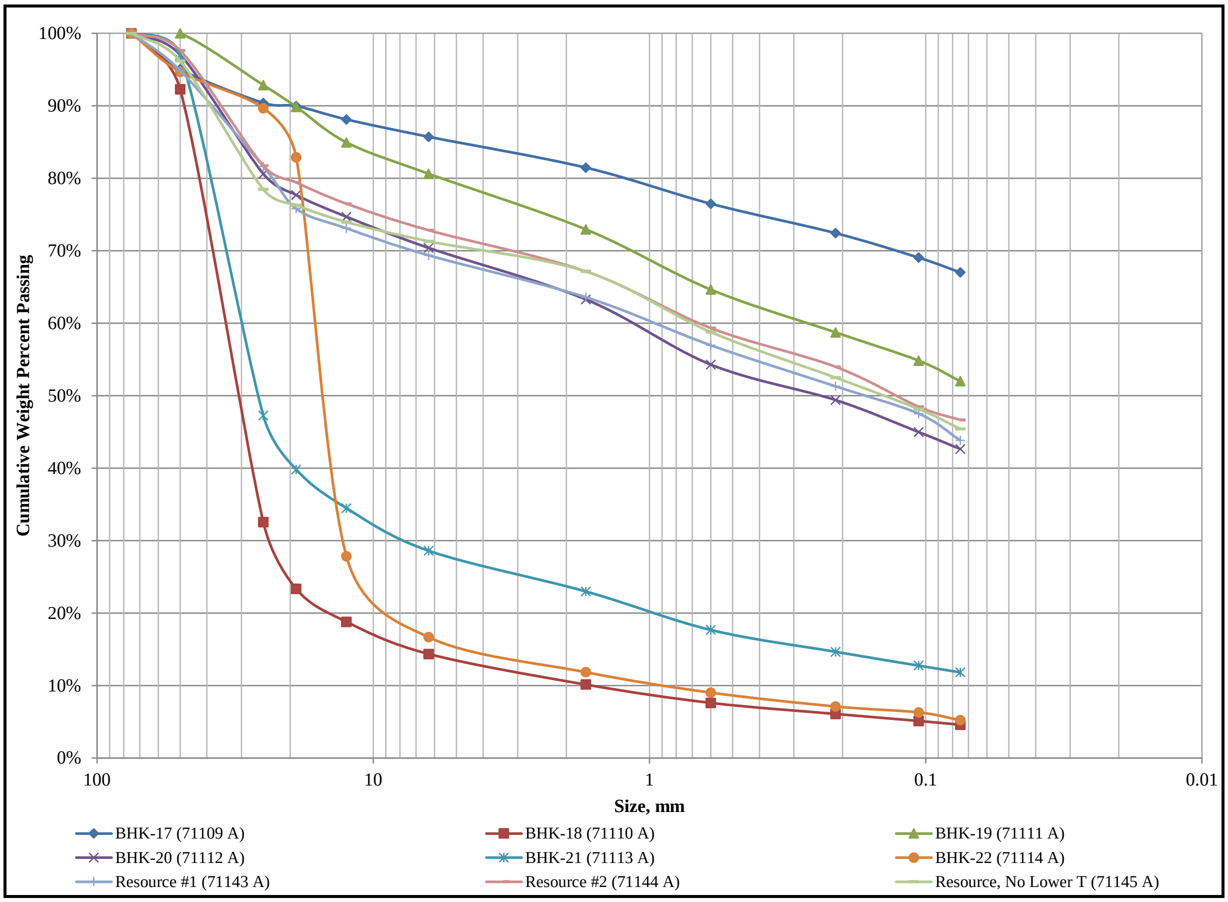
Table 13.37 KCA Hybrid FS – Summary of Head Screen Analyses

KCA Sample No.	Description	Oxide Sub-zone	Calc. P_{80}, mm	Weighted Avg. Head Assay, g/t Au	Weighted Avg. Head Assay, g/t Au
71109 A	BHK-17 High Fines	Ox_U	1.75	0.509	0.94
71110 A	BHK-18 Low Fines	Tr_L	48.31	0.687	1.46
71111 A	BHK-19	Ox_U	5.58	0.976	1.33
71112 A	BHK-20	Ox_L	20.46	0.943	1.21
71113 A	BHK-21	Tr_U	41.41	0.923	1.46
71114 A	BHK-22	Tr_L	19.59	1.052	1.50
71143 A	Resource #1	Ox_U/L, Tr_U/L	21.84	1.107	0.91
71144 A	Resource #2	Ox_U/L, Tr_U/L	18.04	1.219	1.15
71145 A	Resource, No Lower T	Ox_U/L, Tr_U	21.13	0.833	1.23

Ox_U/L: Upper/Lower Oxide, Tr_U/L: Upper/Lower Transition.

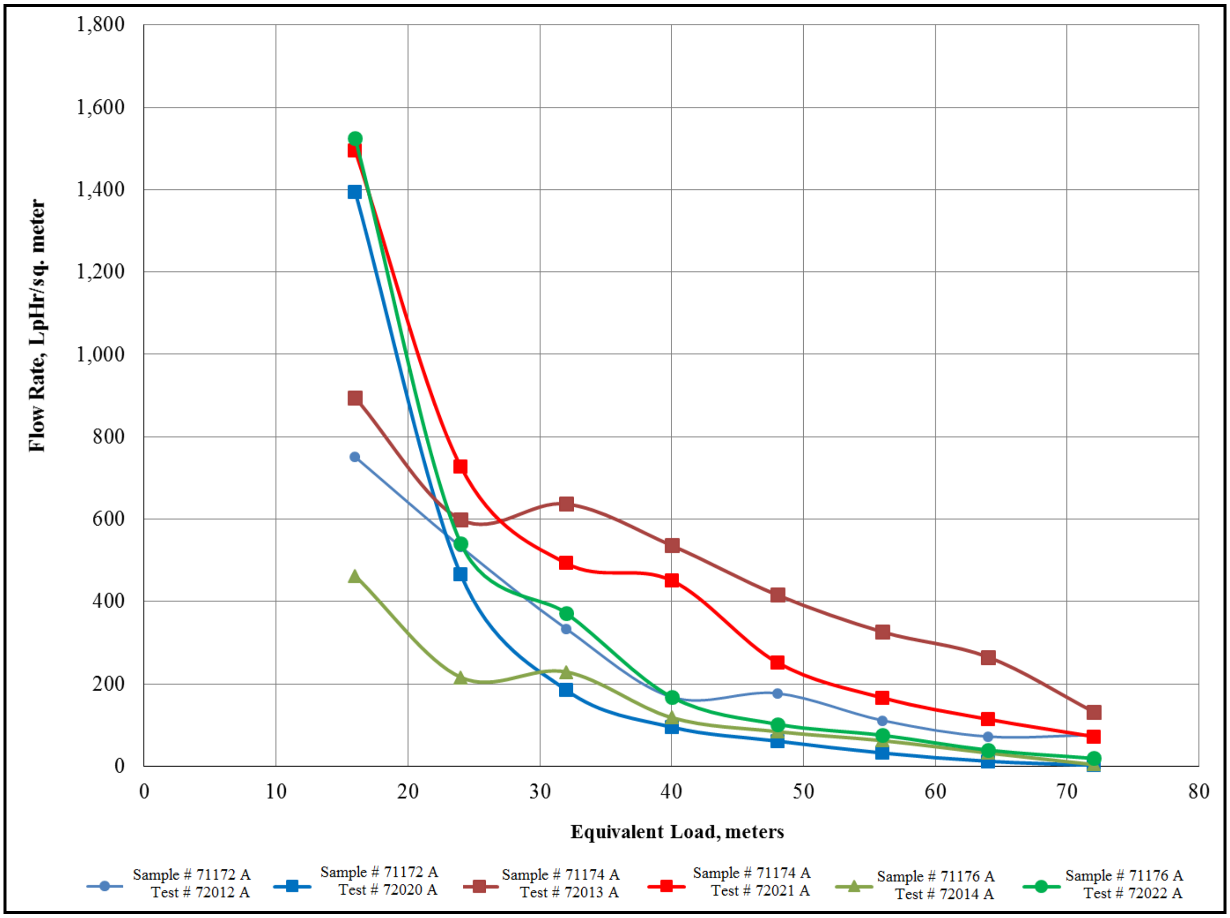
Particle size distributions of the samples are provided in Figure13.18.

Figure 13.18 KCA Hybrid FS – Particle Size Distribution of Samples



A summary of the compacted permeability test results is presented in Figure 13.19.

Figure 13.19 KCA Hybrid FS – Compacted Permeability Tests



A summary of the column leach gold extraction results is presented in Figure 13.20 and Figure 13.21.

Figure 13.20 KCA Hybrid FS – Column Test Leach Curves for BHK-17 to BHK-22

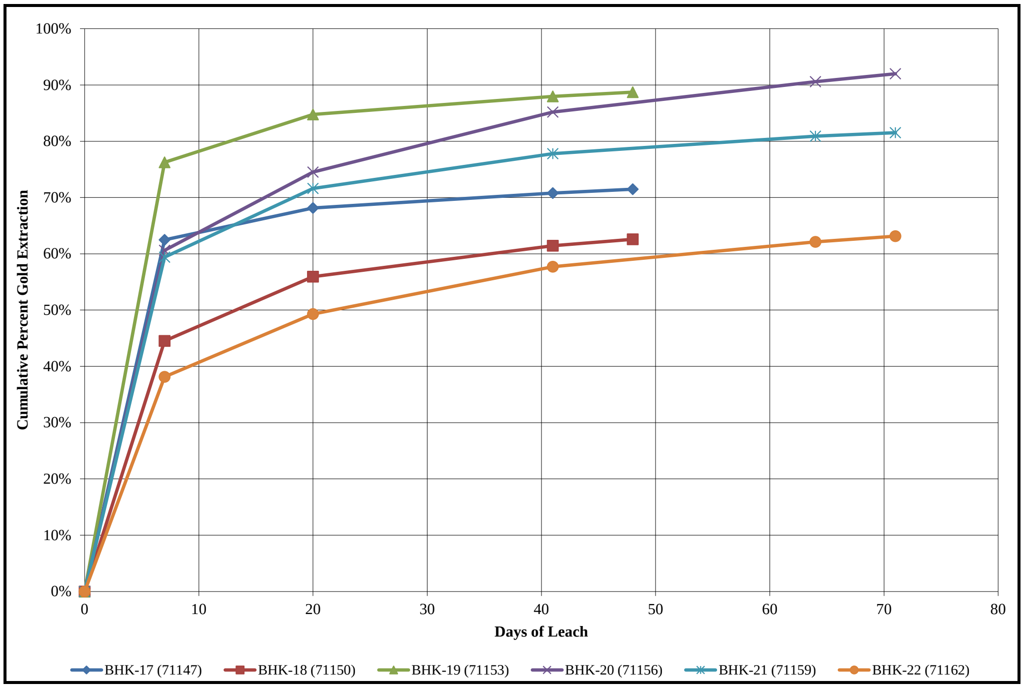
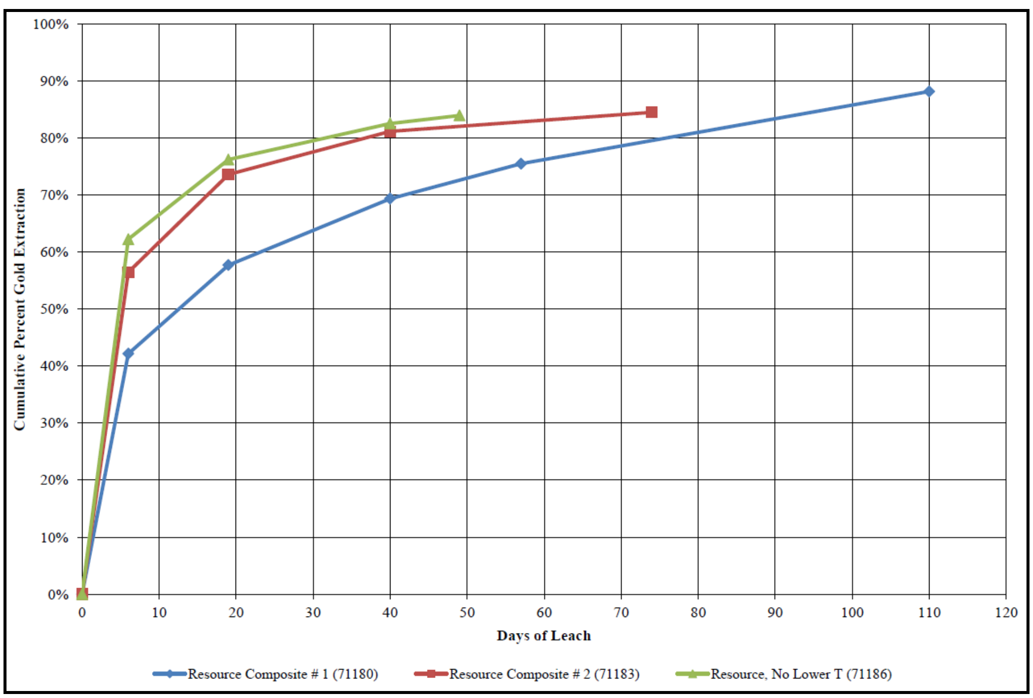
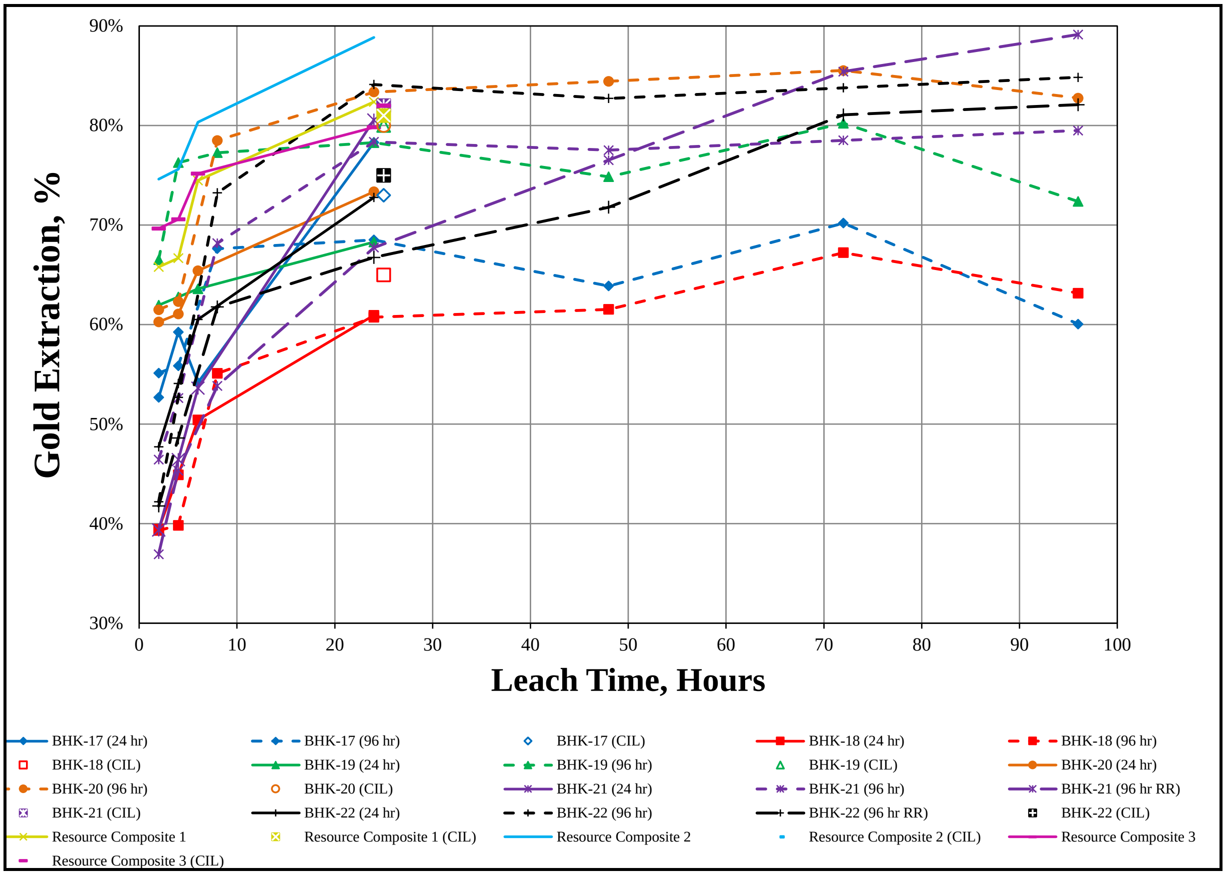


Figure 13.21 KCA Hybrid FS – Column Test Leach Curves for Resource Composites



A summary of all the bottle roll and CIL gold leach extraction curves is presented in Figure 13.22.

Figure 13.22 KCA Hybrid FS – Summary of Bottle Roll / CIL Leach Curves



The overall extraction of the material was calculated by utilizing the weights of the scrubber products and the scrubber solutions. The oversized material was represented in the column leach tests, the undersized material was represented by the 24-hour CIL bottle roll leach tests, and the scrubber solutions were weighed and assayed separately.

The overall extraction calculations are presented in Table 13.38 for all samples and composites.

Table 13.38 KCA Hybrid FS – Overall Extraction Calculation

KCA Test No.	Met ID	Oxide Sub-zone	Wt. Dry +0.212 mm, %	Wt. Dry -0.212 mm, %	Wt. Avg. Head Assay, g/t Au	Calc'd Head, g/t Au	Extraction +0.212mm, g/t Au	Extraction +0.212mm, % Au	Extraction -0.212mm, g/t Au	Extraction -0.212mm, % Au	Extraction Scrubber, g/t Au	Extraction Scrubber, % Au	Overall Extraction, % Au
71124	BHK-17	Ox_U Hi Fines	27.9%	72.1%	0.509	0.598	0.183	31%	0.125	21%	0.169	28%	80%
71125	BHK-18	Tr_L Low Fines	87.0%	13.0%	0.687	0.566	0.291	51%	0.027	5%	0.060	11%	67%
71127	BHK-19	Ox_U	34.8%	65.2%	0.976	1.001	0.467	47%	0.176	18%	0.253	25%	90%
71128	BHK-20	Ox_L	39.8%	60.2%	0.943	1.159	0.701	61%	0.192	17%	0.155	13%	90%
71129	BHK-21	Tr_U	76.8%	23.2%	0.923	0.905	0.546	60%	0.118	13%	0.092	10%	84%
71130	BHK-22	Tr_L	86.0%	14.0%	1.052	0.967	0.506	52%	0.082	8%	0.056	6%	67%
71169	Resource #1	Ox_U/L, Tr_U/L	49.3%	50.7%	1.107	1.177	0.697	59%	0.173	15%	0.173	15%	89%
71170	Resource #2	Ox_U/L, Tr_U/L	47.6%	52.4%	1.219	1.007	0.511	51%	0.179	18%	0.184	18%	87%
71171	Resource #3 , No Lower Transition	Ox_U/L, Tr_U	45.4%	54.6%	0.833	1.080	0.580	54%	0.185	17%	0.164	15%	86%

Ox_U/L: Upper/Lower Oxide, Tr_U/L: Upper/Lower Transition.

The results for the Pocock solid/liquid separation testwork are presented in Table 13.39 to Table 13.40.

Table 13.39 KCA Hybrid FS – Pocock Flocculant Screening and Selection

Material	pH	Temp (°C)	Initial Solids Concentration of Slurry Tested	Minimum Effective Dose Range (g/t)	Flocculant Concentration Used (g/L)(1)	Flocculant Selected
72007 - Leached Composite BK-17 High Fines	10.86	20	12.5%	30-35	0.1	Hychem AF 304(2)
71132 - Scrubbed Composite BK-17 High Fines	9.27	20	10%	40-45	0.1	Hychem AF 304(2)
72008 - Leached Composite Resource #1	10.63	20	12.5%	30-35	0.1	Hychem AF 304(2)
71173 - Scrubbed Composite Resource #1	9.73	20	10%	35-40	0.1	Hychem AF 304(2)

Notes:
(1) Flocculant solution concentration prior to contact with the pulp.
(2) Product select (Hychem AF 304) is a medium to high molecular weight, 15% charge density, anionic polyacrylamide. Products meeting the same description may also serve.

Table 13.40 KCA Hybrid FS – Pocock Thickener Design Parameters

Sample Name	Floc. Type	Temp (C)	Floc Dose(1) (g/t)	Floc Conc(2) (g/L)	Max Thk Feed Solids (%)(3)	Min. Unit Area for Conv. Thick. Sizing(4) (m^2/t/d)	Hydraulic Rate for High Rate Thick. Sizing(5) (m^3/m/h)	Estimated U'Flow Density for Standard Thickener (%)(6)	Thickener Type Recommended
72007-Leached Composite BK-17 High Fines	Hychem AF 304	20	30-35	0.1-0.2	15%-20% (Conv. Type) 10%-12.5% (High Rate)	0.260-0.285 0.273 (Avg.)	3.80-4.20 4.00 (Avg.)	46%-50%	Standard Conventional Type or Standard High Rate
71132-Scrubbed Composite BK-17 High Fines	Hychem AF 304	20	40-45	0.1-0.2	15%-20% (Conv. Type)	0.330-0.360 0.345 (Avg.)	---	46%-50%	Standard Conventional
72008-Leached Composite Resource #1	Hychem AF 304	20	30-35	0.1-0.2	15%-20% (Conv. Type) 10%-12.5% (High Rate)	0.275-0.300 0.288 (Avg.)	4.80-5.30 5.05 (Avg.)	48%-52%	Standard Conventional Type or Standard High Rate
71173-Scrubbed Composite Resource #1	Hychem AF 304	20	35-40	0.1-0.2	10%-15% (Conv. Type)	0.275-0.300 0.288 (Avg.)	---	49%-53%	Standard Conventional

Notes:
General: All tests were conducted at temperature indicated. Flocculant used was Hychem AF 304 a medium to high molecular weight, 15% charge density, anionic polycrylamide. Other products meeting the same description would also serve.
(1) Anticipated minimum flocculant dose range in g/t (grams per metric ton).
(2) Recommended flocculant concentration in g/L prior to contact with feed pulp.
(3) Maximum feed solids concentration range required for thickener operation (wt. %). Note: Maintaining feed solids, concentration in the ranges shown is critical to thickener performance and operation at design rates shown, and may vary by thickener type selected.
(4) Unit Area is for conventional type thickener design only and includes a 1.25 scale-up factor.
(5) Hydraulic loading rate for high-rate thickener design.
(6) Maximum recommended operating underflow solids concentration range for standard conventional thickener based on underflow pulp rheology characteristics.

13.2.13 SGS 2014 Testwork Program

The SGS 2014 testwork program was designed to further investigate the comminution characteristics of the Bomboré ore. Nineteen samples from the Bomboré deposit were submitted for Bond abrasion testing, four samples were submitted for Bond low-energy impact (CWi) test and five samples were submitted for the unconfined compressive strength (UCS) test. Two of the UCS samples were damaged during preparation and therefore only three samples were used. All the samples were considered soft to medium in hardness.

The comminution test statistics are presented in Table 13.41.

Table 13.41 SGS 2014 Test Statistics

Statistics	Ai (g)	CWi (kWh/t)	UCS (MPa)
Results Available	19	4	3
Average (Overall)	0.037	7.6	18.0
Hardness Percentile (1)	11	35	--
Minimum	0.001	5.9	1.5
Maximum	0.137	8.6	45.4

(1) Hardness percentile of the average value relative to the SGS database.

13.2.14 SGS 2016 Testwork Program

The SGS 2016 testwork program was conducted on one fresh rock sample identified as P17S. The program included Bond ball mill work index, gravity separation, cyanidation and flotation testwork.

The P17S sample had a head grade of 3.13 g/t Au, 0.81% $S^{=}$, and 0.10% total carbonaceous matter. The main sulphide minerals were identified as pyrrhotite (1.8%) and arsenopyrite (0.7%). The material was identified as having medium hardness with a Bond work index of 14.2 kWh/t.

Different flowsheet options were tested at a grind size P_{80} of 74 µm. In all options with flotation, the concentrate was reground before leaching. One option included a regrind of the flotation concentrate to P_{80} of 26 µm. The overall metallurgical results for the different flowsheet options are presented in Table 13.42.

Table 13.42 SGS 2016 Overall Metallurgical Results for Flowsheet Options

Flowsheet Options	Overall Au Extraction %						Final Tail	Head
	Gravity	WO CN	Tails CN	Tails Flot.	Conc. CN	Combined	g/t Au	g/t Au
Whole Ore Cyanidation		94.6				94.6	0.17	3.13
Gravity Separation	62.1					62.1	1.18	3.13
Gravity Separation + Grav Tails CN (with pre-aeration)	62.1		33.9			96.0	0.13	3.13
Gravity Separation + Flotation	62.1			34.2		96.3	0.13	3.13
Gravity Sep'n + Flotation + Conc CN (no regrind)	62.1				29.8	91.9	0.23	3.13
Gravity Sep'n + Flotation + Conc CN (with regrind)	62.1				32.2	94.3	0.18	3.13

13.3 Most Recent Testwork for Oxide Plant

In 2017, Orezone approached Lycopodium to conduct a feasibility study for a CIL process on the oxide and upper transition ores at Bomboré. Orezone also contracted Soutex to carry out a gap analysis on the metallurgical testwork programs and results. As a result of the gap analysis, Orezone selected a number of oxide and transition samples to represent different lithologies and grades and submitted these to SGS Quebec for metallurgical testwork. The main objective was to define the Ball mill work index for these lithologies and to optimize the grind size for leaching. Neutralization tests were conducted at SGS Lakefield to determine the lime demand for Bomboré oxide ore.

Dynamic thickening testwork was also conducted by Outotec on an oxide composite sample for the 2017 study.

The results of the more recent testwork programs are summarized in the following sections.

13.3.1 SGS 2017/2018 Testwork Program

A total of 28 low grade and medium grade samples were provided to SGS for characterization of grindability and gold recovery by cyanidation. A summary of the results is presented in Table 13.43.

Table 13.43 SGS 2017/2018 Grindability and Gold Recovery Characterization Results

Sample	Au - Head	LeachWell	Calc Au	As Received (%)		BWI (kWh/t)		Au CN Test	
Name	(g/t)	Au ppm	Head (g/t)	Moisture	+3.35mm	Direct	O'All	P_{80} (μm)	Au Rec,%
Overall Avg.	**0.54**	**0.48**	**0.51**	**0.9**	**85.6**	**8.3**	**7.7**	**87**	**82.1**
MG Avg.	0.70	0.60	0.65	0.9	84.7	8.3	7.7	91	83.9
LG Avg.	0.34	0.31	0.33	1.0	86.7	8.4	7.7	82	79.8
MG Tr U	0.74	0.62	0.69	0.9	87.7	7.4	7.4	92	84.8
LG Tr U	0.37	0.33	0.34	1.0	91.4	8.3	8.3	81	81.6
MG Tr L	0.65	0.58	0.60	0.6	90.9	9.2	9.2	88	82.6
LG Tr L	0.29	0.29	0.35	0.8	92.6	9.1	9.1	84	75.9
MG Ox	0.71	0.61	0.65	1.4	52.8	7.9	3.3	96	85.5
LG Ox	0.36	0.32	0.29	1.3	60.4	6.9	3.0	78	84.8
Oxide	0.54	0.46	0.47	1.3	56.6	7.4	3.2	87	85.2
I1C	0.54	0.45	0.49	0.3	94.0	13.0	13.0	103	73.8
I2	0.53	0.49	0.53	0.7	90.1	7.8	7.8	80	86.1
MI3	0.54	0.47	0.54	1.2	86.7	7.9	7.9	90	80.5
S3	0.54	0.50	0.51	0.8	91.3	6.8	6.8	82	85.9
S4	0.58	0.48	0.52	1.0	91.3	8.2	8.2	82	80.4

Oxide, I2, and S3 lithologies showed the higher gold cyanidation recoveries in the range of 85% Au, while the other three lithologies (I1C, MI3, and S4) showed gold recoveries ranging from 73.8 to 80.5% Au. The low-grade samples generally generated lower gold recoveries.

The neutralization testwork results for determining the lime demand for oxide and transition ores are presented in Figure 13.23 to Figure 13.26 and Table 13.44.

Figure 13.23 SGS 2018 Neutralization Test – pH versus Lime Addition for Oxide Samples

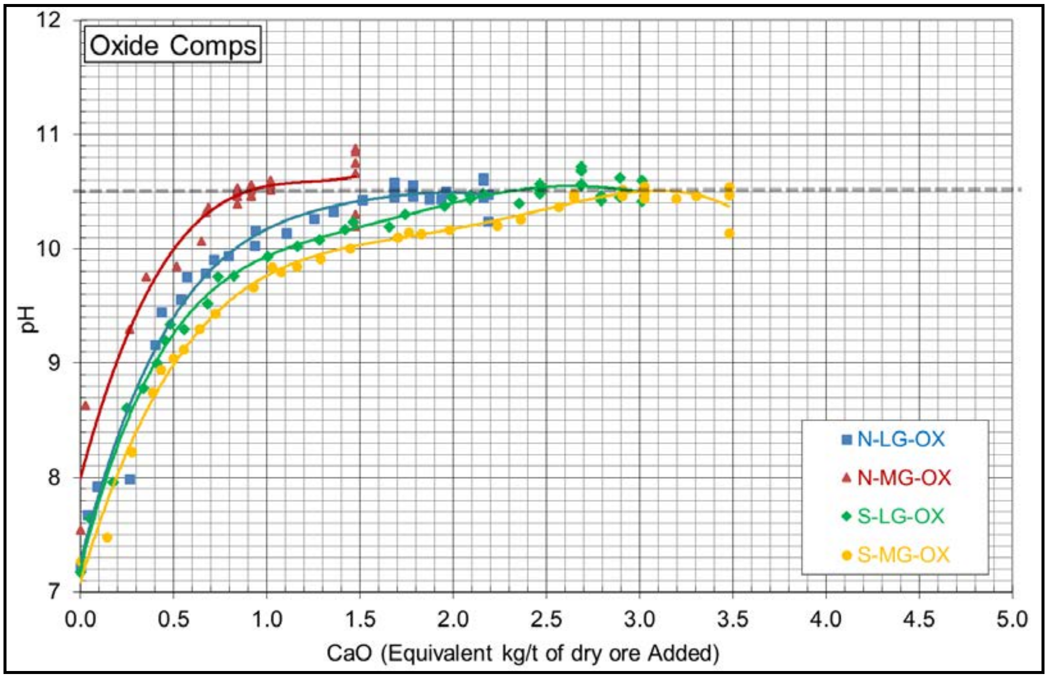


Figure 13.24 SGS 2018 Neutralization Test – pH versus Lime Addition for Transition Samples

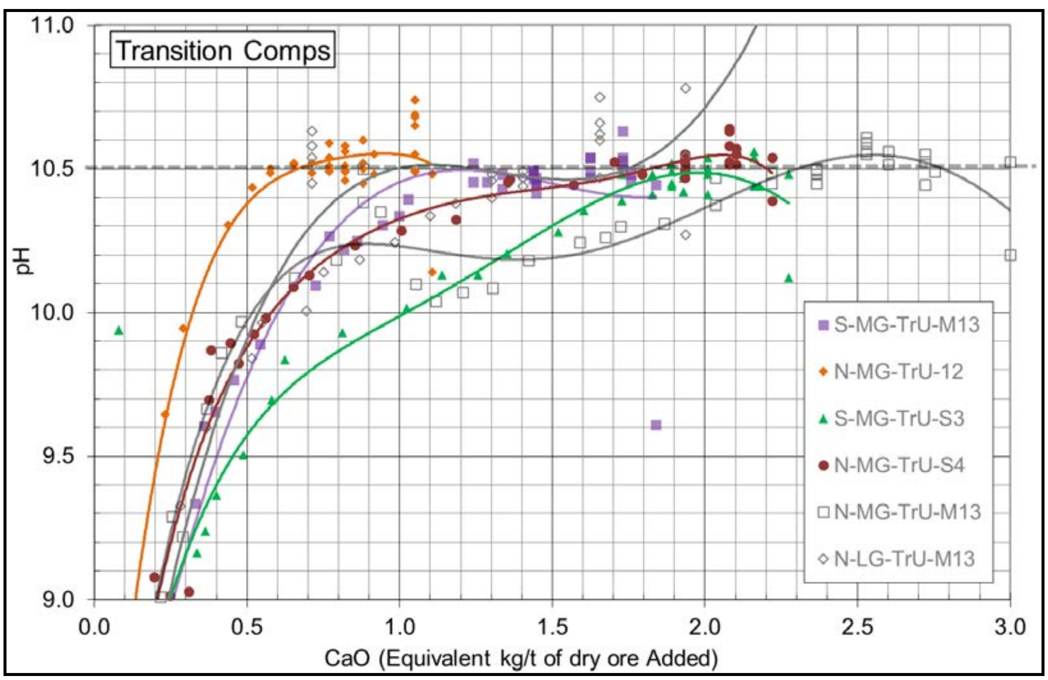


Figure 13.25 SGS 2018 Neutralization Tests – Lime Addition versus Time for Oxide Samples

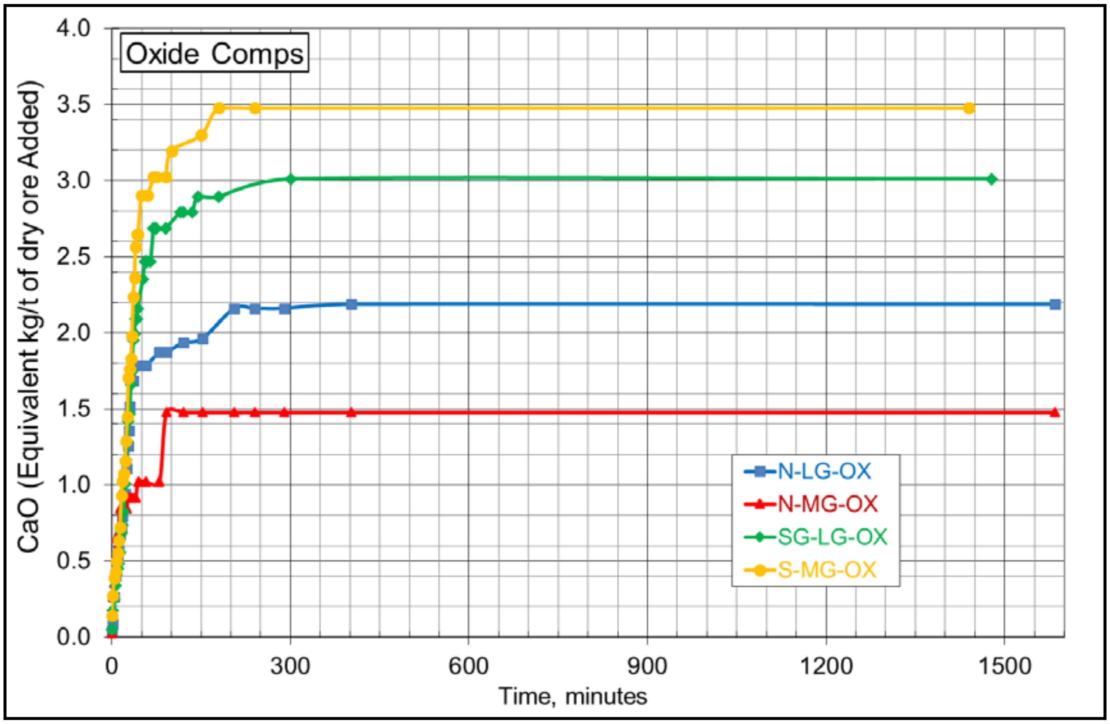


Figure 13.26 SGS 2018 Neutralization Tests – Lime Addition versus Time for Transition Samples

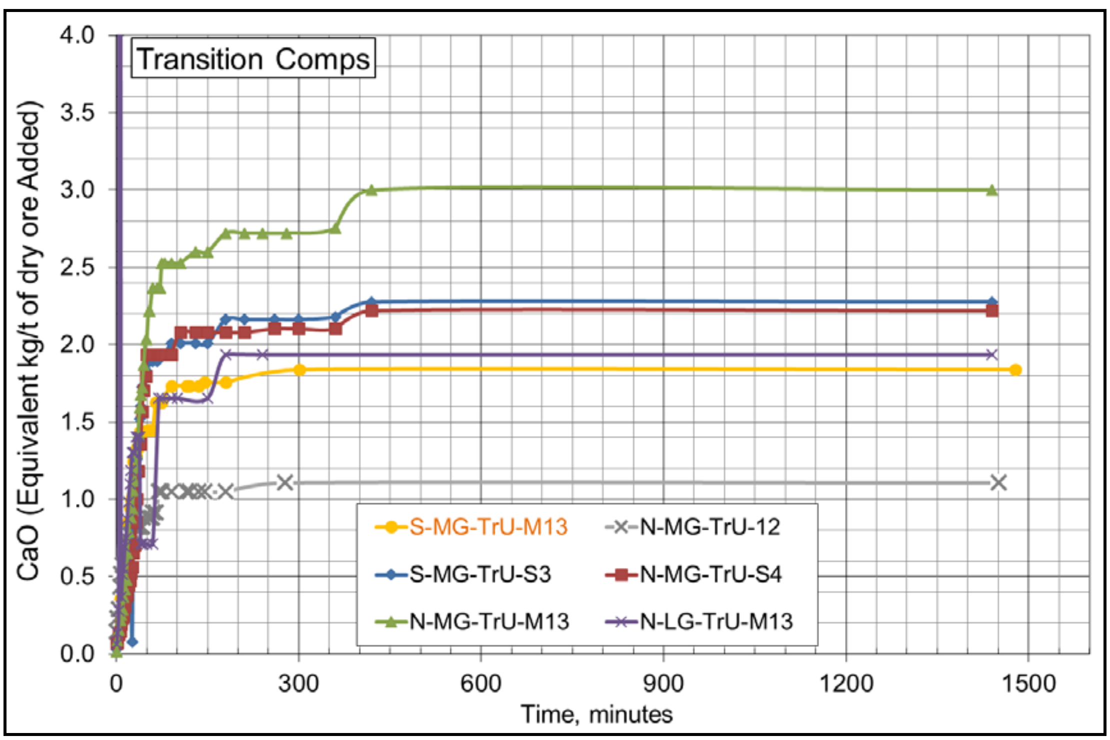


Table 13.44 Summary Results for Neutralization (Lime Demand) Tests

Sample	CaO kg/t ore for 10.5 pH
N-LG-OX	2.19
N-MG-OX	1.48
S-LG-OX	3.01
S-MG-OX	3.48
Oxide Average*	2.54
S-MG-TrU-M13	1.84
N-MG-TrU-12	1.11
S-MG-TrU-S3	2.27
N-MG-TrU-S4	2.22
N-MG-TrU-M13	3.00
N-LG-TrU-M13	1.94
Transition Average*	2.06

** Not weighted average*

13.3.2 Outotec 2018 Testwork Program

A composite sample of the oxide samples from the SGS program above, and after screening out the plus 125 µm materials, was tested by Outotec to examine the flocculant screening and dynamic thickening. Results are shown in Figure 13.27 and Table 13.45. For other results provided by Outotec under this program, refer to Outotec Memorandum No. 11282017-TQ1-TM-001-R0.

Figure 13.27 Outotec Flocculant Screening Results

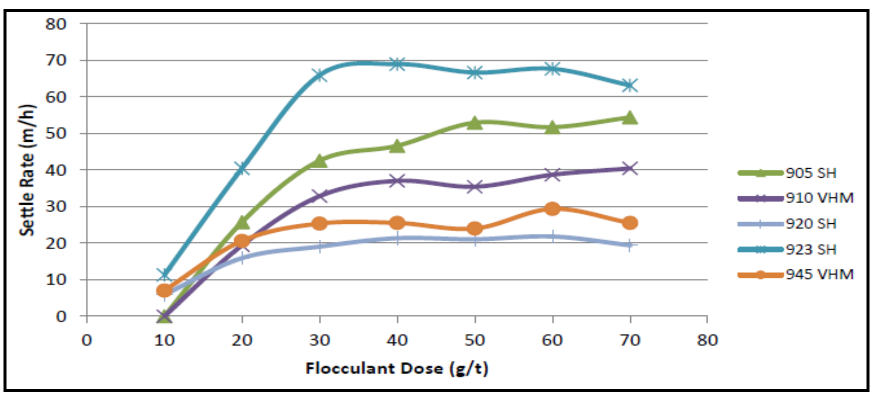


Table 13.45 Outotec Dynamic Thickening Summary Results

Run No.	Feed			Flocculant		Underflow		Overflow
	Flux (t/(m²·h))	Calc. Solids (% (w/w))	Liquor RR (m/h)	Type	Dose (g/t)	Calc. Solids (% (w/w))	YS (Pa)	Solids (mg/L)
1	0.80	12.0	6.17	SNF 923 SH	30	48.4	58	28
2	0.60		4.62		30	48.8	51	14
3	0.40		3.08		30	48.9	58	38
4a	0.20		1.54		30	51.8	90	28
4b *HCT	0.20		1.54		30	54.4	96	28
5	0.20		1.54		20	51.5	79	6
6	0.20		1.54		10	52.1	65	12

As the solids density in the thickener underflow increases, the thickener flux decreases, resulting in increasing thickener diameter size. Due to the saprolitic nature of the Bomboré ore, the maximum density of the thickener underflow is 52.1% at a low flux of 0.2 t/m^{2}h. For thickener underflow density of 48.8% solids, a flux of 0.60 t/m^{2}h can be used for sizing the thickener.

13.3.3 SGS 2019 Testwork Program – Carbon Kinetics Interim Results

In 2019, SGS Lakefield conducted carbon adsorption kinetics and equilibrium isotherm testwork. The results were analyzed to determine the kinetic and equilibrium constants which were used in SGS CIL modelling for validation of the Project CIL design and elution plant selection.

A composite sample was formed using the remaining oxide and upper transition samples from the SGS 2017/2018 program. The make-up for this sample is shown in Table 13.46. The sample's oxide and upper transition ratio is similar to the oxide plant design ore blend of 85% oxide and 15% upper transition.

Table 13.46 SGS 2019 Testwork Composite Sample

Sample	Lithology	Oxidation	Mass (kg)
N-MG-Ox	Mix	Ox	22.0
N-MG-TrU-I1C	I1C	Tr_U	1.0
N-MG-TrU-I2	I2	Tr_U	2.0
N-MG-TrU-MI3	MI3	Tr_U	1.0
N-MG-TrU-S3	S3	Tr_U	0.5
N-MG-TrU-S4	S4	Tr_U	0.5
S-MG-Ox	Mix	Ox	11.0
S-MG-TrU-MI3	MI3	Tr_U	2.0
Total			**40.0**

Table 13.47 presents the various testwork conducted on the composite sample in order to study the leach and carbon adsorption kinetics, and equilibrium isotherm.

Table 13.47 SGS 2019 Testwork Outline

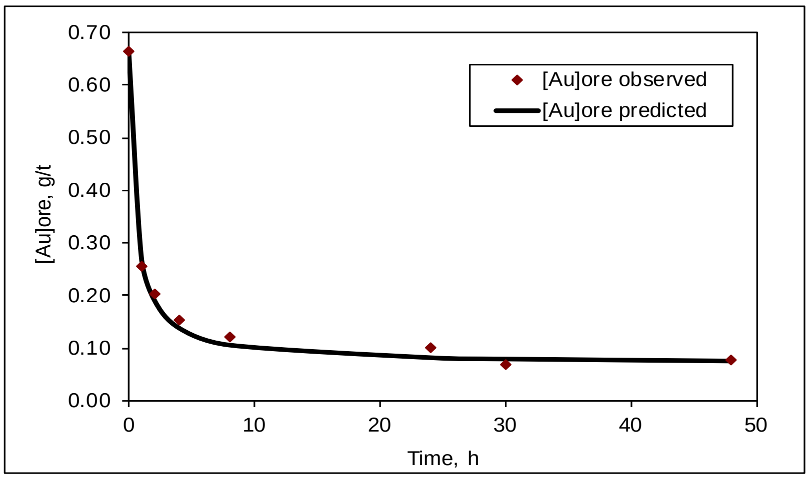
Test No.	Description
CN-1	Initial leach kinetic test (coarse grind)
CN-2	Bulk leach test to create sample for additional testwork
CN-3	Additional leach test using fresh sample and PLS from CN-2
CN-3 AK	Adsorption kinetic test (using CN-3 pulp)
CN-4	Second leach kinetic test (finer grind)
CN-5 A to G	Equilibrium isotherm testwork using CN-2 pulp (spiked solution)
CN-6	Third leach kinetic test (finer grind)

Three leach kinetic tests were performed at grind sizes coarser than the design grind P_{80} of 125 µm (P_{80} of 278 µm, 196 µm, and 177 µm) due to difficulty calibrating the batch grinding mill. A fourth leach test (CN-4) at grind P_{80} of 125 µm is still in progress, however, the result is not expected to be different as the oxide and upper transition ore's leach performance have little dependency on grind size. The result will be included in the final report from SGS.

The available remaining sample from the 2017/2018 SGS testwork program had a lower head grade than the design head grade of 1 g/t Au. In order to achieve the 0.6 mg/L Au solution tenor required for the tests a double leaching technique was used where the PLS from Test No. CN-2 was used to leach the sample in Test No. CN-3.

The leach kinetic constant (k_s) was calculated by SGS to be 3.19 for the composite sample. Figure 13.28 presents the gold leach kinetic curve.

Figure 13.28 SGS 2019 Gold Leach Kinetics



The leach pulp from Test No. CN-3 was then contacted with activated carbon to establish an adsorption profile and for generating the kinetic and equilibrium constants for CIL modelling. The SGS carbon loading kinetic and equilibrium loading models are presented in Figure 13.29 and Figure 13.30, respectively.

Figure 13.29 SGS 2019 Kinetics of Gold Cyanide Extraction by Carbon

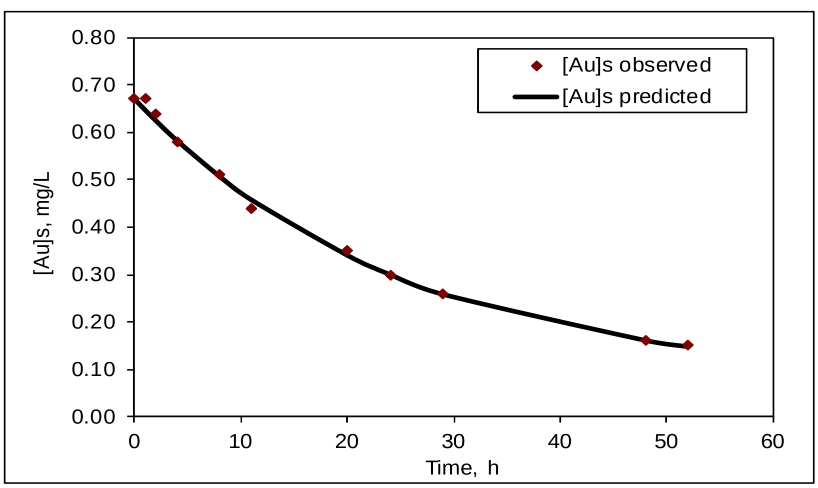


Figure 13.30 SGS 2019 Equilibrium Loading of Gold on Carbon

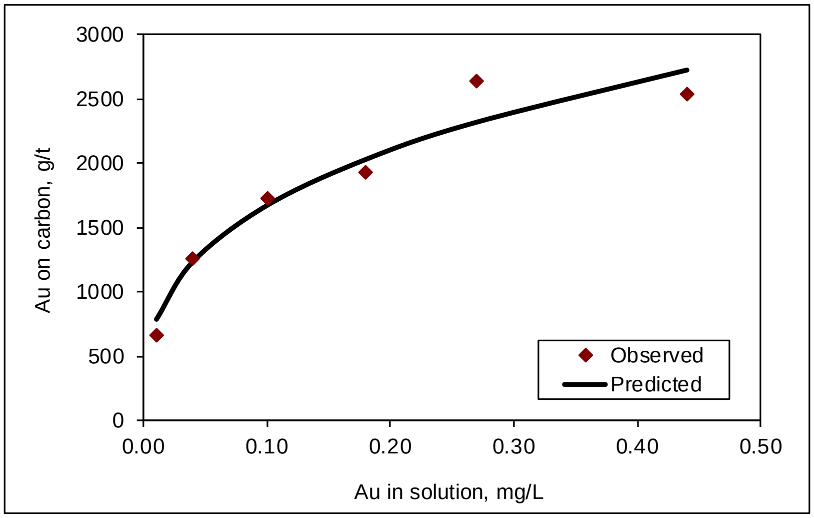


Table 13.48 presents the constants used in the SGS CIL modelling exercise.

Table 13.48 SGS 2019 Modelling Constants

Kinetic Constant (k), h-1	0.005
Equilibrium Constant (K), g/t	15592
Product of Equilibrium and Kinetic Constants (kK)	78

The product of the two constants (kK) provides a useful indication of how well the pulp will perform under CIL or CIP leach conditions. Typically, a kK value of <50 indicates a slow carbon adsorption process is expected, favoring CIP. The kK value for this composite sample is 78, which indicates satisfactory gold adsorption properties for CIL.

The SGS CIL modelling results indicated that the Project oxide plant will respond well to CIL processing. Table 13.49 presents all the modelling inputs for different CIL operating scenarios. The **bold red highlighted values** indicate the parameter that has been changed in each scenario. Table 13.50 presents all the modelling outputs.

Table 13.49 SGS 2019 Design Parameters for Multi-stage CIL Adsorption Circuit

Different Scenarios Inputs	1	2	3	4	5	6	7	8
Slurry feed rate (m^3/h)	1207	1207	1207	1207	1207	1207	1207	1207
Solids (t/h)	650	650	650	650	650	650	650	650
Solution (m^3/h)	975	975	975	975	975	975	975	975
Consider Leach after Carbon addition	Y	Y	Y	Y	Y	Y	Y	N
Gold on stripped carbon, g/t	50	50	50	50	50	**100**	**50**	50
Adsorption tank(s) size, m^3	3621	3621	3621	3621	3621	3621	3621	1207
Carbon frequency advance (% in 24 hours)	28%	33%	**18%**	**14%**	**55%**	**28%**	28%	33%
Leaching								
Au leached before Carbon addition	83.6%	83.6%	83.6%	83.6%	83.6%	83.6%	83.6%	91.7%
Leach time before Carbon addition (h)	3.0	3.0	3.0	3.0	3.0	3.0	3.0	**24**
Leach only total tankage (m^3)	3621	3621	3621	3621	3621	3621	3621	**28971**
Number of Leaching tanks	1	1	1	1	1	1	1	**1**
Volume of Leaching tanks (m^3)	3621	3621	3621	3621	3621	3621	3621	28971
CIP/CIL								
Model output kinetic constant (k)	0.005	0.005	0.005	0.005	0.005	0.005	**0.004**	**0.005**
Model output equilibrium constant (K)	15592	15592	15592	15592	15592	15592	**12474**	**15592**
Product of equilibrium and kinetic constants (	78	78	78	78	78	78	**50**	**78**
Number of stages	7	7	7	7	7	7	7	**7**
Total CIP/CIL volume (m^3)	25350	25350	25350	25350	25350	25350	25350	8450
Slurry residence time in each adsorption tank	3.0	3.0	3.0	3.0	3.0	3.0	3.0	**1.0**
Gold grade in residue (g/t)	0.083	0.083	0.083	0.083	0.083	0.083	0.083	0.083
Gold in final barren solution (mg/L)	0.006	0.006	0.005	0.005	0.011	0.009	0.010	0.004
Gold in loaded carbon (g/t)	1466	1230	1467	1468	1455	1508	1457	1471
Carbon residence time/stage (h)	87	72	130	174	43	87	87	72
Carbon Concentration (g/L pulp)	10	10	**15**	**20**	**5**	**10**	10	**25**
Equivalent transferred carbon unit flowrate (kg	417	500	417	417	417	417	417	417
Daily carbon transfer / batch elution capacity	10000	**12000**	10000	10000	10000	10000	10000	10000
Carbon Inventory per stage (kg)	36214	36214	54321	72429	18107	36214	36214	30179
Carbon inventory all stages (tons)	254	254	380	507	127	254	254	211
Gold Lock-Up on Carbon (kg)	102.8	85.8	144.6	186.2	59.9	115.1	116.5	81.5
CIP/CIL Gold recovery per day (g/day)	14158	14164	14174	14180	14048	14083	14067	14207
Overall Gold Leaching Efficiency	91.7%	91.7%	91.7%	91.7%	91.7%	91.7%	91.7%	91.7%
Overall Gold Adsorption Efficiency	98.9%	98.9%	99.0%	99.1%	98.1%	98.3%	98.2%	99.3%
Overall Gold Recovery	90.7%	90.7%	90.8%	90.8%	89.9%	90.1%	90.0%	91.1%
Au in loaded carbon / Au in feed	1466	1230	1467	1468	1455	1508	1457	1471
Upgrading ratio	2631	2208	2634	2635	2611	2707	2615	2406
Circuit filling time - slurry (days)	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.3
Ramp-up time (days) *	7.3	6.1	10.2	13.1	4.3	8.2	8.3	5.7

* Ramp-up time (days) = Gold lock-up (kg) / Gold Produced (kg/day)

Table 13.50 SGS 2019 CIL Modelling Circuit Profile Data

Interstage data Scenario	1	2	3	4	5	6	7	8
Gold in ore/stage residues (g/t)								
Feed head grade	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Leach tank discharge	0.164	0.164	0.164	0.164	0.164	0.164	0.164	0.083
Adsorption stage 1 discharge	0.120	0.120	0.120	0.120	0.120	0.120	0.120	0.083
Adsorption stage 2 discharge	0.104	0.104	0.104	0.104	0.104	0.104	0.104	0.083
Adsorption stage 3 discharge	0.096	0.096	0.096	0.096	0.096	0.096	0.096	0.083
Adsorption stage 4 discharge	0.091	0.091	0.091	0.091	0.091	0.091	0.091	0.083
Adsorption stage 5 discharge	0.087	0.087	0.087	0.087	0.087	0.087	0.087	0.083
Adsorption stage 6 discharge	0.085	0.085	0.085	0.085	0.085	0.085	0.085	0.083
Adsorption stage 7 discharge	0.083	0.083	0.083	0.083	0.083	0.083	0.083	0.083
Gold on carbon (g/t)								
Adsorption stage 1 discharge	1466	1230	1467	1468	1455	1508	1457	1471
Adsorption stage 2 discharge	661	538	588	545	814	709	789	617
Adsorption stage 3 discharge	313	253	259	231	454	362	428	275
Adsorption stage 4 discharge	166	137	137	123	260	216	241	139
Adsorption stage 5 discharge	102	89	88	83	156	152	144	84
Adsorption stage 6 discharge	73	67	67	65	99	123	93	62
Adsorption stage 7 discharge	58	56	56	56	68	108	65	54
Stripped carbon feed to last stage	50	50	50	50	50	100	50	50
Gold in solution (mg/L)								
Leach tank discharge	0.557	0.557	0.557	0.557	0.557	0.557	0.557	0.611
Adsorption stage 1 discharge	0.243	0.232	0.211	0.192	0.313	0.245	0.301	0.246
Adsorption stage 2 discharge	0.105	0.096	0.081	0.069	0.170	0.107	0.158	0.100
Adsorption stage 3 discharge	0.047	0.042	0.034	0.028	0.092	0.050	0.083	0.042
Adsorption stage 4 discharge	0.023	0.021	0.017	0.014	0.051	0.027	0.045	0.019
Adsorption stage 5 discharge	0.013	0.012	0.010	0.009	0.029	0.016	0.026	0.009
Adsorption stage 6 discharge	0.008	0.008	0.007	0.006	0.017	0.012	0.015	0.006
Adsorption stage 7 discharge	**0.006**	**0.006**	**0.005**	**0.005**	**0.011**	**0.009**	**0.010**	**0.004**

This test program confirmed that the existing CIL design, 1 leach and 7 CIL stages and a 10 t elution plant will achieve satisfactory results and low solution losses to the CIL tails.

The model outputs showed that 6 CIL stages will be sufficient, although having a 7th stage will provide more flexibility for mill feed and head grade fluctuation.

The current plant design calls for a 10 t elution plant with 8.4 strips per week, however, the model outputs predicted that a 10 t elution plant with one elution cycle per day (or 7 strips/week) will be adequate.

The maximum gold carbon loading is estimated to be at 2,200 g/t. The targeted gold carbon loading will range between ~1,200 g/t to ~1,500 g/t.

It is recommended that after start-up, the CIL circuit profiles compared to the modelling results. A sample of the plant CIL feed can then be submitted for confirmatory testing and can be used for further optimization with a focus on lowering operating costs (i.e., less carbon, fewer elution cycles).

13.4 Sulphide Testwork for 2019 Study

At the end of 2018, Orezone approached Lycopodium to expand the 2018 feasibility study to include the treatment of lower transition and fresh ores (sulphide ores). In 2019, Base Metallurgical Laboratories Ltd. (Base Metal) conducted a metallurgical testing program in support of this feasibility study. The objective of the program was to expand the understanding of the response of the lower transition and sulphides to conventional cyanidation, comminution and sedimentation. A blend composite of sulphide and oxide as per the mine plan was also tested to confirm the performance of combined material in the CIL circuit and to provide composite tailings for other testwork.

Four main domain composite samples and 11 variability samples were tested. The cyanidation program explored the effect of grind, cyanide dosage, slurry density and dissolved oxygen levels on the kinetics and final extraction of gold from these samples.

The following sections present key results from this test program. Additional details are provided in the report titled “Metallurgical Testing of Fresh Rock Type in Support of the Feasibility Study”.

13.4.1 Head Analysis

Table 13.51 below lists the composite details and measured head assays, determined through various methods.

Table 13.51 Base Metal 2019 Head Analysis Results for Composite and Variability Samples

Sample ID	Domain/Mine	Weathering	Au	Cu	Fe	Ag	S	C	Cg
Method	**-**	**-**	**SM**	**AR_AA**	**AR_AA**	**AR_AA**	**Leco**	**Leco**	**Leco**
Units	**-**	**-**	**g/t**	**ppm**	**%**	**g/t**	**%**	**%**	**%**
Comp 1	Siga_S	Fresh	1.22	90	7.45	2.5	3.080	1.03	<0.01
Comp 2	P8P9	Fresh	1.30	275	8.85	<1	3.285	0.85	<0.01
Comp 3	P8P9	Lower Trans.	0.96	170	8.25	<1	1.375	0.08	<0.01
Comp 4	P17S_W	Fresh	2.36	20	4.48	2	0.925	0.61	<0.01
Method	**-**	**-**	**SM**	**FAAS**	**FAAS**	**FAAS**	**Leco**	**Leco**	**Leco**
Units	**-**	**-**	**g/t**	**%**	**%**	**%**	**%**	**%**	**%**
Var 1	Maga_H	Fresh	2.0	0.015	7.94	<1	1.97	0.93	<0.01
Var 2	Maga_M	Lower Trans.	0.8	0.003	5.30	<1	0.21	0.46	0.31
Var 3	Siga_S	Lower Trans.	0.8	0.005	6.42	<1	0.72	0.05	<0.01
Var 4	Siga_S	Fresh	1.6	0.003	4.86	<1	1.39	0.93	<0.01
Var 5	Siga_S	Fresh	1.5	0.018	5.88	<1	3.17	1.49	<0.01
Var 6	P8P9	Fresh	0.9	0.023	5.02	<1	2.89	0.69	0.01
Var 7	P8P9	Fresh	1.6	0.026	3.74	<1	1.19	1.23	0.01
Var 8	P8P9	Lower Trans.	0.9	0.022	4.90	<1	1.78	0.03	<0.01
Var 9	P8P9	Lower Trans.	0.8	0.011	3.96	<1	0.25	0.08	<0.01
Var 10	P17S_E	Fresh	1.8	<0.001	5.08	<1	0.94	0.56	0.01
Var 11	P17S_MI3	Fresh	0.5	<0.001	5.44	<1	0.43	0.42	<0.01

13.4.2 Comminution

Table 13.52 summarizes the comminution test results. The samples exhibited a large degree of variation in terms of both hardness and abrasiveness. For instance, the Bond Ball Mill index ranges from 8.6 kWh/t for Var 3 (SigaS) to as high as 17.8 kWh/t for Var 6 (P8P9). Within each domain, the transition samples are generally less abrasive and less resistant to grinding. However, there is some overlap between domains with the transition samples for P8P9 being just as hard as the fresh rock from the other domains.

Table 13.52 Base Metal 2019 Comminution Results

Sample ID		Relative Density	JK Data SMC					Mesh of Grind	BWI parameters				Bond Ai
									F80	P80	Gram/rev	Work Index	Ai
	Rock Type	SMC	A	b	A x b	t_a	DWI (kWh/m^3)		µm	µm		kWh/t	(g)
Comp 1	Siga_S_Fr	2.90	59.0	0.61	36.0	0.32	8.1	150	1,921	80	1.45	13.9	0.245
Comp 2	P8P9_Fr	2.74	68.5	0.50	34.3	0.32	8.1	150	1,906	80	1.33	14.9	0.293
Comp 3	P8P9_Tr_L	2.42	59.4	1.10	65.3	0.70	3.7	150	1,490	79	1.50	13.9	0.069
Comp 4	P17S_W_Fr	2.72	100	0.29	29.0	0.28	9.5	150	1,938	83	1.29	15.7	0.689
Var-1	Maga_H_Fr	2.66	69.4	0.48	33.3	0.32	8.1	150	1,777	80	1.43	14.2	0.261
Var-2	Maga_M_Tr_L	2.31	43.9	1.49	65.4	0.73	3.5	150	1,685	68	2.11	9.4	0.014
Var-3	Siga_S_Tr_L	2.55	49.7	1.81	90.0	0.91	2.8	150	1,629	78	2.62	8.6	0.049
Var-4	Siga_S_Fr_1	2.47	61.8	0.72	44.5	0.47	5.5	150	1,778	80	1.72	12.2	0.110
Var-5	Siga_S_Fr_2	2.38	54.2	0.72	39.0	0.42	6.1	150	1,807	81	1.61	13.0	0.135
Var-6	P8P9_Fr_I1C	2.37	94.5	0.31	29.3	0.32	8.1	150	1,995	83	1.09	17.8	0.673
Var-7	P8P9_Fr_I2	2.32	94.2	0.29	27.3	0.30	8.5	150	1,915	79	1.14	16.8	0.370
Var-8	P8P9_Tr_L_I1	2.54	74.4	0.63	46.9	0.48	5.4	150	1,902	81	1.21	16.3	0.341
Var-9	P8P9_Tr_L_I2	2.58	64.6	0.81	52.3	0.53	4.9	150	1,791	79	1.54	13.3	0.115
Var-10	P17S_E_Fr	2.77	100	0.27	27.0	0.25	10.2	150	2,011	83	1.39	14.6	0.658
Var-11	P17S_MB_Fr	2.91	93.4	0.31	29.0	0.26	10.1	150	1,887	81	1.39	14.5	0.457

Variability: Overall Statistics											
Average	**2.576**	**72.5**	**0.69**	**43.2**	**0.44**	**6.8**	**1,829**	**80**	**1.52**	**13.9**	**0.299**
Std. Dev.	0.20	19.2	0.46	18.2	0.20	2.4	143	4	0.39	2.5	0.231
Rel. Std. Dev.	7.79	26.4	66.6	42.1	45.2	35.5	8	4	25.94	17.9	78
Minimum	2.31	43.9	0.27	27.0	0.25	2.8	1,490	68	1.09	8.6	0.014
10th Percentile	2.34	51.5	0.29	28.0	0.27	3.6	1,651	78	1.17	10.5	0.057
25th Percentile	2.40	59.2	0.31	29.1	0.31	5.2	1,778	79	1.31	13.2	0.112
Median	**2.55**	**68.5**	**0.61**	**36.0**	**0.32**	**8.1**	**1,887**	**80**	**1.43**	**14.2**	**0.261**
75th Percentile	2.73	93.8	0.77	49.6	0.51	8.3	1,918	81	1.57	15.3	0.414
90th Percentile	2.85	97.8	1.33	65.4	0.72	9.8	1,972	83	1.96	16.6	0.667
Maximum	2.91	100.0	1.81	90.0	0.91	10.2	2,011	83	2.62	17.8	0.689

Minimum and Maximum refer to softest and hardest for the grindability tests, respectively

13.4.3 Mineralogy

A mineralogical study indicated quartz and plagioclase as the most abundant minerals with biotite also significant for the Siga samples. The dominant sulphide mineral is pyrite at approximately 4% of the total rock mass. This is accompanied by minor quantities (<1%) of pyrrhotite for all except for composite 2 (P8P9) and composite 4 (P17S). The P17S domain differs from the P8P9 in that it contains very little pyrite but significantly higher pyrrhotite (almost 1.8%) as well as some arsenopyrite (1%).

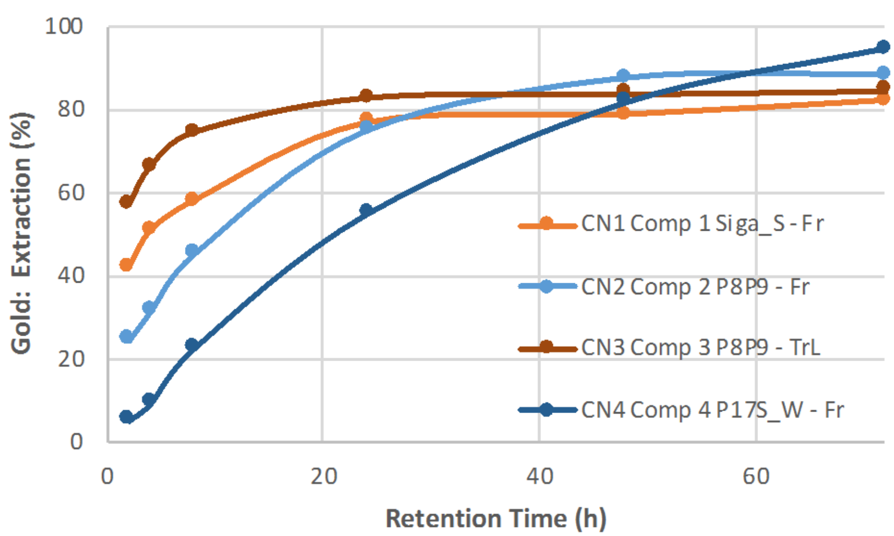
13.4.4 Cyanidation

Cyanidations were performed at the following standard conditions:

- Pulp density of 50% w/w solids.
- Target grind P_{80} of 75 µm.
- Pulp pH of 11.5 (maintained).
- Dissolved oxygen (DO) level maintained through air sparging.
- g/L NaCN (free) of 0.5 (maintained).

Initial optimization testwork varied the grind, NaCN level, pulp density and DO level to determine an optimum suite of operating parameters for the subsequent testing of variability samples. Figure 13.31 shows the leach kinetic curves recorded for the four composites at the standard conditions.

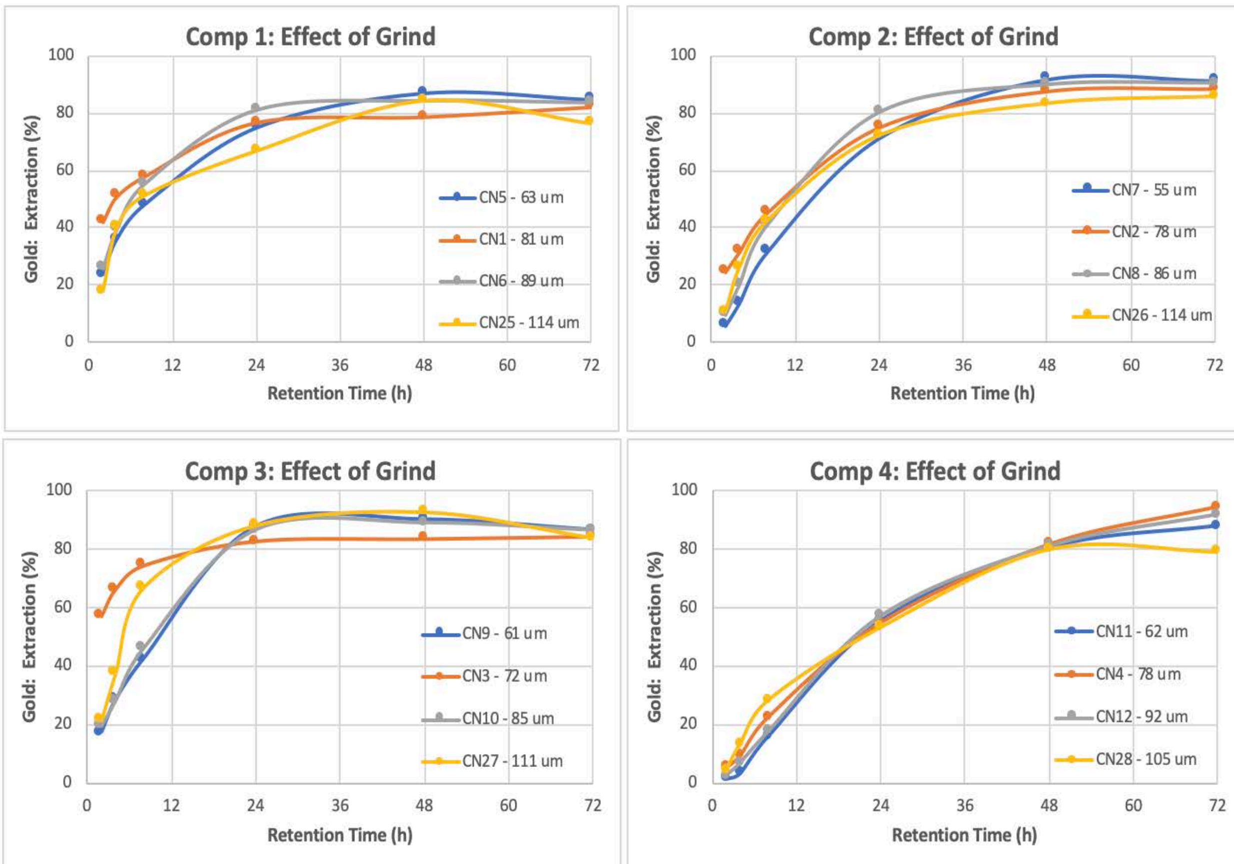
Figure 13.31 Base Metal 2019 Baseline Leaching Kinetics



Moderate (mid 80%) extractions were achieved for Composites 1 to 3 with Composite 4 yielding a high (95%) gold extraction into solution. The rate of gold leaching is fast for the P8P9 transition sample (Composite 3) and still acceptable for Composite 1. However, it is slow for Composite 2 and exceptionally slow for Composite 4. The rate of dissolution appears to be correlated to the pyrrhotite content of the ore as both Composites 2 and 4 have higher level of pyrrhotite than other composites.

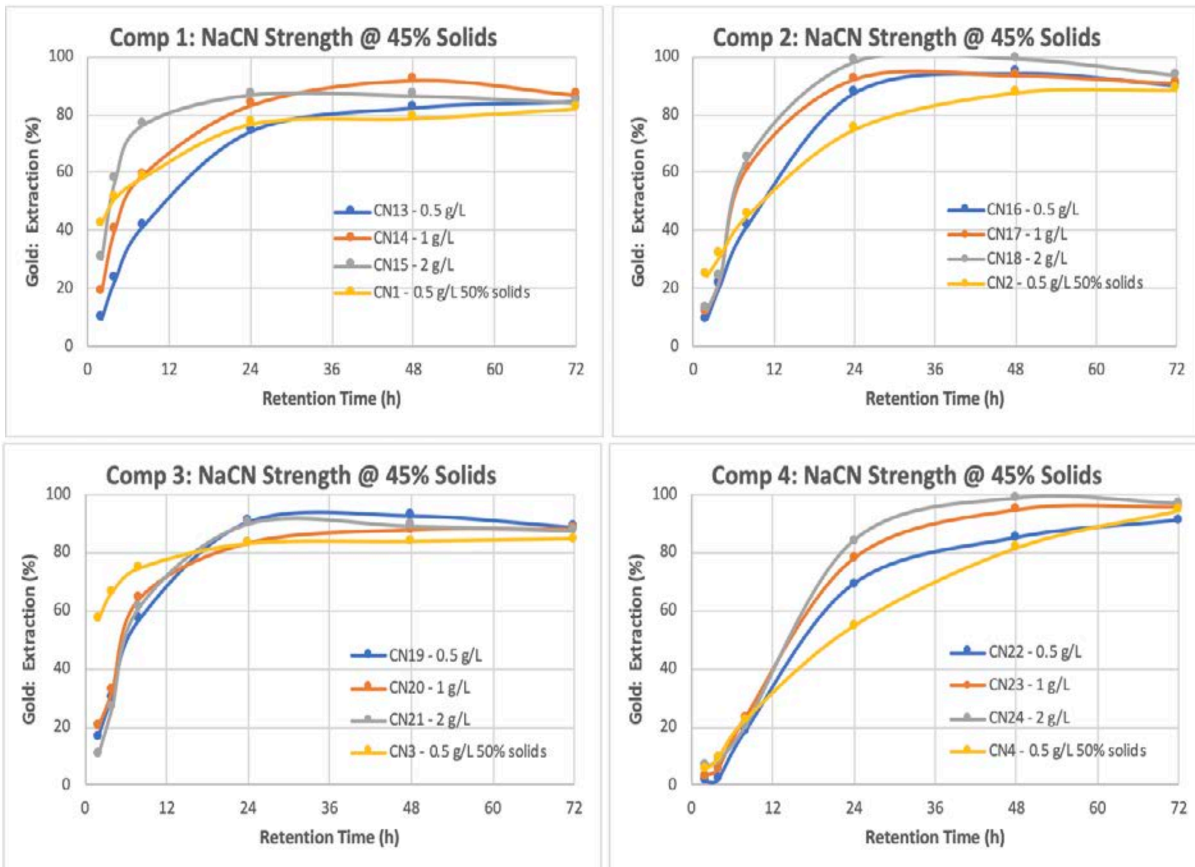
Figure 13.32 shows the effect of grind size on leach kinetics. There is no obvious trend indicating that grind size plays a significant role in the rate of dissolution for these samples. When comparing final residue grades the grind sizes below 90 µm did not show consistent nor significant increases in incremental gold extractions at a finer grind size.

Figure 13.32 Base Metal 2019 - The Effect of Grind Size on Gold Leaching



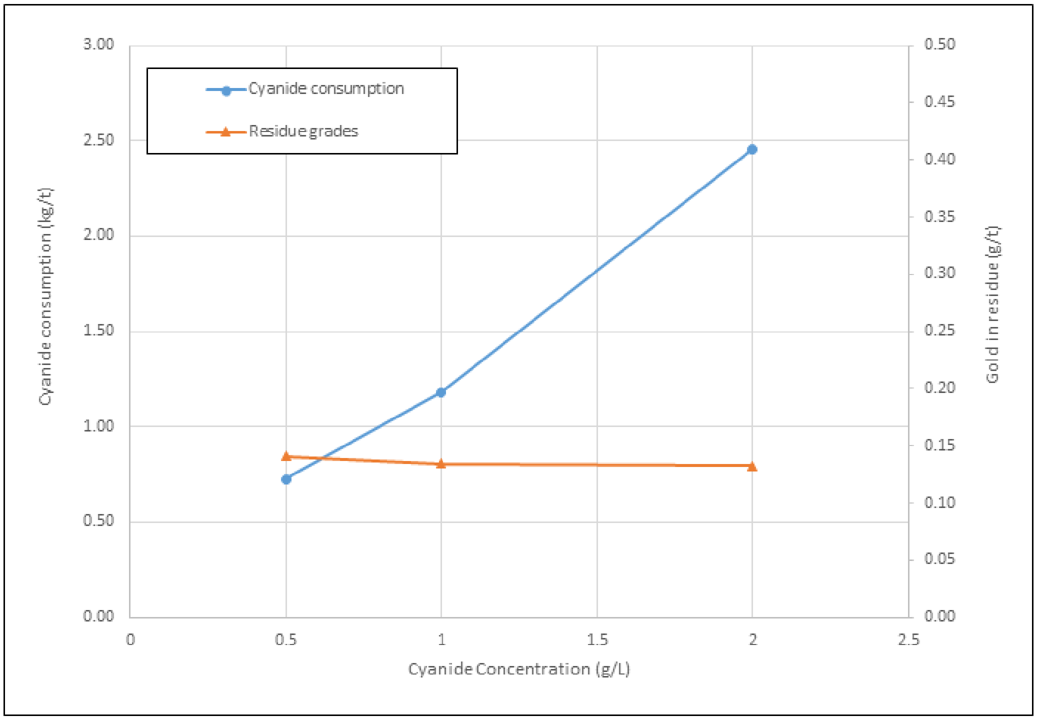
A series of tests at three cyanide levels (0.5, 1.0 and 2.0 g/L) were performed to evaluate the effect of cyanide concentration on leach performance. The kinetics curves presented in Figure 13.33 generally suggest an improvement in initial leach kinetics with an increase in cyanide concentration.

Figure 13.33 Base Metal 2019 - Leach Kinetic Curves at Different Cyanide Concentrations



To further explore the effect of cyanide concentration on leach extraction and consumption, the average residue and consumption rates were calculated for Composites 1, 2 and 3 and are plotted in Figure 13.34. Composite 4 (P17S) was excluded due to its very low residue value (high recovery) regardless of test conditions and also it represents only a small fraction of the ore body. The maximum value for both the primary and secondary y-axis where chosen to be approximately 4 times the minimum value of the data in order to provide a scale that allows direct visual comparison of the strength of the correlations.

Figure 13.34 Base Metal 2019 - Gold Residue and Cyanide Consumption versus Cyanide Concentration

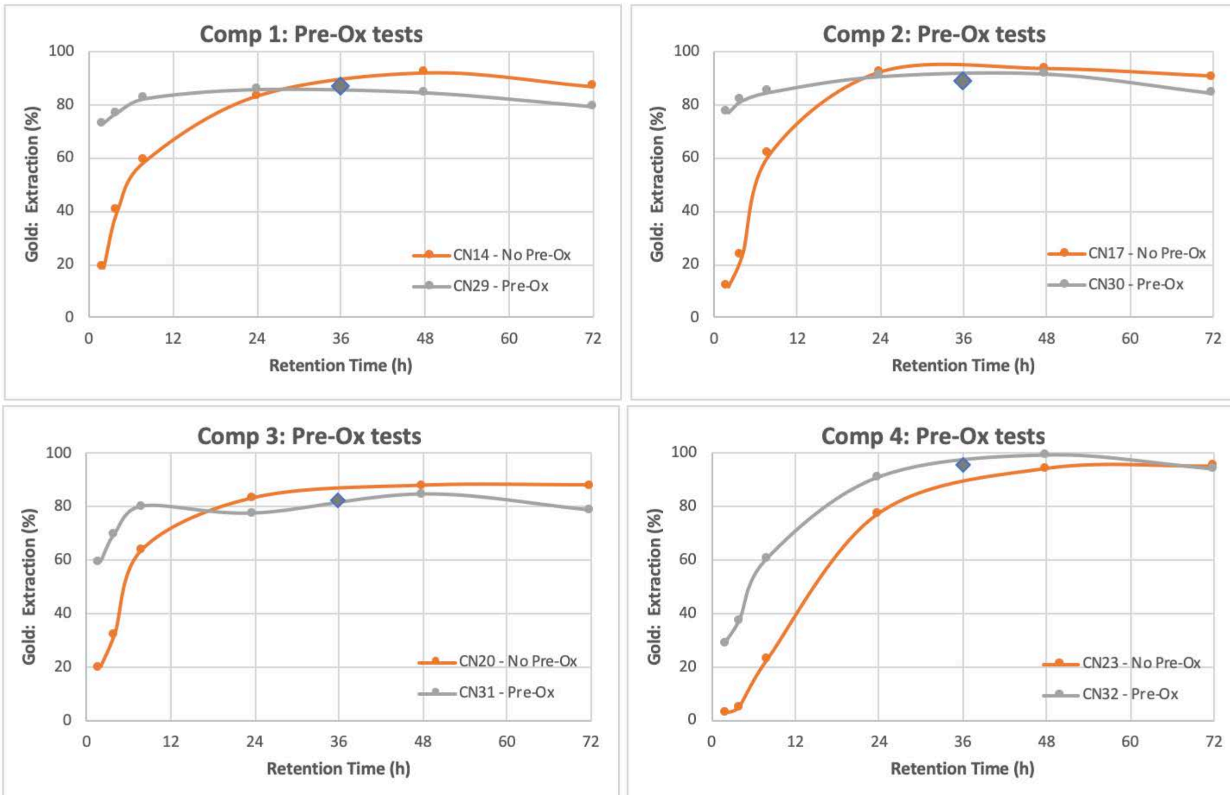


It is evident from this that the average final gold residue (for all composites combined) was largely unaffected by the cyanide concentration used during the test, decreasing from 0.141 g/t at the lowest cyanide concentration to 0.133 g/t at 2 g/L. However, the amount of cyanide consumed during the test was strongly correlated to the cyanide concentration, increasing from 0.72 kg/t at 0.5 g/L to 2.46 kg/t at 2 g/L. As a high cyanide concentration was desirable to improve leach kinetics, pre-oxygenation was investigated to improve the leach kinetics and reduce the impact of cyanide consuming species in the sulphide ores.

Ground samples were pre-oxygenated for 8 hours at a controlled dissolved oxygen concentration of 15 ppm. Cyanide was added and the cyanide and oxygen concentrations were maintained at 1 g/L and 15 ppm respectively throughout the duration of the leach.

Figure 13.35 demonstrates how the pre-treatment step enhanced leach kinetics for all of the composites with much more rapid leaching in the first 12 hours. In addition, the measured cyanide consumption reduced significantly with the use of pre-oxygenation. The average cyanide consumption reduced from 1.2 kg/t with air sparging to less than 0.5 kg/t for tests using pre-oxygenation and oxygen sparging.

Figure 13.35 Base Metal 2019 - Comparison of Pre-oxygenation and Oxygen Sparging versus Air Only



13.4.5 Variability Testing

Variability testing was performed at the following conditions:

- Grind P_{80} of 75 µm.
- Pulp density of 50% w/w solids.
- Pulp pH of 11.5.
- Pre-oxygenation for 8 hours.
- DO of 15 ppm maintained.
- g/L NaCN (free) of 1.0 maintained.
- Leach duration of 48 hours.

The average gold extraction achieved for the variability samples was 91.6%, which is higher than the predicted recoveries from the equations developed from LeachWell database.

The measured cyanide consumption ranged from 0.25 kg/t to 0.9 kg/t with a median value of 0.36 kg/t. Lime consumption ranged from 0.68 kg/t to 2.99 kg/t with a median value of 1.38 kg/t. Generally, the variability testing did not yield an unusually large degree of variability, which augurs well for a smooth operation.

13.4.6 Sedimentation Testwork

Dynamic settling tests were conducted with a bench-scale raked thickener. The three tests were performed at a constant feed rate equivalent to a loading of 0.3 $t/h/m^2$. One of the tests used a different flocculant (MF10 instead of MF351) while the third test was performed at an elevated pH (11.3 versus 10.5). All three tests yielded similar underflow densities within the range 57 to 58 % w/w solids. The test at a higher pH yielded less suspended solids in the overflow. Further sedimentation tests are required to determine the expected loading rate for the targeted pre-leach thickener underflow density of 50% w/w solids.

13.5 Results Interpretation

The following sections describe the main results that contributed to the development of the process design criteria for the Bomboré Project.

13.5.1 Ore Comminution Characteristics

OMC compiled and analysed all the historical comminution testwork and provided ore characteristic values for use in the oxide plant comminution design as per Table 13.53. The oxide plant comminution design parameters were reviewed and endorsed by Soutex on behalf of Orezone.

Table 13.53 Ore Characteristics for Oxide Plant Comminution Design

Parameter	Units	Process Design Criteria			Notes
		Oxide	Transition Upper	Blend	
ROM Ore Types					
Design Ore Blend	%	85	15	100	Client
ROM Distribution					
ROM Fresh Feed, F_{80}	mm	7.4	23.2	9.8	KCA014009/96
ROM Properties					
Moisture Content	$\%H_2O$	5.70	2.80	5.27	RPA Resource Update 2017
Specific Gravity	-			2.83	Outotec 11282017-TQ1-TM-001-R0
AI	g	0.028	0.052	0.031	Testwork Ave
CWI	kWh/t	7.5	8.6	7.7	Testwork Max
RWI	kWh/t	5.4	7.8	5.8	Testwork - 1 Sample
BWI	kWh/t	4.3	7.8	4.8	85th Percentile

The sulphide comminution design, also provided by OMC, is summarized in Table 13.54.

Table 13.54 Ore Characteristics for Sulphide Plant Comminution Design

	Relative Dist. %	Axb		CWI (kWh/t)		RWI (kWh/t)		BWI (kWh/t)		SG		AI (g)	
		Ave	85th %	Ave	85th %	Ave	85th %	Ave	85th %	Ave	85th %	Ave	85th %
Transition Wt Ave	24%	55.7	52.7	-	-	-	-	12.7	13.3	2.55	2.61	0.148	0.253
Fresh Wt Ave	76%	38.4	28.9	15.1	19.8	16.2	17.1	15.0	16.3	2.75	2.87	0.309	0.433
Resource Wt Ave	100%	42.5	36.6	*15.1*	*19.8*	*16.2*	*17.1*	14.1	15.2	2.68	2.79	0.258	0.377
Overall Testwork	-	41.9	27.0	15.1	19.8	16.2	17.1	14.8	16.9	2.74	2.89	0.305	0.608

**Due to low number of tests, statistics with italicized font includes AM01 and AMCT testing.*

13.5.2 Grind Size Selection

McClelland and AMMTEC both conducted extensive leach testwork at varying grind sizes. Results from the two programs were used in a grind size selection exercise for the oxide plant. Figure 13.36 and Figure 13.37 show the grind sizes plotted against the residue grades for oxide and transition ores, respectively. Considering the life-of-mine head grade, only data with head grades below 2 g/t were used in the analysis. The trends indicate that the residue grades do not vary significantly with grind size and there appears to be little benefit from fine grinding. Based on this finding, Lycopodium and OMC recommends that the Bomboré oxide comminution circuit be designed for a grind size P_{80} of 125 µm.

Figure 13.36 Grind Size versus Residue Grade for Oxide Material <2 g/t

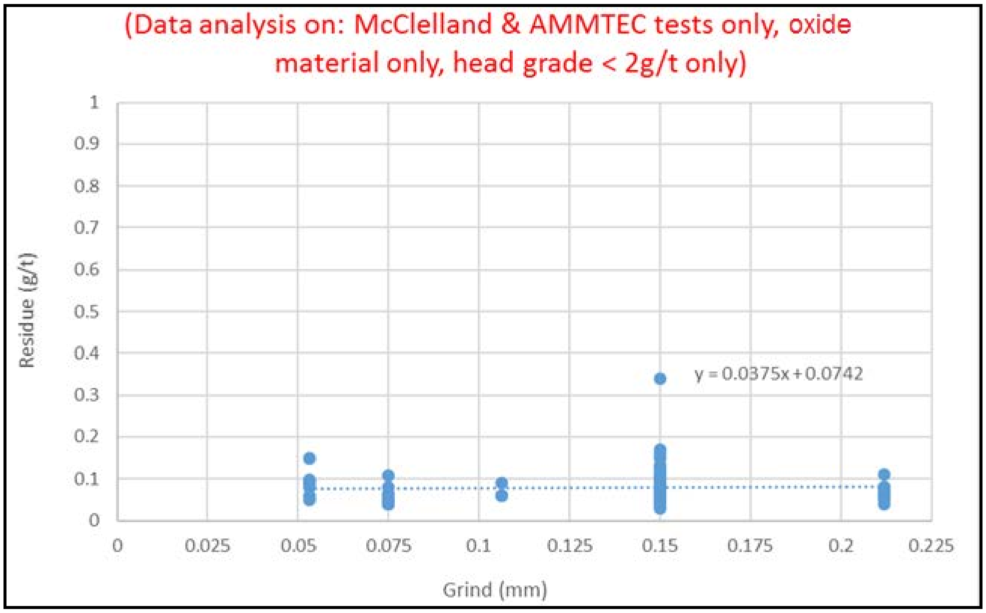
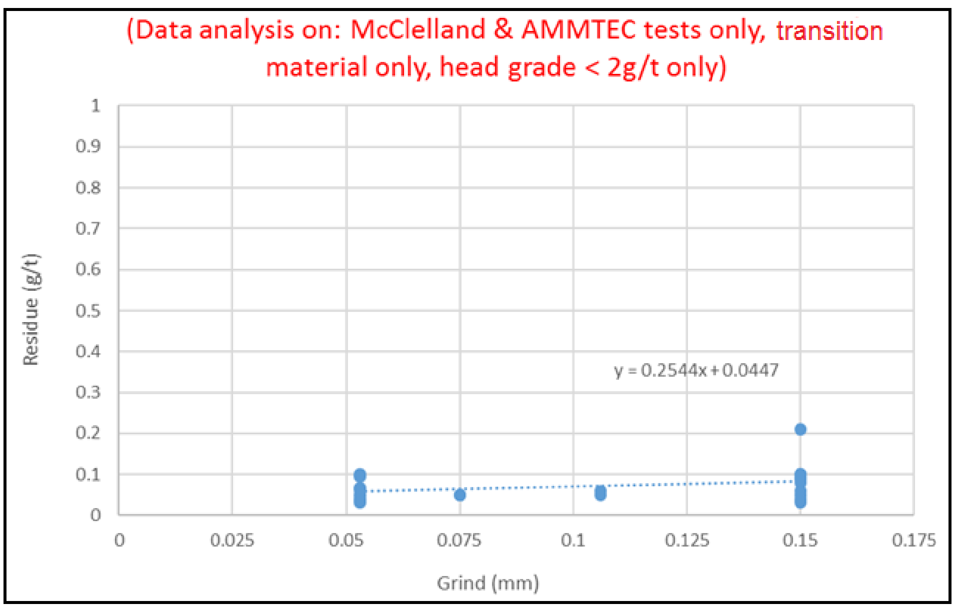
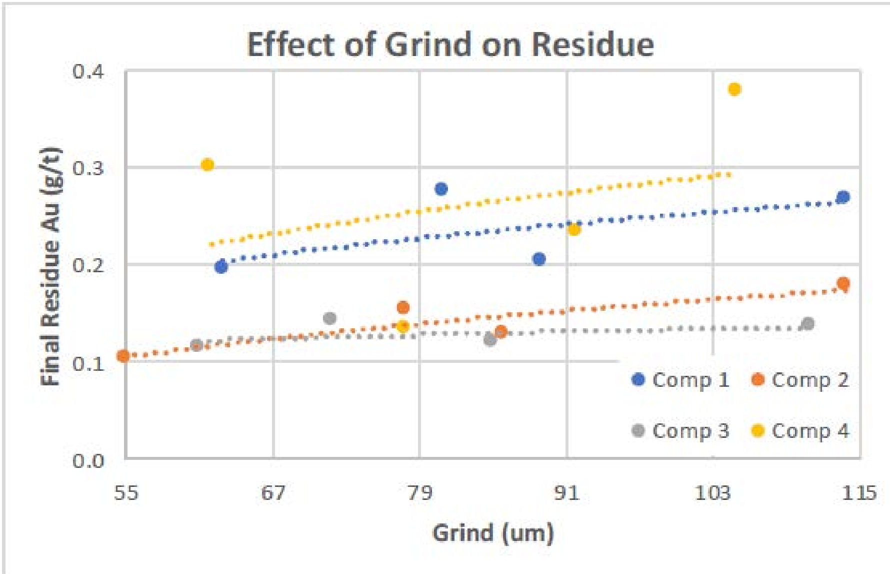


Figure 13.37 Grind Size versus Residue Grade Transition Material <2g/t



The 2019 testwork completed by Base Metal for the sulphide samples showed a trend for grind size versus residue grade. Refer to Figure 13.38. Although the trend is weak, a grind size of P_{80} of 75 µm for the sulphide plant was selected as most historical testwork supports this grind size and it provides a conservative approach for the SSAG design for optimum gold recovery. Further grind size optimization could be conducted prior to detail design.

Figure 13.38 Base Metal 2019 - Grind versus Final Au Residue



13.5.3 Oxide Plant Gold Extraction Models

Available metallurgical data were reviewed and screened based on certain criteria pertinent to the Project feasibility study:

- Ore type: Upper Oxide, Lower Oxide and Upper Transition.
- Grind size: 74 µm, 106 µm and 150 µm.
- Process: cyanidation leach at 24 hours and 48 hours.

Gold head grade and leach residue grade were plotted for each ore type. It was recognized that the data plotted were somewhat scattered resulting in poor correlation, therefore, no relationship can be formed for the head grades and the residue grades.

Two approaches were taken for the prediction of gold extraction:

1. Variable residue grade (using weak correlation between head grade and residue grade).
2. Constant residue grade (using average residue for each ore type).

The first approach of using variable residue grade appears to provide a lower gold extraction prediction for high-grade ores and higher recovery prediction for low-grade ore when compared to the second approach of using a constant residue (see Figure 13.39 to Figure 13.41).

The predicted gold production is essentially the same regardless of which approach is used. Since the correlation between head grade and residue (R^2) is low, it has been concluded that a constant residue approach (average residue) is more suitable for use. The gold extraction equations are presented in Table 13.55.

Table 13.55 Gold Extraction Equations

Upper Oxide	Extraction % = (Head Grade – 0.073) / Head Grade x 100
Lower Oxide	Extraction % = (Head Grade – 0.075) / Head Grade x 100
Upper Transition	Extraction % = (Head Grade – 0.078) / Head Grade x 100

Additional losses, equivalent to ~0.005 g/t, to account for gold losses via solution and carbon fines have been applied to the extraction equations in Table 13.55. Table 13.56 presents the gold recovery equations with these losses incorporated.

Table 13.56 Gold Recovery Equations

Upper Oxide	Extraction % = (Head Grade – 0.078) / Head Grade x 100
Lower Oxide	Extraction % = (Head Grade – 0.080) / Head Grade x 100
Upper Transition	Extraction % = (Head Grade – 0.083) / Head Grade x 100

Figure 13.39 Gold Extraction Equation for Upper Oxide Ore

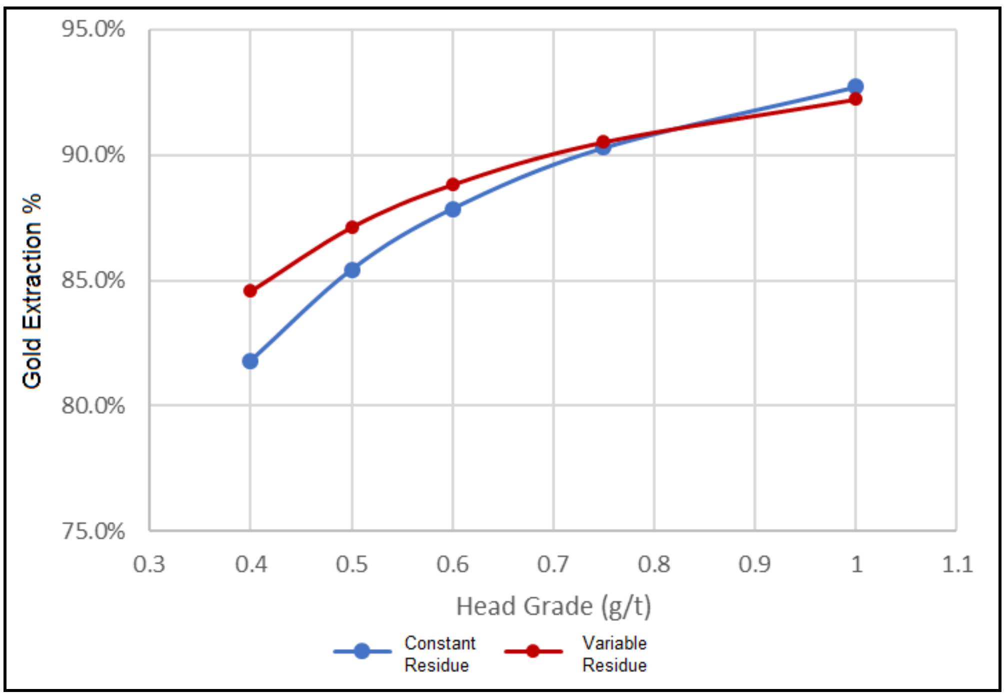


Figure 13.40 Gold Extraction Equation for Lower Oxide Ore

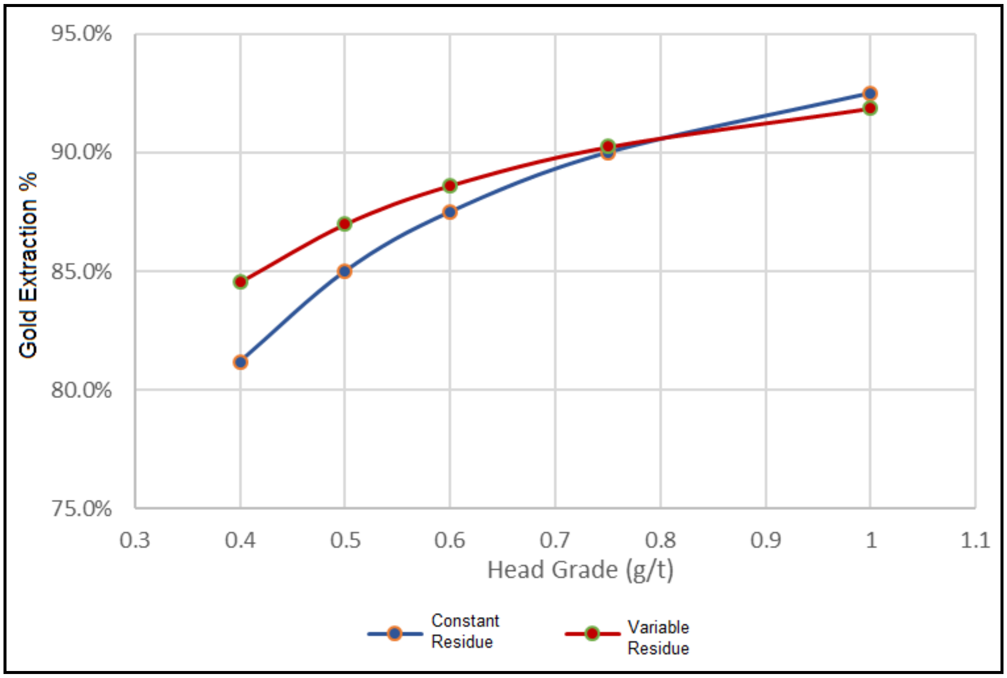
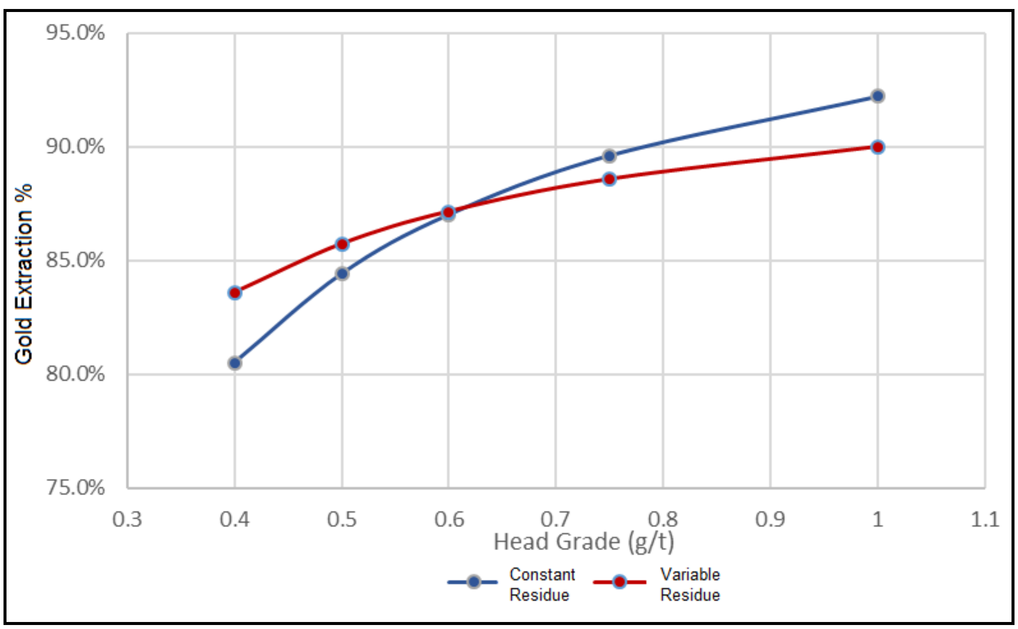


Figure 13.41 Gold Extraction for Upper Transition Ore



13.5.4 Sulphide Plant Gold Extraction Models

The gold extraction model for the sulphide plant was developed using drill core LeachWell results from Orezone's database. The results from the 2019 Base Metal testwork have not been considered in this model; however, the results were compared with the database.

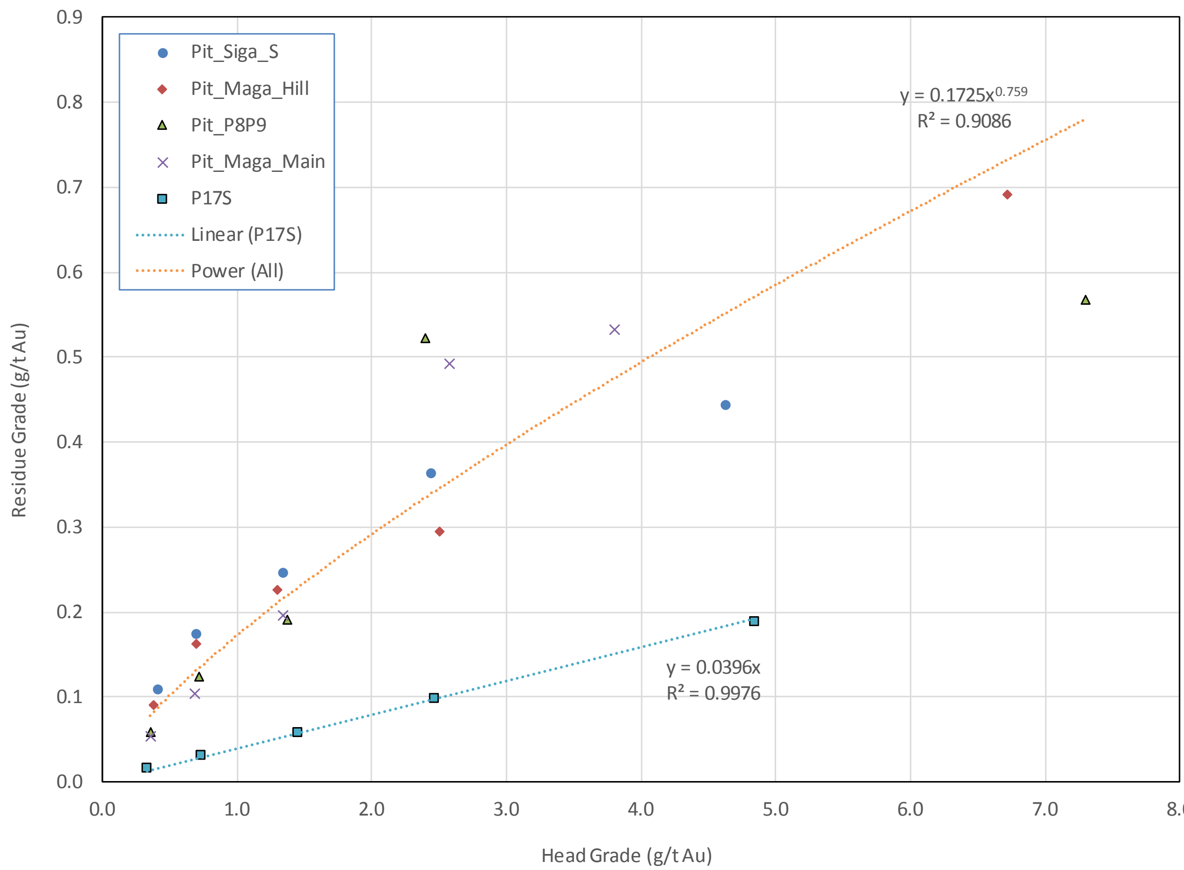
This LeachWell procedure employs an extended leach time, high cyanide concentration and powerful oxidizing agents to yield the best achievable gold recovery. The database contains head and residue grade analyses as determined by duplicate fire assays. The data was grouped into grade ranges (bins) to facilitate mine planning. Table 13.57 below provides the count of the number of samples in each grade bin per pit area. Note that this dataset includes only drill core intercepts which were labelled as fresh ore and which falls within the pits' boundaries.

Table 13.57 Number of Individual Intercept Samples per Grade Bin and Pit Domain (fresh ore)

Origin of sample	0.2 to 0.5 g/t	0.5 to 1.0 g/t	1.0 to 2.0 g/t	2.0 to 3.0 g/t	> 3.0 g/t
P17S	261	182	249	140	229
Pit_Maga_Hill	51	30	25	15	18
Pit_Maga_Main	33	50	36	8	8
Pit_Siga_S	82	177	77	28	13
Pit_P8P9	246	192	132	27	19
Totals:	**673**	**631**	**519**	**218**	**287**

The total number of samples within this population is 2,328. Approximately half of these are from the P17S pit. With respect to grade, about 12% of the samples fell within the >3 g/t range. Figure 13.42 summarizes the averaged residue grade versus averaged head grade for each grade bin per domain.

Figure 13.42 Residue Grade versus Head Grade from Leachwell Tests (Sulphide Samples)



It is evident from the graph that the P17S pit samples behaved differently from all of the other samples in that they yielded noticeably lower residue grades at comparable head grades. The LeachWell test results for this pit shows a near perfect (R^2 = 0.998) straight-line correlation between residue grade and head grade. The implication is that gold extraction is constant at 96% and independent of head grade.

The data for all of the other pits/domains can be grouped together and described by a single residue versus head grade trendline. In this instance, a power equation yielded a marginally better R2 than a simple straight line fit. This equation is as follows:

Residue grade (g/t) = 0.1725*(Head Grade in g/t)$^{0.76}$

This equation can be rewritten to derive a model which describes gold extraction as a function of head grade.

The gold extraction models for the sulphide ore are as follows:

- Pit P17S: Gold extraction % = 96%.
- All other pits: Gold extraction % = 100 - 17.25*HG$^{-0.24}$

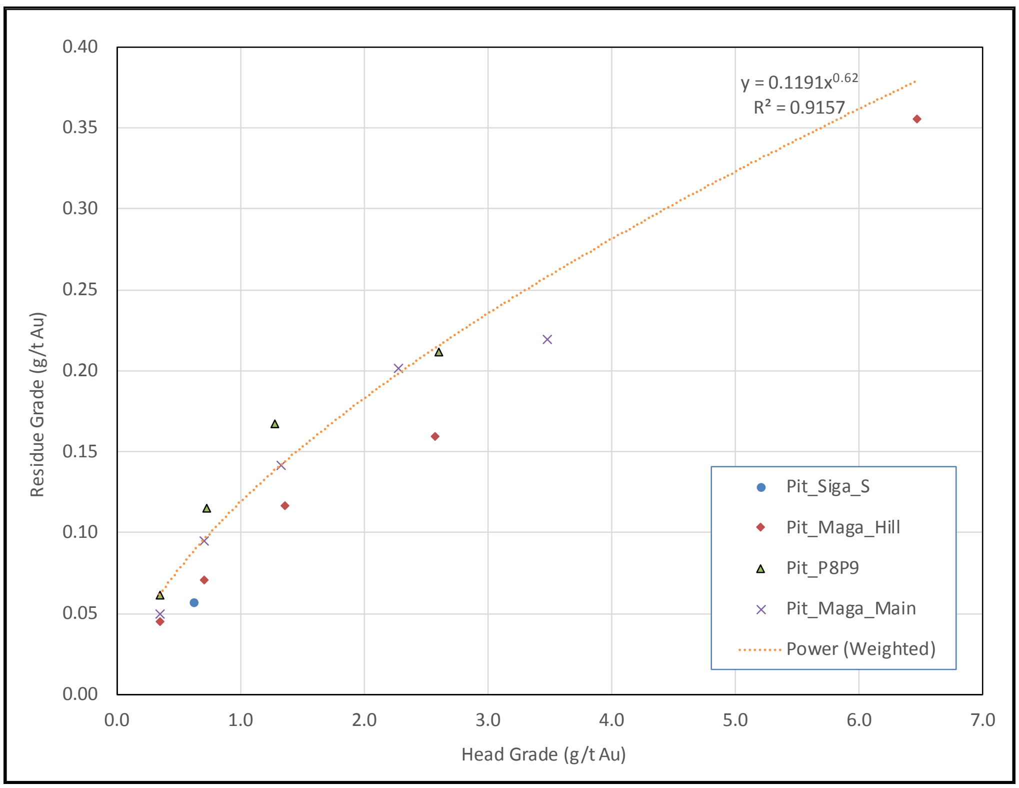
A similar exercise was performed for the transition ore samples. Table 13.58 summarizes the number of transition ore samples within each grade bin per domain/pit.

Table 13.58 Number of Individual Intercept Samples per Grade Bin and Pit Domain (Transition Ore)

Origin of sample	0.2 to 0.5 g/t	0.5 to 1.0 g/t	1.0 to 2.0 g/t	2.0 to 3.0 g/t	> 3.0 g/t
P17S	2	3	3	0	1
Pit_Maga_Hill	46	27	17	10	11
Pit_Maga_Main	43	44	34	6	3
Pit_Siga_S and _E	77	39	16	3	2
Pit_P8P9	156	118	52	15	9
Totals :	**324**	**231**	**122**	**34**	**26**

The total number of samples within this population was 737. Almost half of these samples are from pit P8P9, which contains approximately two thirds of the transition ore reserves. Figure 13.43 summarizes the averaged residue grade versus averaged head grade for each grade bin per domain for the transition ore dataset.

Figure 13.43 Residue Grade versus Head Grade from LeachWell Tests (Transition Samples)



Given the low number of transition samples from P17S (10 compared to 1079 sulphide samples) and the fact that it appears to be similar to the sulphides from the pit it is concluded that the assumption of a constant 96% recovery is sufficient to be used for both sulphide and transition samples from this domain.

The best-fit trendline for all other domains is different from the best fit recorded for fresh ore. The shape of the curves is similar, as evidenced by the similarity in the exponent values (0.759 versus 0.62). However, the difference in the constant value indicates that the transition ores will yield gold extractions that are approximately 6% higher than fresh ore of the same head grade. It should be noted though that the trendline shown in Figure 13.43 is not just an arithmetic mean of all of the data points, but, rather, a weighted average fit which accounts for the abundance of the transition ores in the original mine plan. This weighting shifts the trendline towards the P8P9 data points as they represent approximately two thirds of all of the transition ores.

The gold extraction models for the transition ores are:

- For pit P17S: Gold extraction % = 96%
- All other pits: Gold extraction % = 100 - 11.91*$HG^{-0.38}$

Gold recovery models are derived from gold extraction minus the predicted losses and inefficiency discount. The gold losses from solution and carbon fines combined equates to approximately 0.55%. An additional 0.5% discount is then applied in order to account for LeachWell tests being performed under optimum laboratory leach conditions.

After considering losses and inefficiency discount, the gold recovery models for the fresh ore, transition ore and P17S ore are presented by the following equations:

- Fresh Ore Gold Recovery % = 100 - 17.25*$HG^{-0.24}$ - 1.05
  = 98.95 - 17.25*$HG^{-0.24}$
- Transition Ore Gold Recovery % = 100 - 11.91*$HG^{-0.38}$ – 1.05
  = 98.95 - 11.91*$HG^{-0.38}$
- Pit P17S Only Gold Recovery % = 96% - 1.05%
  = 94.95%

Figure 13.44 and Figure 13.45 show individual data points plotted for the entire database of sulphide and transition test results. The graphs are truncated at a maximum head grade of 2.0 g/t as most of the data points fall within this range. The two extraction model curves are shown, as well as individual results from the latest (2019) testwork. The data exhibits a substantial degree of scatter from the average.

Figure 13.44 Residue Grade versus Head Grade from LeachWell Tests (Sulphide Samples)

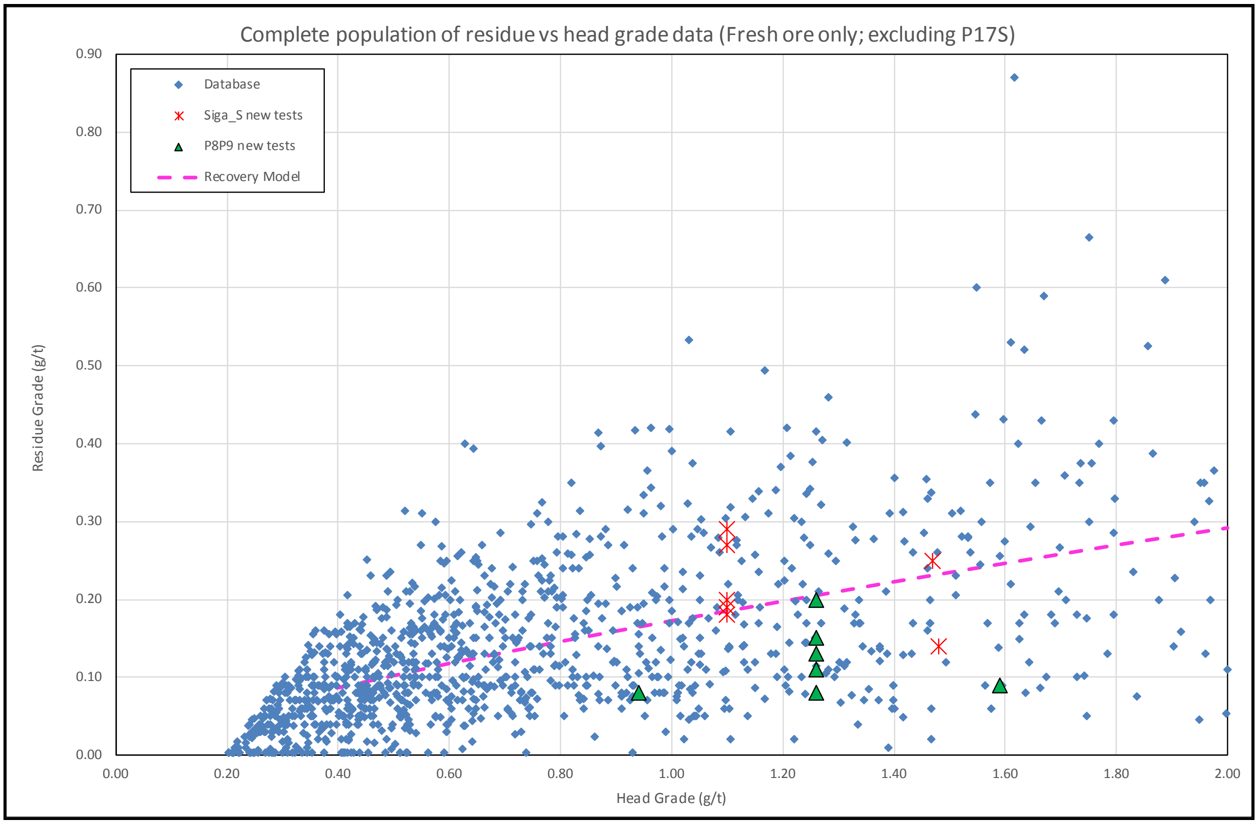
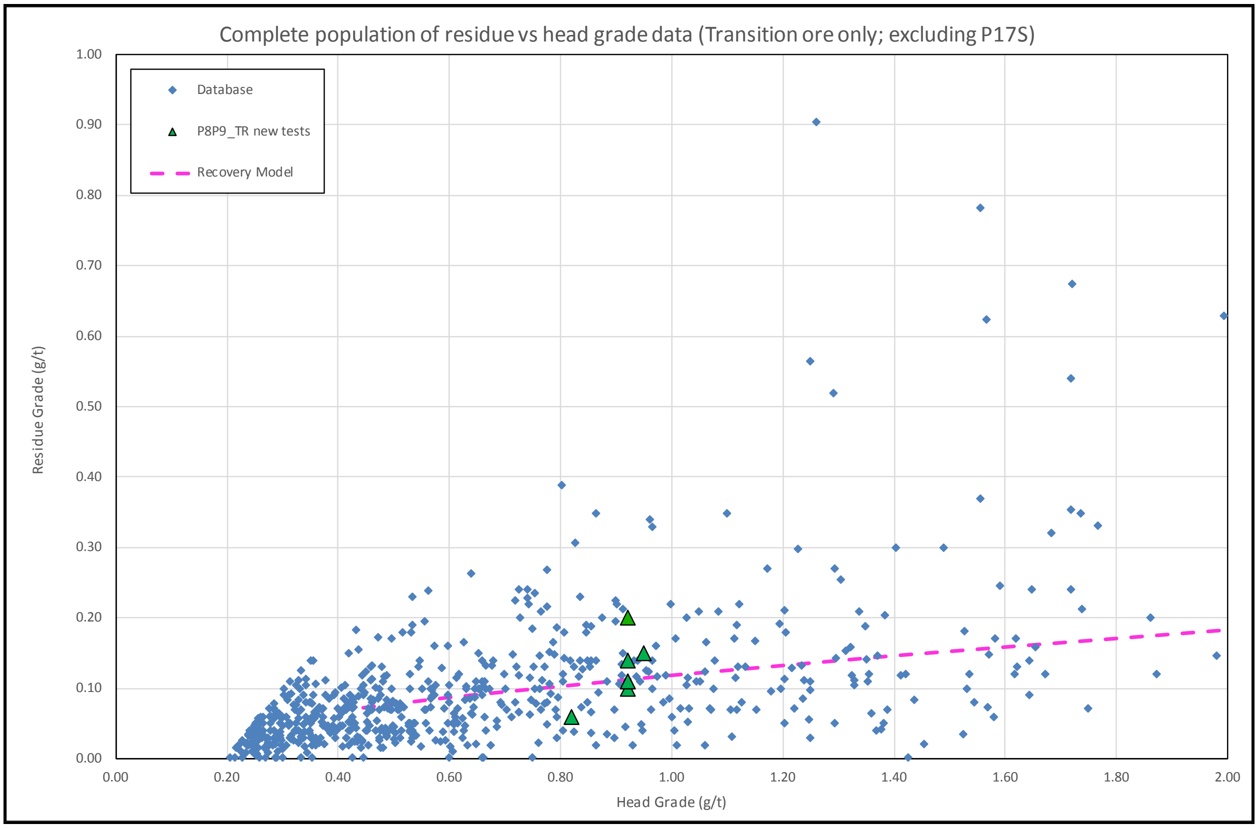


Figure 13.45 Residue Grade versus Head Grade from LeachWell Tests (Transition Samples)



13.5.5 Validation of Sulphide Recovery Equations against Testwork Database

A dataset, provided by Orezone and containing all historical testwork results, was used to validate the recovery equations for the Sulphide ores. Table 13.59 summarizes the average gold extractions recorded within this database by pit and ore type. Note that the database was filtered to extract a dataset containing only samples that fall within the pit boundaries and were labelled as either sulphide (fresh) ore or lower transition ore.

Table 13.59 Gold Extraction from Historical Metallurgical Testwork Database

	Gold Extraction (%)		Tonnes Distribution (%)	
	Sulphide	Lower transition	Sulphide	Lower transition
Maga	84.8	87.9	19.6	5.7
P17S	93.1	-	5.8	0.2
P8P9	77.4	86.4	23.5	10.3
Siga S	83.7	79.2	27.3	7.7

The weighted average gold extraction, when applying the tonnes distribution as shown above, is 83.0%. By comparison, the gold extraction models derived from the database above yields a weighted average gold extraction of 85.6% at a head grade of 1.25 g/t.

13.5.6 Oxide Plant Major Reagent Consumption

The SGS lime demand test was conducted by adding lime to the Bomboré ore slurry to maintain a pH of 10.5 for a period of 24 hours. The results show some lime buffering at the target pH. Lycopodium reviewed the results and estimated the consumption for maintaining ore slurry at 10.3 pH instead of 10.5 pH. In order to estimate this, the cumulative lime consumption value for reaching 10.3 pH was added to the incremental lime addition values required to maintain a constant pH for the 24-hour leach period. (Table 13.60).

Table 13.60 Estimated Lime Additions for 10.3 pH for Oxide Ore

Sample	CaO kg/t ore for 10.5 pH	Estimated CaO kg/t ore for 10.3 pH	Diff. (kg/t) pH 10.3	%Diff.
N-LG-OX	2.19	1.76	0.43	
N-MG-OX	1.48	1.30	0.17	
S-LG-OX	3.01	2.66	0.35	
S-MG-OX	3.48	3.19	0.29	
Oxide Average*	2.54	2.23	0.31	-12%
S-MG-TrU-M13	1.84	1.46	0.38	
N-MG-TrU-12	1.11	0.97	0.14	
S-MG-TrU-S3	2.27	2.07	0.21	
N-MG-TrU-S4	2.22	1.47	0.75	
N-MG-TrU-M13	3.00	2.64	0.36	
N-LG-TrU-M13	1.94	1.74	0.20	
Transition Average*	2.06	1.72	0.34	-16%

** Not weighted average*

Cyanide (NaCN) consumptions from all the relevant test programs were first listed for consideration. For leach tests that were conducted for longer than 24 hours, only the consumption from the 24-hour leach time was used from the raw data sheet in the test report. Afterward, any possible outliers were identified and excluded from the calculated averages. Lastly, only unique samples were considered because any sample with multiple tests conducted were averaged to give only one data point for the analysis.

The following were also considered when estimating the cyanide consumption:

- AMMTEC tests were removed despite g/L NaCN being maintained at 0.5 because oxygen sparging during the test was too aggressive.
- Only tests with 0.5 g/L NaCN were kept from the McClelland tests. If multiple tests were done on one sample, they were averaged first prior to being used as a data point in the final average.
- 2017/2018 SGS tests conducted prior to January 2018 were removed as excess cyanide was used in those tests. Lower transition samples were also excluded since they are not currently included in the mine plan.

The cyanide consumption data analyzed are summarized in Table 13.61 along with the average consumption value.

Table 13.61 Estimation of Cyanide Consumption for Oxide Ore

Test Program	Sample ID	Type of Test	Ore Type	NaCN Consumption (kg/t)
2012 McClelland	MGO	Bottle-roll (Direct)	Oxide	0.09
2012 McClelland	HGO	Bottle-roll (Direct)	Oxide	0.09
2012 McClelland	Oxide Master Composite	Leach/CIP	Oxide	0.42
2018 SGS	S-MG-OX	Bottle-roll	Oxide	0.23
2018 SGS	S-LG-OX	Bottle-roll	Oxide	0.14
			Average:	**0.19**
2018 SGS	N-LG-TrU-S3	Bottle-roll	Transition	0.28
2018 SGS	N-MG-TrU-I2	Bottle-roll	Transition	0.50
2018 SGS	S-MG-TrU-MI3	Bottle-roll	Transition	0.01
2018 SGS	N-LG-TrU-I2	Bottle-roll	Transition	0.22
2018 SGS	N-LG-TrU-MI3	Bottle-roll	Transition	0.09
2018 SGS	N-MG-TrU-S4	Bottle-roll	Transition	0.03
2018 SGS	N-LG-TrU-I1C	Bottle-roll	Transition	0.38
2018 SGS	N-LG-TrU-S4	Bottle-roll	Transition	0.26
2018 SGS	N-MG-TrU-I1C	Bottle-roll	Transition	0.31
2018 SGS	S-MG-TrU-S3	Bottle-roll	Transition	0.15
			Average:	**0.22**

13.5.7 Sulphide Plant Major Reagent Consumption

Similar to the oxide plant, a review of the historical and new metallurgical testwork was completed to estimate the major reagents consumption for the sulphide plant. Table 13.62 summarizes the reagent consumption averages from the review.

Table 13.62 Major Reagents Consumption for Sulphide Ore

Area	NaCN (kg/t)	Quicklime (kg/t)
Old Tests - P17	1.42	0.66
Old Tests - All other pits	0.35	0.93
Old Tests - Weighted Average	0.41	0.91
New Tests - Composites 1-4	0.67	1.69
New Tests – Variability Samples	0.41	1.45
Average of Old and New	**0.50**	**1.35**

Note that the cyanide consumption from the old tests presented in Table 13.62 are from tests performed at the lowest constant cyanide concentration of 0.5 g/L only. It was observed that tests performed at higher cyanide levels yielded significantly elevated consumption rates for no discernable improvement in recovery. The weighted average cyanide consumption from the old tests was 0.41 kg/t and the quicklime consumption was 0.91 kg/t. These consumption rates were then combined with the new 2019 testwork results to yield design consumption rates of 0.5 kg/t NaCN and 1.35 kg/t CaO for adoption into the sulphide plant process design criteria.

13.5.8 Summary of Metallurgical Design Criteria

A summary of the metallurgical inputs to the oxide plant and sulphide plant process design criteria are presented in Table 13.63 and Table 13.64, respectively.

Table 13.63 Summary of Metallurgical Criteria for Oxide Plant

Criteria	Units	Design	Notes/Source
Plant Throughput	tpa	5,200,000	Orezone
Ore Type	-	Upper & Lower Oxide Upper Transition	Mine plan
Design Ore Blend - Upper & Lower Oxide - Lower Transition	% %	85 15	Mine plan Mine plan
Head Grade - Gold (Design) - Gold (LOM average)	g/t Au g/t Au	1.0 0.67	Lycopodium/Orezone Mine plan
Gold Recovery Estimation at 1 g Au/t - Upper Oxide - Lower Oxide - Upper Transition - Per Design Ore Blend	 % % % %	 92.2 92.0 91.7 92.0	 Recovery plan Recovery plan Recovery plan Calculated
Ore Specific Density Ore Bulk Density Crushing Work Index (CWi) Rod Mill Work Index (RWi) Bond Ball Mill Work Index (BWi) Bond Abrasion Index (Ai)	t/m^3 t/m^3 kWh/t kWh/t kWh/t g	2.8 1.65 7.7 5.8 4.8 0.031	Testwork Lycopodium/Orezone Testwork Testwork Testwork Testwork
Grind Size P_{80} CIL Circuit Residence Time CIL Slurry Density (for saprolitic ore)	µm hrs % solids	125 24 ~40%	Lycopodium Testwork Lycopodium
Thickener Solids Loading	$t/m^2 \cdot h$	0.60	Testwork
Sodium Cyanide Addition	kg/t NaCN	0.28	Testwork/Calculated
Quicklime Addition	kg/t CaO	1.86	Testwork/Calculated

Table 13.64 Summary of Metallurgical Criteria for Sulphide Plant

Criteria	Units	Design	Notes/Source
Plant Throughput	tpa	2,200,000	Orezone
Ore Type	-	Lower Transition Upper & Lower Sulphide	Mine plan
Design Ore Blend - Lower Transition - Upper & Lower Sulphide	% %	24 76	Mine plan Mine plan
Head Grade - Gold (Design) - Gold (LOM average)	g/t Au g/t Au	1.25 1.22	Lycopodium/Orezone Mine plan
Gold Recovery at 1.25 g/t Au - Lower Transition - Upper & Lower Sulphide - Pit P17S Only* - Per Design Ore Blend	 % % % %	 88.0 82.6 94.95 84.6	 Recovery model Recovery model Recovery model Calculated
Ore Specific Density Ore Bulk Density Crushing Work Index (CWi) Rod Mill work Index (RWi) Bond Ball Mill Work Index (BWi) A x b Parameter Bond Abrasion Index (Ai)	t/m^3 t/m^3 kWh/t kWh/t kWh/t g	2.8 1.65 19.8 17.1 16.9 27.0 0.258	Testwork Lycopodium/Orezone Testwork Testwork Testwork Testwork Testwork
Grind Size P_{80} CIL Circuit Residence Time CIL Slurry Density	µm hrs % solids	75 24 ~50%	Testwork Testwork Lycopodium
Thickener Solids Loading	$t/m^2 \cdot h$	0.50	Orezone
Sodium Cyanide Addition	kg/t NaCN	0.78	Testwork/Calculated
Quicklime Addition	kg/t CaO	1.35	Testwork/Calculated

** Amount of total processed ore for sulphide plant coming from Pit P17 is 6% per mine plan.*

13.6 Conclusions and Recommendations

13.6.1 Conclusions

The following conclusions can be drawn from the metallurgical testwork:

- Oxide, transition and sulphide ores at the Project are readily amenable to CIL whole ore cyanidation.
- Oxide plant: Gold recoveries are predicted to be over 90% for head grades over 0.80 g/t Au, high 80%'s for head grades of 0.55 g/t Au to 0.80 g/t Au, and low 80%'s for head grades of 0.4 g/t Au to 0.55 g/t Au.
- Sulphide plant: Gold recoveries are predicted to be over 80% for head grades over 0.70 g/t Au, and stay in the high 70%'s even for lower head grades.
- Optimum grind size for the oxide plant was determined to be P_{80} of 125 µm based on grind size and recovery relationship.

- Optimum grind size for the sulphide plant was selected to P_{80} of 75 µm however; testwork may allow further grind optimization.
- Leach extraction rates are essentially complete within 24 hours based on the observed leach kinetics.
- Oxygen addition is beneficial for sulphide ore leaching.
- Cyanide consumption rates are expected to be low, averaging about 0.19 kg/t NaCN ore for the oxide ore and about 0.37 kg/t NaCN ore for the sulphide ore.
- Lime consumption rates are expected to be moderate, averaging about 1.86 kg/t CaO ore for the oxide ore and about 1.35 kg/t CaO ore for the sulphide ore.

13.6.2 Recommendations

It is recommended that the following additional testwork be considered prior to finalizing the sulphide plant design criteria, as there is opportunity to improve recoveries and optimize reagent and power consumptions:

Sulphide Plant

- Confirm the grind size optimization to investigate opportunity of a coarser grind.
- Confirm the oxygen uptake to investigate whether air sparging can replace oxygen.
- Dynamic thickening tests at different loading rates to optimize the thickener diameter.
- Diagnostic leaching on selected composites to investigate samples with low LeachWell recoveries – give priority to results showing under 80% recovery.
- Comparison of bottle roll and LeachWell cyanidations at optimum conditions to validate the use of LeachWell data.
- Additional cyanide and lime demand tests at optimum leach conditions of 0.5 g/L NaCN, and 15 ppm DO at 48-hour leaching.
- Conduct additional variability testing; the 2019 program had only 8 variability samples.

In addition, it is recommended that during plant operations:

- Natural cyanide attenuation (free and WAD) be tested and monitored in the tailings storage facility.
- Site water quality (raw and process) be tested and monitored during the initial wet and dry seasons to document any seasonal impact of water quality.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Summary

Gold mineralization has been defined at shallow depths by reverse circulation (RC) drilling, diamond drilling, and trenching along a strike length of over 12 km. The gold mineralized zones have been modelled as a large number of sub-parallel, tabular zones that gradually change in strike from north-northwest, to northeast. Most of the mineralization wireframes are interpreted to dip moderately to the east or southeast. Review of the lithologic models shows that gold values are contained within all host rock types and can be seen to follow a stratiform orientation.

In order to keep the size of the various block model files within functionally manageable limits, the gold mineralization has been split into five separate block model areas, referred to as the North, South, P16, P17 and P17S areas. Together, the North and South block models contain the majority of the Mineral Resources. Low grade mineralized wireframe models were created for the 2016 Mineral Resource estimate using a grade threshold of approximately 0.20 g/t Au, and high-grade mineralized wireframe models that were created using a grade threshold of approximately 0.45 g/t Au.

Following completion of the 2016 Mineral Resource estimate, an additional set of LG mineralized wireframes was created for the North and South model areas using only the lower grade threshold of 0.20 g/t Au to capture material remaining outside the 2016 Mineral Resource estimate wireframes. There was also a further grade estimate completed for selected material outside all wireframes on an unconstrained basis (“third domain”) for the North, South, P16, and P17 model areas. The low-grade mineralized wireframe models and the third domain were used to extract a total of 3,207 and 146,372 assay results, respectively, from the four drill hole databases (North, South, P16, and P17) for analysis. The P17S high-grade deposit was modelled and interpolated in December 2018 following additional drilling and mineralization wireframe interpretations. The P17S drill hole database includes 108 diamond drill holes (16,423 m) and 54 reverse circulation holes (1,979 m), totalling 162 drill holes (18,402 m), and seven channels totalling 23.4 m.

Orezone has elected to use the capping method to reduce the influence of high-grade assay values. The selection of the various capping values was guided by the goal of achieving a target coefficient of variation (CoV) of less than approximately 2.0. This resulted in capping values that ranged from 1.50 g/t Au to 48.97 g/t Au for the low and high grade mineralized wireframe domain assays for the 2016 Mineral Resource estimate. A universal value of 5.00 g/t Au was used by RPA for both the January 2017 estimate low-grade mineralization wireframe domain and third domain assays. Capped assays were composited within the domain boundaries at 1.5 m length. Parts of P17 were composited at 1 m length. For the P17S resource, assays were composited to 1 m lengths with a minimum length of 0.25 m for each individual mineralization wireframe.

Gold grades within the 2016 Mineral Resource estimate mineralized wireframe models (low-grade, high-grade) for the North, South, and P16 areas were estimated using the ordinary kriging (OK) interpolation algorithm. The gold grades within the 2016 and 2017 Mineral Resource estimate wireframe models (low-grade, high-grade) for the P17 model area were estimated using the inverse distance squared (ID^2) interpolation algorithm. The gold grades inside the January 2017 additional low-grade mineralized wireframe models for the North and South areas were also estimated using the OK interpolation algorithm; there were no additional low-grade wireframe domain models in the P16 and P17 model areas. Hard boundaries were used to constrain the source composite files such that only those composite samples that are present within a specified wireframe were used to estimate block grades. Similarly, hard boundaries were used to constrain coding of the block model where only those blocks that are contained within the specified mineralized wireframe model were permitted to receive estimated gold grades. Gold grades for the January 2017 estimate third domain in all model areas were estimated using a two-step process using the inverse distance cubed (ID^3) interpolation algorithm. The first step used only composites outside wireframes and above 0.20 g/t Au to flag blocks with a grade above 0.00 g/t Au from a minimum of two composites, then on the second step used all composites outside wireframes to estimate the gold grade of the previously flagged blocks. Gold grades for the P17S December 2018 estimate were estimated within the wireframe models by ID^2.

Data from 276 new holes totalling 15,387 m, located within the resource area but outside P17S, were received after the resource database was finalized for the January 5, 2017 resource statement. RPA reviewed the results and is of the opinion that the resource model is still appropriate to be used as the basis for this Technical Report and that the effective date should remain at January 5, 2017.

Measured Mineral Resources comprise that mineralized material that has been outlined with a drill hole density of at least 25 m x 25 m. Indicated Mineral Resources comprise that mineralized material that has been outlined with a nominal drill hole density of 25 m x 50 m. Inferred Mineral Resources comprise the mineralized material that has been outlined with a nominal drill hole density of 100 m x 100 m and to within a depth of 100 m below the bottom of the drill hole coverage. Clipping polygons representing the various Mineral Resource categories were created for each of the oxidation layers to ensure the continuity and consistency of the classification category. These clipping polygons were used to code final classification into each of the four block models.

A number of cut-off grades were developed for the Project that reflect the varying processing costs and metallurgical recoveries of the different oxidation layers and the additional transportation costs for mineralized material that is located distant to the proposed processing plant. A gold price of US$1,400/oz was used for all cut-off grades for reporting of the Mineral Resources. To fulfill the NI 43-101 requirement of "reasonable prospects for eventual economic extraction", RPA prepared a preliminary open pit shell to constrain the block models for resource reporting purposes. Additional criterion to constrain the Mineral Resource report included several "non-permitted" areas related to environmentally sensitive areas and mineralized areas being set aside for the benefit of local artisanal miners.

RPA provided an updated Mineral Resource estimate with an effective date of January 5, 2017 ("2017 Mineral Resources") by incorporating the oxide material within the previously excluded "Restricted Zones", the sulphide resources comprising lower transition and fresh layers and all drilling completed to that date on the high-grade P17S deposit.

The Mineral Resource estimate for the P17S area has an effective date of December 21, 2018. RPA notes that the effective date of the deposit as a whole remains January 5, 2017 since the bulk of the Mineral Resources (North, South, P16, and P17) has not been updated since that estimate. A fifth block model has been added for the P17S deposit. The P17S deposit is described in detail in section 14.15.

The updated 2017 Mineral Resource estimate comprises five separate block models, including P17S, which have been combined into a global resource as shown in Table 14.1. The 2017 Mineral Resource estimate conforms to Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM (2014) definitions).

Table 14.1 Summary of the Mineral Resources as of January 5, 2017

Classification		Measured			Indicated			Measured + Indicated			Inferred		
	Cut-off	Tonnage	Grade	Contained	Tonnage	Grade	Contained	Tonnage	Grade	Contained	Tonnage	Grade	Contained
	g/t Au	000 t	g/t Au	Au koz	000 t	g/t Au	koz Au	000 t	g/t Au	Au koz	000 t	g/t Au	koz Au
Oxides	0.20	31,600	0.62	628	75,300	0.53	1,273	106,900	0.55	1,901	20,900	0.40	265
Sulphides	0.2/0.38	9,000	0.90	260	113,600	0.79	2,894	122,600	0.80	3,154	32,400	0.81	842
TOTAL		**40,600**	**0.68**	**888**	**188,900**	**0.69**	**4,167**	**229,400**	**0.69**	**5,055**	**53,300**	**0.65**	**1,107**

Notes:

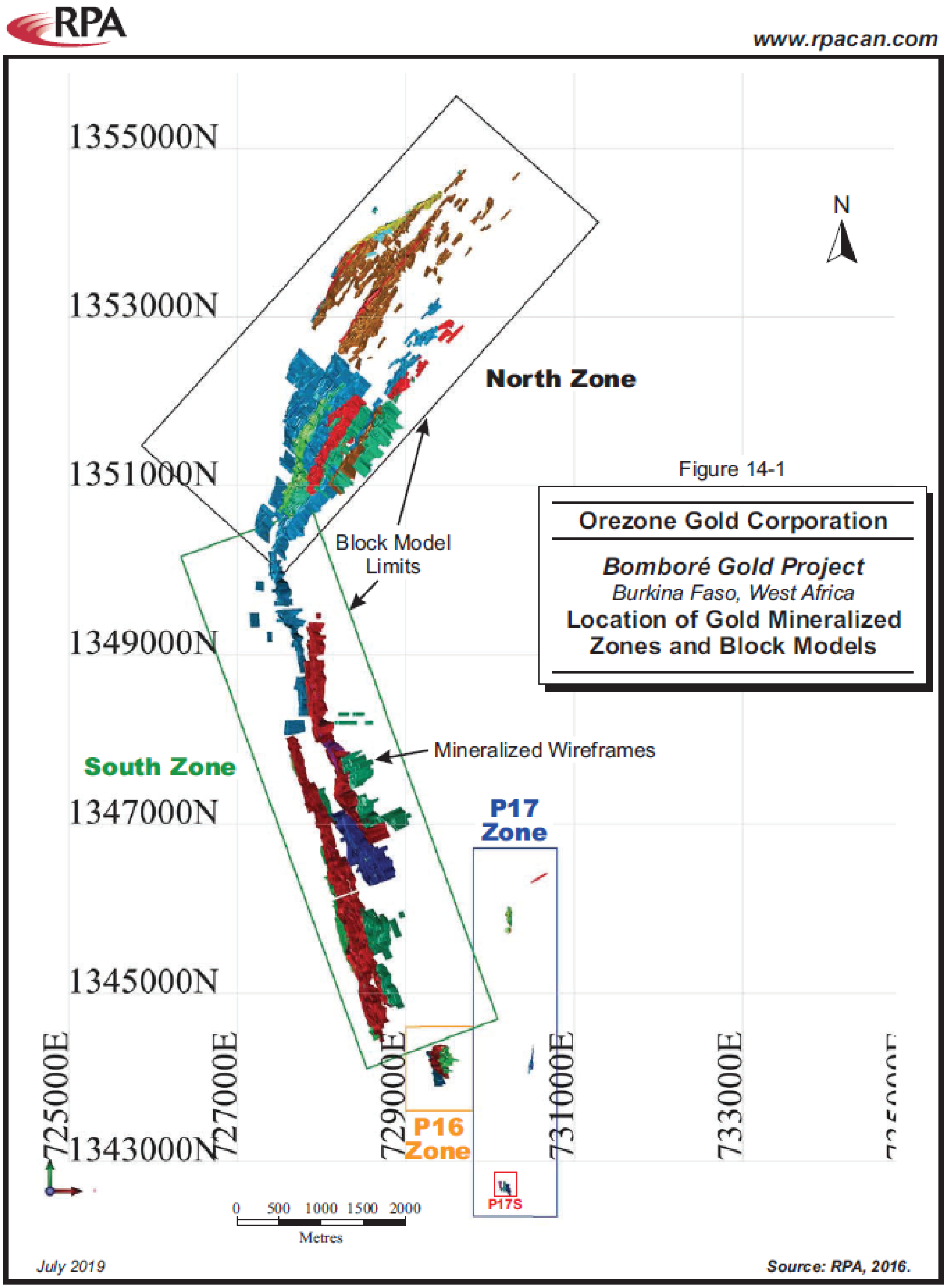
1. *CIM (2014) definitions were followed for Mineral Resources.*
2. *Mineral Resources are inclusive of Mineral Reserves.*
3. *Oxide resources are made up of the regolith, saprolite, and upper transition layers reported at a cut-off of 0.2 g/t Au.*
4. *Sulphide resources are made up of lower transition and fresh layers reported at 0.2 g/t Au and 0.38 g/t Au respectively.*
5. *Mineral Resources have been constrained within a preliminary pit shell generated in Whittle software.*
6. *Mineral Resources are estimated using a long-term gold price of US$1,400/oz.*
7. *A minimum mining width of approximately 3 m was used.*
8. *Bulk densities vary by material type.*
9. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
10. *Numbers may not add due to rounding.*

RPA is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the 2017 Mineral Resource estimate.

14.2 Description of Databases

Gold mineralization at shallow depths has been defined by RC drilling, diamond drilling, and trenching along a strike length of over 12 km. For the most part, the gold mineralized zones have been modelled as a large number of sub-parallel, tabular zones that gradually change in strike from north-northwest to north-south to northeast (Figure 14.1). Most of the mineralization wireframes are interpreted to dip moderately to the east or southeast.

Figure 14.1 Location of Gold Mineralized Zones and Block Models



The gold mineralization along the 12 km strike length has been split into five separate block model areas in order to keep the size of the various block model files within functionally manageable limits. The block model areas are referred to as the North, South, P16 P17 and P17S areas. Together, the North and South block models contain the majority (95%) of the 2017 Mineral Resources. The mineralized wireframes have been grouped into various sub-domains within each of the block model areas.

The modelling of the host lithologies and the extents of the gold mineralization at the Project is based on drill hole data only. No information obtained from trenching programs has been used, and outcrop data is too sparse to provide any meaningful information.

The drill hole data are contained in four separate databases that have been prepared using the GEMS mine modelling software package as the source database files. Drill hole data were converted and imported into the Surpac mine modelling package on an as-needed basis.

Each of the four drill hole databases was constructed using a similar structure to store information for lithology, weathering profile, oxidation level, gold assays, density measurements, and various working tables that are required to be created as part of the block model estimation process. In the vast majority of cases, all drill holes were angled towards the west or the northwest.

The North model drill hole database contains the largest proportion of the drill hole data (5,502 drill hole records), followed by the South model (2,001 drill hole records). The P16 and P17 models are defined by 165 and 170 drill holes, respectively (Table 14.2). In total, the four databases contain drill hole information for 7,838 drill holes. The cut-off date for the drill hole database is January 5, 2017. The location of the drill holes, which were used to prepare the 2017 Mineral Resource estimate, are shown in Figure 14.2.

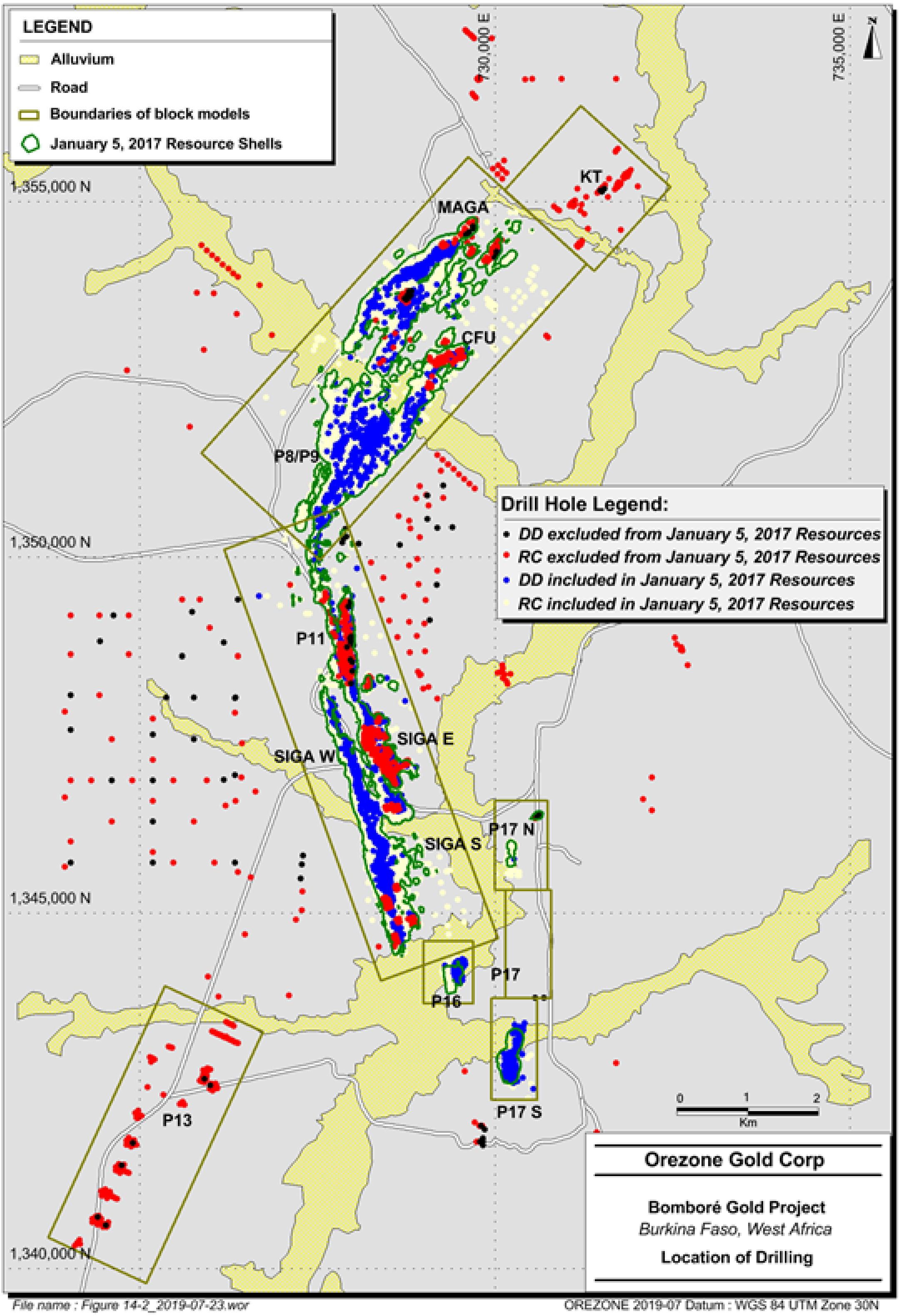
Table 14.2 Summary of the Resource Database

Table Name	Data Type	Table Type	Records
North Model			
collar			5,502
survey			16,507
au_3d_domain	interval	time-independent	9,489
au_assay_waste	interval	time-independent	183,086
Drillholes_NITON_lithologies	interval	time-independent	279,807
Drillholes_NITON_oxidation	interval	time-independent	279,807
Drillholes_NITON_weathering	interval	time-independent	279,807
Drillholes_assays	interval	time-independent	267,443
Drillholes_comp200ppb	interval	time-independent	12,181
Drillholes_comp450ppb	interval	time-independent	5,249
Drillholes_density	Point	time-independent	48,326
Drillholes_structure_Foliations	Point	time-independent	0
Drillholes_structure_beddings	Point	time-independent	0
Drillholes_structure_fract_jts	Point	time-independent	0

Table Name	Data Type	Table Type	Records
ResEstim_intersect_450_envelops	interval	time-independent	3,232
ResEstim_intersect_LG_envelops	interval	time-independent	8,485
ResEstim_intersect_R_envelops	interval	time-independent	3,914
flagging_orezones_200	interval	time-independent	61,614
flagging_orezones_450	interval	time-independent	18,619
flagging_orezones_R	interval	time-independent	8,121
styles			291
translation			0
South Model			
collar			2,001
survey			8,560
Drillholes_NITON_lithologies	interval	time-independent	156,046
Drillholes_NITON_oxidation	interval	time-independent	133,011
Drillholes_NITON_weathering	interval	time-independent	133,011
Drillholes_assays	interval	time-independent	133,011
Drillholes_comp200ppb	interval	time-independent	6,086
Drillholes_comp450ppb	interval	time-independent	2,810
Drillholes_density	Point	time-independent	37,010
Drillholes_structure_Foliations	Point	time-independent	15,084
Drillholes_structure_beddings	Point	time-independent	3,262
Drillholes_structure_fract_jts	Point	time-independent	460
ResEstim_intersect	interval	time-independent	0
ResEstim_intersect_450_envelops	interval	time-independent	1,989
ResEstim_intersect_LG_envelops	interval	time-independent	4,067
ResEstim_intersect_R_envelops	interval	time-independent	1,802
composites	interval	time-independent	35,562
flagging_orezones_200	interval	time-independent	29,130
flagging_orezones_450	interval	time-independent	11,418
flagging_orezones_R	interval	time-independent	5,753
styles			267
translation			0
P16 Model			
collar			165
survey			573
Drillholes_NITON_lithologies	interval	time-independent	11,959
Drillholes_NITON_oxidation	interval	time-independent	11,444
Drillholes_NITON_weathering	interval	time-independent	11,444
Drillholes_assays	interval	time-independent	11,444
Drillholes_comp200ppb	interval	time-independent	359
Drillholes_comp450ppb	interval	time-independent	225

Table Name	Data Type	Table Type	Records
Drillholes_density	Point	time-independent	3,347
Drillholes_structure_Foliations	Point	time-independent	17,452
Drillholes_structure_beddings	Point	time-independent	3,777
Drillholes_structure_fract_jts	Point	time-independent	623
ResEstim_intersect_450_envelops	interval	time-independent	284
ResEstim_intersect_LG_envelops	interval	time-independent	431
ResEstim_intersect_R_envelops	interval	time-independent	161
comps_autot_200	interval	time-independent	888
comps_autot_450	interval	time-independent	856
flagging_orezones_200	interval	time-independent	1,538
flagging_orezones_450	interval	time-independent	1,629
flagging_orezones_R	interval	time-independent	539
translation			0
P17 Model			
collar			170
survey			748
Drillholes_NITON_lithologies	interval	time-independent	13,157
Drillholes_NITON_oxidation	interval	time-independent	11,240
Drillholes_NITON_weathering	interval	time-independent	11,240
Drillholes_assays	interval	time-independent	11,240
Drillholes_comp200ppb	interval	time-independent	373
Drillholes_comp200ppb_b	interval	time-independent	223
Drillholes_comp450ppb	interval	time-independent	8,361
Drillholes_density	Point	time-independent	3,275
Drillholes_structure_Foliations	Point	time-independent	17,452
Drillholes_structure_beddings	Point	time-independent	3,777
Drillholes_structure_fract_jts	Point	time-independent	17,452
ResEstim_intersect	interval	time-independent	170
flag_orezone_200	interval	time-independent	701
flag_orezone_450	interval	time-independent	701
flag_orezone_r	interval	time-independent	486
p17_comps_450	interval	time-independent	904
styles			88
translation			0

Figure 14.2 Drill Hole Plan Map



The spacing of the drill holes varies for each of the four model areas and is summarized in Table 14.3.

Table 14.3 Summary of the Drill Hole Spacing

Model	Drill Hole Spacing	Remarks
North	25 m spacing on 50 m spaced section planes.	Some portions of the mineralization are defined by drill holes at 25 m x 25 m spacing.
South	25 m spacing on 50 m spaced section planes.	
P16	25 m x 25 m.	
P17	25 m x 25 m.	

14.3 Lithology and Mineralization Wireframes

14.3.1 Weathering Profile

The gold mineralization at the Project is located in a non-glaciated terrain. Consequently, the weathering profile has largely remained in place except within paleo- and current drainage channels where the saprolite layer has been eroded to a various extent, and for a small amount of re-working of the upper portions of the weathering profile to form a thin alluvium layer in places. The weathering model consists of seven units:

- Regolith unit (discontinuous thin layer of secondary deposits).
- Saprolite: Upper and Lower units (upper and lower half of the completely oxidized zone).
- Saprock: Upper and Lower units (upper and lower half of the transition or partially oxidized zone).
- Sulphide: Upper and Lower units (primary or non-weathered sulphide zones, subdivided as the top 25 m sub-zone and the deeper sub-zone).

Digital surface models for each of these weathering units have been created from the visual information collected during logging of the drill core or RC chips and from geochemical and other petro-physical indicators.

14.3.2 Host Rock Lithology

The host rocks at the Project are comprised of a large sequence of metamorphosed clastic sediments that are Lower Proterozoic in age. The clastic sediments are composed of conglomerate, greywackes, and argillite/siltstone units that may contain graphitic material in places. The clastic sedimentary package has been intruded by rocks of gabbroic and granodioritic compositions. Many of the rock units contain a weakly to moderately well-developed foliation or fabric that strikes in a north-northwesterly direction in the southern portions of the mineralized trend. The strike gradually changes to a northeasterly direction in the northern portions of the mineralized trend. The foliations of the host rocks dip consistently moderately to the northeast, east, and southeast. A late-tectonic intrusion of quartz-feldspar porphyritic granite is present in the southern portions of the South model area.

Digital solids models of the major rock units were prepared on cross sections using information collected during logging of the drill core or RC chips. The cross-sectional lithology contacts were then linked together from section to section to form the final solid models that were subsequently used to code the block model. Examples of the lithologic interpretation and host rocks for the South model area are provided in Figures 14.3 and 14.4. Examples of the lithologic interpretation and host rocks for the North model area are provided in Figures 14.5 and 14.6.

Figure 14.3 Representative Cross Section, South Zone Area

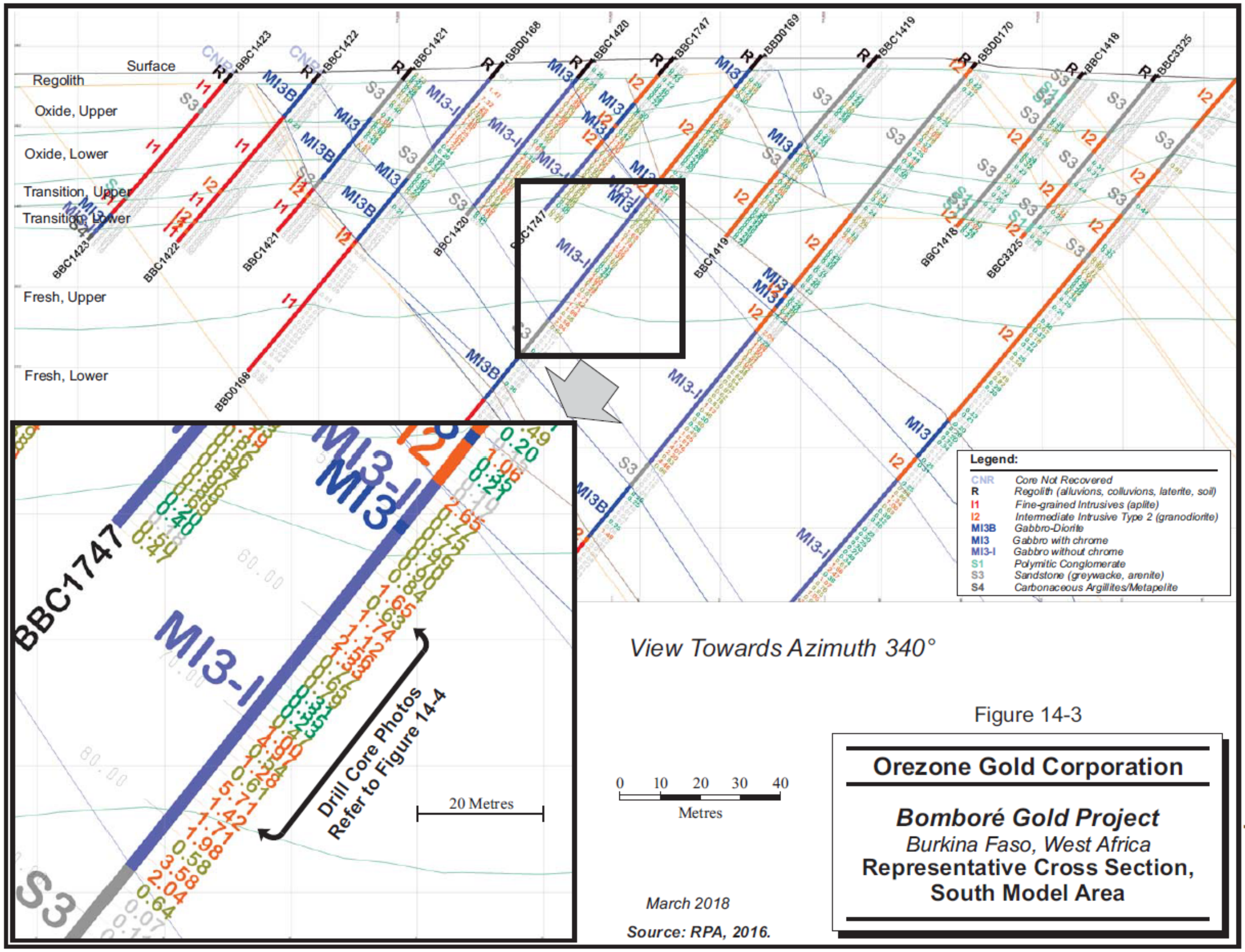


Figure 14.4 Mineralized Gabbro, Drill Hole BBD0169, South Zone Area

Figure 14-4

Orezone Gold Corporation

Bomboré Gold Project

Burkina Faso, West Africa

Mineralized Gabbro, Drill Hole BBD0169, South Model Area

March 2018

Source: RPA, 2016.

Figure 14.5 Representative Cross Section, North Zone Area

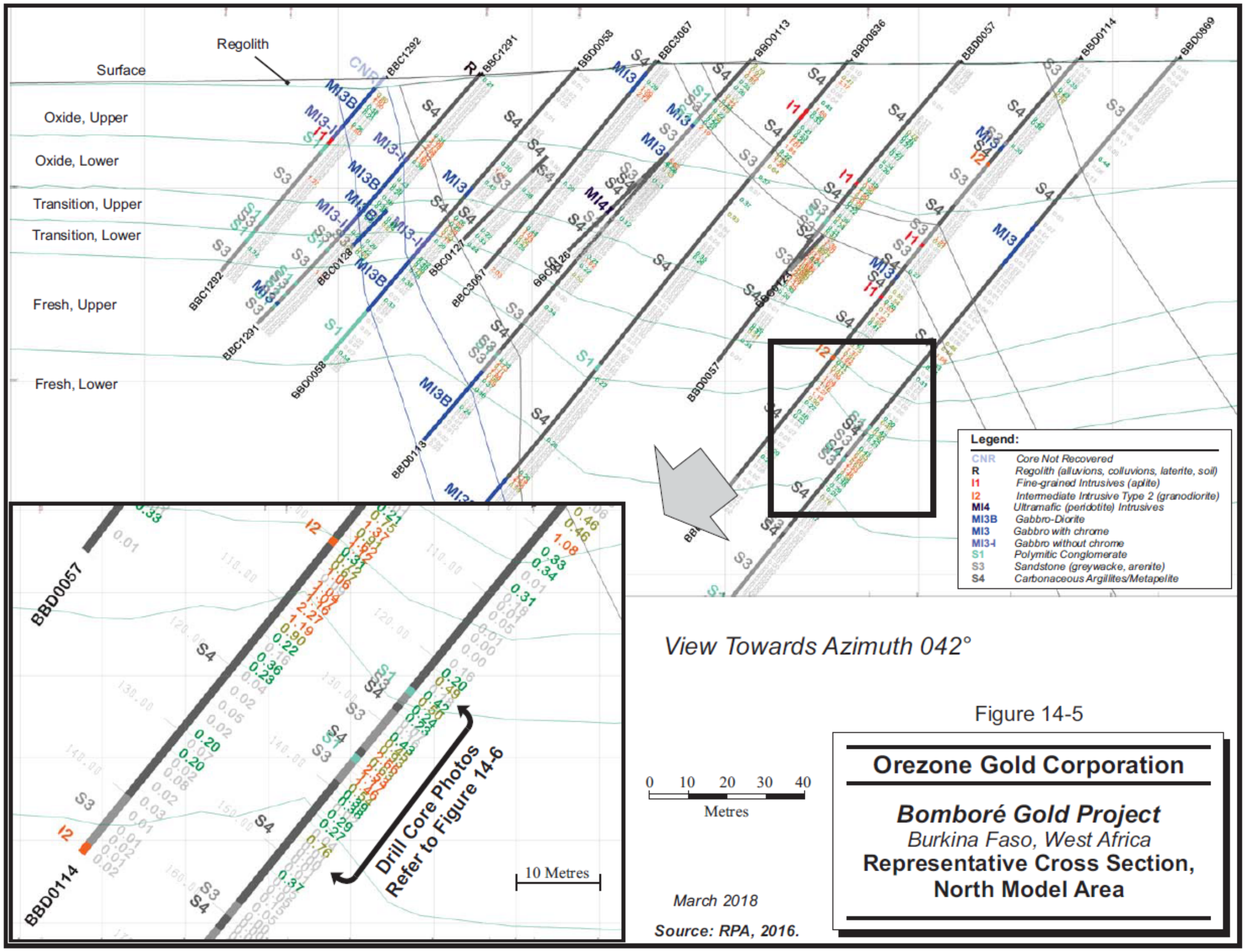


Figure 14.6 Mineralized Sediments, Drill Hole BBD0869, North Zone Area

Figure 14-6

Orezone Gold Corporation

Bomboré Gold Project

Burkina Faso, West Africa

Mineralized Sediments
Drill Hole BBD0869,
North Model Area

March 2018

Source: RPA, 2016.

14.3.3 Mineralization Wireframes

The mineralization wireframes for the September 2016 estimate were prepared by Orezone on cross sectional views using a two-tiered cut-off grade approach and a minimum width of three metres for the high-grade domains. A minimum width of 5 m was used for creation of the September 2016 estimate low grade domains.

For the 2017 Mineral Resources estimate RPA produced 391 new low-grade wireframe-constrained domain models in those areas where a minimum of three intervals longer than three metres and an overall grade above 0.20 g/t Au can be observed to be present in adjacent drill sections and there are a minimum of two such intersects in at least one section. For the P17S deposit, Orezone produced 22 new wireframe-constrained domain models at a nominal cut-off of 0.20 g/t Au.

A minimum width of 3 m was used by RPA for the creation of the new low grade domains for the January 2017 estimate. Sample intervals with assays results less than the nominated cut-off grade were included within the January 2017 estimate new low grade mineralized wireframes if the overall grade of the samples in that intercept was above 0.20 g/t Au.

The September 2016 estimate, January 2017 estimate low grade domain models, and December 2018 P17S estimate were created using a lower grade limit of approximately 0.20 g/t Au, while the September 2016 estimate high grade domain models were created using a grade limit of approximately 0.45 g/t Au. RPA considers the selection of 0.20 g/t Au to be appropriate for construction of mineralized wireframe outlines, as this value well reflects the lowest cut-off grade that is expected to be applied for reporting of the Mineral Resources in an open pit operating scenario. The selection of the threshold for the high-grade population was set as the inflection point in a plot of the coefficient of variation of composite samples as a function of cut-off grade.

Review of the drill hole information supports the interpretation that the gold mineralization follows a stratiform orientation for the most part. In some locales, the elevated gold grades can clearly be seen to be related to one specific host rock type. Consequently, the mineralization wireframes were constructed to sub-parallel the lithological contacts (Figures 14.7 and 14.8). The mineralization wireframe interpretations were cross-checked for continuity from section to section prior to construction of the final solid model.

Figure 14.7 Example of the Mineralized Wireframe Interpretation, South Model Area

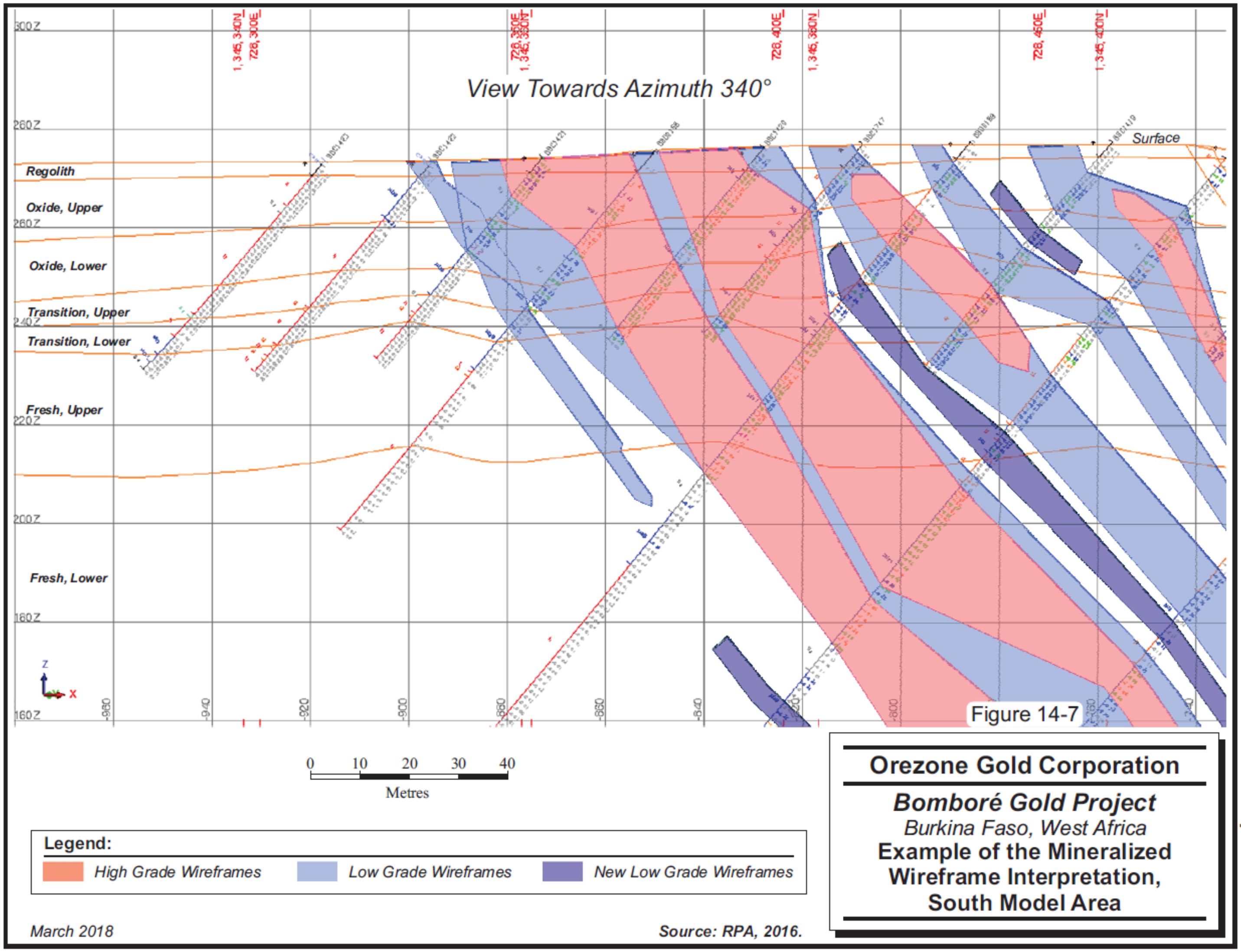
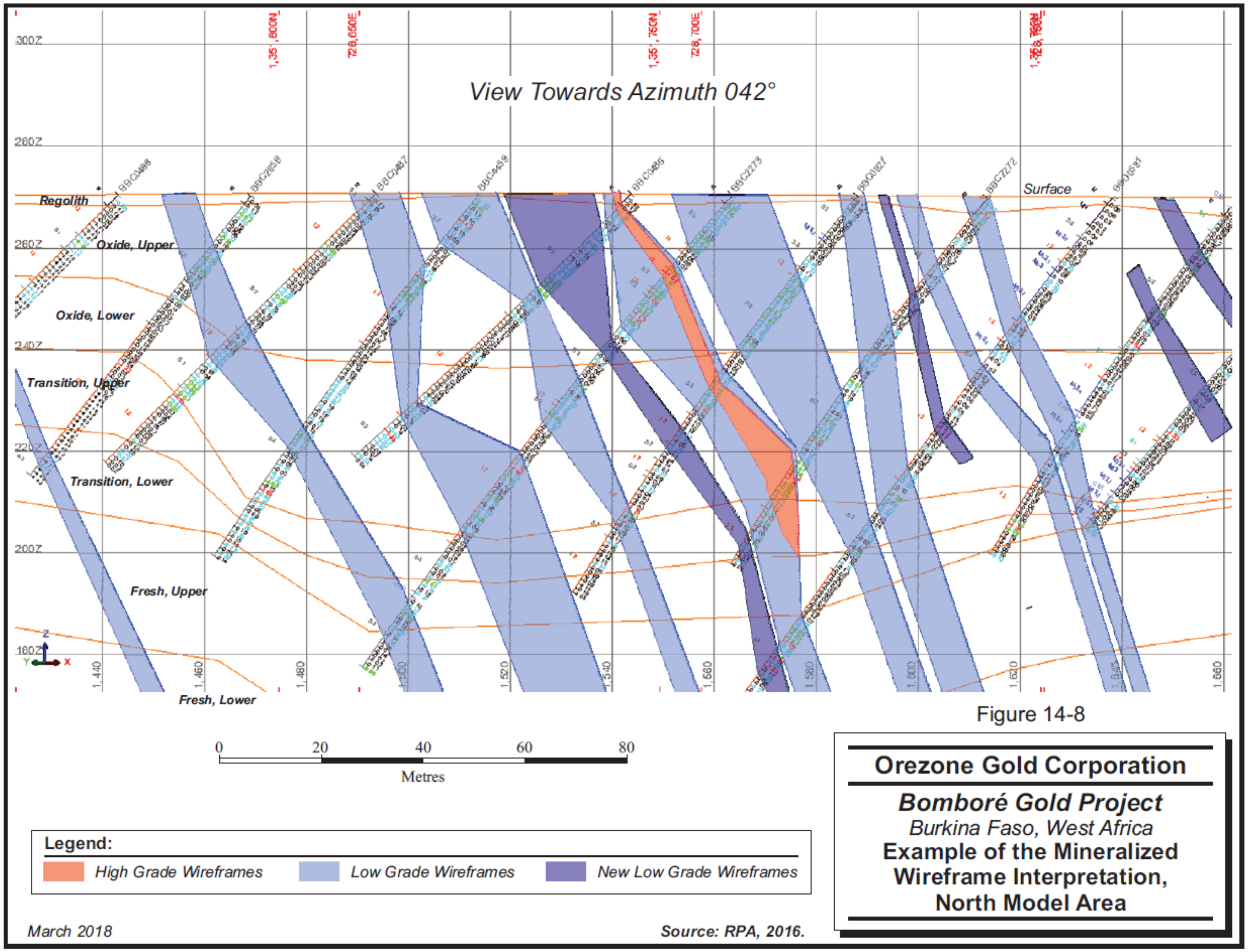


Figure 14.8 Example of the Mineralized Wireframe Interpretation, North Model Area



14.4 Mined Out Areas

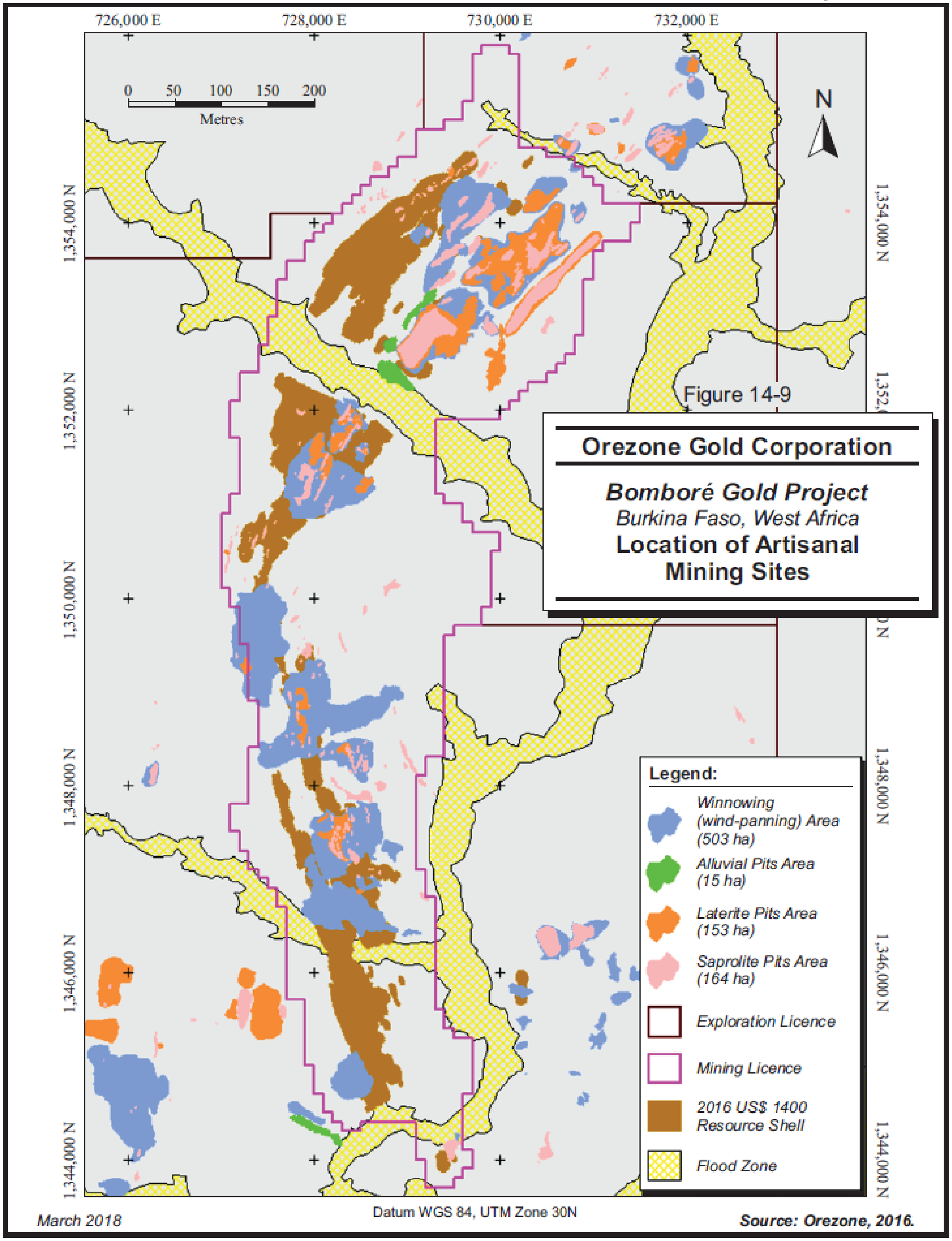
Artisanal gold mining (*zones d'orpaillage*) on the Project was described for the first time during the 1989 to 1994 period when Générale de Mines et de Carrières (GMC) was the operator of the Project. Channel Mining (Barbados) Company, Ltd. (Channel) prepared initial maps and detailed descriptions of the various gold prospects mined by the artisanal miners during the 1994 to 2000 period. Artisanal mining is no longer permitted in the areas where the Mineral Resources are located.

Orezone mapped the artisanal gold mining zones according to a standard definition which is based on the identification of the style of artisanal gold mining (Table 14.4). The locations of the individual pits and areas of exploitation are identified by means of handheld GPS units (Figure 14.9).

Table 14.4 Description of the Artisanal Mining Types

Work Type	Description
Limite de vannage (LV)	Area of winnowing, a very superficial type of exploitation. The top surface is brushed and gold concentrated by winnowing, using the wind to blow away the light particles with heavy ones progressively concentrated over several cycles. These areas are easily identified by the shallow winnowing mounds (typically a conical shallow mound of about one square meter) and such areas can cover several square kilometres they are typically developed over areas of residual soils.
Limite de puits alluvionnaires (LPA)	Area of alluvial material mining. Typically found in the active drainages, but can also be found in the paleo-drainages. Gravels are typically exploited to a multi-metre depth. In places, the softer saprolite underlying the gravels will be mined first, and the gravel horizon then mined from bottom up, until the gold contents decline. These deposits are typically linear and the alluvial gold can be transported or in contact with the primary (Birimian) source. Individual pits are picked as "puits alluvionnaires" (PA), and the outline of an extensive zone of PA is picked as an LPA. There are three knows alluvial zones at Bomboré, each located essentially outside of the resource pit shells.
Limite de puits latéritiques (LPL)	Area of pits exploiting ferrugineous laterite or duricrust. These are typically found within the LV areas, over more restricted areas where gold has been residually concentrated in this horizon. At Bomboré, this horizon is discontinuous and where present, is typically less than 2 m thick. Individual pits are picked as PL, and the outline of an extensive zone of PL is picked as an LPL. Such an area can cover several hectares.
Limite de puits dans la saprolite (LPS)	Area where gold is mined from shafts reaching the saprolite, and in place even the saprock or sulphide zones. Individual shafts are picked as PS, and the outline of an extensive zone of PS is picked as an LPS. At Bomboré, most of these zones are narrow and follow developed along a quartz vein structure, the longest being 700 m long on the hanging wall of the Maga prospect. LPS zones can be found within LPL or LV areas, or where the Birimian is exposed. LPS can also be developed over areas of several hectares. The P16, P17 and P17S gold deposits are the only ones where a relatively large pit has been developed.

Figure 14.9 Location of Artisanal Mining Sites



14.5 Topography Models

In April 2011, Orezone acquired the initial satellite topographic data from PhotoSat Information Ltd. (PhotoSat) for the Project area. Satellite topographic data was subsequently acquired from PhotoSat for the area to the west and south of the original data in January 2012 and was merged with it. In October 2012, PhotoSat re-processed the data to produce a topographic model that better fit ground control point co-ordinates and to eliminate north-south strips that appeared in the contoured topographic model (Marquis, 2014). An improved series of raster images and 50 cm topographic contours using more control points than the original surveys were delivered to Orezone in November 2012.

Orezone has created a comprehensive database of survey points within the Project area including 20 control points within the PhotoSat topographic model area. These control points were used to geo-reference the November 2012 topographic model. In 2014, Geozone Geomatics validated the Project survey data against the PhotoSat topographic surface.

14.6 Sample Statistics and Grade Capping

The 2016 Mineral Resource estimate mineralization wireframe models and the 2017 mineralization wireframe models were used to code the drill hole database and identify those samples within all the mineralized wireframes. All remaining samples in the third domain were considered as potentially being part of the third domain population. These groups of samples were then extracted from the database on a group-by-group basis, subjected to statistical analyses into their respective domains, and then subjected to analysis by means of histograms and probability plots. A total of 146,372 samples were contained within the 2016 Mineral Resource estimate low and high-grade domain mineralized wireframes in all four block model areas. A total of 15,003 samples were contained within the 2017 Mineral Resources estimate new low-grade domain mineralized wireframes in the North and South model areas. A total of 272,058 samples remained outside all wireframes for the four block model areas. The sample statistics are summarized in Table 14.5.

Table 14.5 Summary Statistics of the Uncapped Assays

Wireframe	Mean (g/t Au)	StDev	CoV	Min	Max	Count
North Model						
R	0.22	0.57	2.61	0	27.24	7,403
Cfu_200 main	0.96	7.05	7.35	0.02	171.32	1,055
Cfu_200hw	0.44	0.86	1.98	0.03	6.64	73
Cfu_200fw	0.79	5.20	6.58	0.01	163.77	1,407
Cfu_450main	1.22	1.76	1.44	0.02	16.97	208
Cfu_450hw	0.86	0.53	0.62	0.08	2.67	72
Cfu_450fw	2.02	7.28	3.6	0.03	102.81	246
Maga_200main	0.45	1.14	2.56	0	66.26	4,629
Maga_200hw	0.46	0.95	2.06	0	50.19	14,751
Maga_200fw	0.53	0.64	1.21	0.01	7.89	572
Maga_450main	1.18	1.73	1.46	0.01	46.9	3,392
Maga_450hw	1.36	2.89	2.13	0.01	93.39	2,485
Maga_450fw	1.60	2.25	1.40	0.03	13.54	75
P8/P9_200main_c	0.44	0.85	1.91	0	37.38	6,494
P8/P9_200main_e_s_w	0.39	0.52	1.34	0	12.77	7,965
P8/P9_200hw	0.45	1.28	2.87	0	71.05	6,629
P8/P9_200fw	0.43	1.14	2.68	0	55	15,147
P8/P9_450main_c	1.10	1.94	1.76	0	59.56	4,340
P8/P9_450main_e_s_w	0.87	2.10	2.41	0.01	104.52	3,000
P8/P9_450hw	1.38	2.85	2.08	0.01	50.16	1,597
P8/P9_450fw	0.99	2.02	2.04	0.01	37.19	1,856
3D (new wireframes)	0.47	0.97	2.03	0.001	60.40	10,550
Waste (incl. third domain)	0.12	0.49	4.10	0.001	122.82	172,020
South Model						
R	0.20	0.71	3.63	0	33.81	5,314
Siga SW_200main	0.40	0.68	1.67	0	42.6	12,779
Siga SW_200hw	0.39	0.59	1.52	0.01	15.6	2,539
Siga SW_200fw	0.40	1.00	2.51	0.01	28.93	1,128
Siga SW_450main	1.24	21.95	17.77	0.01	1,7836.88	6,639
Siga SW_450hw	1.01	1.73	1.72	0.06	21.44	268
Siga SW_450fw	1.00	2.18	2.17	0.03	26.97	172
Siga East_200main	0.47	0.91	1.94	0	29.50	3,080
Siga East_200hw	0.47	1.08	2.31	0	32.28	1,420
Siga East_200fw	0.41	1.41	3.45	0	49.63	2,455
Siga East_450main	1.03	2.62	2.55	0	67.04	1,797
Siga East_450hw	1.21	2.09	1.72	0	31.98	554

Wireframe	Mean (g/t Au)	StDev	CoV	Min	Max	Count
Siga East_450fw	0.83	1.44	1.74	0.01	28.00	526
P11_200main	0.42	0.97	2.28	0	36.72	2,430
P11_200fw	0.38	0.70	1.82	0	15.09	2,318
P11_450main	1.06	1.95	1.83	0	32.19	1,042
P11_450fw	1.01	1.48	1.47	0.02	14.54	369
3D (new wireframes)	0.48	1.15	2.39	0.001	31.95	4,453
Waste (incl. third domain)	0.11	0.45	4.04	0.001	89.93	82,584
P16 Model						
P16_200main	0.35	0.60	1.72	0	12.32	603
P16_200hw	0.45	0.93	2.08	0	12.68	624
P16_200fw	0.32	0.36	1.10	0.02	3.21	385
P16_450main	1.41	2.68	1.91	0.01	44.33	770
P16_450hw	1.76	5.78	3.28	0	114.73	698
P16_450fw	0.87	1.22	1.41	0.02	13.59	175
Waste (incl. third domain)	0.06	0.32	5.15	0.001	21.09	7,711
P17 Model						
P17c	1.10	2.21	2.01	0	41.63	884
P17n	0.80	2.01	2.51	0.02	36.42	409
P17s	2.15	2.34	1.09	0	15.52	298
Waste (incl. third domain)	0.06	0.29	4.77	0.001	12.06	9,743

Orezone elected to cap high assay values to reduce the influence of erratic high-grade assay values. The selection of the various capping values was partially guided by the goal of achieving a target CoV of less than approximately 2.0. For the 2016 Mineral Resource estimate the capping values were selected separately for the high grade (0.45 g/t Au) and low grade (0.20 g/t Au) domains for the North, South, and P16 block model areas, partially guided by the goal of achieving a target CoV of less than approximately 2.0. Capping values were also applied to the three grade domains for the P17 block model area. For the 2017 estimate a uniform capping value of 5.00 g/t Au was applied by RPA to the new low-grade mineralized wireframes and the third domain samples. The summary statistics for the capped assays are provided in Table 14.6.

Table 14.6 Summary Statistics of the Capped Assays

Wireframe	Mean (g/t Au)	StDev	CoV	Min	Max	Count
North Model						
R	0.21	0.40	1.87	0	5.02	7,403
Cfu_200 main	0.57	1.09	1.90	0.02	10.00	1,055
Cfu_200hw	0.33	0.31	0.94	0.03	1.50	73
Cfu_200fw	0.57	1.08	1.89	0.01	10.00	1,407
Cfu_450main	1.13	1.11	0.99	0.02	7.00	208
Cfu_450hw	0.85	0.51	0.60	0.08	2.25	72
Cfu_450fw	1.39	1.99	1.43	0.03	10.00	246
Maga_200main	0.43	0.57	1.33	0	9.00	4,629
Maga_200hw	0.46	0.79	1.72	0	16.69	14,751
Maga_200fw	0.52	0.55	1.06	0.01	3.50	572
Maga_450main	1.15	1.28	1.11	0.01	18.30	3,392
Maga_450hw	1.30	1.86	1.43	0.01	20.00	2,485
Maga_450fw	1.56	2.02	1.30	0.03	10.00	75
P8/P9_200main_c	0.44	0.69	1.57	0	10.00	6,494
P8/P9_200main_e_s_w	0.39	0.51	1.30	0	8.00	7,695
P8/P9_200hw	0.43	0.73	1.72	0	10.00	6,629
P8/P9_200fw	0.41	0.65	1.60	0	10.00	15,147
P8/P9_450main_c	1.08	1.49	1.38	0	20.00	4,340
P8/P9_450main_e_s_w	0.84	0.89	1.07	0.01	8.50	3,000
P8/P9_450hw	1.29	2.11	1.63	0.01	15.00	1,597
P8/P9_450fw	0.97	1.67	1.73	0.01	20.00	1,856
3D (new wireframes)	0.43	0.52	1.19	0.001	5.00	10,550
Waste (inc. third domain)	0.11	0.22	1.95	0.001	5.00	172,020
South Model						
R	0.19	0.38	2..07	0	7.79	5,314
Siga SW_200main	0.40	0.60	1.48	0	22.20	12,779
Siga SW_200hw	0.38	0.45	1.18	0.01	6.00	2,539
Siga SW_200fw	0.38	0.58	1.54	0.01	9.00	1,128
Siga SW_450main	0.97	1.62	1.67	0.01	48.97	6,639
Siga SW_450hw	0.92	1.00	1.09	0.06	7.00	268
Siga SW_450fw	0.86	0.84	0.98	0.03	5.00	172
Siga East_200main	0.46	0.75	1.62	0	10.00	3,080
Siga East_200hw	0.45	0.71	1.57	0	10.00	1,420
Siga East_200fw	0.37	0.68	1.82	0	10.00	2,455
Siga East_450main	0.93	1.20	1.30	0	10.00	1,797
Siga East_450hw	1.16	1.57	1.36	0	10.00	554

Wireframe	Mean (g/t Au)	StDev	CoV	Min	Max	Count
Siga East_450fw	0.79	0.91	1.15	0.01	10.00	526
P11_200main	0.41	0.62	1.50	0	8.00	2,430
P11_200fw	0.37	0.49	1.34	0	6.24	2,318
P11_450main	0.98	1.26	1.28	0	8.00	1,042
P11_450fw	0.95	1.08	1.14	0.02	6.70	369
3D (new wireframes)	0.44	0.59	1.35	0.001	5.00	4,453
Waste (incl. third domain)	0.10	0.23	2.15	0.001	5.00	82,584
P16 Model						
P16_200main	0.34	0.37	1.10	0	3.00	603
P16_200hw	0.42	0.72	1.71	0	5.56	624
P16_200fw	0.32	0.31	0.99	0.02	2.00	385
P16_450main	1.29	1.53	1.19	0.01	10.00	770
P16_450hw	1.37	2.15	1.56	0	11.87	698
P16_450fw	0.84	1.02	1.20	0.02	10.00	175
Waste (incl. third domain)	0.06	0.17	2.70	0.001	5.00	7,711
P17 Model						
P17c	1.06	1.74	1.64	0	12.03	884
P17n	0.74	1.06	1.43	0.02	9.17	409
P17s	2.14	2.28	1.07	0	12.27	298
Waste (incl. third domain)	0.05	0.20	3.71	0.001	5.00	9,743

RPA agrees that the influence of high-grade gold assays must be reduced or controlled and uses a number of industry best practice methods to achieve this goal, including capping of high-grade values. RPA employs a number of statistical analytical methods to determine an appropriate capping value including preparation of frequency histograms, probability plots, decile analyses, and capping curves. Using these methodologies, RPA examined the selected capping values for the mineralized domains in the four model areas. In RPA's opinion, the selected capping values are reasonable and have been correctly applied to the raw assay values for all four-model areas.

14.7 Compositing Methods

Composited samples were created from the capped, raw assay values using the downhole compositing function of the Surpac mine modelling software package. In this function, compositing begins at the point in a drill hole at which the zone of interest is encountered and continues down the length of the hole until the end of the zone is reached. Composite samples were created for each individual mineralization wireframe solid model and the resulting data stored in separate files. In the case of the third domain, a composite file for each zone was created along the length of the holes using only assays outside all wireframes. For the 2016 Mineral Resource estimate a composite length of 1.5 m was used for the North, South, and P16 block model areas while a composite length of 1 m was used for the P17 block model area. The 2017 Mineral Resources estimate used a composite length of 1.5 m for the new low-grade mineralization wireframe domains in the North and South block model areas and the same composite lengths as in the 2016 Mineral Resource estimate for the new third domain in all four-block model areas. The summary statistics for the composited samples are presented in Table 14.7.

The thickness of the mineralized zone encountered by any given drill hole is not an even multiple of the composite length. The remaining samples that were less than 100% of the composite length (i.e., the “tails”) were retained as part of the data set so as to enable a more accurate estimate of the grades for the various elements along the bottom contacts of the respective domain models.

Table 14.7 Summary Statistics of the Composited Assays

Wireframe	Mean (g/t Au)	StDev	CoV	Min	Max	Count
North Model (1.5 m composites)						
R	0.23	0.38	1.66	0	5.02	6,670
Cfu_200 main	0.57	0.87	1.52	0.02	10.00	809
Cfu_200hw	0.33	0.22	0.68	0.07	1.08	54
Cfu_200fw	0.57	0.91	1.60	0.01	10.00	1,086
Cfu_450main	1.12	0.94	0.84	0.03	7.00	169
Cfu_450hw	0.86	0.38	0.44	0.18	2.08	56
Cfu_450fw	1.47	1.86	1.27	0.04	10.00	198
Maga_200main	0.43	0.47	1.09	0.01	6.76	3,526
Maga_200hw	0.45	0.65	1.43	0	16.69	11,186
Maga_200fw	0.51	0.46	0.89	0.01	3.5	439
Maga_450main	1.14	1.06	0.93	0.01	12.93	2,520
Maga_450hw	1.30	1.63	1.25	0.01	20.00	1,925
Maga_450fw	1.58	1.80	1.14	0.21	9.22	55
P8/P9_200main_c	0.43	0.57	1.31	0	9.41	5,052
P8/P9_200main_e_s_w	0.38	0.40	1.05	0	6.70	5,943
P8/P9_200hw	0.42	0.62	1.46	0	10.00	5,090
P8/P9_200fw	0.40	0.55	1.35	0	10.00	11,178

Wireframe	Mean (g/t Au)	StDev	CoV	Min	Max	Count
P8/P9_450main_c	1.05	1.15	1.09	0	15.88	3,359
P8/P9_450main_e_s_w	0.83	0.73	0.89	0.01	8.08	2,241
P8/P9_450hw	1.29	1.79	1.40	0.01	15.00	1,284
P8/P9_450fw	0.98	1.60	1.64	0.01	20.00	1,441
3D	0.45	0.46	1.02	0.01	5.00	7,880
Waste (incl. third domain)	0.11	0.20	1.75	0.01	5.00	128,305
South Model (1.5 m composites)						
R	0.20	0.36	1.80	0	7.79	4,657
Siga SW_200main	0.40	0.50	1.25	0	20.20	10,083
Siga SW_200hw	0.38	0.38	1.01	0.02	6.00	1,822
Siga SW_200fw	0.38	0.47	1.23	0.01	6.18	840
Siga SW_450main	0.96	1.29	1.34	0.01	33.02	5,276
Siga SW_450hw	0.92	0.76	0.83	0.22	4.90	194
Siga SW_450fw	0.89	0.81	0.91	0.19	5.00	126
Siga East_200main	0.45	0.63	1.38	0	10.00	2,589
Siga East_200hw	0.50	0.74	1.47	0	10.00	1,225
Siga East_200fw	0.37	0.59	1.57	0	10.00	1,904
Siga East_450main	0.92	1.08	1.18	0	10.00	1,515
Siga East_450hw	1.13	1.31	1.16	0	8.51	430
Siga East_450fw	0.77	0.67	0.87	0.01	7.02	431
P11_200main	0.40	0.50	1.25	0	6.33	1,836
P11_200fw	0.37	0.40	1.08	0	6.24	1,647
P11_450main	0.98	1.04	1.07	0.01	8.00	803
P11_450fw	0.93	0.86	0.93	0.05	5.37	268
3D	0.44	0.50	1.12	0.01	5.00	3,441
Waste (incl. third domain)	0.10	0.19	1.78	0.01	5.00	61,885
P16 Model (1.5 m composites)						
P16_200main	0.40	0.39	0.97	0	2.70	457
P16_200hw	0.50	0.93	1.87	0	11.87	479
P16_200fw	0.30	0.23	0.77	0.02	2.00	270
P16_450main	1.32	1.42	1.07	0.01	10.00	531
P16_450hw	1.53	2.11	1.37	0.01	11.87	423
P16_450fw	0.83	0.65	0.77	0.04	3.57	130
Waste (incl. third domain)	0.06	0.13	2.02	0.001	3.35	5,950
P17 Model (1.0 m composites)						
P17c	1.15	1.70	1.48	0	12.03	916
P17n	0.71	1.04	1.46	0.01	9.17	424
P17s	2.27	2.32	1.02	0	12.27	304
Waste (incl. third domain)	0.05	0.19	3.61	0.001	5.00	9,745

14.8 Bulk Density

The specific gravity of a single piece of core, 10 cm to 15 cm in length, selected from each core box prior to splitting is determined by the water immersion method on site. A wax or film coating is applied to the sample if necessary (e.g., for samples of oxidized material or from the transition zone). The results of the specific gravity measurements are classified by rock and material type. Specific gravity determinations have been completed on average every two metres in all historical and current boreholes. The specific gravity is seen to increase with depth through the weathering profile and is fairly homogeneous within the fresh zone for any given lithology.

A total of 91,958 density determinations are contained in the drill hole databases for each of the four block model areas.

To estimate the moisture content, samples are weighed at the sample preparation laboratory before and after drying. The average loss of moisture is 5.7% in the oxide core samples, 2.8% in the transition core samples, and 0.2% in the fresh core samples. A reduction factor of 5% and 2% is applied to individual samples in the oxide and transition zones, respectively, for the block model density calculations.

The bulk densities were extracted from each of the four drill hole databases and were grouped according to their host rock lithology and weathering profile. Separate capping values were determined for each lithology type and oxidation state. The density values for each block in a given block model were then estimated using the capped density values and the OK interpolation algorithm. For those areas of limited sample data, the average density value of the specific lithology type and oxidation state was applied.

14.9 Variography

14.9.1 North Model Area

Variography studies for the North model areas were carried out on two test areas of the Maga sub-domain that were selected as representative of the mineralization distribution in that area. The two sub-domains used for variogram analysis together outline gold mineralization along a strike length of approximately two kilometres and include four wireframe models: Maga 200 Main Upper, Maga 450 Main Upper, Maga 200 Main Lower, and the Maga 450 Main Lower. Normal scores variograms were created for both of the test areas, however, meaningful variograms were derived for only the Maga 200 Main Upper wireframe model (Test Area 2). The drill spacing that outlines the gold mineralization for this area is approximately 25 m x 25 m. The normal scores variogram models from this test area were back-transformed and the resulting parameters were used to estimate the gold grades in the remainder of the wireframe models in the North block model area. The geometries of the search ellipses were suitably modified to account for the changes in the strike of each of the individual mineralized wireframe models separately. The range of the major axis (along strike direction) was determined to be approximately 100 m for Test Area 2.

Given the complex geometries and the large number of wireframe models present in the North model area, RPA agrees with the approach of selecting representative areas for variography studies. Using a combined data set of the low-grade and high-grade composite values, RPA was able to generate an along-strike variogram with a range of approximately 50 m.

As a cross-check of the selected variography parameters, RPA proceeded to examine the gold distribution for the two test areas by means of conventional longitudinal projections (Figures 14.10 and 14.11). Comparison of the two images clearly shows that, other than the high-grade values occurring as small isolated pods, the gold grade distributions follow different patterns between the two test areas. The gold grades in Test Area 1 occur for the most part as larger pods of medium-grade mineralization that is surrounded by lower gold grades. In contrast, the gold grades in Test Area 2 are dominated by a zone of higher gold grades located in the central and northern portion of the area. As no clear, consistent overall rakes or plunges are apparent in either of the two test areas examined, RPA agrees with the selection of the search ellipse parameters.

Figure 14.10 Longitudinal Projection of the Gold Values for Test Area 1, North Model Area

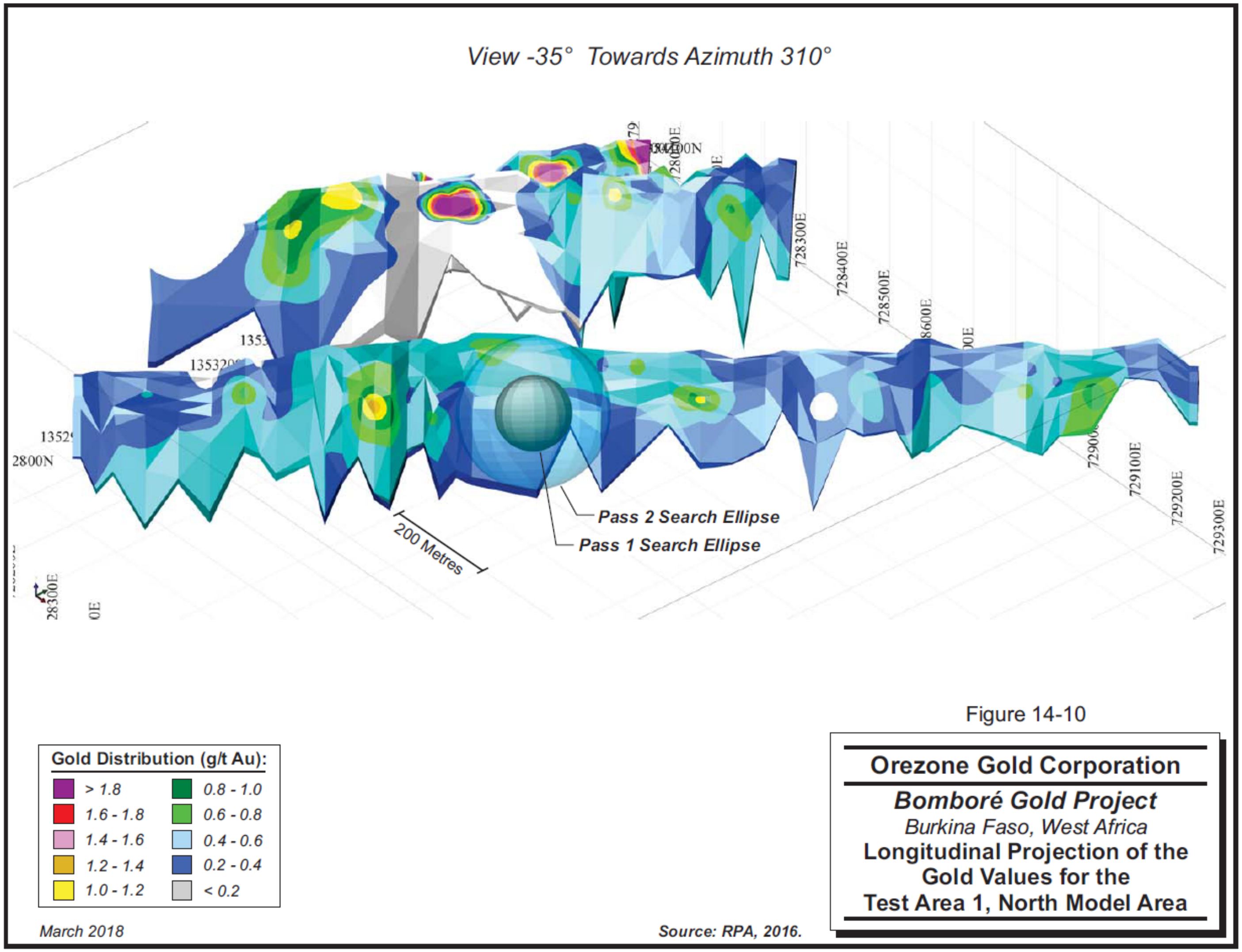
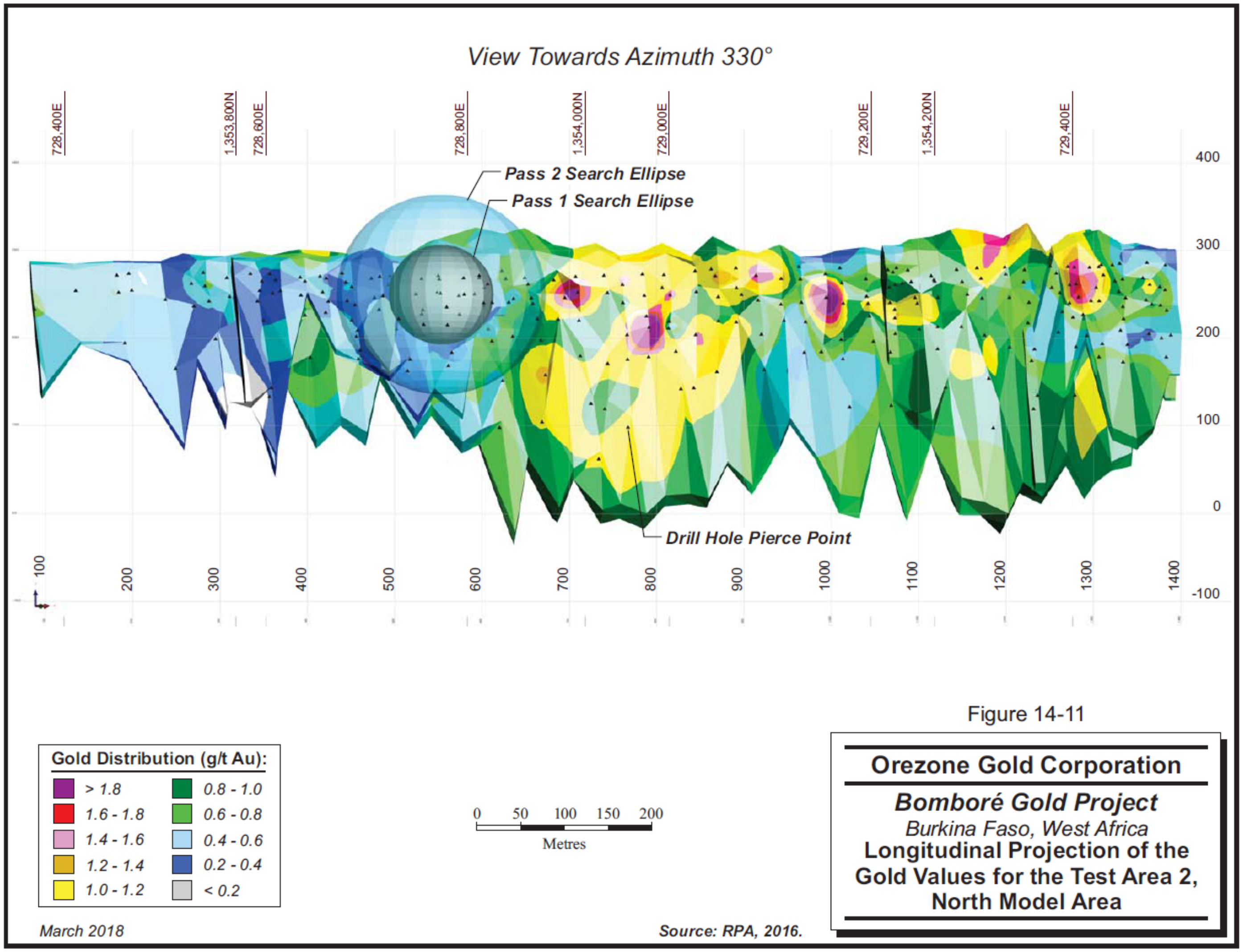


Figure 14.11 Longitudinal Projection of the Gold Values for Test Area 2, North Model Area



14.9.2 South Model Area

Variography studies for the South model areas were carried out on composite samples from the Siga SW wireframe model. The mineralized zones contained within this sub-domain define gold mineralization along a strike length of approximately 3,700 m. The drill spacing that outlines the gold mineralization for this area is approximately 25 m x 50 m. Normal scores variograms were created to measure the spatial variability. The normal scores variogram models from this test area were back-transformed and the resulting parameters were used to estimate the gold grades in the remainder of the wireframe models in the South block model area. The geometries of the search ellipses were suitably modified to account for the specific orientations of the wireframe in question. The range of the major axis (along strike direction) was determined to be approximately 100 m at 90% of the sill with a range of almost 500 m at 100% of the sill.

Using a combined data set of the low grade and high-grade composite values, RPA was able to generate an along-strike variogram with a range of approximately 60 m to 70 m.

As a cross-check of the selected variography parameters, RPA proceeded to examine the gold distribution for the test area by means of conventional longitudinal projections for the P11 and Siga SW portions of the South model area (Figures 14.12 and 14.13). Located at the northern portion of the South model area, the gold grades in the P11_450 Main sub-domain are seen to be distributed in a series of higher-grade pods that exhibit a reasonable degree of continuity from section to section. In contrast, the higher gold grades in the Siga SW_450 Main sub-domain are much more discontinuous in their distribution.

Figure 14.12 Longitudinal Projection of the Gold Values for the P11_450 Main Wireframe, South Model Area

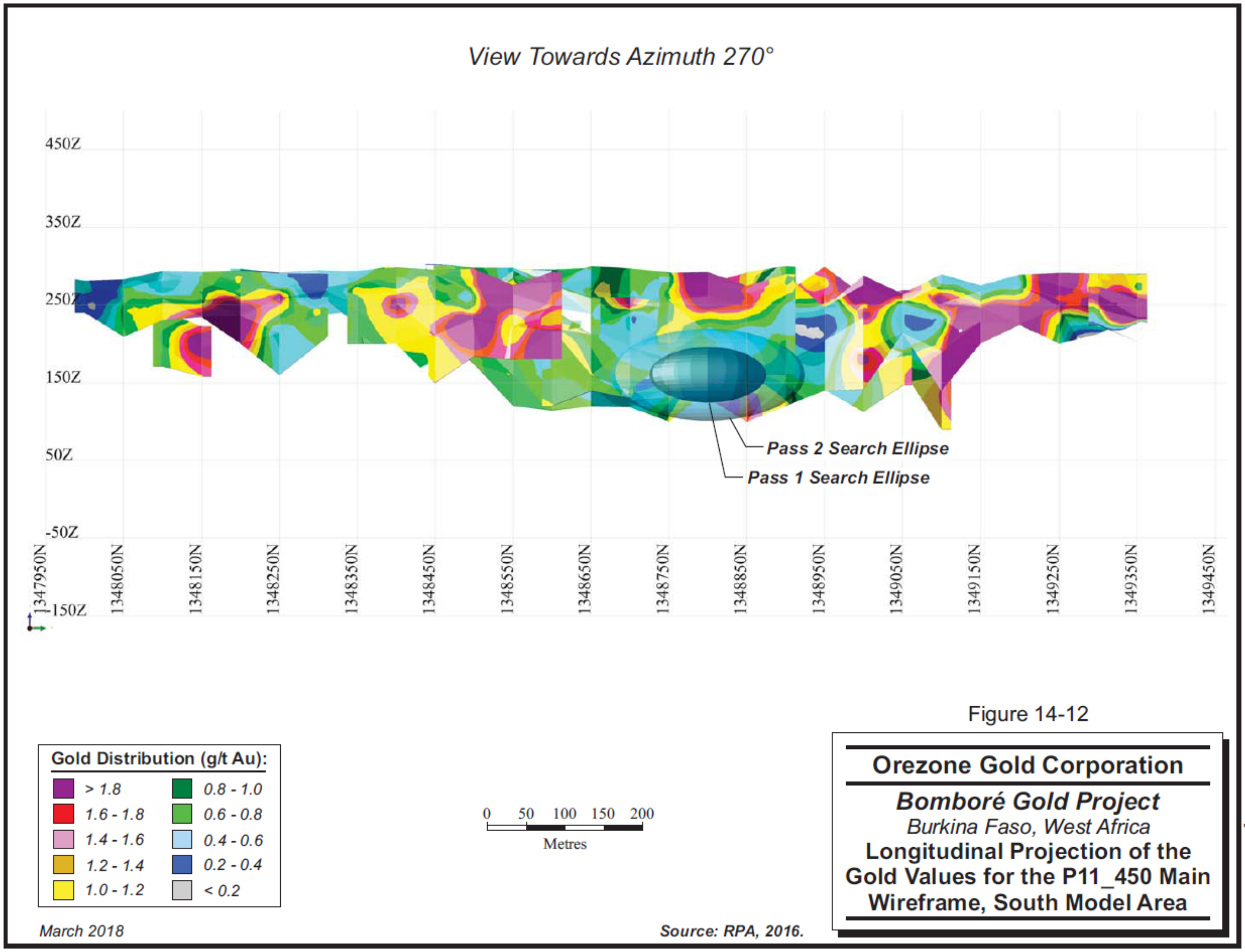
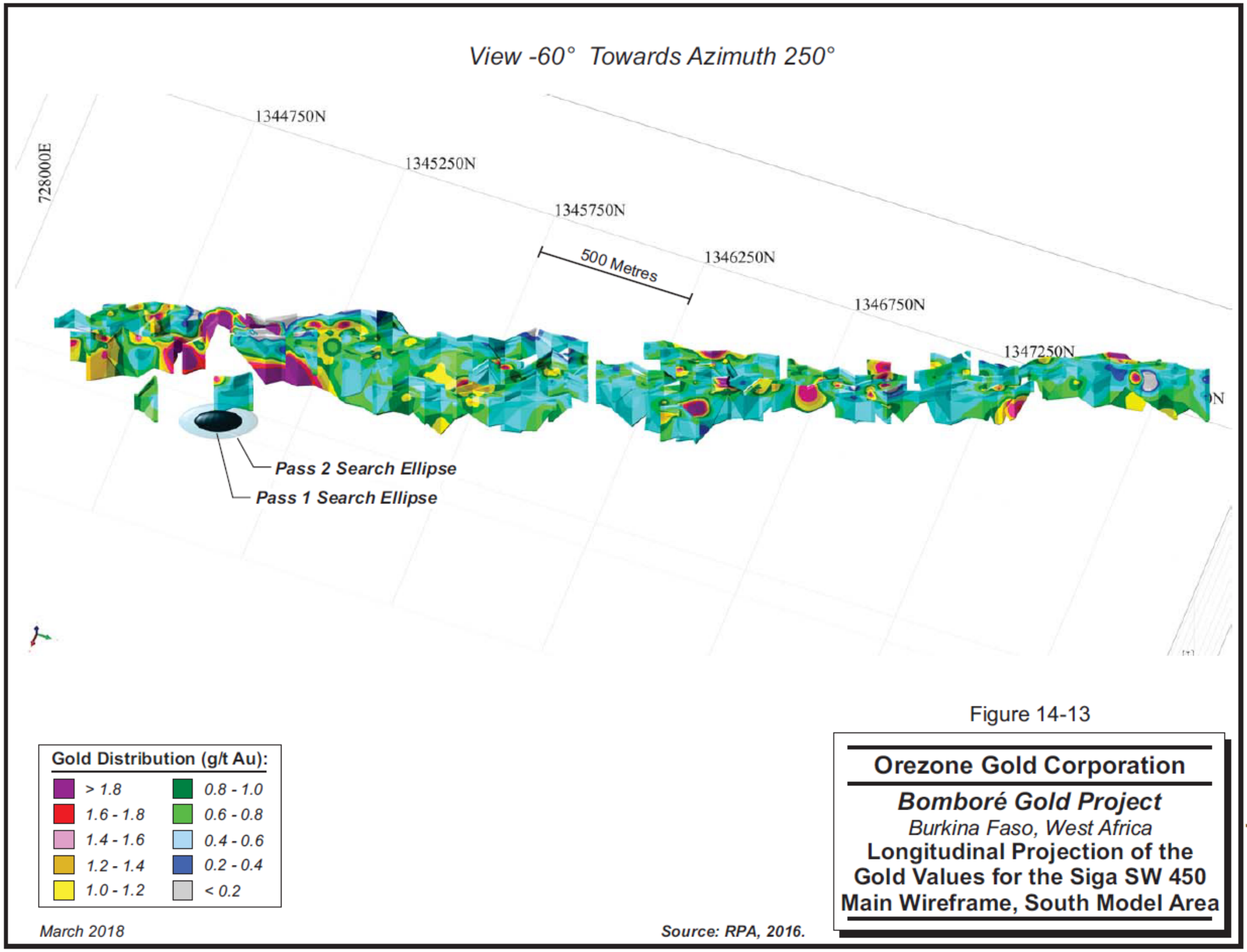


Figure 14.13 Longitudinal Projection of the Gold Values for the Siga SW_450 Main Wireframe, South Model Area



14.9.3 Other Areas

Given the relatively small number of drill holes that are available for the P16 model area, no variogram models were prepared for these mineralized wireframes. The variogram models for the Siga SW wireframe models were used to estimate the grades for this model area.

Variogram analyses were not carried out for the 2017 Mineral Resources estimate new low-grade mineralization domains as RPA believes that those variograms would be nearly identical to the variograms of their associated models. A variogram for the unconstrained third domain would be irrelevant due to the widely spaced distribution and unconstrained nature of the data.

RPA recommends that examination of the gold distribution be carried out on a number of selected mineralized wireframes in the remaining areas of the North and South model areas, as knowledge gained from this exercise will improve the accuracy of the local estimate.

In-fill drilling to confirm the continuity of the gold grades in the high-grade pods contained within the resource pit shell is warranted in the South model area. The information gained from this work will improve the variogram models in these areas and will improve the accuracy and level of confidence of the local estimate.

14.10 Block Model Construction

Individual block models were created for each of the four model areas using a parent block size of 12.5 m (along strike) x 4 m (across strike) x 6 m (bench height) and sub-blocks that measured 6.25 m (along strike) x 2 m (across strike) x 3 m (bench height). The azimuths of each of the four block models were appropriately rotated so as to align with the overall strike of the mineralization within the given model area.

A number of attributes were created to store such information as host rock lithology, oxidation state, material density, estimated gold grades, wireframe code, prospect code, Mineral Resource classification, etc., for each block model area.

A summary of the block model extents of each of the four model areas is provided in Table 14.8.

Table 14.8 Block Model Extents

Type	Y (Northing)	X (Easting)	Z (Elevation)
North Model Area:			
Minimum Coordinates	1,351,465	725,871	-100
Maximum Coordinates	1,357,065	728,123	332
Parent Block Size	12.5	4	6
Sub-Block Size	6.25	2	3
Rotation	0.0	0.0	+42.0 (clockwise)
South Model Area:			
Minimum Coordinates	1,344,097.837	728,538.423	0
Maximum Coordinates	1,350,547.837	730,190.423	348
Parent Block Size	12.5	4	6
Sub-Block Size	6.25	2	3
Rotation	0.0	0.0	-20.0 (counter-clockwise)
P16 Model Area:			
Minimum Coordinates	1,343,600	729,000	100
Maximum Coordinates	1,344,600	729,800	292
Parent Block Size	12.5	4	6
Sub-Block Size	6.25	2	3
Rotation	0.0	0.0	0.0
P17 Model Area:			
Minimum Coordinates	1,342,350	729,800	100
Maximum Coordinates	1,346,712.5	730,800	292
Parent Block Size	12.5	4	6
Sub-Block Size	6.25	2	3
Rotation	0.0	0.0	0.0

Gold grades were estimated using the OK interpolation algorithm for the 2016 Mineral Resource estimate in the low grade and high-grade mineralization domains in the North, South, and P16 model areas. The gold grades in the P17 model area were estimated using the ID2 interpolation algorithm in both the 2016 and 2017 Mineral Resource estimates. Gold grades were estimated using the OK interpolation algorithm for the 2017 Mineral Resource estimate within the new LG mineralization domains in the North, South and P16 areas and using the ID3 interpolation algorithm for the third domain in the North, South, P16 and P17 areas.

Hard boundaries were used to constrain the source composite files such that only those composite samples that are present within a specified wireframe were used to estimate block grades. Similarly, hard boundaries were used to constrain coding of the block model where only those blocks that are contained within the specified mineralized wireframe model were permitted to receive estimated gold grades. In the case of the third domain gold grades were estimated in all model areas using a two-step process using the ID^3 interpolation algorithm, which on the first step used only composites outside wireframes. Above 0.2 g/t Au to flag blocks with a grade above 0.00 g/t Au from a minimum of two composites, then on the second step used all composites outside wireframes to estimate the gold grade of the previously flagged blocks.

RPA recommends that the gold grades for the North, South, and P16 model areas be estimated using Leapfrog, ID^3 and the nearest neighbour (NN) interpolation algorithms to provide a cross check of the accuracy of the initial grade estimate.

RPA recommends that the gold grades for the P17 model area be estimated using the NN interpolation algorithm to provide a cross check of the accuracy of the initial grade estimate.

A summary of the search strategies used for the mineralized wireframe domains in each of the four model areas is provided in Table 14.9; a summary of the search strategies used for the third domain by model area is provided in Table 14.10. A multiple-pass search strategy was used to estimate the block grades, and the sizes and orientations of the search ellipses for the first two passes for the North and South model areas are presented in Figures 14.10 through 14.13 above.

Table 14.9 Summary of Variography and Interpolation Parameters in the Mineralized Wireframes Domains

Item	North	South	P16	P17
Variogram Model Type	Normal Scores	Normal Scores	Same as South	N/A
Nugget (C0)	0.29	0.24	Same as South	N/A
Sill (Pass No., C1)	1: 0.27 2: 0.27 3: 0.27	1: 0.441 2: 0.441 3: 0.441	Same as South	N/A
Sill (Pass No., C2)	1: 0.22 2: 0.22 3: 0.22	1: 0.19 2: 0.19 3: 0.19	Same as South	N/A
Sill (Pass No., C3)	1: 0.22 2: 0.22 3: 0.22	1: 0.129 2: 0.129 3: 0.129	Same as South	N/A
Interpolation Algorithm	Ordinary Kriging	Ordinary Kriging	Ordinary Kriging	Inverse Distance Squared
Ellipse Type	Octant	Octant	Octant	Octant

Item	North	South	P16	P17
Maximum Number of Adjacent Octants	1: 2 2: 4 3: Ellipsoid	1: 2 2: 4 3: Ellipsoid	1: 2 2: 4 3: Ellipsoid	1: 2 2: Ellipsoid
Orientation	Varies by wireframe	Varies by wireframe	Varies by wireframe	Varies by wireframe
Length of Major Axis (Pass No., m)	1: 60 2:120 3: 6,000	1: 75 2: 125 3: 300	1: 30 2: 60 3: 1000	1: 30 2: 60
Anisotropy Ratio (Major/Semi-Major)	1: 1 2: 1 3: 1	1: 2 2: 2 3: 2	1: 3 2: 3 3: 2	1: 1 2: 1
Anisotropy Ratio (Major/Minor)	1: 4 2: 4 3: 5	1: 4 2: 6 3: 6	1: 4.2 2: 6 3: 6	1: 8 2: 8
Minimum Number of Samples	1: 6 2: 4 3: 2	1: 4 2: 4 3: 2	1: 4 2: 2 3: 2	1: 5 2: 2
Maximum Number of Samples	1: 22 2: 16 3: 12	1: 22 2: 22 3: 16	1: 16 2: 12 3: 12	1: 10 2: 15
Maximum Number of Samples/Hole	1: 4 2: 4 3: 2	1: 4 2: 4 3: N/A	1: 4 2: 4 3: N/A	1: 3 2: N/A

Table 14.10 Summary of Variography and Interpolation Parameters in the Third Domain

Item	North	South	P16	P17
Interpolation Algorithm	Inverse Distance Cubed	Inverse Distance Cubed	Inverse Distance Cubed	Inverse Distance Cubed
Ellipse Type	Ellipse	Ellipse	Ellipse	Ellipse
Orientation	Varies by wireframe	Varies by wireframe	Varies by wireframe	Varies by wireframe
Length of Major Axis (Pass No., m)	1: 10 2:35	1: 10 2: 35	1: 10 2: 35	1: 10 2: 35
Anisotropy Ratio (Major/Semi-Major)	1: 0.666 2: 1	1: 0.666 2: 1	1: 0.666 2: 1	1: 0.666 2: 1
Anisotropy Ratio (Major/Minor)	1: 4 2: 14	1: 4 2: 14	1: 4 2: 14	1: 4 2: 14
Minimum Number of Samples (Block Tag)	1: 2 2: 2	1: 2 2: 2	1: 2 2: 2	1: 2 2: 2

Item	North	South	P16	P17
Minimum Number of Samples (Block Estimate)	1: 3 2: 3	1: 3 2: 3	1: 3 2: 3	1: 3 2: 3
Maximum Number of Samples (Block Tag)	1: 8 2: 10	1: 8 2: 10	1: 8 2: 10	1: 8 2: 10
Maximum Number of Samples (Block Estimate)	1: 10 2: 10	1: 10 2: 10	1: 10 2:10	1: 10 2: 10
Maximum Number of Samples/Hole	1: 3 2: 3	1: 3 2: 3	1: 3 2: 3	1: 3 2: 3

14.11 Block Model Validation

RPA compared the mean values of the estimated block grades with the corresponding composite samples for selected wireframes from each of the four block models. No significant discrepancies were noticed.

Similarly, RPA created a number of swath plots for selected wireframes from each of the four block models. While some local variations were observed between the composite average grades and the block average grades, no material discrepancies were noted (Figures 14.14 to 14.21).

RPA also carried out a visual comparison of the estimated block grades relative to the contoured longitudinal gold grades for four selected areas located in the North and South model areas (Figures 14.22 to 14.25). The results of the visual examination suggest that local discrepancies are present between the contoured gold grades and the estimated block model grades. RPA attributes this in part to the differences in the methods used to estimate the gold grade distribution in each case, and in part to the density of the drill hole information, particularly in the South model area.

In RPA's opinion, the final block models provide a reasonable estimate of the distribution of the gold mineralization at the Project and are of suitable quality for use in estimation of the 2017 Mineral Resources.

Figure 14.14 Swath Plot by Northing for the P8/P9 High Grade Domains, North Model Area

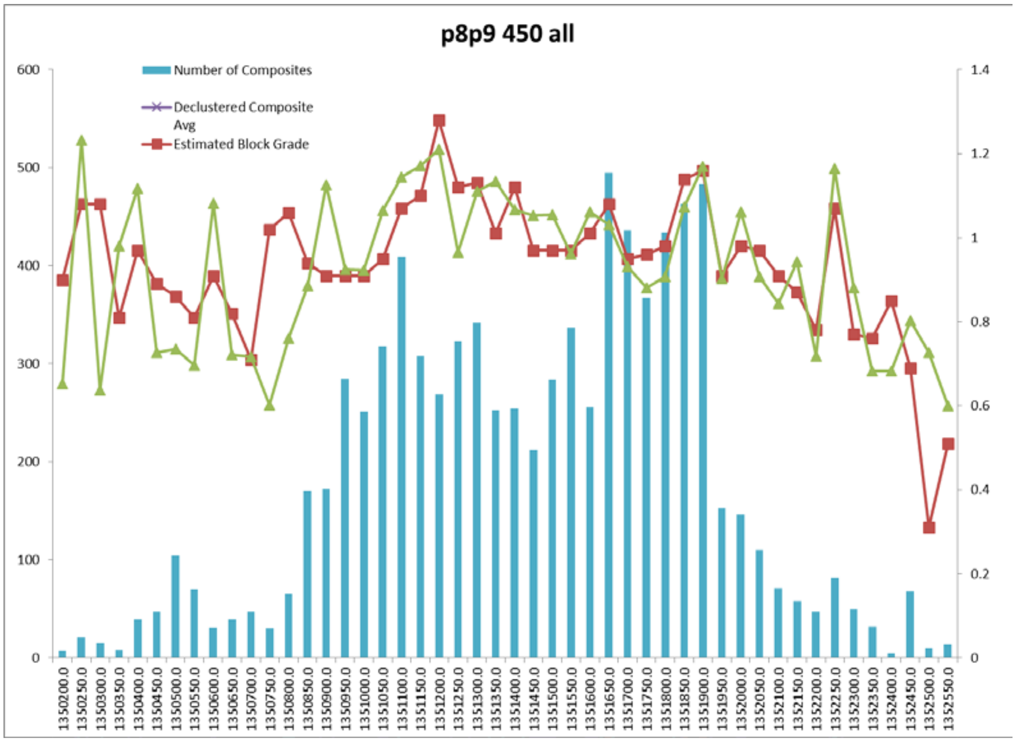


Figure 14.15 Swath Plot by Northing for the Siga SW High Grade Domains, South Model Area

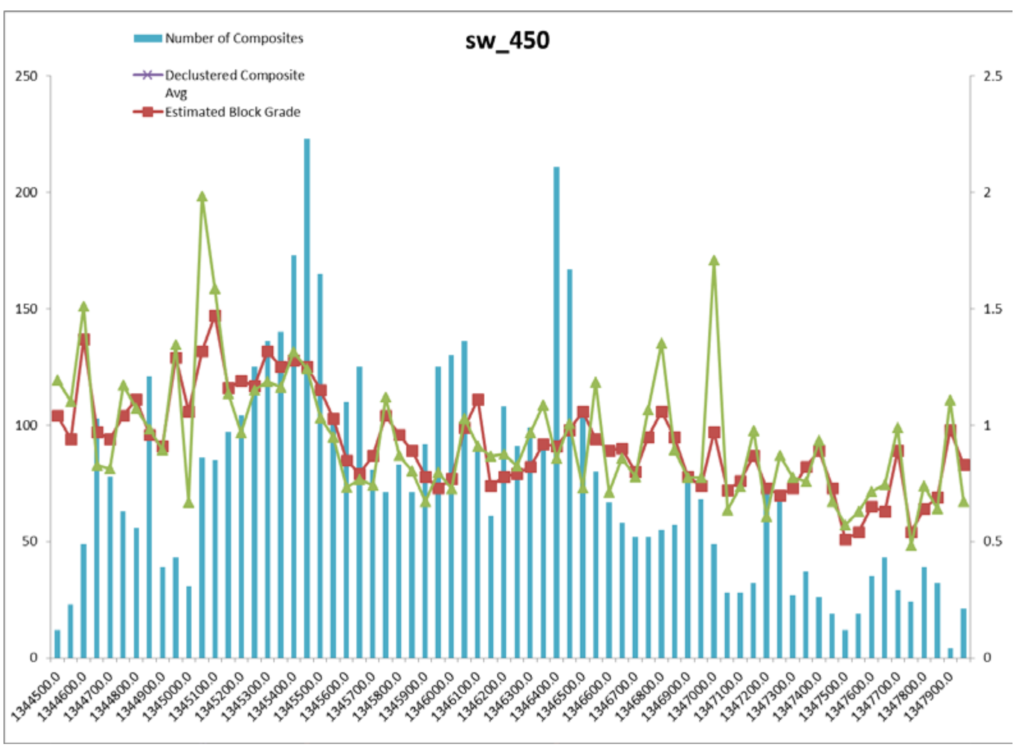


Figure 14.16 Bomboré North Zone, New Low-Grade Domain Swath Plot (by Eastings), Rotated 40° Anti-clockwise

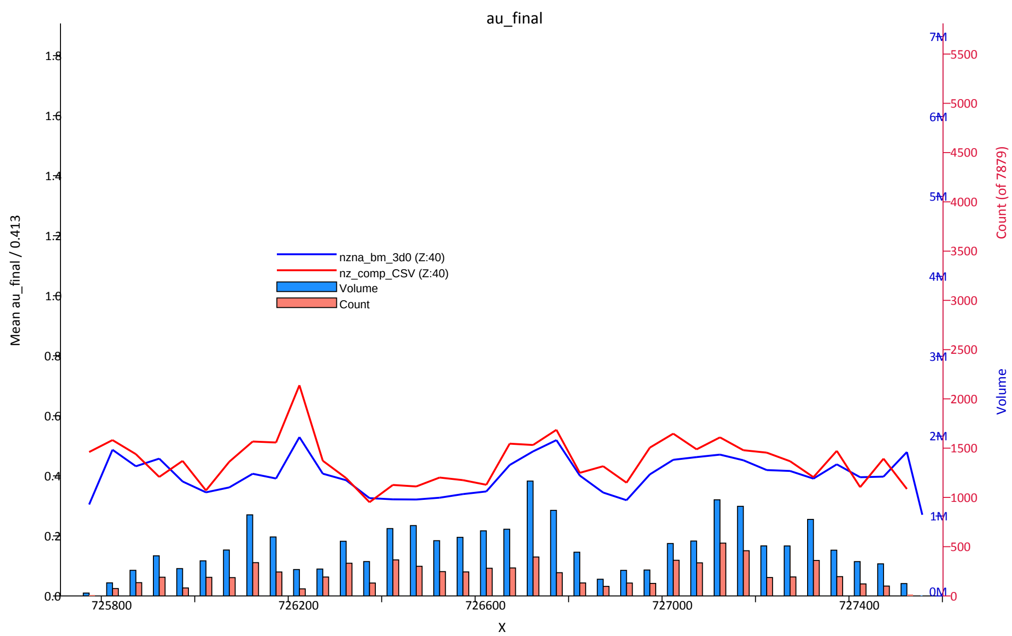


Figure 14.17 Bomboré North Zone, New Low-Grade Domain Swath Plot (by Northings), Rotated 40° Anti-clockwise

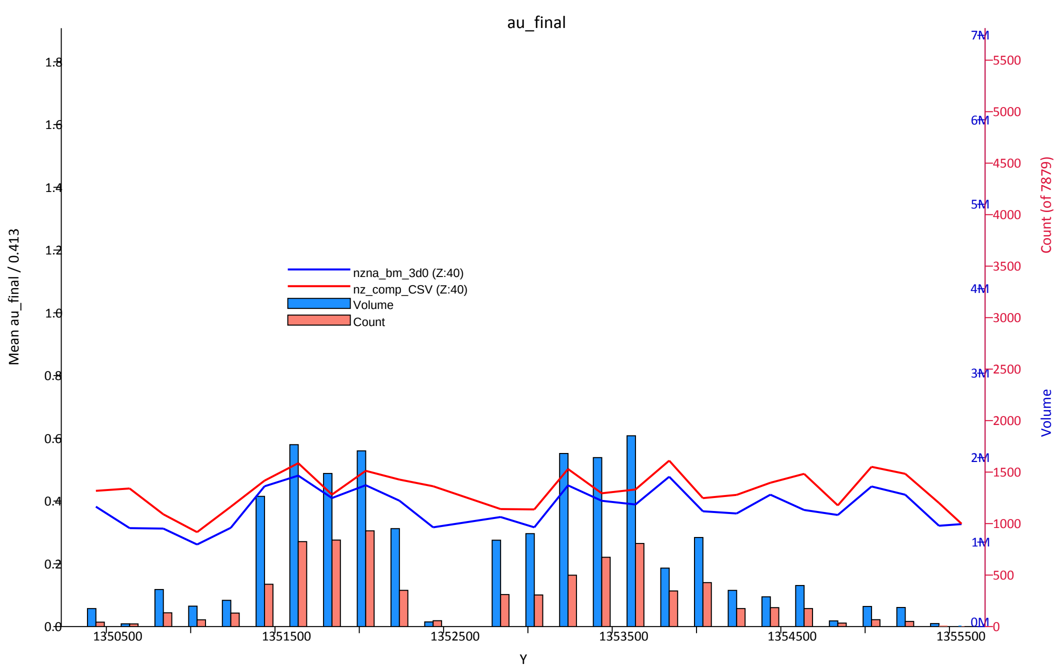


Figure 14.18 Bomboré North Zone, New Low-Grade Domain Swath Plot (by Elevation)

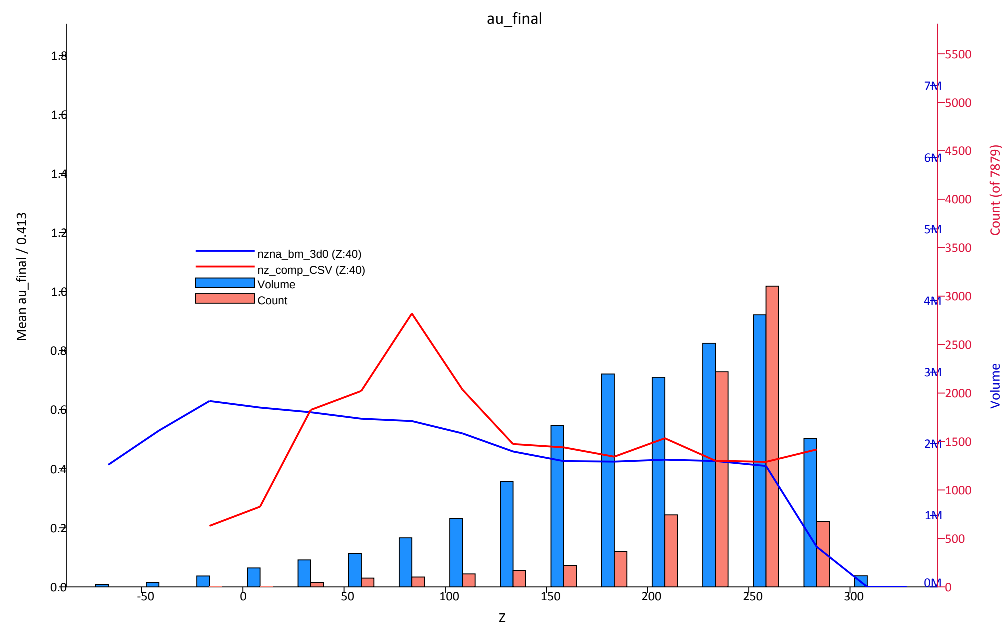


Figure 14.19 Bomboré South Zone, New Low-Grade Domain Swath Plot (by Eastings), Rotated 13° Clockwise

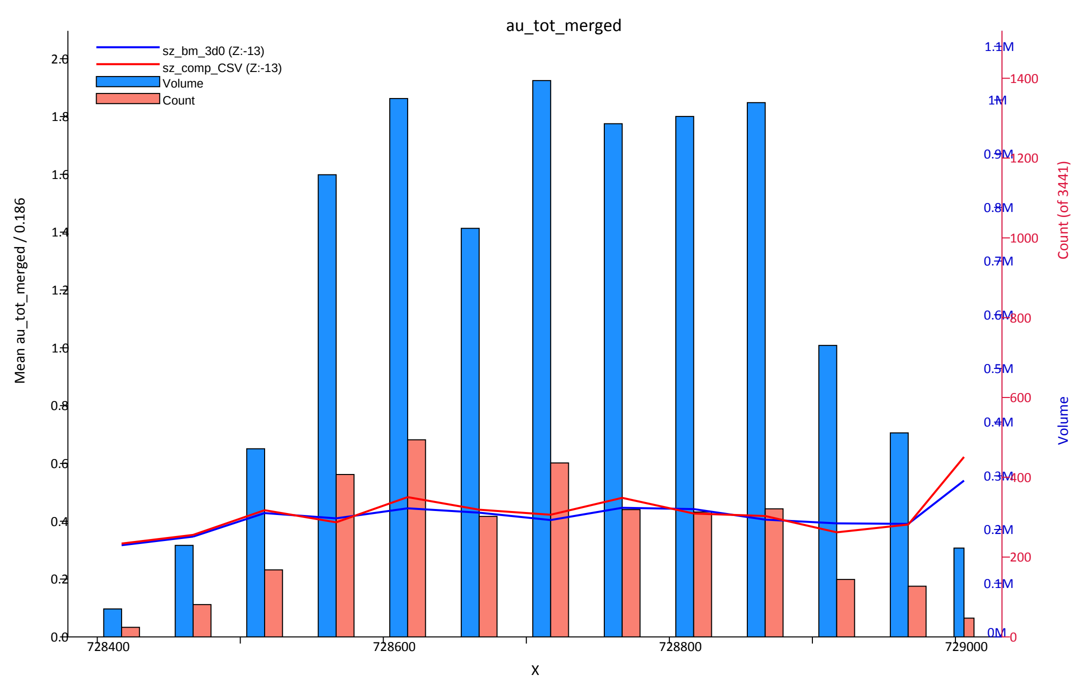


Figure 14.20 Bomboré South Zone, New Low-Grade Domain Swath Plot (by Northings), Rotated 13° Clockwise

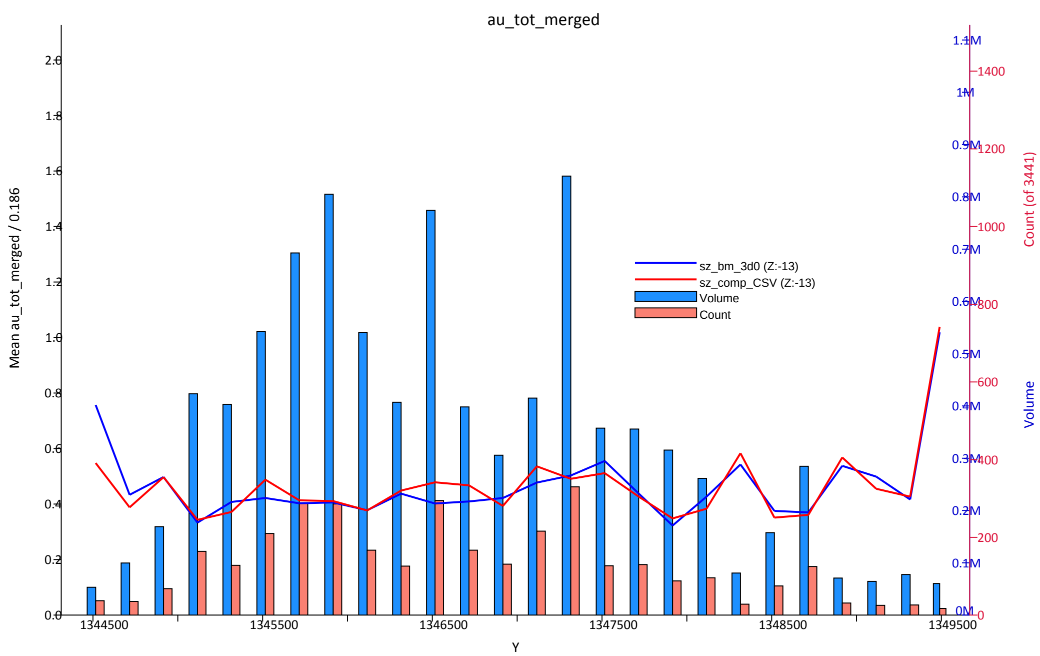


Figure 14.21 Bomboré South Zone, New Low-Grade Domain Swath Plot (by Elevation)

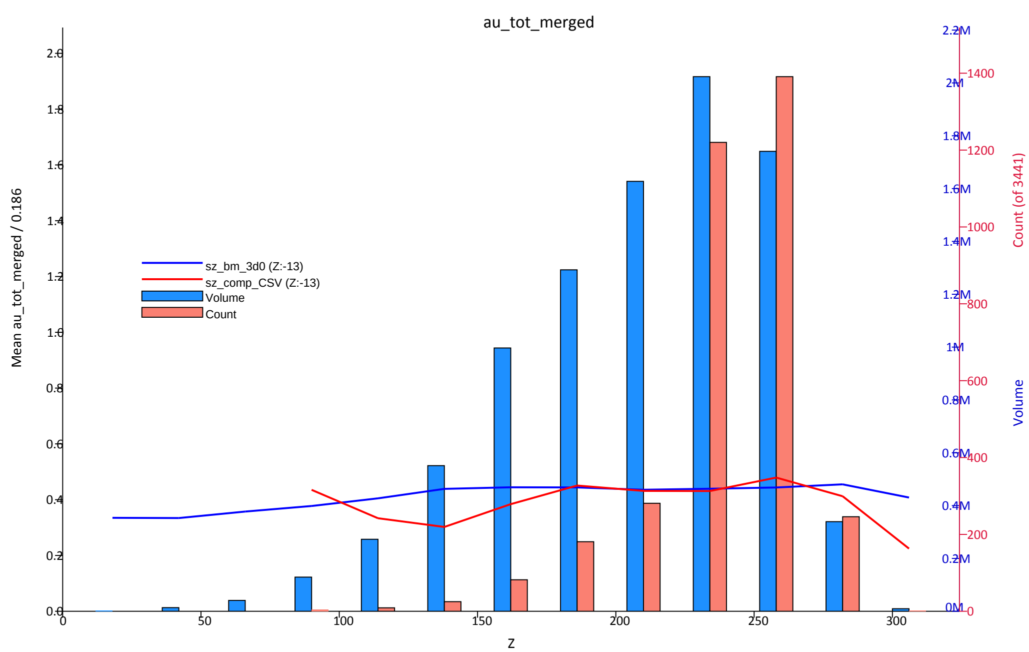


Figure 14.22 Comparison of Block Grades versus Contoured Gold Values, Test Area 1, North Model Area

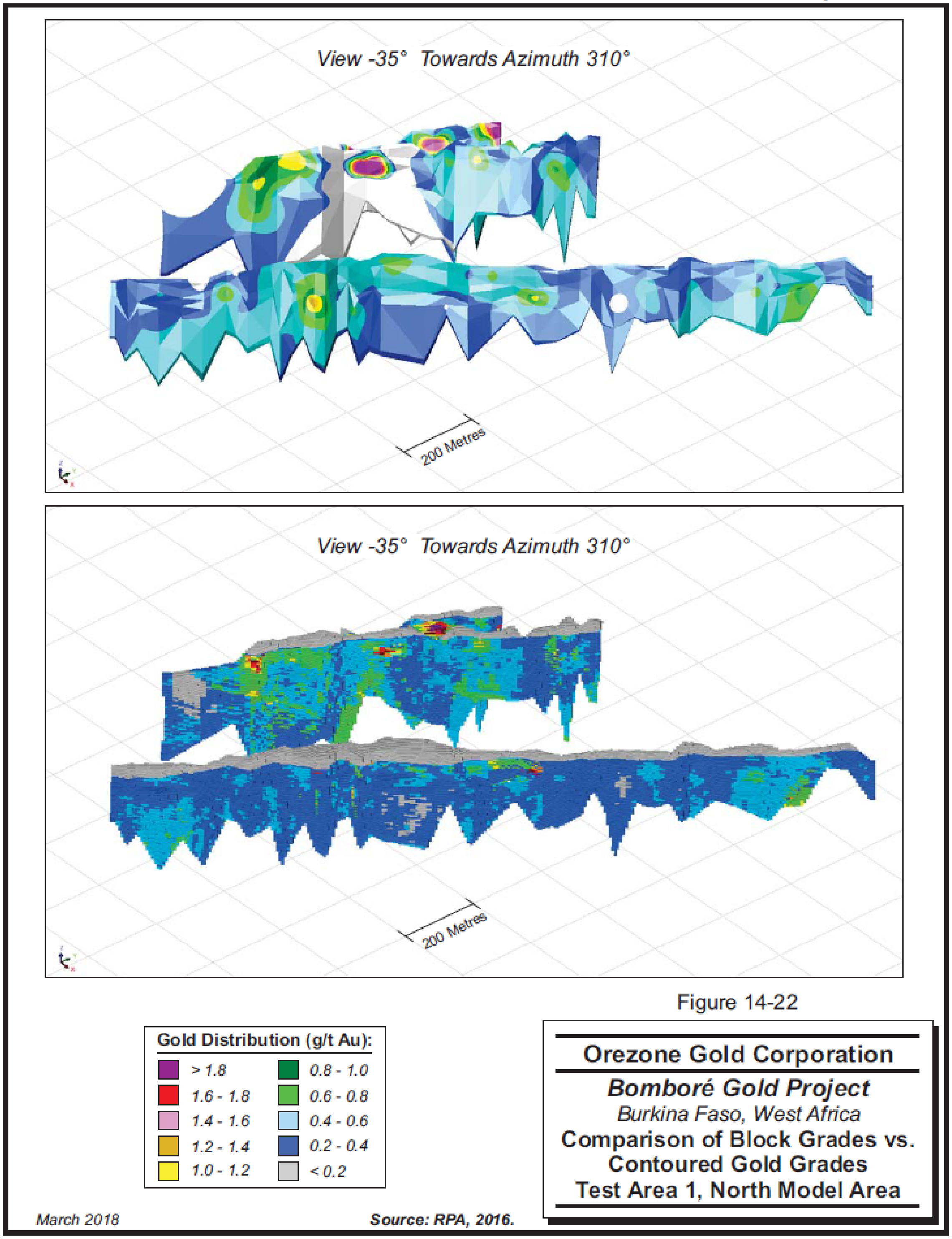


Figure 14.23 Comparison of Block Grades versus Contoured Gold Values, Test Area 2, North Model Area

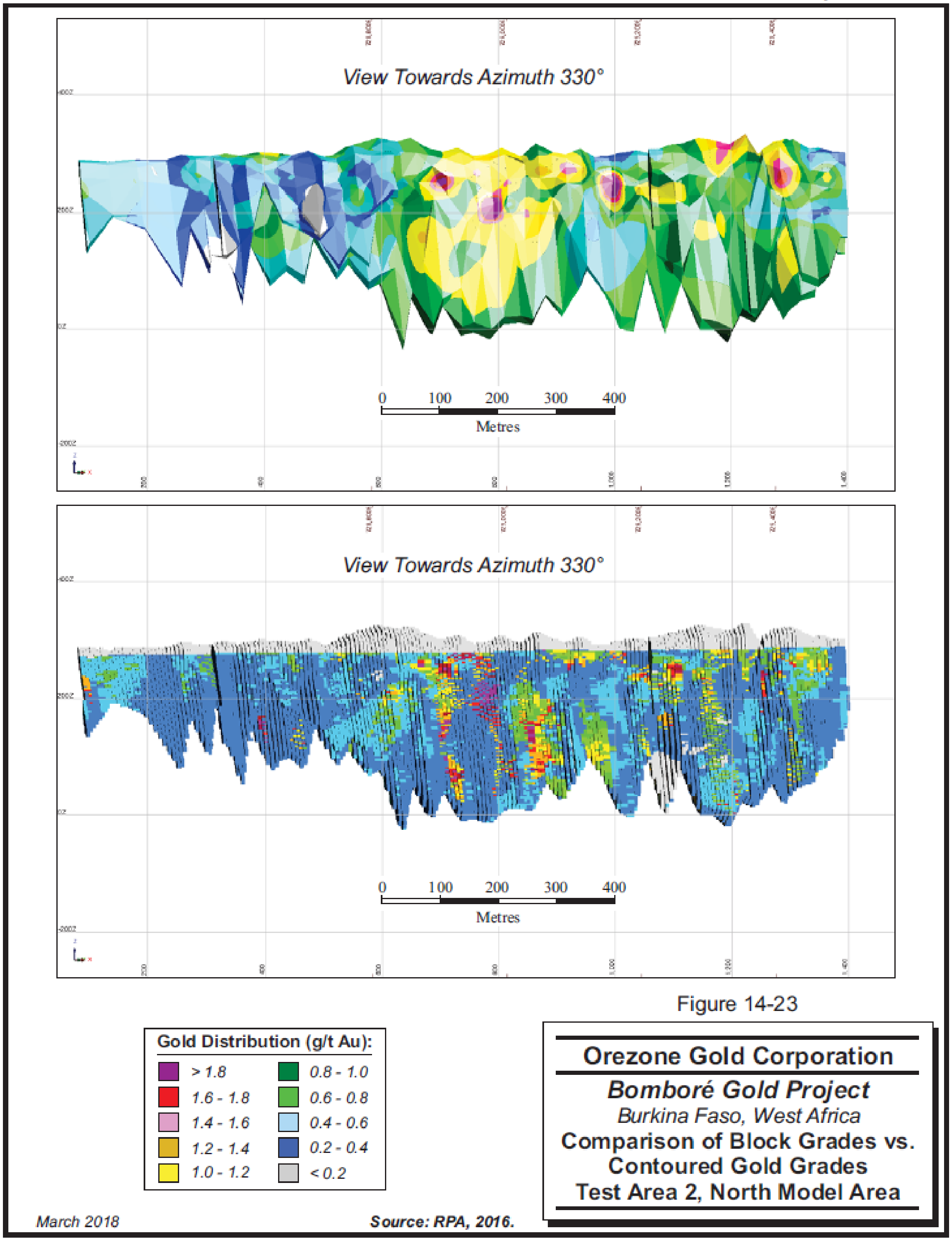


Figure 14.24 Comparison of Block Grades versus Contoured Gold Values, P11 450 Main Sub-Domain, South Model Area

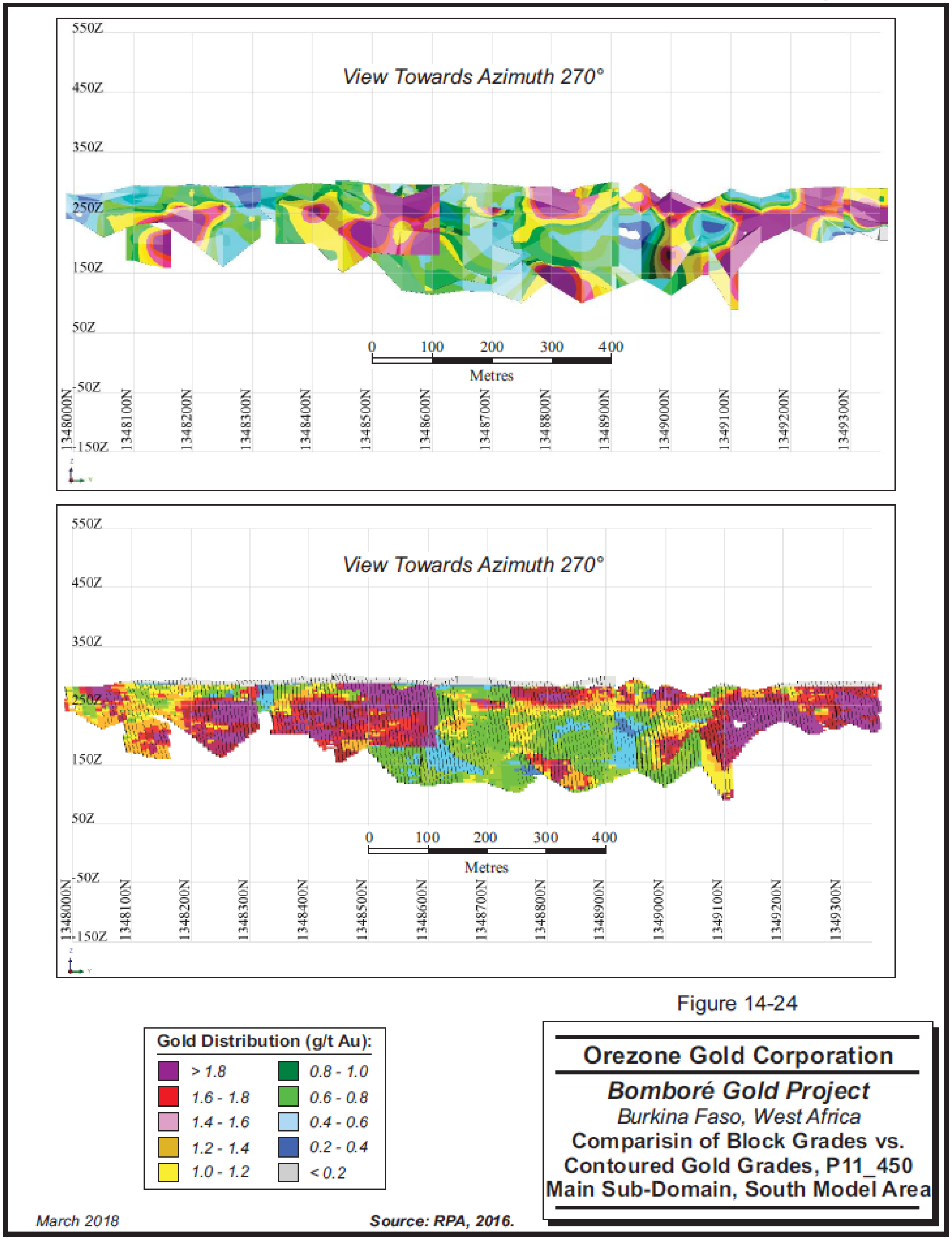
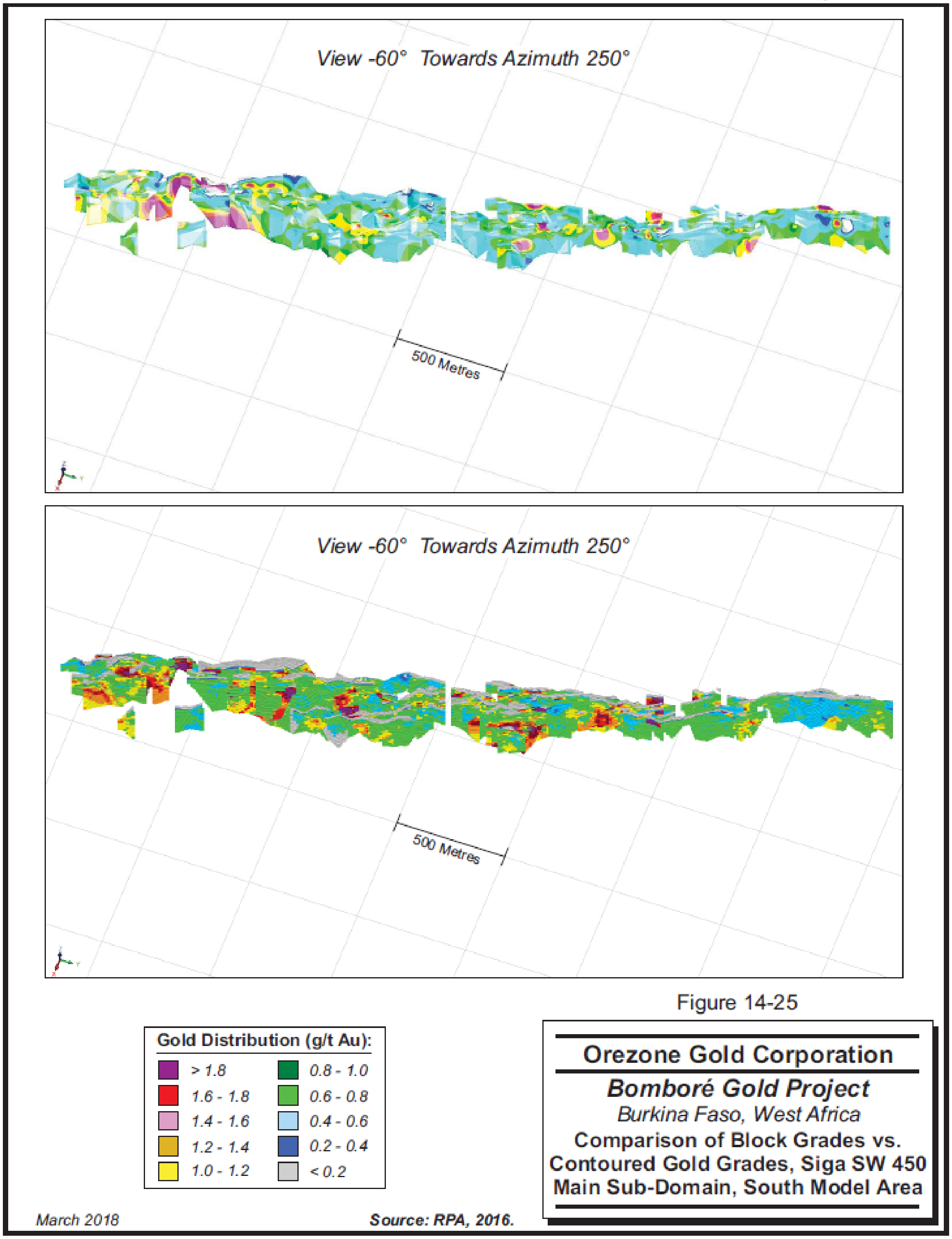


Figure 14.25 Comparison of Block Grades versus Contoured Gold Values, Siga SW 450 Main Sub-domain, South Model Area



14.12 Mineral Resource Classification Criteria

Definitions for resource categories used in this report are consistent with those defined by CIM (2014) and adopted by NI 43-101. In the CIM classification, a Mineral Resource is defined as "a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction". Mineral Resources are classified into Measured, Indicated, and Inferred categories.

The four block models were classified by RPA in accordance with the definitions contained in CIM (2014). The mineralized material for each wireframe was classified into the Measured, Indicated, or Inferred Mineral Resource category on the basis of the density of drill hole information and the location of the open pit shells generated using the Whittle software package.

Variography studies by RPA indicate that the range of gold grade continuity is approximately 50 m at 90% to 100% of the variogram sill and approximately 25 m at 50% of the sill. These observations, combined with the existing drill spacing, deposit type, and geometric continuity of the mineralized zones, led to RPA's recommendation of assigning Measured to areas supported by approximately 25 m by 25 m spaced drilling and Indicated to areas with approximately 50 m by 25 m spaced drilling.

The classification criteria were coded into the four block models on a layer-by-layer basis that followed the level of the oxidation state of the host material. For the oxide and transition layers, the upper and lower units were grouped together into one layer for each unit. A 25 m thick layer representing the top of the fresh rock unit was coded into each of the four block models, and this was retained for classification purposes.

On the basis of these criteria, Measured Mineral Resources comprise that mineralized material that has been outlined with a drill hole density of a maximum of 25 m x 25 m. Indicated Mineral Resources comprise that mineralized material that has been outlined with a nominal drill hole density of 25 m x 50 m. Inferred Mineral Resources comprise the mineralized material that has been outlined with a nominal drill hole density of 100 m x 100 m and to within a depth of 100 m below the bottom of the drill hole coverage. Examples of the final classification of the mineralized material are presented in Figures 14.26 and 14.27.

Orezone has carried out additional drilling within the resource area since the current Mineral Resource was estimated. RPA reviewed the results and is of the opinion that current resource model is still appropriate to be used as the basis for the 2018 FS, and that the effective date of the estimate should remain at January 5, 2017.

Figure 14.26 Plan View of the Classified Resources, North Model Area

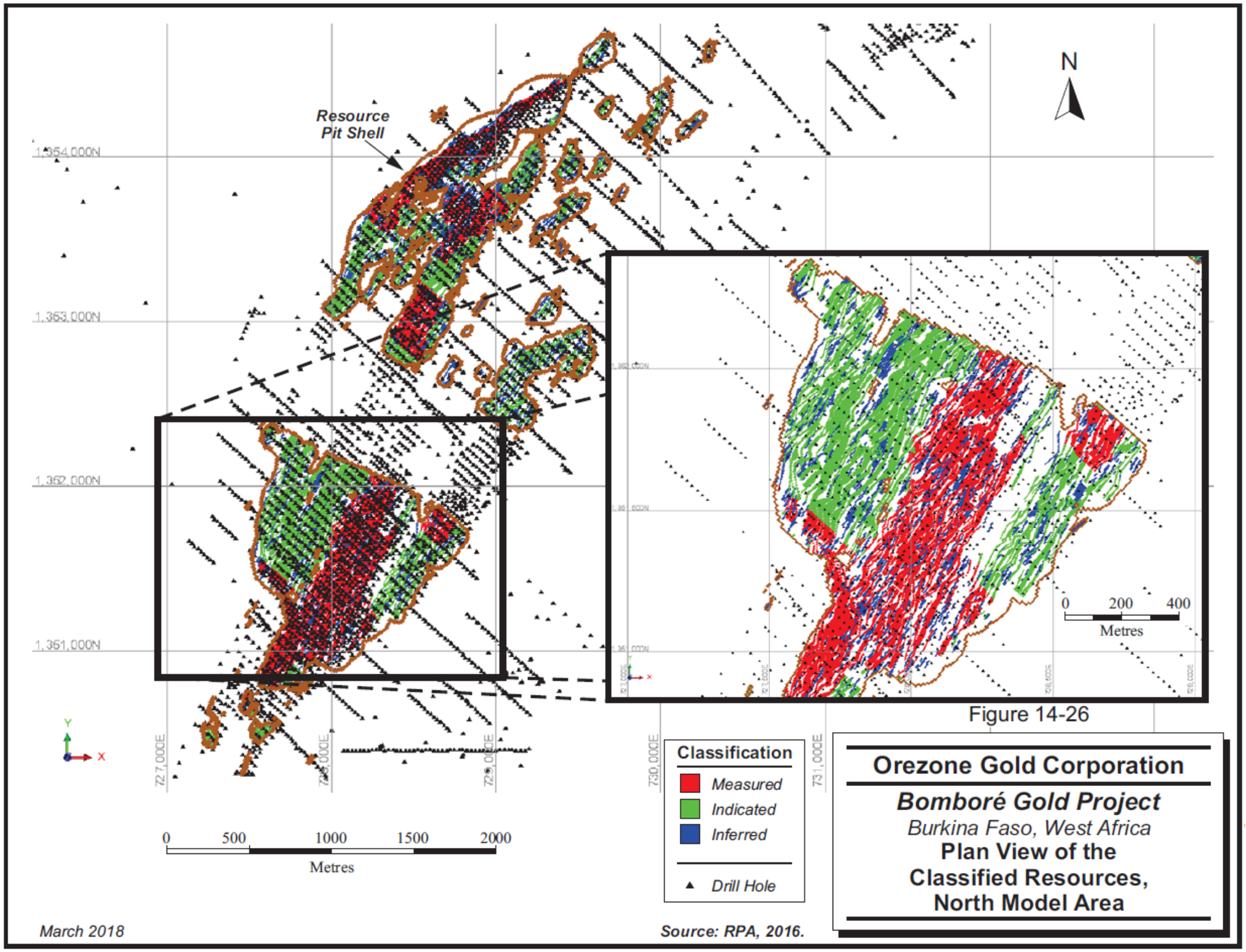
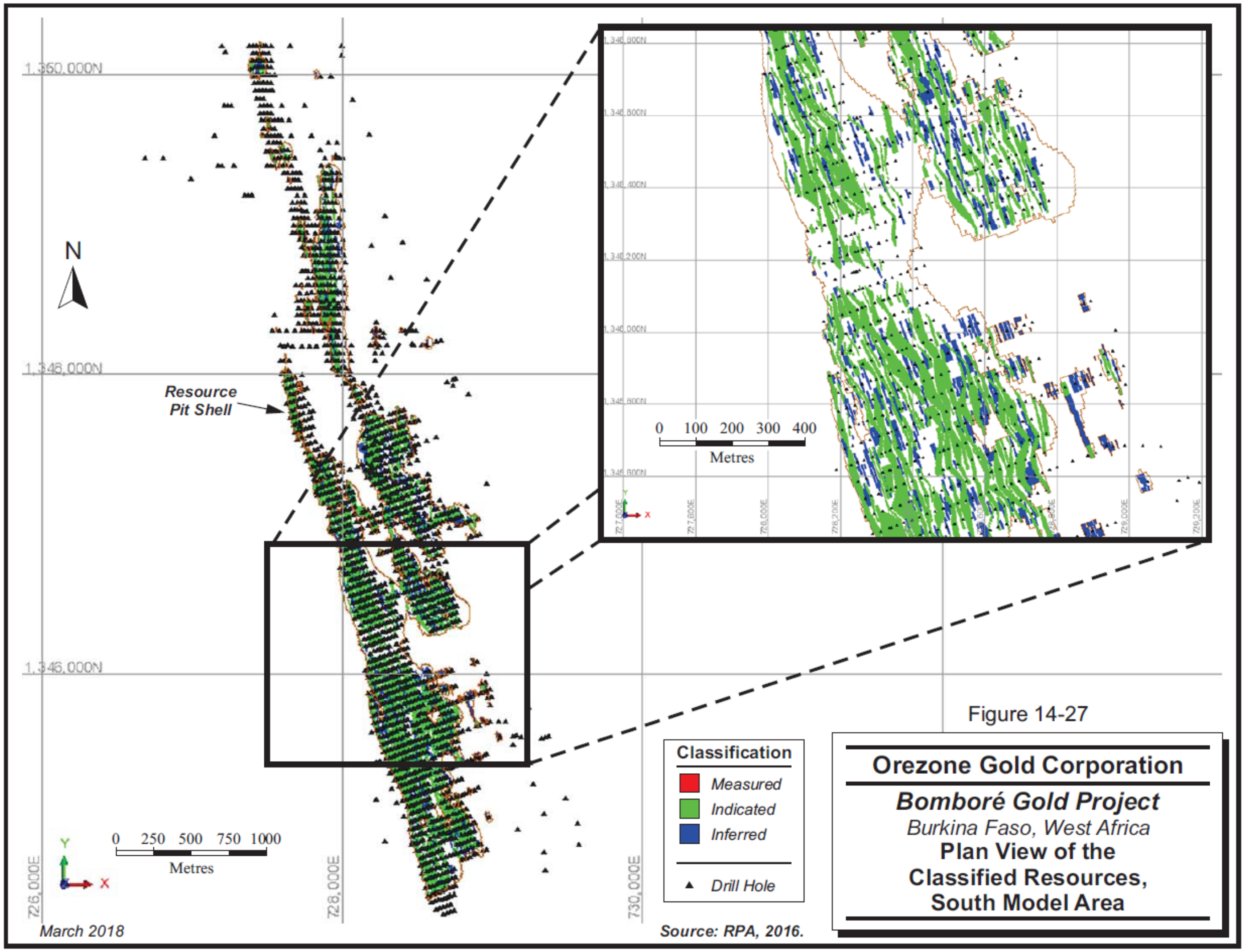


Figure 14.27 Plan View of the Classified Resources, South Model Area



14.13 Responsibility for the Estimate

The estimate of the Mineral Resources for the Project presented in this Technical Report was prepared by Mr. Babacar Diouf, Consulting Mineral Resource Geologist, under the supervision of Mr. Pascal Marquis, Geo., Senior Vice President with Orezone. This work included the development of the geological model, block model, and the grade estimation. Mr. Reno Pressacco, M.Sc.(A), P.Geo., Principal Geologist with RPA, carried out audits, classified and reported the 2016 Mineral Resource estimate. Mr. Pressacco, Mr. José Texidor Carlsson, M.Sc., P.Geo., Senior Geologist with RPA, and Mr. Tudorel Ciuculescu, M.Sc., P.Geo, Senior Geologist with RPA, share responsibility for the current 2017 Mineral Resource estimate, which includes the addition of the new 391 low-grade mineralized wireframe domains for the North and South areas and an unconstrained third domain for all four model areas to capture material remaining outside the September 2016 estimate wireframes.

14.14 Cut-Off Grade

A number of cut-off grades were developed for the Project that reflect the varying processing costs and metallurgical recoveries of the different oxidation layers and the additional transportation costs for mineralized material that is located distant to the proposed processing plant (Table 14.11). A gold price of US$1,400/oz was used for all cut-off grades for reporting of the 2017 Mineral Resources.

Metal prices used for mineral resources are based on consensus, long term forecasts from banks, financial institutions, and other sources.

To fulfil the CIM Definitions requirement of “reasonable prospects for eventual economic extraction”, RPA prepared preliminary open pit shells to constrain the block models for resource reporting purposes. The preliminary pit shells were generated using Whittle software, and the assumptions are listed in Table 14.11.

Additional criteria to constrain the Mineral Resource statement included some “non-permitted” areas related to flood plains, environmentally sensitive areas, and mineralized areas being set aside for the benefit of local artisanal miners.

Table 14.11 Whittle Parameters for Resource Pit Shells

Operating Parameter/Assumption	Units	Upper Saprolite & Regolith	Lower Saprolite	Upper Transition	Lower Transition	Fresh
Pit Wall Slopes:						
Maximum overall	Degrees	40	40	45	45	50
Mining Parameters:						
Waste Rock Mining Cost	$/t	1.30	1.30	1.60	1.60	1.75
"Ore" Mining Cost	$/t	2.45	2.45	2.75	2.75	2.75
Mining Recovery	%	97	97	97	97	95
Mining Dilution	%	3	3	3	3	5
Mining Dilution Grade	g/t Au	0.00	0.00	0.00	0.00	0.00
Processing Parameters:						
Process Cost	$/t	4.50	4.50	4.50	4.50	10.30
Re-handle Cost	$/t	0.25	0.25	0.25	0.25	0.25
Process Sustaining Cost	$/t	0.80	0.80	0.80	0.80	0.80
Recovery, Au	%	90.7	88.4	86.0	82.5	81.7
G&A Cost	$/t	1.80	1.80	1.80	1.80	1.25
Closure Cost	$/t	0.35	0.35	0.35	0.35	0.35
Revenue Parameters:						
Sale Price	US$/oz Au	1,400	1,400	1,400	1,400	1,400
Payable	% Au	99.9	99.9	99.9	99.9	99.9
Burkina Faso NSR: >US$1,300/oz Au	%	5.0	5.0	5.0	5.0	5.0
CSR	%	1.0	1.0	1.0	1.0	1.0
Selling Costs	US$/oz Au	2.50	2.50	2.50	2.50	2.50
North and South Estimated Cut-off Grade:	g/t Au	0.202	0.208	0.214	0.224	0.381
P16 and P17 Estimated Cut-off Grade:	g/t Au	0.216	0.222	0.228	0.239	0.396

The saprolite, transition, and fresh rock horizons have been split into upper and lower zones in the 2016 model. In the case of fresh rock, a single column is presented in Table 14.11, because values for the upper and lower fresh rock are identical.

In addition to Table 14.11 input parameters, RPA has applied incremental costs for mining depth and haul distance. The incremental mining cost for depth is an additional $0.025/t per 10 m vertical mined applied to all models. In the case of P16 and P17 models only, $0.50/t processed was added to account for the additional haulage distance to the proposed process facilities.

Assumptions for on-site diesel cost, heavy fuel oil (HFO) cost, and cost of electricity (HFO powered generators), which are significant input parameters to mining and processing operating costs, were $1.01/L, $0.59/L, and $0.14/kWhr, respectively.

14.15 P17S Resource Update

The P17S deposit is the south extension of the P17 area. Additional drilling and mineralization wireframe interpretations have been completed in the P17S deposit. The gold mineralized zones have been modelled as a number of tabular zones centred on a folded and dislocated granodiorite unit that is intruded in a meta-gabbro unit itself intruding a W-shape meta-sedimentary folded sequence. Gold values are contained by various rock types but are most commonly within granodioritic intrusives and can be seen to follow a stratiform orientation. The P17S deposit extends approximately 1,000 m from north to south and to a depth of 250 m. The P17S deposit is open to the north and at depth. Surface weathering has affected the rocks in the P17S deposit to an average depth of between five metres and ten metres.

Mineral Resources at P17S are reported within a preliminary pit shell generated in Whittle software at a reporting cut-off grade of 0.20 g/t Au for oxidized and transition rocks and a 0.38 g/t Au cut-off grade for fresh rocks. It is possible that the gold cut-off values are adjusted based on the engineering work as part of the Phase II Sulphide Expansion study. Mineral Resources are contained within vertical/sub-vertical and northeast dipping high grade lenses. Overall, the deposit is approximately 1,000 m in the north-south direction by 180 m in the east-west direction by 200 m thick. The deposit is located on a relatively flat-lying topographic plain.

The P17S Mineral Resource estimate, effective December 21, 2018, is listed in Table 14.12. The Mineral Resource estimates conform to CIM (2014) definitions. RPA notes that the effective date of the deposit as a whole remains January 5, 2017 since the bulk of the Mineral Resources (North, South etc.) have not been updated since that estimate.

Table 14.12 Mineral Resources at Bomboré P17S, as of December 21, 2018

Material Type	Measured			Indicated			Measured + Indicated			Inferred		
	Tonnes kt	Gradeg/t	Gold koz	Tonnes kt	Grade g/t	Gold koz	Tonnes kt	Grade g/t	Gold koz	Tonnes kt	Grade g/t	Gold koz
Total Oxide+Tran	26	1.89	2	21	1.22	1	46	1.59	2	38	0.68	1
Total Fresh	1,045	2.26	76	617	1.89	38	1,662	2.13	114	1,563	1.07	54
Oxide+Tran +Fresh NE Zone	1,071	2.26	78	638	1.87	38	1,708	2.11	116	1,601	1.06	55

Notes:

1. *CIM (2014) definitions were followed for Mineral Resources.*
2. *Mineral Resources are reported above a cut-off of 0.20 g/t Au in oxide and transition material and 0.38 g/t Au in fresh material. It is possible that the gold cut-off values are raised.*
3. *Mineral Resources are estimated using a long-term gold price of US$1,450 per ounce.*
4. *Bulk density varies by material type.*
5. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
6. *Numbers may not add due to rounding.*

RPA is not aware of any environmental, permitting, legal, title, taxation, socio-political, marketing, and other relevant factors that would affect the Bomboré Mineral Resource estimates at P17S.

The P17S drill hole database includes 108 diamond drill holes (16,423 m) and 54 RC holes (1,979 m), totalling 162 drill holes (18,402 m), and seven channels totalling 23.4 m. The three-dimensional (3D) wireframe mineralization models were generated using Leapfrog software on a cut-off value of 0.20 g/t Au. Assays were composited to 1 m lengths with a minimum length of 0.25 m for each individual mineralization wireframe. Block model grades within the wireframe models were interpolated by ID^2. Density was estimated to be between 1.94 t/m^3 and 3.20 t/m^3 for the mineralization and between 1.55 t/m^3 and 3.37 t/m^3 for the waste based on density measurements from core samples. The Mineral Resources were assigned Measured, Indicated, and Inferred category in the mineralization based on drill hole spacing.

14.15.1 Resource Database at P17S

RPA received header, survey, assay, alteration, and geology data from P17S imported into Seequent's Leapfrog Version 4.2.2 for Mineral Resource modelling. Orezone provided Leapfrog wireframe files (msh) containing the latest geological, oxidation, and mineralization domain interpretations. RPA used these files to develop a series of resource wireframes. The latest drill hole included in the resource database is BBD1000. A summary of records for all drilling on the P17S deposit is listed below:

- Holes: 162
- Channels: 7
- Surveys: 2,465
- Assays: 28,162
- Composites 2,101

- Lithology: 3,387
- Density measurements: 2,385

Data verification of the drill hole database included verification against original digital sources, a series of digital queries, and a review of Orezone's quality assurance and quality control (QA/QC) procedures and results. No discrepancies were identified and RPA is of the opinion that the drill hole database is valid and suitable to estimate Mineral Resources for Bomboré's P17S deposit.

14.15.2 Geological Interpretation and 3D Solids

With recent drilling, Orezone improved the understanding of the lithological setting of the gold mineralization at the Project. Orezone used the new geological model to better delineate the mineralized zones within the host rocks.

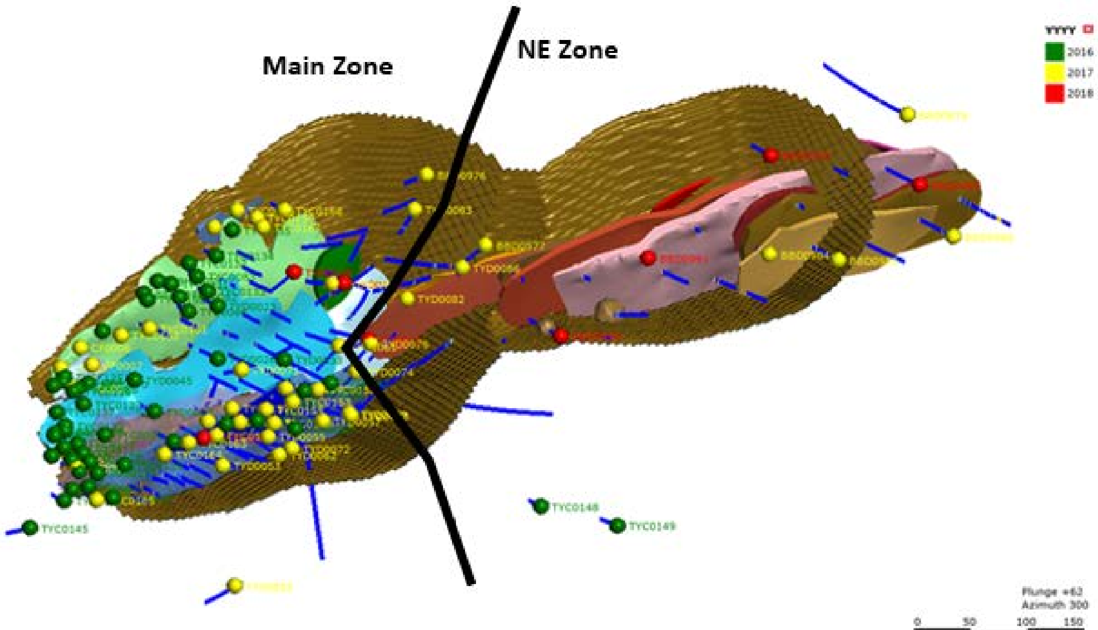
Wireframe models of mineralized zones were used to constrain the block grade interpolation process. A Leapfrog Geo 4.4.2 vein system modelling tool was used to generate interpretation of the mineralization at a nominal cut-off of 0.20 g/t Au for the mineralization. No minimum thickness was applied, with most mineralized sections being several metres wide and up to 20 m wide. Occasionally, lower grade intersections were included to maintain continuity. At model extremities, the wireframe models in the Main Zone were extrapolated up to 15 m beyond the last drill hole section, with some areas of good continuity along trend where they were extrapolated up to 50 m. In the Northeast (NE) Zone, the wireframe models were extrapolated up to 25 m beyond the last drill hole section, with some areas of good continuity along trend where they were extrapolated up to 80 m.

RPA reviewed 22 domains in two zones, the Main Zone and the NE Zone (Figure 14.28). A description of each modelled domain follows:

- The Main Zone extends for 600 m north-south and it is the highest gold grade area within P17S. It is made up of 11 parallel domains, three sub-vertical and eight dipping to the east. The thickness of individual domains ranges from 3 m to 10 m, with some zones pinching out and some zones as wide as 15 m. The Main Zone was predominantly drilled between 2016 and 2017.

The NE Zone extends for 600 m north-south and contains the largest volume of mineralization wireframes in P17. It is made up of 11 parallel vertical/sub-vertical domains, ten dipping to the east and one dipping to the west, ranging in length from 200 m to 490 m and up to 240 m below surface. The thickness of individual domains ranges from 3 m to 10 m, with some zones pinching out and some zones as wide as 20 m. The NE Zone was drilled between 2017 and 2018.

Figure 14.28 3D View of Bomboré P17S Wireframe Domains and Zones, including drill holes by year



14.15.3 Statistical Analysis

Assay values located inside the wireframe models were tagged with domain identifiers and exported for statistical analysis. Results were used to help verify the modelling process. Statistics by zone are summarized in Table 14.13.

Table 14.13 Descriptive Statistics of Resource Assay Values

Zone and Statistic	Au (g/t)
Main Zone	
Count	1,549
Min	0.001
Max	32.70
Average	2.07
Variance	7.10
StDev	2.66
CoV	1.28
NE Zone	
Count	941
Min	0.001
Max	87.75
Average	1.02
Variance	9.44
StDev	3.07
CoV	2.99

14.15.4 Capping High Grade Values

Where the assay distribution is skewed positively or approaches log-normal, erratic high-grade assay values can have a disproportionate effect on the average grade of a deposit. One method of treating these outliers in order to reduce their influence on the average grade is to cut or cap them at a specific grade level. In the absence of production data to calibrate the capping level, inspection of the assay distribution can be used to estimate a “first pass” capping level.

Review of the resource assay exploratory data analysis (EDA) including histograms (Figure 14.29) and probability plots within the wireframe domains (Figure 14.30), and a visual inspection of high-grade values on vertical sections suggest that cutting of erratic values at 10 g/t Au is warranted and appropriate for the P17S deposit. Table 14.14 lists the descriptive statistics of capped resource assay values by zone.

Figure 14.29 Au Histogram Distribution by Grade

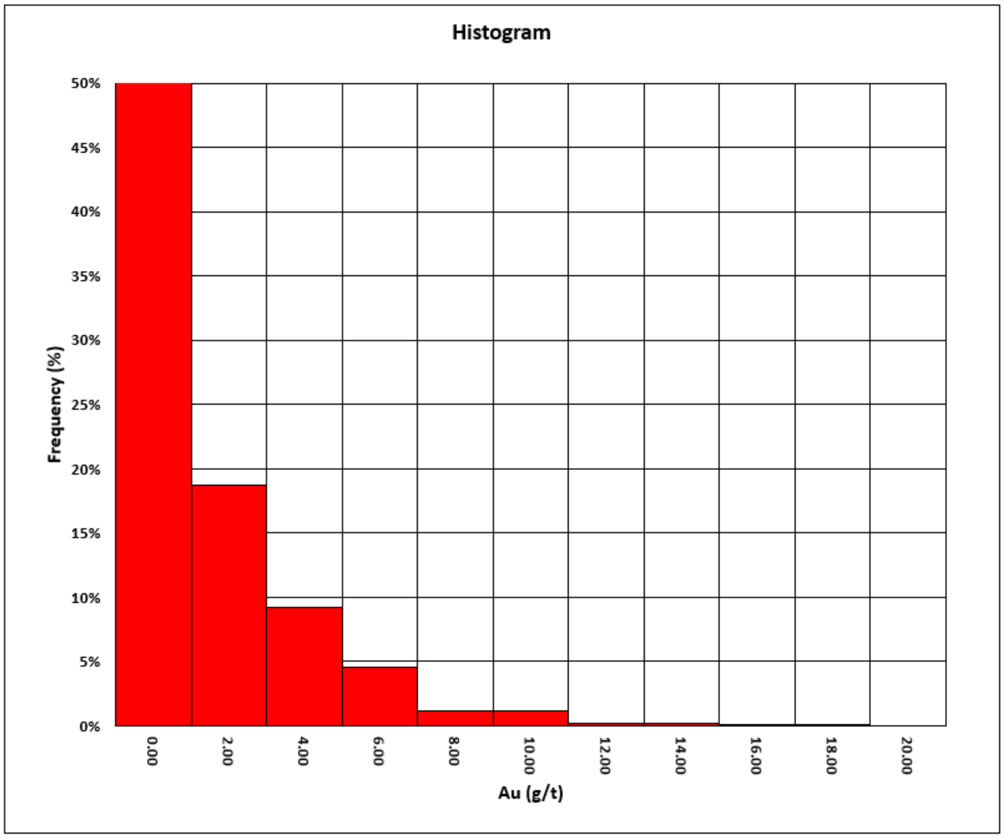


Figure 14.30 Au Probability Plots

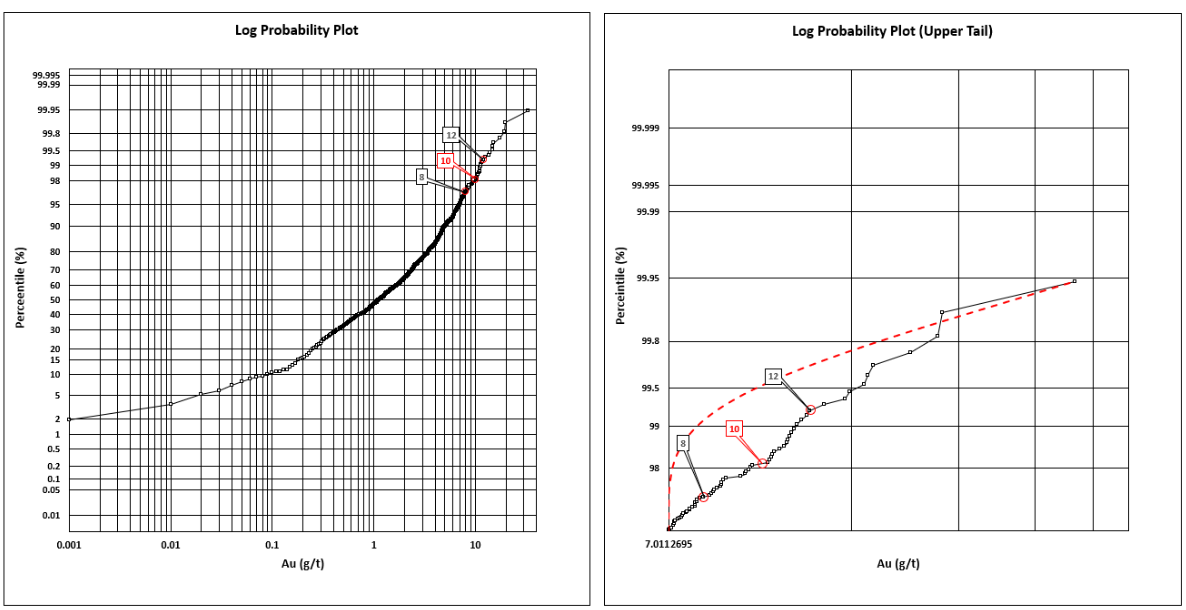


Table 14.14 Descriptive Statistics of Capped Resource Assay Values

Zone and Statistic	g/t Au
Main Zone	
Count	1,549
Min	0.001
Max	10.00
Average	1.99
Variance	5.11
StDev	2.26
CoV	1.13
NE Zone	
Count	941
Min	0.001
Max	10.00
Average	0.93
Variance	2.01
StDev	1.41
CoV	1.52

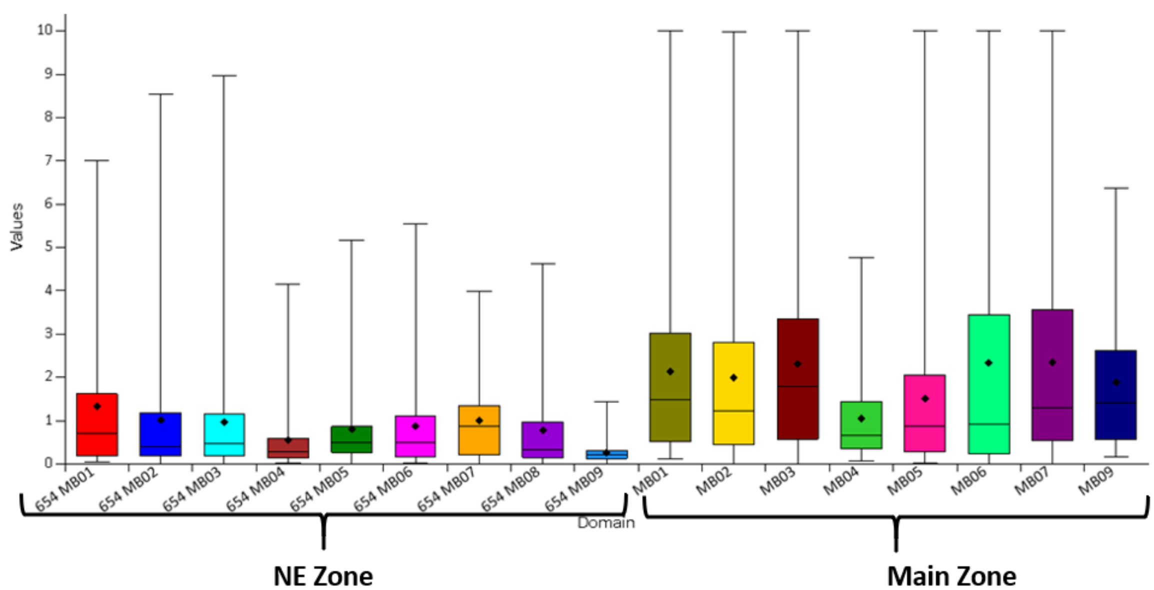
14.15.5 Compositing

Sample lengths range from 0.1 m to 4.8 m within the resource domain wireframe models with 92% of samples taken at approximately 1 m. Given these distributions, and considering the width of the mineralization, RPA chose to composite to one metre lengths. Assays within the wireframe domains were composited using the downhole compositing method, which starts at the first mineralized wireframe boundary from the collar and resets at each new wireframe boundary. Composites less than 0.25 m, located at the bottom of the mineralized intercept, were removed from the database. Table 14.15 lists descriptive statistics of the composites by zone. Figure 14.31 shows the grade distribution of capped composites by wireframe domain and indicates that the average grades of the Main Zone domains is higher than the average grade domains of the NE Zone. Overall, the average grade of east dipping domains is higher than that of vertical or sub-vertical domains.

Table 14.15 Descriptive Statistics of Composite Values

Zone and Statistic	Au (g/t)
Main Zone	
Count	1,549
Min	0.001
Max	10.00
Average	1.99
Variance	5.11
StDev	2.26
CoV	1.13
NE Zone	
Count	941
Min	0.001
Max	10.00
Average	0.93
Variance	2.01
StDev	1.41
CoV	1.52

Figure 14.31 Au Grade Distribution of Capped Composites by Domain and Zone



Variography was assessed on Au composites to determine the search ellipsoid dimensions for each of the mineralization domains. Grades were interpolated by ID^2 and two passes, with a minimum of five to a maximum of ten composites per block estimate for first pass, and a minimum of two to a maximum of fifteen composites per block estimate in the second pass. A maximum of three composites per drill hole was applied during the first pass to inform interpolated blocks by at least two boreholes. The first pass search distance was equal to the range at 100% of the variogram sill (variance), and this distance was doubled for the second pass to fill remaining blocks (Table 14.16). The search ellipse orientation varied by domain (Table 14.17).

Hard boundaries were used to limit the use of composites between wireframe boundaries. Figures 14.32 and 14.33 show examples of the interpolation results of the block grade estimates. Gold was interpolated for all the domains in the Main Zone except MB08 and MB10 and for all domains in the NE Zone except MB10 and MB11. The Main Zone's MB08 and MB10 domains and the NE Zone's 654MB10 domain were not interpolated due to their small volumes and lack of support data. The NE Zone's 654MB11 domain was not interpolated as it is a very small volume and it is only intersected by two drill holes.

Table 14.16 Block Estimate Parameters and Search Ellipsoid Distances, by Domain and Zone

Zone	Domain	Min. No. of Comps.		Max No. of Comps.		Max Comps. per Drill Hole		Distance X (m)		Distance Y (m)		Distance Z (m)	
		P1	P2	P1	P2	P1	P2	P1	P2	P1	P2	P1	P2
Central	MB01	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
Central	MB02	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
Central	MB03	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
Central	MB04	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
Central	MB05	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
Central	MB06	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
Central	MB07	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
Central	MB09	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
Central	MB11	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
NE	654MB01	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
NE	654MB02	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
NE	654MB03	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
NE	654MB04	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
NE	654MB05	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
NE	654MB06	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
NE	654MB07	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
NE	654MB08	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50
NE	654MB09	5	2	10	15	3	N/A	30	60	30	60	3.75	7.50

** Note: Leapfrog Dip, Dip Azimuth and pitch nomenclature is used above.*
P1 = pass one, P2 = pass two

Table 14.17 Block Estimate Search Ellipse Orientation

Zone	Mineralization Domain	Search Ellipse Orientation in Leapfrog		
		Dip (°)	Dip Azimuth (°)	Pitch (°)
Central	MB01	55.46	61.22	130
Central	MB02	49.75	71.12	40
Central	MB03	37.56	62.04	40
Central	MB04	79.86	87.24	10
Central	MB05	85.29	86.05	15
Central	MB06	50.20	73.60	15
Central	MB07	47.54	65.23	25
Central	MB09	56.09	73.49	35
Central	MB11	80.40	81.68	5
NE	MB01	87.04	94.42	10
NE	MB02	73.21	85.98	25
NE	MB03	86.83	89.72	15
NE	MB04	74.57	83.47	30
NE	MB05	74.10	82.37	20
NE	MB06	79.74	87.90	20
NE	MB07	79.74	87.90	20
NE	MB08	78.47	293.31	155
NE	MB09	77.68	87.42	30

** Note: Leapfrog Dip, Dip Azimuth and Pitch nomenclature is used above.*

Figure 14.32 Main Zone, Vertical Cross Section at N1342718, Looking North

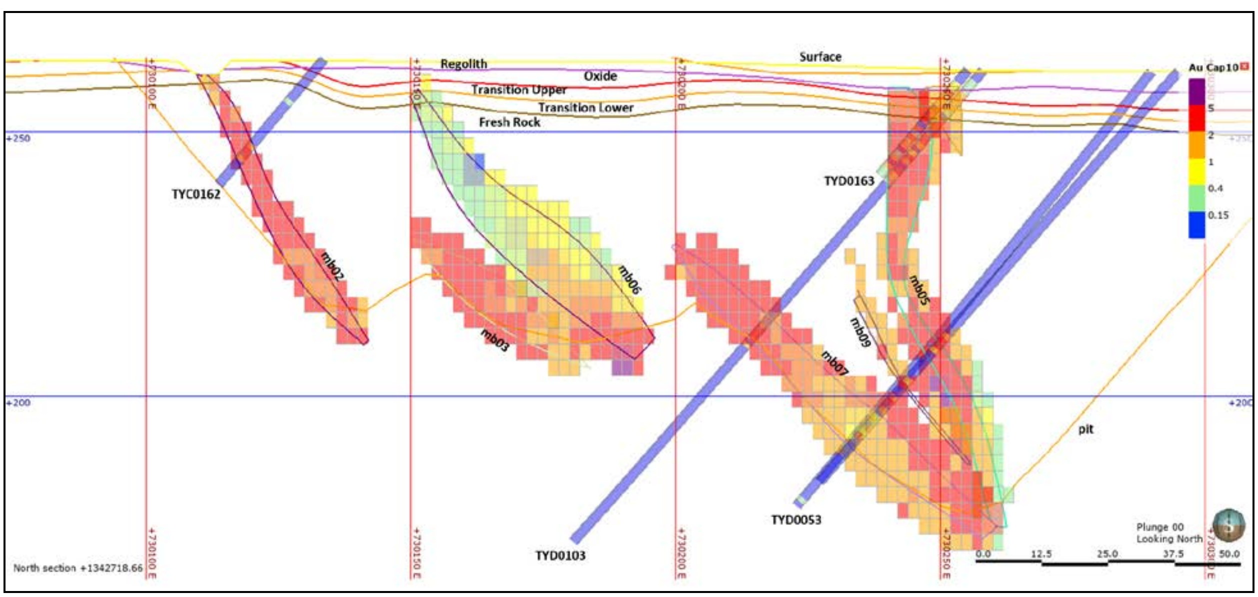
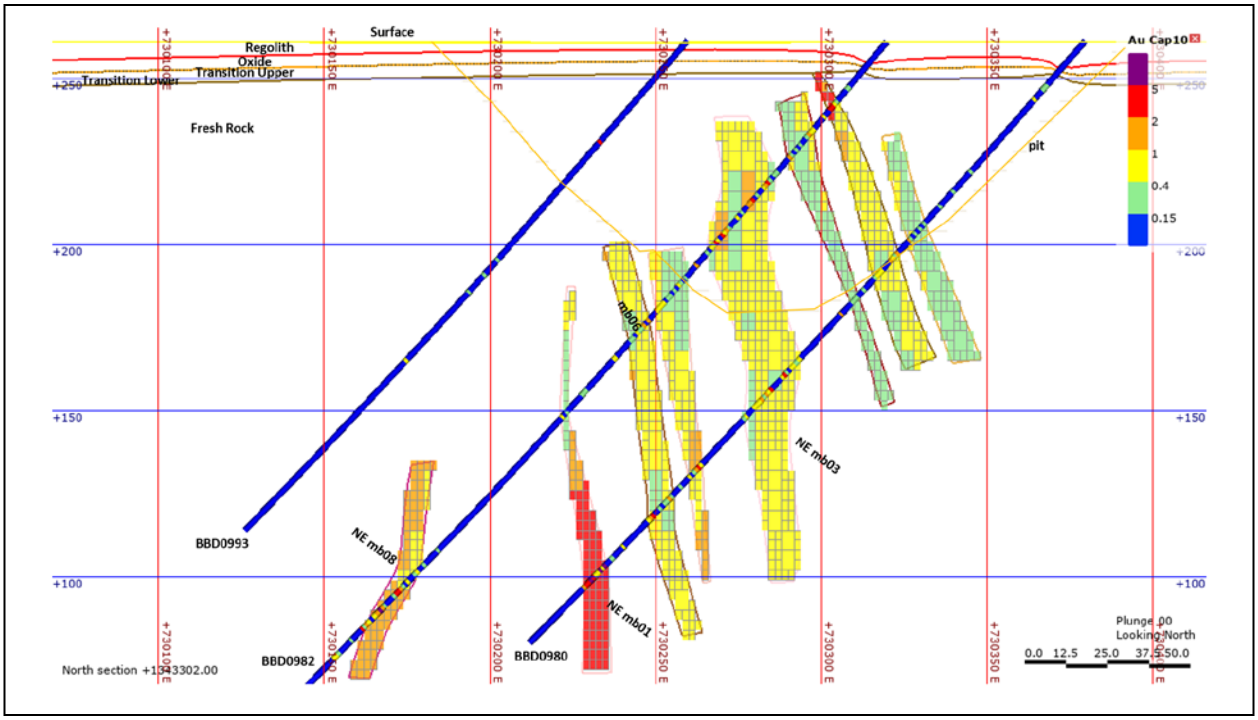


Figure 14.33 NE Zone, Vertical Cross Section at 1343302, Looking North



14.15.6 Density

A total of 2,385 density measurements are located within the wireframe models representing the mineralized domains and a total of 12,629 density measurements are located within the waste. After removing outliers from the low and high ends of the distribution (three highest and three lowest values for each rock type, in mineralization and waste), RPA used the average of these measurements combined with a moisture factor for oxide and transition materials and the average of these measurements for fresh material, without any moisture factor (Table 14.18). Due to some uncertainty on the exact extent of the I1C lithology (felsic intrusive) and based on the statistics of fresh density values, it was decided to allocate the same value to fresh Mineralization and waste within the I1C lithology as the calculated average for fresh-all (except I1C, MI3, S3).

Table 14.18 Descriptive Statistics of Densities (t/m³)

Material	Oxidation	Count	Minimum	Maximum	Average	Assigned
Mineralization	Oxide	2	2.04	2.38	2.02*	2.02
Mineralization	Upper Trans.	10	1.94	2.69	2.39*	2.39
Mineralization	Lower Trans.	15	1.94	2.81	2.54*	2.54
Mineralization	Fresh-All, except I1C, MI3	1,313	2.37	3.33	2.80	2.77
Mineralization	Fresh, I1C	832	NC	NC	2.77	2.77
Mineralization	Fresh, MI3	213	NC	NC	2.96	2.96
Waste	Oxide	255	1.43	3.52	1.84*	1.84
Waste	Upper Trans.	201	1.43	3.14	2.35*	2.35
Waste	Lower Trans.	229	1.61	3.14	2.59*	2.59
Waste	Fresh-All, except I1C, MI3, S3	6,949	1.74	3.67	2.77	2.77
Waste	Fresh, I1C	674	NC	NC	2.78	2.77
Waste	Fresh, MI3	4,321	NC	NC	3.05	3.05
Waste	Fresh, S3	1,187	NC	NC	2.77	2.77

Notes:
**Average with moisture factor calculation*
NC: value not calculated

14.15.7 Block Model

A model of 1,561,953 blocks was built using Leapfrog (v 4.4.2). The block model is made up of 174 rows, 103 columns, and 46 levels. The model origin (lower-left corner at highest elevation) is at coordinates 729,840 mE, 1,342,350 mN and 282 m elevation. Each block is 4 m wide by 12.5 m long by 6 m high, with sub-blocks 2 m wide by 6.25 m long by 3 m high. The block model contains the following information:

- Mineralization domain.
- Oxidation.
- Estimated grade of gold.
- Density.
- Lithology.
- Classification.
- Presence within the Resource Pit.
- Number of samples used to estimate grade.
- Distance to the closest composite used to interpolate the block grade.

14.15.8 Cut-off Grades

Pit optimization analyses were run on the block model to determine the potential economics of extraction by open pit methods. The parameters used in the pit optimization runs using Whittle software are presented in Table 14.19.

Whittle calculates a final breakeven pit shell based on all operating costs (mining, processing, and general and administrative (G&A)) required to mine a given block of material. Since all blocks within the breakeven pit shell must be mined (regardless if they are waste or mineral), any block that has sufficient revenue to cover the costs of processing and G&A is sent to the processing plant. An incremental, or pit discard, cut-off grade is calculated using only the processing and G&A costs.

Gold grades were calculated by RPA for the purposes of geological interpretation and resource reporting. Net smelter return (NSR) is the estimated value per tonne of mineralized material after allowance for metallurgical recovery and consideration of smelter terms, including payables, treatment charges, refining charges, price participation, penalties, smelter losses, transportation, and sales charges. These assumptions are dependent on the processing scenario, and will be sensitive to changes in inputs from further metallurgical testwork. Key assumptions are listed below. Assumed recoveries are based on testwork and experience from other operations.

Table 14.19 Whittle Pit Parameters

Operating Parameter/Assumption	Units	Upper Saprolite & Regolith	Lower Saprolite	Upper Transition	Lower Transition	Fresh
Pit Wall Slopes:						
Maximum overall	Degrees	40	40	45	45	50
Mining Parameters:						
Waste Rock Mining Cost	$/t	1.30	1.30	1.30	1.60	1.75
"Ore" Mining Cost	$/t	2.35	2.35	2.35	2.65	2.80
Mining Recovery	%	98	98	98	98	98
Mining Dilution	%	3	3	3	3	5
Mining Dilution Grade	g/t Au	0.00	0.00	0.00	0.00	0.00
Processing Parameters:						
Process Cost	$/t	5.00	5.00	5.00	10.05	10.05
Re-handle Cost	$/t	0.25	0.25	0.25	0.25	0.25
Process Sustaining Cost	$/t	1.00	1.00	1.00	1.00	1.00
Recovery, Au	%	90.0	90.0	90.0	89.0	82.0
G&A Cost	$/t	2.22	2.22	2.22	2.22	2.22
Closure Cost	$/t	0.15	0.15	0.15	0.15	0.15

Operating Parameter/Assumption	Units	Upper Saprolite & Regolith	Lower Saprolite	Upper Transition	Lower Transition	Fresh
Revenue Parameters:						
Sale Price	US$/oz Au	1,450	1,450	1,450	1,450	1,450
Payable	% Au	99.9	99.9	99.9	99.9	99.9
Burkina Faso NSR: >US$1,300/oz Au	%	5.0	5.0	5.0	5.0	5.0
CSR	%	1.0	1.0	1.0	1.0	1.0
Selling Costs	US$/oz Au	3.50	3.50	3.50	3.50	3.50
P17S Estimated Cut-off Grade:	**g/t Au**	**0.258**	**0.258**	**0.258**	**0.424**	**0.391**

Metal prices used for reserves are based on consensus, long term forecasts from banks, financial institutions, and other sources. For resources, metal prices used are slightly higher than those for reserves.

14.15.9 Classification

Definitions for resource categories used in this Technical Report are consistent with those defined by CIM (2014) and incorporated by reference into NI 43-101. In the CIM classification, a Mineral Resource is defined as "a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction". Mineral Resources are classified into Measured, Indicated, and Inferred categories.

The classification criteria used to define the Indicated Mineral Resources included spatial analysis, drill hole spacing, and continuity of the mineralization. The drill hole spacing within a resource area assigned the Indicated category is commonly ranges from 40 m to 70 m. The limits of the Mineral Resource remain open in several directions.

14.15.10 Mineral Resource Validation

RPA validated the block model by visual inspection, volumetric comparison, and statistical comparison of block grades to assay and composite grade. Visual comparison on vertical sections and plan views, and a series of swath plots found good overall correlation between the block grade estimates and supporting composite grades.

The estimated total volume of the wireframe models is 2,377,271 m^3, while the volume of the block model at a zero-grade cut-off is 2,376,807 m^3. Results of individual volume comparisons are listed by zone and domain in Table 14.21.

Table 14.20 Volume Comparisons

Zone	Domain	Block Volume (m^3 x 1,000)	Wireframe Volume (m^3 x 1,000)	Volume Difference %
Central	MB01	36,518	36,825	0.8%
Central	MB02	211,860	212,175	0.1%
Central	MB03	189,540	190,612	0.6%
Central	MB04	10,991	10,762	-2.1%
Central	MB05	84,992	85,387	0.5%
Central	MB06	91,093	91,200	0.1%
Central	MB07	86,816	87,150	0.4%
Central	MB09	16,698	16,612	-0.5%
Central	MB11	6,511	6,525	0.2%
NE	654 MB01	144,050	142,162	-1.3%
NE	654 MB02	241,690	241,125	-0.2%
NE	654 MB03	524,840	525,450	0.1%
NE	654 MB04	110,830	110,325	-0.5%
NE	654 MB05	130,350	130,050	-0.2%
NE	654 MB06	257,090	258,000	0.4%
NE	654 MB07	87,202	87,375	0.2%
NE	654 MB08	65,966	66,787	1.2%
NE	654 MB09	78,748	77,887	-1.1%
	Total	**2,376,807**	**2,377,271**	**0.0%**

Statistical analysis comparing estimated block grades versus composite grades is shown in a histogram in Figure 14.34. RPA created a number of swath plots for selected wireframes. While some local variations were observed between the composite average grades and the block average grades, no material discrepancies were noted (Figures 14.33 to 14.36).

Figure 14.34 Histogram of Estimated Block versus Composite Grades

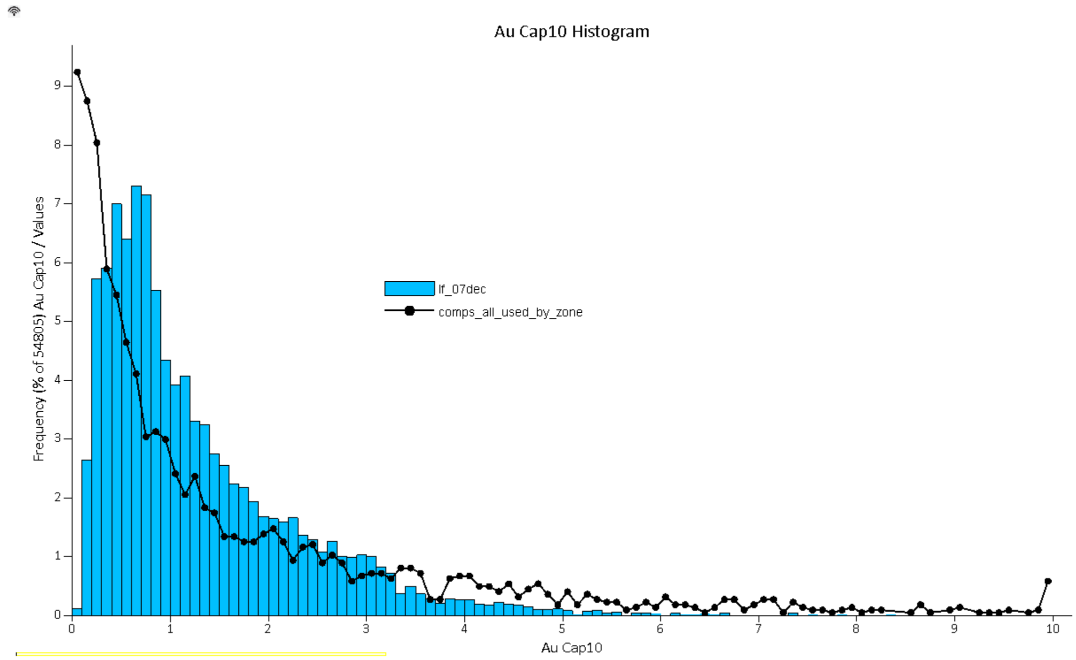


Figure 14.35 Swath Plot by Easting

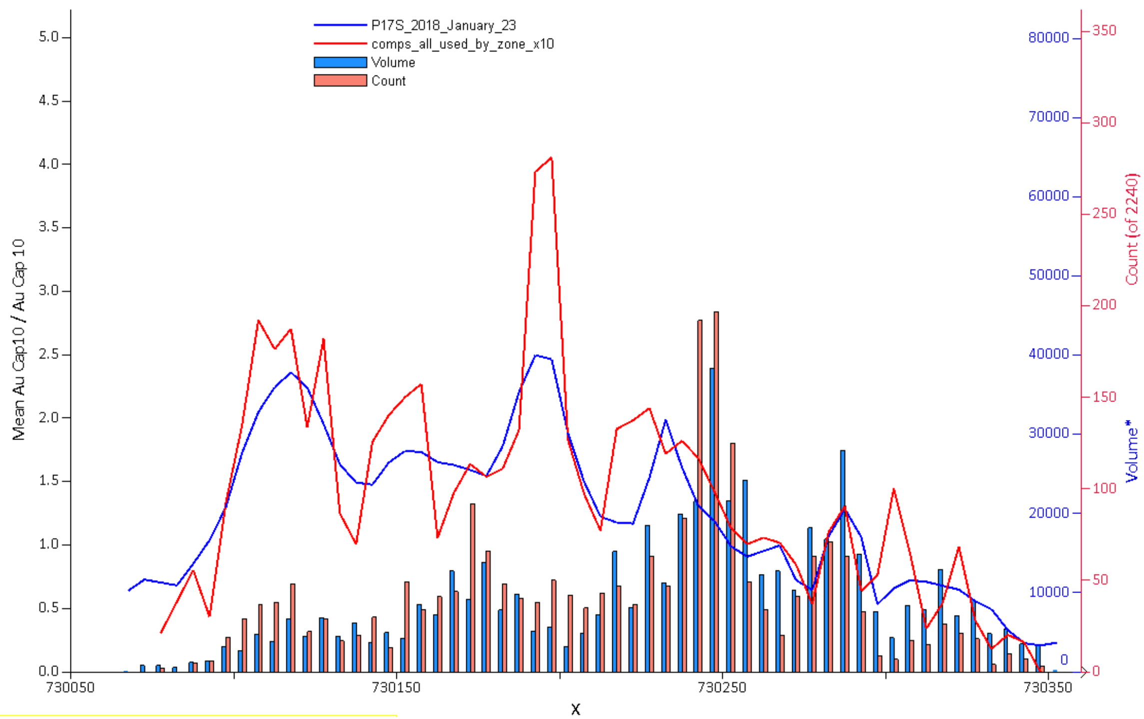


Figure 14.36 Swath Plot by Northing

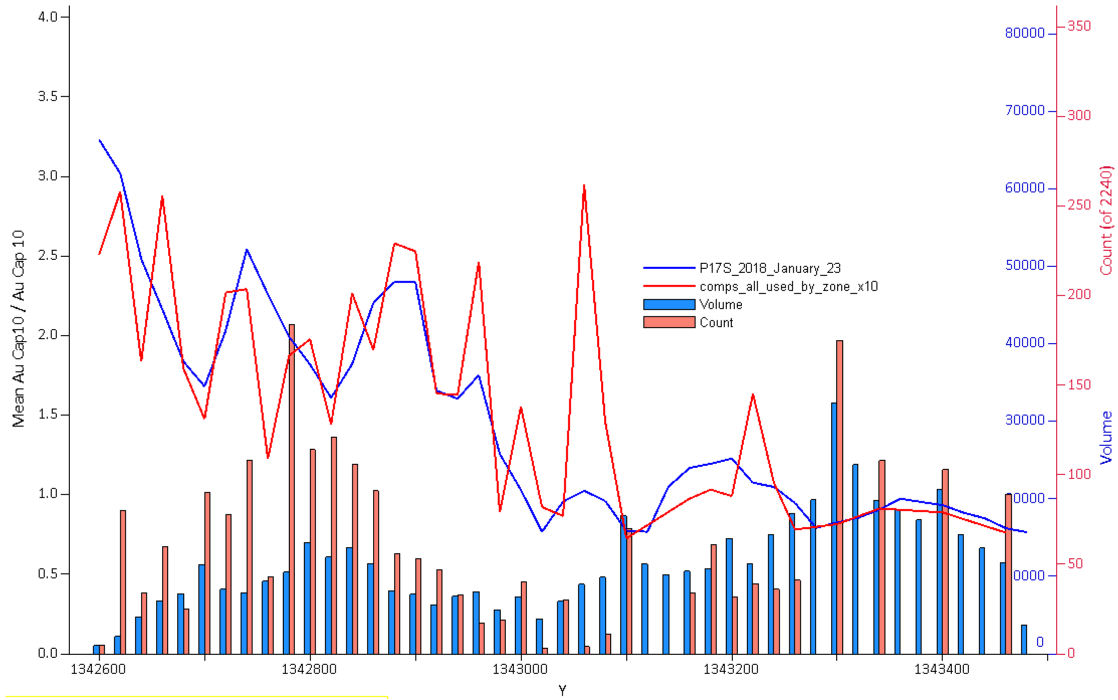
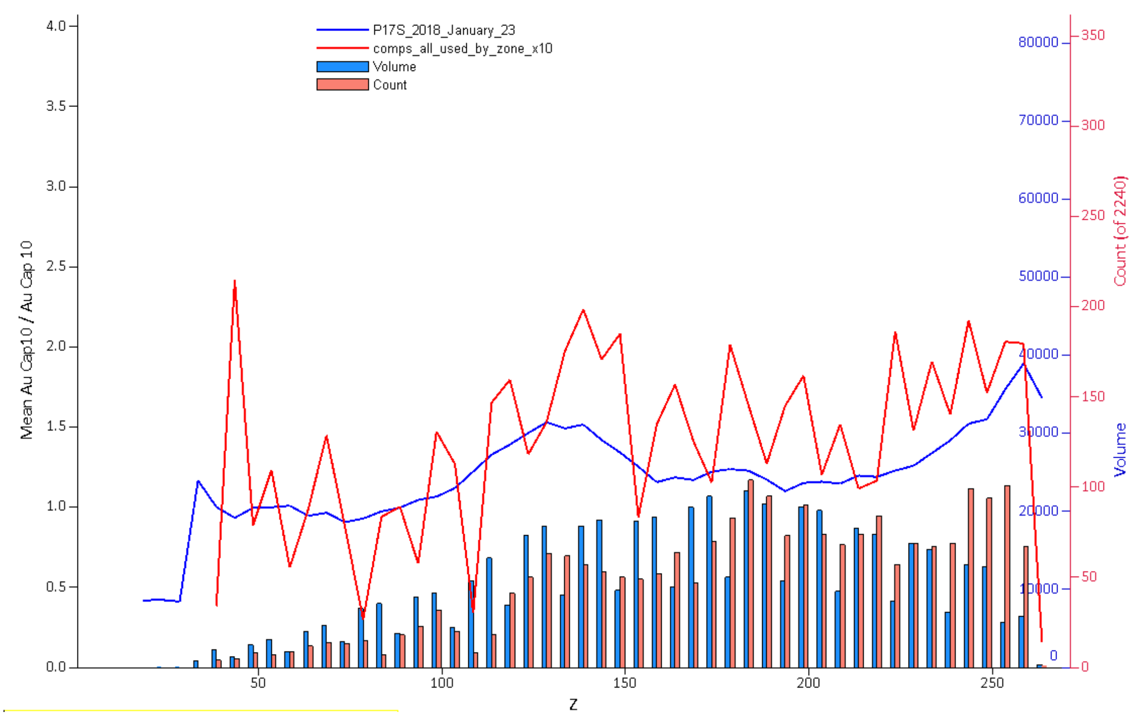


Figure 14.37 Swath Plot by Elevation



14.16 Mineral Resource Estimate

The 2017 Mineral Resource estimate comprises five separate block models, which have been combined into a global resource as shown in Table 14.21s below.

Table 14.21 Summary of the Mineral Resources as of January 5, 2017

Classification		Measured			Indicated			Measured + Indicated			Inferred		
	Cut-off	Tonnage	Grade	Contained	Tonnage	Grade	Contained	Tonnage	Grade	Contained	Tonnage	Grade	Contained
	g/t Au	000 t	g/t Au	koz Au	000 t	g/t Au	koz Au	000 t	g/t Au	koz Au	000 t	g/t Au	koz Au
Oxides	0.20	31,600	0.62	628	75,300	0.53	1,273	106,900	0.55	1,901	20,900	0.40	265
Sulphides	0.2/0.38	9,000	0.90	260	113,600	0.79	2,894	122,600	0.80	3,154	32,400	0.81	842
TOTAL		**40,600**	**0.68**	**888**	**188,900**	**0.69**	**4,167**	**229,400**	**0.69**	**5,055**	**53,300**	**0.65**	**1,107**

Notes:

1. *CIM (2014) definitions were followed for Mineral Resources.*
2. *Mineral Resources are inclusive of Mineral Reserves.*
3. *Oxide resources are made up of the regolith, saprolite, and upper transition layers reported at a cut-off of 0.2 g/t Au.*
4. *Sulphide resources are made up of lower transition and fresh layers reported at 0.2 g/t Au and 0.38 g/t Au respectively.*
5. *Mineral Resources have been constrained within a preliminary pit shell generated in Whittle software.*
6. *Mineral Resources are estimated using a long-term gold price of US$1,400/oz.*
7. *A minimum mining width of approximately 3 m was used.*
8. *Bulk densities vary by material type.*
9. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
10. *Numbers may not add due to rounding.*

The Mineral Resource estimate for the P17S area has an effective date of December 21, 2018. RPA notes that effective date of the deposit as a whole remains January 5, 2017 since the bulk of the Mineral Resources (North, South etc.) have not been updated since that estimate.

RPA is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the 2017 Mineral Resource estimate.

15.0 MINERAL RESERVE ESTIMATES

The Mineral Reserve Estimate is based on the updated 2017 Mineral Resource estimate prepared by RPA (with an effective date of 5 January 2017), which incorporates the oxide material within the previously excluded "Restricted Zones" and takes into account all drilling completed to December 2018 in the P17S deposit.

AMC used the following four separate resource block models.

- North model including the Maga, CFU and P8P9 deposits.
- South model including the P11, Siga E, and Siga W deposits.
- P16 model, a standalone deposit at the southern end of the Project.
- P17S model, a standalone deposit at the south-east end of the Project.

This Technical Report incorporates all available Measured and Indicated Mineral Resources material in the 2017 Mineral Resource Estimate within the oxide, transition, and sulphide horizons. AMC developed mine models by applying modifying factors to the resource block models using Datamine's™ Studio OP software (Datamine). Pit optimizations were conducted on the mine models using Gemcom's Whittle™ 4.X software (Whittle). The pit optimization was then used as basis for producing practical mine designs. The weathered saprolite and upper transition (UT) horizons, which reach a thickness of up to 90 m across the site, can be excavated without the need for prior blasting (free-dig material).

AMC assumed that 70% of the Lower Transition (LT) material would require ripping prior to being loaded onto the haul trucks, while the remaining 30% will have to be blasted.

The sulphide material below the weathered horizons requires drill-and-blast.

15.1 Open Pit Optimization

AMC completed open-pit optimization using Whittle™. The following sections detail the modifying factors, optimization inputs, and results obtained.

15.1.1 Optimization Block Models

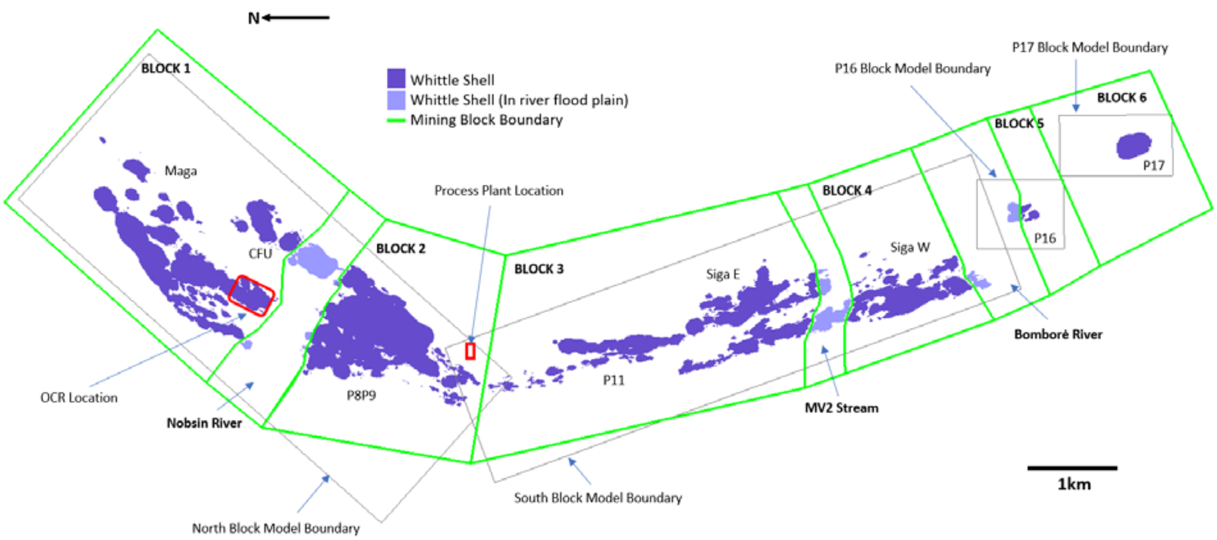
AMC validated the RPA resource block models and divided them into seven mining blocks separated geographically as follows:

- Block 1 – North model: North of the Nobsin River.
- Block 2 – North model: South of the Nobsin River.
- Block 3 – South model: North of the MV2 Stream.

- Block 4 – South Model: South of the MV2 Stream.
- Block 5 – P16 Model.
- Block 6 – P17S Model.
- Restricted Zones - Material contained in the Nobsin, MV2, and Bomboré river flood plains.

Figure 15.1 shows a plan view of the mining block boundaries.

Figure 15.1 Plan View of Mining Block Boundaries



15.1.2 Optimization Inputs

The pit optimization included both oxide and sulphide horizons, with inputs varied depending on the proposed mining method. Inferred Mineral Resources were treated as waste, and only Measured and Indicated Mineral Resources were considered as feed to the processing plant. The optimization inputs are shown in Table 15.1.

Table 15.1 Pit Optimization Inputs

Input	Units	Value
Global Inputs		
• Gold Price	US$/oz Au	$1,250
• Payable Gold	%	99.9%
• Royalty	%	4.0
• Local Development Fund Tax	%	1.0
• Offsite Charges	US$/oz Au	3.50
Mining Inputs		
• Mining Recovery Oxides	%	98
• Mining Recovery Sulphides	%	96-98
• Mining Dilution	%	Variable
• Contractor Mining Base Rate (Oxide)	US$/t mined	1.30
• Contractor Mining Base Rate (Lower Transition)	US$/t mined	1.60
• Contractor Mining Base Rate (Sulphide)	US$/t mined	2.30
• Owners Team and Dewatering Cost	US$/t ore	0.49
• Grade Control	US$/t ore	0.13
• Haulage Rate	US$/t km	0.09
• Backfill Cost (Nobsin River)	US$/t ore	0.35
• Backfill Cost (MV2 Stream)	US$/t ore	0.32
• Backfill Cost (Bombore River)	US$/t ore	0.28
• Revegetation Cost (Restricted Zones)	US$/t ore	0.03
Process Inputs		
• Process Throughput	Mtpa	5.2
• Process Recovery Regolith & Upper Saprolite	%	(Head Grade – 0.068)/Head Grade x 100
• Process Recovery Lower Saprolite	%	(Head Grade – 0.070)/Head Grade x 100
• Process Recovery Upper Transition	%	(Head Grade – 0.073)/Head Grade x 100
• Process Recovery Lower Transition & Sulphide	%	100 - 17.25 x (Head Grade)^-0.241 - 2
• Process Recovery P17S Lower Transition & Sulphide	%	94
• Process Cost (Oxide)	US$/t ore	4.83
• Process Cost (Lower Transition & Sulphide)	US$/t ore	12.43
• Process Sustaining Cost (TSF development)	US$/t ore	1.00
Other Costs		
• General & Administration Cost	US$/t ore	1.92
• Closure Cost	US$/t ore	0.15

Orezone provided the gold price of $1,250/oz and associated offsite charges.

Gold royalties in Burkina Faso are calculated as follows:

- Less than US$ 1,000/oz: 3% of the NSR + 1% Local Development Mining Fund (“FMDL”) tax.
- Equal to or greater than US$ 1,000/oz and less than or equal to US$ 1,300/oz: 4% of the NSR + 1% FMDL tax.
- Greater than US$ 1,300/oz: 5% of the NSR + 1% FMDL tax.

Royalties are applied to the totality of the gold sold.

AMC varied the mine operating cost by mining block by accounting for the average ore and waste haulage distances to the plant; waste rock dumps (WRDs) and TSF. A mining contractor base rate of $1.30/t was estimated for oxide, $1.60 for lower transition and $2.30 for sulphide material. In addition, a $0.09 per t-km adjustment was applied to ore to model the additional ore haulage distance to the plant, and to 80% of oxide waste from Blocks 1 to 4 to simulate the longer hauls for TSF construction requirements. The mine operating costs used in the pit optimization are summarized in Table 15.2 and Table 15.3.

Table 15.2 Pit Optimization – Mine Operating Cost Inputs (Blocks 1 to 6)

	Waste Unit Cost				Ore Unit Cost				
Block	**Waste Unit Cost to WRD (US$/t)**	**Waste Unit Cost to TSF (US$/t)**	**Waste to TSF (%)**	**Total Waste Cost (US$/t)**	**Base Rate Contractor Cost (US$/t)**	**Additional Haulage (US$/t)**	**Owners Team (US$/t)**	**Grade Control (US$/t)**	**Total Ore Cost (US$/t)**
Oxide									
Block 1	1.30	1.64	80	1.57	1.30	0.30	0.49	0.13	**2.22**
Block 2	1.30	1.40	80	1.38	1.30	0.00	0.49	0.13	**1.92**
Block 3	1.30	1.56	80	1.51	1.30	0.21	0.49	0.13	**2.13**
Block 4	1.30	1.77	80	1.67	1.30	0.46	0.49	0.13	**2.38**
Block 5	1.30	-	-	1.30	1.30	0.57	0.49	0.13	**2.49**
Block 6	1.30	-	-	1.30	1.30	0.68	0.49	0.13	**2.60**
LT									
Block 1	1.60	-	-	1.60	1.60	0.30	0.49	0.13	**2.52**
Block 2	1.60	-	-	1.60	1.60	0.00	0.49	0.13	**2.22**
Block 3	1.60	-	-	1.60	1.60	0.21	0.49	0.13	**2.43**
Block 4	1.60	-	-	1.60	1.60	0.46	0.49	0.13	**2.68**
Block 5	1.60	-	-	1.60	1.60	0.57	0.49	0.13	**2.79**
Block 6	1.60	-	-	1.60	1.60	0.68	0.49	0.13	**2.90**
Sulphide									
Block 1	2.30	-	-	2.30	2.30	0.30	0.49	0.13	**3.22**
Block 2	2.30	-	-	2.30	2.30	0.00	0.49	0.13	**2.92**
Block 3	2.30	-	-	2.30	2.30	0.21	0.49	0.13	**3.13**
Block 4	2.30	-	-	2.30	2.30	0.46	0.49	0.13	**3.38**
Block 5	2.30	-	-	2.30	2.30	0.57	0.49	0.13	**3.49**
Block 6	2.30	-	-	2.30	2.30	0.68	0.49	0.13	**3.60**

In the Restricted Zones, owing to a different mining method, mine-operating costs also include waste backfill, compaction and revegetation. Mining and backfilling activities in the restricted zones are limited to the dry season (approximately 133 days). Only oxide material is to be mined in the Restricted Zones.

The operating costs used for pit optimization of restricted zones are shown in Table 15.3.

Table 15.3 Pit Optimization – Mine Operating Cost Inputs (Restricted Zones)

	Waste Unit Cost			Ore Unit Cost						
River	**Base Rate Contractor Cost (US$/t)**	**Re-handle & compacting cost (US$/t)**	**Total Waste Cost (US$/t)**	**Base Rate Contractor Cost (US$/t)**	**Additional Haulage (US$/t)**	**Owners Team (US$/t)**	**Grade Control (US$/t)**	**Backfill Cost (US$/t)**	**Revegetation Cost (US$/t)**	**Total Ore Cost (US$/t)**
Nobsin	1.23	0.33	1.56	1.23	0.23	0.49	0.13	0.35	0.03	**2.46**
MV2	1.23	0.33	1.56	1.23	0.43	0.49	0.13	0.32	0.03	**2.63**
Bombore	1.23	0.33	1.56	1.23	0.59	0.49	0.13	0.28	0.03	**2.76**

Mining Dilution and Recovery

The Bomboré block models have a parent block size of 4 m x 12.5 m x 6 m (X, Y, Z) with a minimum sub-block size of 2 m x 6.25 m x 3 m. The Bomboré mine will be excavated on 6 m benches mined as two 3 m flitches in the free-dig oxide and fresh rock. AMC evaluated that, based on the mineralization geometry, equipment size and grade control procedures, the minimum selective mining unit (SMU) will be 3 m x 6.25 m x 3 m.

AMC applied dilution differently based on mineralization geometry and mining method. A description of the dilution and ore loss method is provided below.

Mining Dilution and Recovery in Blocks 1 to 5 and Restricted Zones

For Mining Blocks 1 to 5 and the Restricted Zones, AMC generated diluted recovered mining block models using the proprietary Drill dilution and ore loss algorithm in Datamine™. This algorithm modified the block models by:

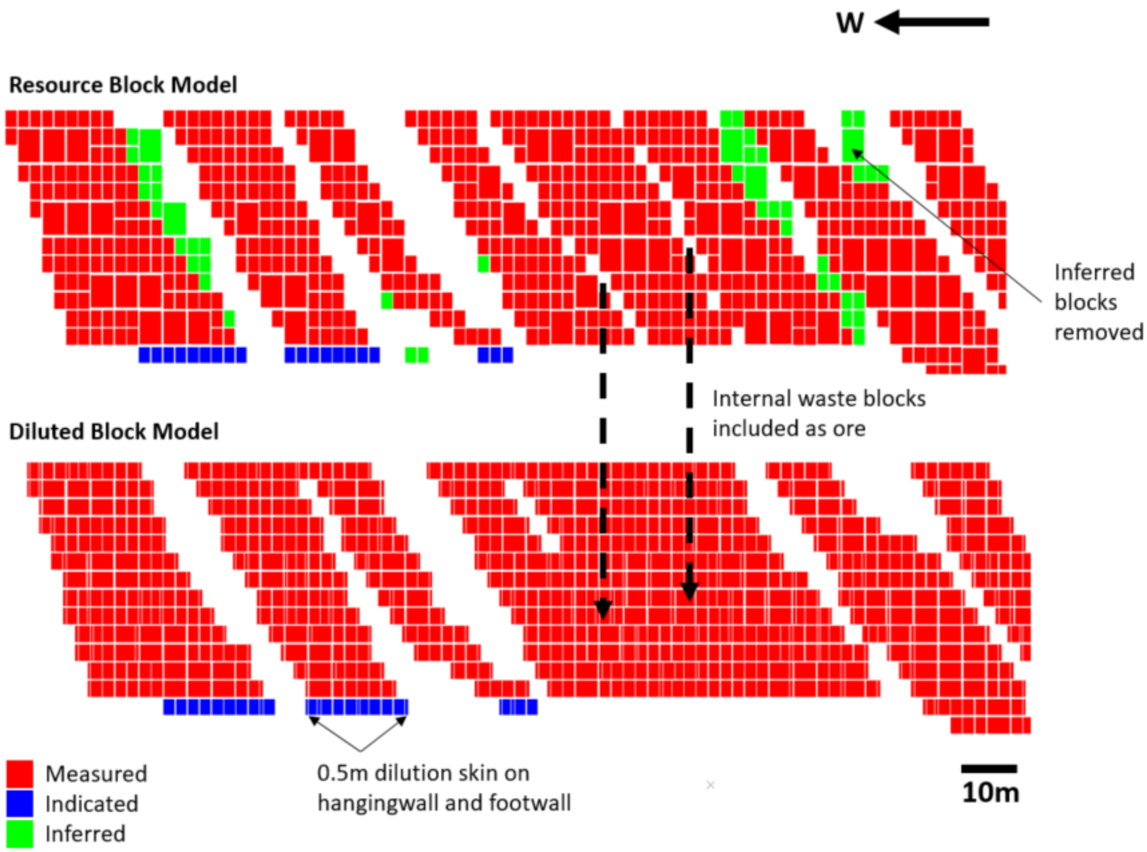
- Adding 0.5 m wide dilution skins to the footwall and hanging wall contacts of the modelled mineralization. This dilution accounts for exceeding dig lines and spalling of waste into delineated ore zones.

- Applying a minimum mining width of 3 m to include internal waste areas between mineralized zones which could not be mined selectively.

Based on the grade tonnage curves and drill assay results outside the mineralized domains, a dilutant grade of 0.1 g/t Au was applied to calculate mining dilution. AMC treated Inferred material as waste in the dilution process with a zero grade.

A typical cross section through the North block model is shown in Figure 15.2 to illustrate the results of diluting the resource block model.

Figure 15.2 Typical Cross Section through Block Model before and after Dilution Process



For sulphide material in Maga, P8P9, and SigaS, AMC assessed the potential blast movement and mixing at ore boundaries applicable to most zones and an additional 4% dilution was applied as a factor in Whittle™ to account for blast-induced dilution.

AMC applied a mine recovery factor of 98% for oxides to represent a 2% operational ore loss during the load, haul, and dump cycle. A 96% mine recovery factor was applied to sulphides in the Whittle™ optimization to account for higher potential ore loss due to blasting.

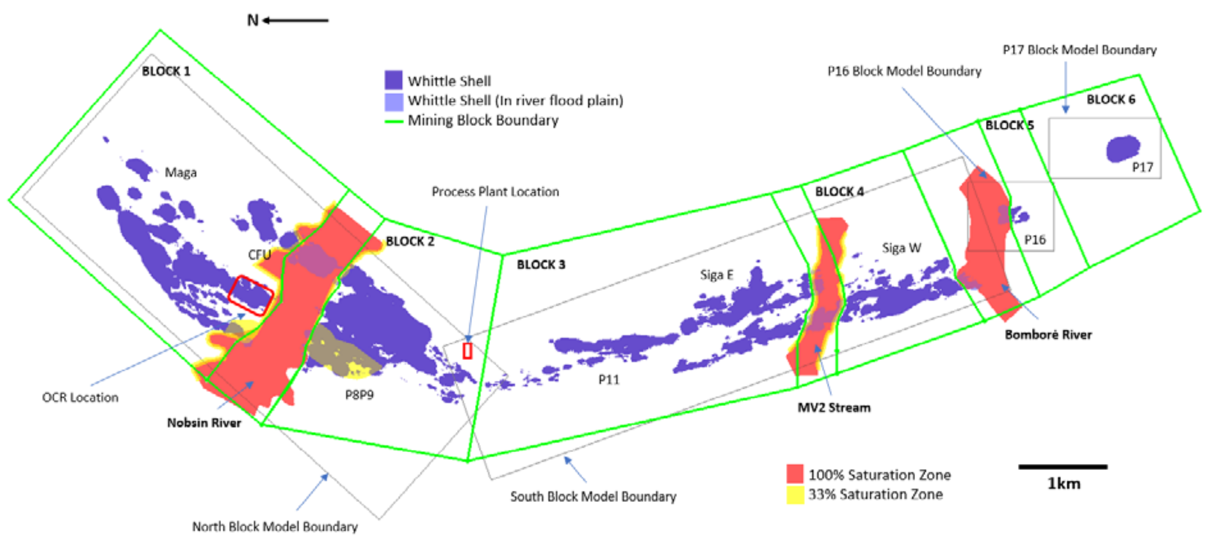
Mining Dilution and Recovery in Block 6

A different dilution method was applied for Block 6 (the P17S deposit) to reflect its specific orebody characteristics. The sulphide mineralization at P17S is wide, shallow dipping in some areas, more continuous and higher grade than the other deposits. For P17S, a dilution factor of 14% and ore loss of 2% was applied in the Whittle™ optimization. The 14% dilution factor is equivalent to 0.3 m footwall & 0.5 m hanging wall dilution on steeply dipping zones, and 0.3 m footwall and 0.9 m hanging wall dilution in shallow dipping zones. The 0.3 m of dilution on the footwall was justified by the visually distinguishable nature of the contact. The 0.9 m of dilution in the hanging wall models, the difficulty associated with mining a shallow dipping contact. Blast hole sampling will provide grade definition on a 3.5 m X x 4 m Y grid. With blast movement tracking and stringent grade control it can be expected that the contact will be lost by no more than 0 – 0.5 times the burden after the blast (0 – 1.75 m, or 0.9 m on average).

Slope Recommendations

Three water saturation zones were identified by Golder where 10%, 33% and 100% of the saprolite thickness is assumed to be saturated. The location of the 33% (yellow) and 100% (red) saturation zones is shown in Figure 15.3. The rest of the material is assumed to be within the 10% saturation zone.

Figure 15.3 Groundwater Saturation Zones



The pits in Blocks 1 to 6 lie within the 10% saturation zone with some areas adjacent to the river basins encroaching into the 33% saturation zone. The 100% saturation zone, which coincides with the river flood plains was treated as a separate domain for optimization (Restricted Zones) with separate slope recommendations.

Slope Recommendations Oxide and Upper Transition

Golder has recommended inter-ramp slope angles for the 10% and 33% saturation zones are shown in Table 15.4.

Table 15.4 Golder Slope Recommendations by Saturation Zone and Slope Height

		10% Saturation Zone			33% Saturation Zone		
Maximum Slope Height (m)	**Bench Height (m)**	**Bench Face Angle (°)**	**Catch Bench Width (m)**	**Inter-Ramp Slope Angle (°)**	**Bench Face Angle (°)**	**Catch Bench Width (m)**	**Inter-Ramp Slope Angle (°)**
6	6	63.4	-	63.4	63.4	-	63.4
12	6	63.4	2	50	63.4	2	50
18	6	63.4	2	50	63.4	2.2	49
24	6	63.4	2	50	63.4	3.4	43
30	6	63.4	2.8	46	63.4	4.7	38
36	6	63.4	3.7	42	63.4	5.6	35
42	6	63.4	4.1	40	63.4	6.6	32
48	6	63.4	4.4	39	63.4	7.8	29
54	6	63.4	4.9	37	63.4	8.3	28
60	6	63.4	5.3	36	63.4	8.8	27
66	6	63.4	5.6	35	63.4	9.3	26
72	6	63.4	5.9	34	63.4	9.9	25
78	6	63.4	6.2	33	63.4	9.9	25
84	6	63.4	6.2	33	63.4	9.9	25
90	6	63.4	6.6	32	63.4	10.5	24
96	6	63.4	6.6	32	63.4	10.5	24

The slope regions and slope angles applied to each region based on anticipated pit depth are shown in Figure 15.4 and Table 15.5.

Figure 15.4 Numbered Slope Regions by Pit Depth (Oxide)

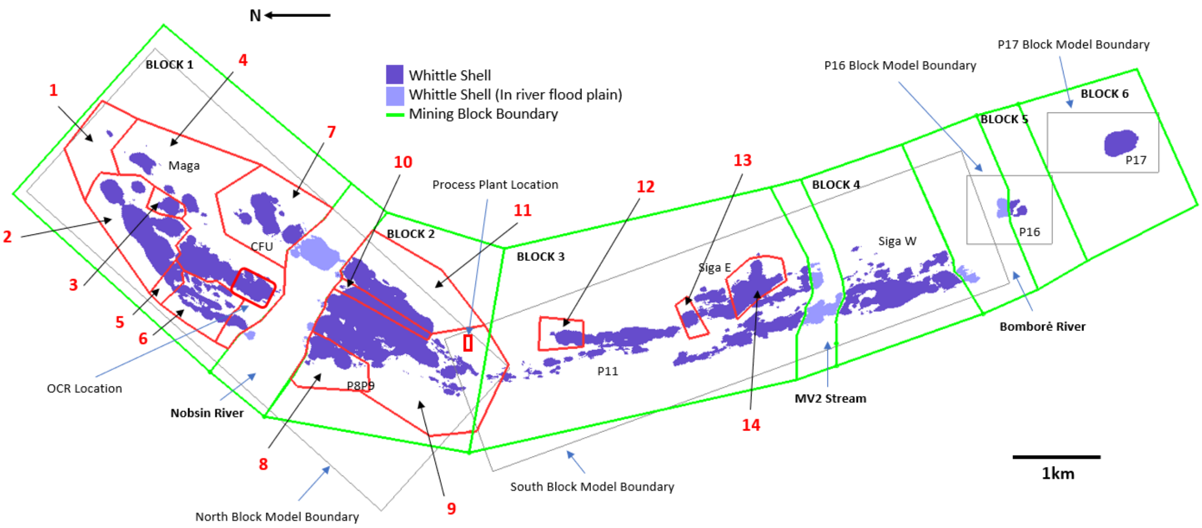


Table 15.5 Oxide Slope Regions and Applied Inter-ramp and Overall Slope Angles

		10% Saturation zone		33% Saturation Zone	
Slope Region	**Max Slope Height (m)**	**Inter-Ramp Slope Angle (°)**	**Overall Slope Angle Including 14 m wide ramp (°)**	**Inter-Ramp Slope Angle (°)**	**Overall Slope Angle Including 14 m wide ramp (°)**
1	20-30	46	35	38	30
2	60-70	34	31	25	23
3	50-60	36	32	27	24
4	40-50	39	33	29	26
5	20-30	46	35	38	30
6	20-30	46	35	38	30
7	50-60	36	32	27	24
8	70-80	33	30	25	23
9	20-30	46	35	38	30
10	40-50	39	33	29	26
11	70-80	33	30	25	23
12	40-50	39	33	29	26
13	40-50	39	33	29	26
14	40-50	39	33	29	26
South Model	20-30	46	35	38	30
P16 Model	20-30	46	35	38	30

Table 15.5 shows the recommended inter-ramp slope angle along with a reduced angle for the inclusion of 14 m wide ramps. AMC applied the inter-ramp slope angle to the Western slopes and the reduced angle to the Eastern slopes to account for requirements to transport the majority of the waste to the Eastern side of the pits.

For the Restricted Zones, AMC based slope angles on the median angle of the Golder recommendations for 33% and 100% saturation zones, which are summarized in Table 15.6.

Table 15.6 Restricted Zones Slope Recommendations (AMC, 2019)

Max Slope Height (m)	Berm Width (m)	BFA (°)	Bench Height (m)	IRA (°)
24	6	63.4	6	33.7
30	7.9	63.4	6	28.8
36	9.1	63.4	6	26.4
42	11.1	63.4	6	23

Slope Recommendations Lower Transition and Sulphide

The geotechnical characteristics of the fresh rock across the project are similar and fresh rock was treated as a single slope domain in the pit optimization. AMC used the inter-ramp angle and bench face angle from Golder to determine the berm width for the transitional and fresh rock domains based on bench heights (vertical distance between berms) of 6 m and 12 m. The slope recommendations are shown in Table 15.7.

Table 15.7 Fresh Rock Slope Recommendations (AMC, 2019)

Location	Maximum Slope Height (m)	Operating Practices	Bench Configuration and Height (m)	Catch Berm Width (m)	Bench Face Angle (°)	Design Inter-Ramp Slope Angle (°)
Transition – All Areas						
All	40	Trim Blasting	Double Bench 2 x 6 m; 12 m between benches	6	63.4	45
All	50	Trim Blasting	Double Bench 2 x 6 m; 12 m between benches	8	63.4	40
Fresh Rock – North Area						
West (Footwall sectors)	Any	Good Quality Trim Blasting	Double Bench 2 x 6 m; 12 m between benches	5	68	50
East (Hanging Wall), North and South (End Wall) Sectors	Any	Good Quality Trim Blasting	Double Bench 2 x 6 m; 12 m between benches	5	68	50
East (Hanging Wall), North and South (End Wall) Sectors	Any	Excellent Quality Pre-Split Blasting	Double Bench 2 x 6 m; 12 m between benches	5	75	55
Fresh Rock – South Area						
West (Footwall sectors)	Any	Good Quality Trim Blasting	Double Bench 2 x 6 m; 12 m between benches	5	60	45
East (Hanging Wall), North and South (End Wall) Sectors	Any	Good Quality Trim Blasting	Double Bench 2 x 6 m; 12 m between benches	5	68	50
East (Hanging Wall), North and South (End Wall) Sectors	Any	Excellent Quality Pre-Split Blasting	Double Bench 2 x 6 m; 12 m between benches	5	75	55

AMC assessed preliminary sulphide shells and reduced the Inter-ramp slope angles to account for likely required ramp access. The resulting overall slope angles (OSA) used in the optimization are shown in Table 15.8.

Table 15.8 Overall Slope Angles (Fresh Rock)

Block	Eastern Slope Angle (°)	Western Slope Angle (°)
Block 1	43	43
Block 2	43	43
Block 3	43	39
Block 4	43	39
Block 5	-	-
Block 6	43	39

** Note: Block 5 has insignificant sulphide tonnage for extension of pit below oxide.*

15.1.3 Optimization Results

AMC ran Whittle™ optimizations at multiple gold prices to produce a series of nested pit shells. For each mining block, AMC selected shells with the aim of maximizing NPV. Table 15.9 summarizes the selected pit shells which were used as guides for pit design.

Table 15.9 Pit Shells Tonnes and Grades

				Oxide and Upper Transition			Lower Transition			Sulphide		
Mining Block	**Revenue Factor**	**Waste Tonnes (Mt)**	**Strip Ratio (W:O)**	**Cut Off Grade (g/t Au)**	**Ore Tonnes (Mt)**	**Gold Grade (g/t Au)**	**Cut Off Grade (g/t Au)**	**Ore Tonnes (Mt)**	**Gold Grade (g/t Au)**	**Cut Off Grade (g/t Au)**	**Ore Tonnes (Mt)**	**Gold Grade (g/t Au)**
OCR	-	7.0	2.32	0.29	3.0	0.46						
Block 1	0.9	44.1	2.69	0.29	13.0	0.66	0.55	0.89	1.22	0.55	2.5	1.35
Block 2	0.9	50.2	1.75	0.29	22.9	0.65	0.54	1.80	1.07	0.54	4.0	1.12
Block 3	0.9	24.6	1.91	0.29	10.2	0.68	0.55	0.80	1.13	0.55	1.9	1.14
Block 4	0.85	17.3	1.55	0.30	5.9	0.68	0.56	0.68	1.08	0.56	4.6	1.11
Block 5	0.95	0.4	1.94	0.31	0.2	0.91						
Block 6	0.95	9.8	7.58	0.31	0.0	1.23	0.47	0.03	1.63	0.47	1.2	1.90
Restricted zones	0.7 to 0.9	7.1	3.30	0.30	2.2	0.75						
Total		**160.6**	**2.12**		**57.4**	**0.66**		**4.20**	**1.12**		**14.2**	**1.23**

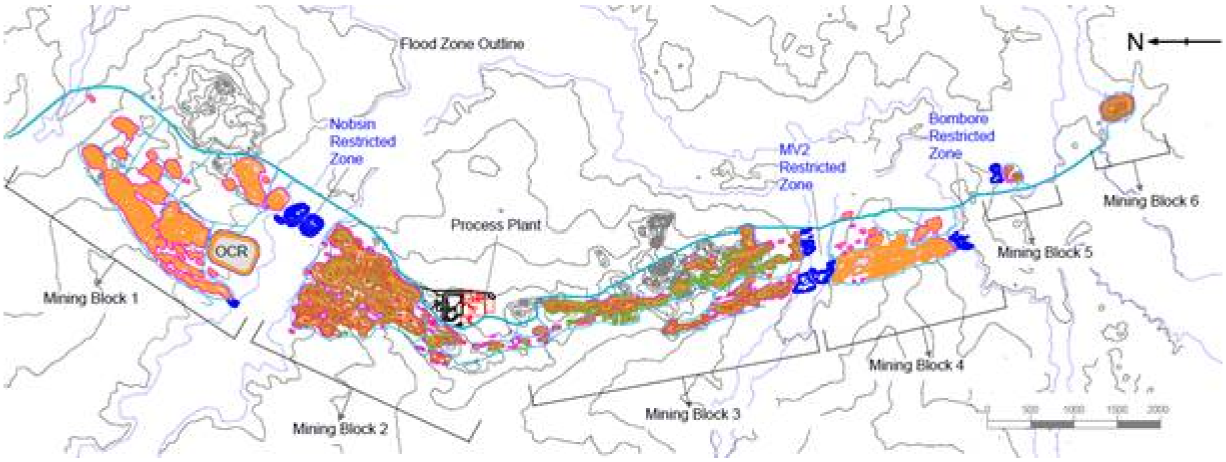
The cut-off grades shown in Table 15.9 represent the break-even cut-off grades at the time of pit optimization.

AMC allowed the optimization to mine both the oxide material (free dig) and fresh rock material (drill and blast) in mining blocks 1 to 6. The Restricted Zones were limited to free-dig material only in this Technical Report.

15.2 Open Pit Design

AMC created an ultimate pit design based on the selected Whittle™ Pit Shells. The design incorporates over 60 separate pits varying in depth from 18 m to 140 m along a 12.2 km long and 3 km wide strike. A plan view of the design is shown in Figure 15.5.

Figure 15.5 Plan View of Ultimate Pit Design



In summary, the design in oxide is based on 6 m benches which will be mined in two 3 m flitches with berm widths varied according to the geotechnical parameters recommended by Golder in Table 15.4. In fresh rock, mining will take place on 6 m benches double stacked to 12 m in the final pit walls. The design incorporates ramps 7 m in width for single lane haulage and 14 m wide for dual lane haulage which predominantly exit on the Eastern sides of the pits.

15.2.1 Mining Dilution and Recovery within Pit Designs

The resulting overall mining dilution and ore losses within pit designs are summarized in Table 15.10. The resulting dilution and ore losses were derived by comparing tonnages and grades against the resources contained within the pit designs only.

Table 15.10 Dilution and losses within pit design

	Oxide & Upper Transition		Lower Transition		Sulphide		Total	
Mining Block	**Dilution**	**Losses**	**Dilution**	**Losses**	**Dilution**	**Losses**	**Dilution**	**Losses**
Block 1	4.7%	3.4%	5.0%	3.4%	8.0%	4.4%	5.3%	3.7%
Block 2	3.5%	3.7%	5.7%	3.9%	8.0%	4.9%	4.3%	4.0%
Block 3	4.6%	4.8%	5.6%	4.6%	8.9%	6.3%	5.4%	5.1%
Block 4	1.8%	5.0%	1.2%	5.3%	3.6%	4.4%	2.4%	4.7%
Block 5	5.7%	1.8%	10.5%	1.1%	11.0%	0.0%	5.9%	1.8%
Block 6	13.7%	2.1%	13.0%	2.3%	12.2%	2.5%	12.2%	2.5%

15.2.2 Break even cut-off grades

The break-even cut-off grades used to estimate Mineral Reserves are based on the following gold recoveries by weathering unit:

- Process recovery regolith & upper saprolite (%) = (Head Grade – 0.078)/Head Grade x 100.
- Process recovery lower saprolite (%) = (Head Grade – 0.080)/Head Grade x 100.
- Process recovery upper transition (%) = (Head Grade – 0.083)/Head Grade x 100.
- Process recovery lower transition in Mining Block 1-5 (%) = 100 – 11.91 x (Head Grade)^-0.38 – 1.05.
- Process recovery sulphide in Mining Block 1-5 (%) = 100 – 17.25 x (Head Grade)^-0.241 – 1.05.
- Process recovery lower transition & sulphide in Mining Block 6 only (%) = 94.95.

Table 15.11 summarizes the updated cut-off grades used for Mineral Reserves estimation.

Table 15.11 Cut-off grades by weathering unit

Description	Block 1	Block 2	Block 3	Block 4	Block 5	Block 6	Restricted Zones
Regolith	0.303	0.300	0.302	0.304	0.317	0.320	0.310-0.317
Upper saprolite	0.303	0.300	0.302	0.304	0.317	0.320	0.310-0.317
Lower saprolite	0.305	0.302	0.304	0.306	0.319	0.322	0.312-0.319
Upper transition	0.308	0.305	0.307	0.309	0.322	0.325	0.315-0.322
Lower transition	0.517	0.507	0.514	0.521	0.524	0.466	Not applicable
Sulphide	0.547	0.537	0.544	0.552	0.555	0.466	Not applicable

15.2.3 Reconciliation between Pit Optimization and Design

A comparison of the Whittle optimization to ultimate pit design tonnages is shown in Table 15.12.

Table 15.12 Pit Shell Recommendation

	Mineralized Tonnes (Mt)	Gold Grade (g/t Au)	Waste Tonnes (Mt)	Strip Ratio (W:O)
Whittle Selected Shells (Including OCR Design)	75.8	0.79	160.6	2.12
Ultimate Pit Design (Including OCR Design)	71.8	0.80	164.4	2.29
Difference	**-4.0**	**0.01**	**3.8**	**0.2**
Difference %	-5.2%	1.8%	2.4%	8.0%

The ultimate pit design has 5.2% less mineralized tonnes and 2.4% more waste due primarily to small satellite pits not taken through to design and smoothing of pit floors in larger shallow areas. Some additional waste was incorporated with the inclusion of additional ramps both in the Eastern and Western final pit walls.

Designs inventory are based on the updated break-even cut-off grades that vary slightly from the ones used for pit optimization and account for dilution assumptions in wide sulphide zones.

Restricted Zones designs contain less mineralized tonnes than the pit optimization. Large pit shells in the Nobsin and MV2 areas that could not be fully mined and backfilled within one dry season were split into two smaller pits, thus resulting in ore loss between the smaller pits.

The mine design evaluation includes 1.7 Mt of mineralized oxide material that remains in the low-grade stockpiles at the end of the mine life; as this material is not processed it has not been included in the Mineral Reserve Estimate.

15.3 Mineral Reserve Estimate

The Bomboré Mineral Reserve Estimate is summarized in Table 15.13. The Mineral Reserve includes diluted recovered Measured and Indicated Resources constrained by the ultimate pit design. The Mineral Reserve Estimate excludes 1.7 Mt of mineralized low-grade oxides that are stockpiled and not included in the current mill feed schedule.

Table 15.13 Summary Mineral Reserve Estimate – June 26, 2019

Classification	**Proven**			**Probable**			**Proven & Probable**		
	Tonnes	Gold Grade	Contained Gold	Tonnes	Gold Grade	Contained Gold	Tonnes	Gold Grade	Contained Gold
	000 t	g/t Au	000 oz Au	000 t	g/t Au	000 oz Au	000 t	g/t Au	000 oz Au
Material type									
Oxides	20,213	0.73	473	32,326	0.66	687	52,539	0.69	1,161
Sulphides	3,241	1.31	136	14,320	1.17	538	17,561	1.19	675
Total	**23,453**	**0.81**	**610**	**46,647**	**0.82**	**1,225**	**70,100**	**0.81**	**1,835**

1. *Oxides include regolith, saprolite and upper transition material.*
2. *Sulphides include lower transition and fresh material.*
3. *Mineral Reserves have been estimated in accordance with the CIM Definition Standards.*
4. *Mineral Reserves are based on cut-off grades that range from 0.300 to 0.325 g/t Au for oxides, and 0.466 to 0.555 g/t Au for sulphides.*
5. *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability.*
6. *There are 1.7Mt of low-grade mineralized oxide material above cut-off grade remaining in the stockpiles that are not included in the Mineral Reserves Estimate.*
7. *Mineral Reserves are estimated at an average long-term gold price of US$ 1,250/troy oz.*
8. *Mineral Reserves are reported effective June 26, 2019.*
9. *Mining recovery factors estimated at 98% for oxides and 96%-100% for sulphides.*
10. *Processing recovery varies by grade, weathering unit and location.*
11. *Rounding of some figures may lead to minor discrepancies in total.*

Table 15.14 presents the Mineral Reserve Estimate by weathering unit.

Table 15.14 Mineral Reserve Estimate by Weathering Unit – June 26, 2019

Classification	Units	Block 1	Block 2	Block 3	Block 4	Block 5	Block 6	Restricted Zones	Total
Summary of Mineral Reserves									
Proven									
Ore	000 t	9,022	13,211	-	-	166	838	215	**23,453**
Gold Grade	g/t Au	0.79	0.74	-	-	0.95	1.99	0.91	**0.81**
Contained Gold	000 oz Au	229	316	-	-	5	54	6	**610**
Probable									
Ore	000 t	9,310	12,671	12,355	10,356	9	428	1,518	**46,647**
Gold Grade	g/t Au	0.81	0.79	0.77	0.90	0.66	1.61	0.71	**0.82**
Contained Gold	000 oz Au	242	320	308	299	0	22	35	**1,225**
Proven & Probable									
Ore	**000 t**	**18,333**	**25,883**	**12,355**	**10,356**	**175**	**1,266**	**1,733**	**70,100**
Gold Grade	**g/t Au**	**0.80**	**0.76**	**0.77**	**0.90**	**0.94**	**1.86**	**0.73**	**0.81**
Contained Gold	**000 oz Au**	**471**	**636**	**308**	**299**	**5**	**76**	**41**	**1,835**
Mineral Reserves by Material Type									
Proven									
Regolith									
Ore	000 t	375	945	-	-	29	1	22	**1,372**
Gold Grade	g/t Au	0.62	0.49	-	-	0.47	1.50	0.44	**0.53**
Contained Gold	000 oz Au	7	15	-	-	0	0	0	**23**
Saprolite									
Ore	000 t	6,353	8,317	-	-	115	5	154	**14,944**
Gold Grade	g/t Au	0.71	0.74	-	-	1.09	1.11	0.88	**0.73**
Contained Gold	000 oz Au	144	197	-	-	4	0	4	**350**
Upper Transition									
Ore	000 t	1,264	2,573	-	-	15	7	39	**3,897**
Gold Grade	g/t Au	0.91	0.74	-	-	0.84	1.55	1.32	**0.80**
Contained Gold	000 oz Au	37	61	-	-	0	0	2	**100**
Lower Transition									
Ore	000 t	550	1,011	-	-	6	16	-	**1,583**
Gold Grade	g/t Au	1.18	0.98	-	-	0.91	1.90	-	**1.06**
Contained Gold	000 oz Au	21	32	-	-	0	1	-	**54**
Sulphide									
Ore	000 t	480	365	-	-	1	810	-	**1,657**
Gold Grade	g/t Au	1.22	0.98	-	-	0.88	2.00	-	**1.55**
Contained Gold	000 oz Au	19	12	-	-	0	52	-	**83**

Classification	Units	Block 1	Block 2	Block 3	Block 4	Block 5	Block 6	Restricted Zones	Total
Probable									
Regolith									
Ore	000 t	452	666	838	424	9	1	77	**2,467**
Gold Grade	g/t Au	0.52	0.50	0.51	0.50	0.66	1.20	0.51	**0.51**
Contained Gold	000 oz Au	8	11	14	7	0	0	1	**40**
Saprolite									
Ore	000 t	5,798	6,488	7,068	3,438	-	2	1,009	**23,803**
Gold Grade	g/t Au	0.63	0.60	0.69	0.70	-	0.86	0.71	**0.65**
Contained Gold	000 oz Au	118	125	156	77	-	0	23	**499**
Upper Transition									
Ore	000 t	758	1,424	1,876	1,559	-	8	431	**6,056**
Gold Grade	g/t Au	0.73	0.77	0.77	0.76	-	1.13	0.74	**0.76**
Contained Gold	000 oz Au	18	35	47	38	-	0	10	**148**
Lower Transition									
Ore	000 t	331	778	823	610	-	10	-	**2,552**
Gold Grade	g/t Au	1.15	1.09	1.09	1.06	-	1.19	-	**1.09**
Contained Gold	000 oz Au	12	27	29	21	-	0	-	**90**
Sulphide									
Ore	000 t	1,971	3,316	1,750	4,325	-	407	-	**11,768**
Gold Grade	g/t Au	1.37	1.15	1.12	1.12	-	1.63	-	**1.19**
Contained Gold	000 oz Au	87	122	63	156	-	21	-	**449**
Subtotals Proven & Probable									
Regolith									
Ore	000 t	827	1,611	838	424	38	2	100	**3,839**
Gold Grade	g/t Au	0.56	0.50	0.51	0.50	0.51	1.32	0.49	**0.51**
Contained Gold	000 oz Au	15	26	14	7	1	0	2	**63**
Saprolite									
Ore	000 t	12,152	14,805	7,068	3,438	115	7	1,163	**38,747**
Gold Grade	g/t Au	0.67	0.68	0.69	0.70	1.09	1.03	0.73	**0.68**
Contained Gold	000 oz Au	262	322	156	77	4	0	27	**849**
Upper Transition									
Ore	000 t	2,021	3,997	1,876	1,559	15	15	471	**9,954**
Gold Grade	g/t Au	0.85	0.75	0.77	0.76	0.84	1.32	0.78	**0.78**
Contained Gold	000 oz Au	55	96	47	38	0	1	12	**248**
Lower Transition									
Ore	000 t	881	1,789	823	610	6	26	-	**4,135**
Gold Grade	g/t Au	1.17	1.03	1.09	1.06	0.91	1.63	-	**1.08**
Contained Gold	000 oz Au	33	59	29	21	0	1	-	**143**

Classification	Units	Block 1	Block 2	Block 3	Block 4	Block 5	Block 6	Restricted Zones	Total
Sulphide									
Ore	000 t	2,451	3,681	1,750	4,325	1	1,216	-	**13,425**
Gold Grade	g/t Au	1.34	1.13	1.12	1.12	0.88	1.88	-	**1.23**
Contained Gold	000 oz Au	106	134	63	156	0	73	-	**531**

1. *Mineral Reserves have been estimated in accordance with the CIM Definition Standards.*
2. *Mineral Reserves are based on cut-off grades that range from 0.300 to 0.325 g/t Au for oxides, and 0.466 to 0.555 g/t Au for sulphides.*
3. *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability.*
4. *There are 1.7Mt of low-grade mineralized oxide material above cut-off grade remaining in the stockpiles that are not included in the Mineral Reserves Estimate*
5. *Mineral Reserves are estimated at an average long-term gold price of US$ 1,250/troy oz.*
6. *Mineral Reserves are reported effective June 26, 2019*
7. *Mining recovery factors estimated at 98% for oxides and 96%-100% for sulphides.*
8. *Processing recovery varies by grade, weathering unit and location.*
9. *Rounding of some figures may lead to minor discrepancies in totals.*

16.0 MINING METHODS

The Bomboré mine will be developed as an open pit operation mining oxide and sulphide material from over 60 separate pits of variable size and depth across a mineralized zone approximately 12.2 km long and 3 km wide. The oxides include the regolith, upper saprolite, lower saprolite, and upper transition weathering units. The oxide material can be readily excavated in situ (free-dig material). The sulphides include the lower transition and fresh rock weathering units, which will require a varying degree of drill and blast prior to being loaded onto trucks.

This Technical Report considers mining of the Restricted Zones which are mining areas located within the floodplains of the Nobsin River, MV2 Stream, and Bomboré River. Mining within the Restricted Zones targets the free-dig oxide material and will only take place during the dry seasons. Mining, backfilling, and rehabilitation of the pits within these areas is to be fully completed prior to re-establishing river flow by the start of the next wet season.

The production schedule is based on the Mineral Reserve Estimate described in Section 15. Mining is planned to span 13.3 years with run-of-mine (ROM) ore delivered to the plant, followed by processing of low-grade stockpiles at the end of the mine life.

The key project life of mine (LOM) highlights are:

- 236.2 Mt total material mined:
  - 71.8 Mt of mineralized material:
  - 70.1 Mt of ore at 0.81 g/t Au mined and processed, including 52.5 Mt of oxides at 0.69 g/t Au and 17.6 Mt of sulphides at 1.19 g/t Au.
  - 1.7 Mt of mineralized low-grade material remaining on stockpiles and not processed at the end of the mine life.

    164.4 Mt waste

    2.34 strip ratio

    13.3-year mine life.
- Pre-production mining of 1.5 years, including excavation of the Off-Channel Reservoir (OCR) for water storage and supply.
- Total production:
  - 54.5 Mt at 0.88 g/t Au ROM ore
  - 15.6 Mt at 0.60 g/t Au low-grade ore re-handled from stockpiles
  - 1.6 Moz Au produced.

Mining of ore and waste will be contracted out with an owner's team responsible for site management, grade control, and mine planning activities. Mining of oxides will be undertaken with 4.5 m^3 hydraulic excavators (i.e. Komatsu PC850) and 30-50 t highway dump trucks. The sulphides will be mined using a separate fleet (i.e. Komatsu PC1250 and 50 t Volvo FMX rigid body trucks) to account for the increased density, abrasion and hardness of the material.

ROM ore will be hauled to the process plant and low-grade material hauled to the low-grade stockpiles.

Separate waste rock dumps (WRD) will be constructed for oxides and sulphides. Testwork is currently underway to determine acid generating and metal leaching potential of fresh rock, however results to date do not indicate that metal leaching or acid rock drainage (ARD) control will be an issue. If, however if final testwork indicates that there are any such zones, they will be dealt with then and within an updated closure plan. Approximately 53% of the oxide waste produced will be used in the construction of the Tailings Storage Facility (TSF) with the remainder being hauled to the oxide WRDs and four environmental barriers.

The project area generally consists of flat terrain crossed by wide, shallow river flood plains, which flood during the wet season. Low hills are occasionally encountered along the Eastern flank. The hills which border the pits are composed of weathered saprolite material with laterite caps typically 0.5 m thick.

The climate consists of wet and dry seasons with rainfall generally occurring in the four months between June and September. On average, 800 mm of rainfall occurs each year in daily short bursts of heavy rainfall during the wet season. The impacts of the wet season on the mining operation have been taken into consideration in the mine schedule.

16.1 Mine Planning

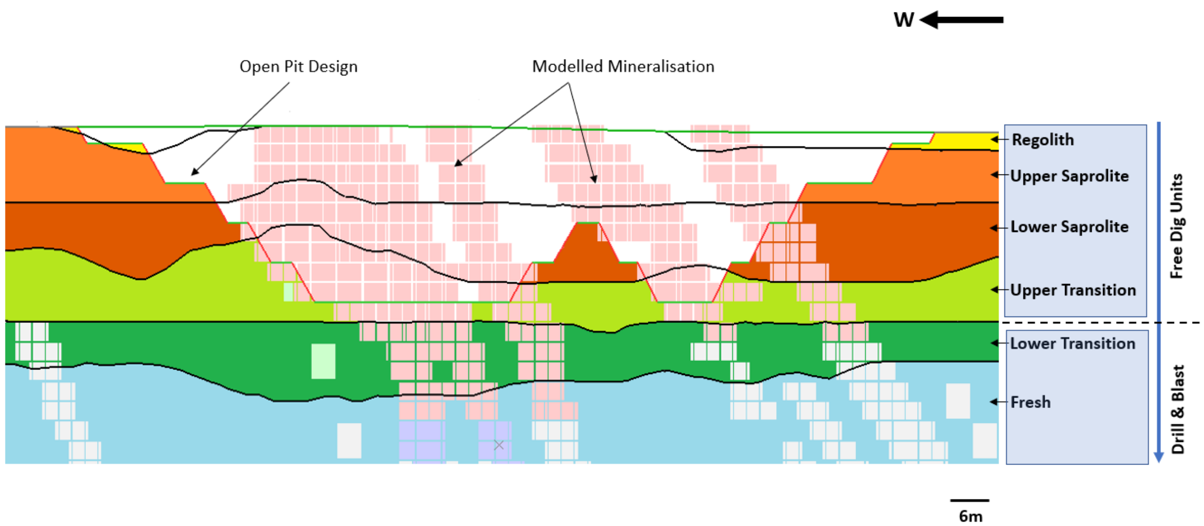
16.1.1 Material Types

The Bomboré block models are divided into weathering horizons based on geological logging and multi-element XRF analyses. The horizons take the form of smoothly undulating layers and include the following materials:

- Surface soil.
- Regolith.
- Upper saprolite.
- Lower saprolite.
- Upper transition.
- Lower Transition (no dig).
- Fresh Rock (no dig).

The Mineral Reserve Estimate is composed of ore from all weathering horizons. A typical cross section through a pit showing the different weathering horizons is shown in Figure 16.1.

Figure 16.1 Typical Section of Pit Design through Weathering Horizons



Mineralization continues through all weathering horizons and each material may be classified as ore or waste based on the cut-off grade. The upper saprolite and lower saprolite units account for approximately 55% of the mill feed, the fresh rock accounts for 19%, the upper transition for 14%, the lower transition for 6% and the regolith accounts for the remaining 5%.

Weathering profiles are generally deeper in the Northern areas of the project, gradually shallowing towards the South. This leads to thicker deposits of oxide material in the North and shallower occurrences of sulphide material in the South.

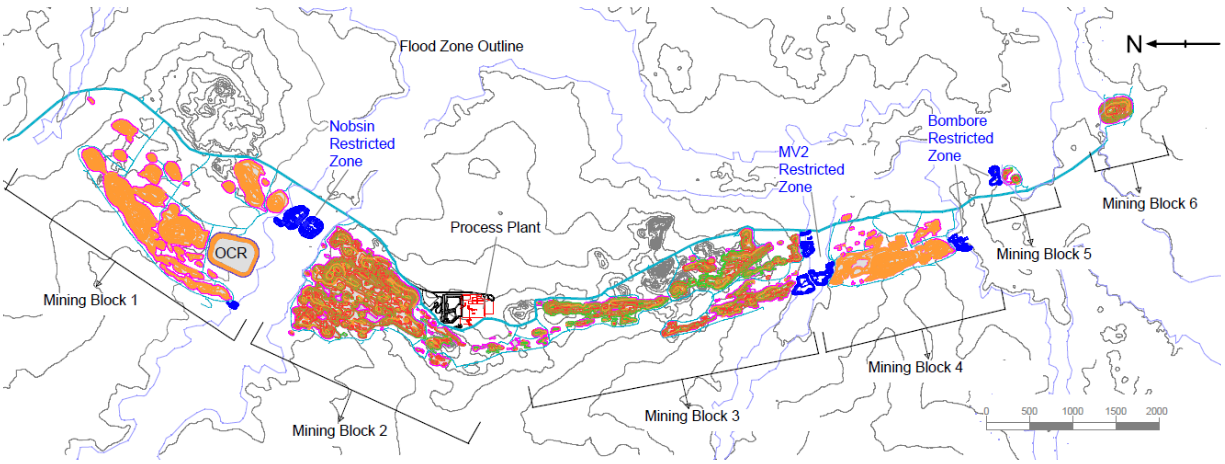
Surface soil, approximately 20 cm deep will be stripped during the dry season using tracked dozers and stored around the perimeters of the WRDs and the TSF for use in reclamation. The remaining material is classified as follows:

- High-grade ore – Hauled to the ROM pad or the process plant.
- Low-grade ore – Hauled to low-grade stockpiles.
- Waste – Hauled to the TSF for construction, environmental barriers, and WRDs.

16.1.2 Open Pit Design

AMC created an ultimate pit design using the selected Whittle pit shells as a guide. The design incorporates over 60 separate pits varying in depth from 18 m to 140 m along a 12.2 km strike. A plan view of the pit designs is shown in Figure 16.2.

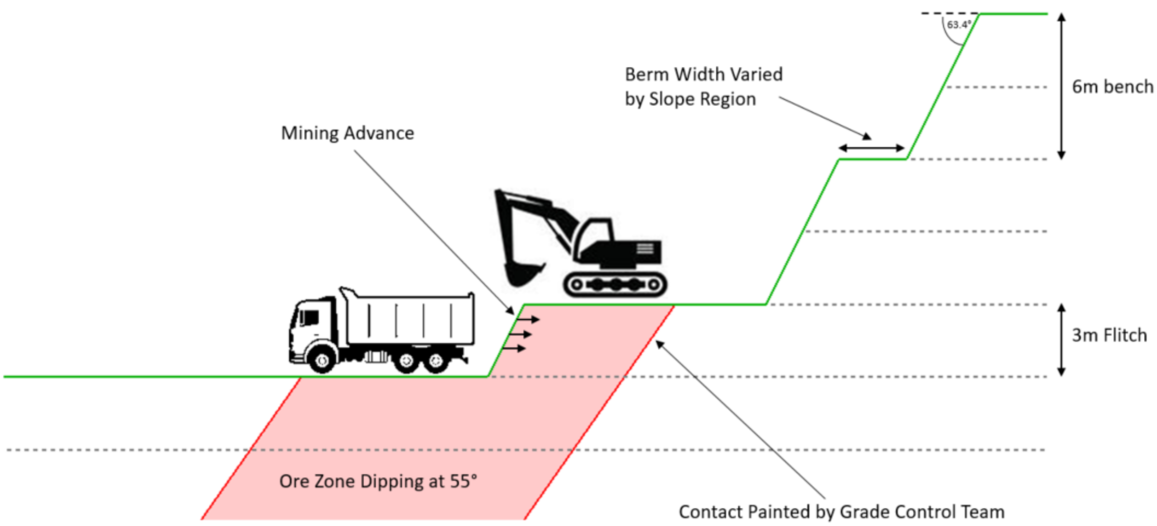
Figure 16.2 Plan View of Ultimate Pit Design



The pit design is based on 6 m benches for oxides and double stack 2 x 6 m bench height for sulphides. Starter pits containing high-grade oxide ore were designed in Maga, P8P9, and SigaS.

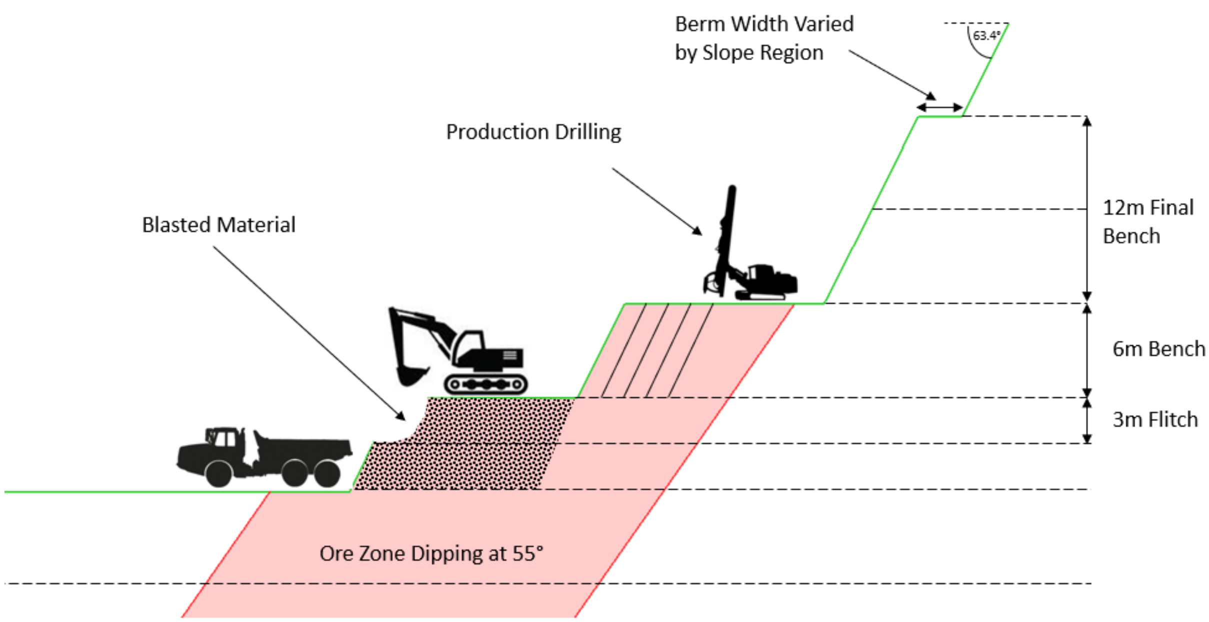
In the oxide horizons, ore will be excavated in 3 m flitches to increase mining selectivity. A typical layout of an oxide mining area annotated with design parameters is shown in Figure 16.3.

Figure 16.3 Cross Section of a Typical Bench Layout (Oxides)



Free-dig operations will continue down to the lower transition horizon as far as practically achievable by the mining fleet making use of mechanical ripping where required. As mining progresses into the sulphide horizon, drill & blast operations will be required, and a separate larger mining fleet used. A typical layout of a sulphide mining area is shown in Figure 16.4.

Figure 16.4 Cross Section of Typical Bench Layout (Sulphides)



Slope Parameters

AMC was provided with oxide slope recommendations by Golder (Golder, 2018). These slope parameters are presented in Table 15.4. AMC divided the project into slope regions for optimization (Figure 15.4 and Table 15.5) and carried these slope regions through to design. In order to maintain the recommended inter-ramp slope angles, AMC varied the berm width for oxide material by slope region as shown in Table 16.1.

Table 16.1 Berm Widths Varied by Slope Region (oxides)

		10% Saturation zone		33% Saturation Zone	
Oxide Slope Region	**Max Slope Height (m)**	**Inter-Ramp Slope Angle (°)**	**Berm Width Applied (m)**	**Inter-Ramp Slope Angle (°)**	**Berm Width Applied (m)**
1	20-30	46	2.8	38	4.7
2	60-70	34	5.9	25	9.9
3	50-60	36	5.3	27	8.8
4	40-50	39	4.4	29	7.8
5	20-30	46	2.8	38	4.7
6	20-30	46	2.8	38	4.7
7	50-60	36	5.3	27	8.8
8	70-80	33	6.2	25	9.9
9	20-30	46	2.8	38	4.7
10	40-50	39	4.4	29	7.8
11	70-80	33	6.2	25	9.9
12	40-50	39	4.4	29	7.8
13	40-50	39	4.4	29	7.8
14	40-50	39	4.4	29	7.8
South Model	20-30	46	2.8	38	4.7
P16 Model	20-30	46	2.8	38	4.7

The slope design for the sulphides are designed based on the geotechnical recommendations summarized in Table 15.7

All benches were designed at 63.4˚ bench face angles as per Golder’s recommendations.

The pits in the restricted zones were designed based on 6 m benches and berm widths that vary by pit depth, as shown in Table 15.6. A minimum buffer distance of 20 m from crest-to-crest between the restricted and non-restricted pits was included during pit design.

Mining Footprint

The ultimate pit design generated by AMC has a surface footprint of approximately 446 ha; a breakdown of surface area by mining block is presented in Table 16.2.

Table 16.2 Surface Footprint by Mining Block

Block Model	Mining Block	Surface Footprint (ha)
North	Block 1	136
North	Block 2	126
South	Block 3	98
South	Block 4	46
P16	Block 5	3
P17S	Block 6	9
North / South / P16	Restricted Zones	27
Total		**446**

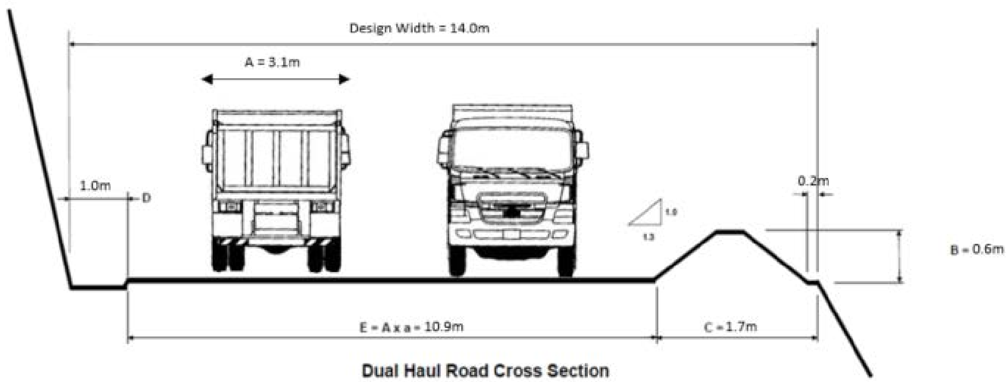
16.1.3 In Pit Ramp Design

AMC designed in pit ramps at a maximum 10% gradient. In pit ramps were designed with widths of 7 m for single lane haulage and 14 m for dual lane haulage. A cross section of a typical haul road design is shown in Figure 16.5.

Figure 16.5 Dual and Single in-pit Ramp Design

Dual Lane Number of Truck Widths "a" = 3.5

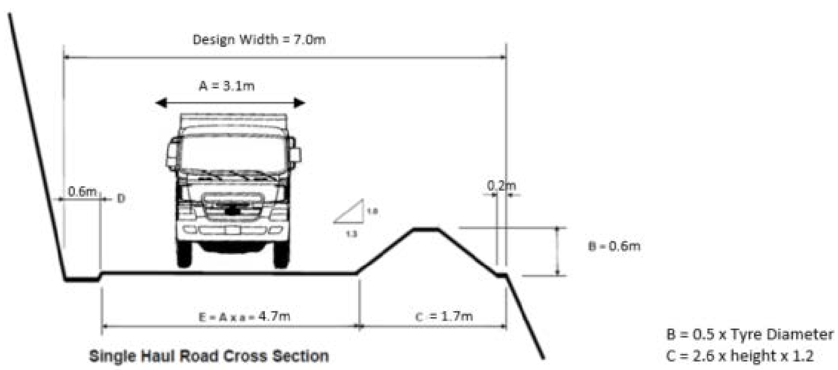
Truck	A Operating Width (m)	Tyre Diameter (m)	B Bund Height (m)	C Bund width (m)	D Drain width (m)	E Pavement Width (m)	F Total (m)
35t Dump Truck	3.1	1.1	0.6	1.7	1.0	10.9	13.6



Dual Haul Road Cross Section

Single Lane Number of Truck Widths "a" = 1.5

Truck	A Operating Width (m)	Tyre Diameter (m)	B Bund Height (m)	C Bund width (m)	D Drain width (m)	E Pavement Width (m)	F Total (m)
35t Dump Truck	3.1	1.1	0.6	1.7	0.6	4.7	7.0



Single Haul Road Cross Section

Multiple truck models were put forward during the mining contract request for quote process; truck size ranges between 20 t and 35 t for the oxide fleet. The design criteria for sulphide pits was based on 50 t rigid body trucks. All the trucks have an operating width of between 2.5 m and 3.5 m. Dual lane 14 m haul roads will allow trucks to operate safely and provide some flexibility should larger trucks be considered in the future.

Due to the free-dig nature of the oxide units, the exact location of ramps and pit exits of oxide-only pits may be optimized during detailed mine planning. Ramp routes may also be modified during operations when required to suit changing mining conditions. Many of the smaller satellite pits may be mined using a temporary ramp along the orebody strike. Where possible, AMC designed the haul roads along the Eastern pit slopes to reduce haulage distances.

16.1.4 Surface Haul Roads

The mine will be served by a trunk road approximately 14.7 km in length running along the eastern side of the pits. The trunk road will consist of a haul road and parallel service road between the north camp and the process plant. The parallel service road will not continue south beyond the process plant. The trunk road will have one permanent bridge crossing the Nobsin River to allow for continuous haulage from the north of the Project during the wet season.

Surface roads will connect the trunk road to the pit ramp exits and will be 14 m wide where dual lane haulage is required (approximately 90%) and 7 m wide for single lane. Construction material for the surface roads will be sourced from mine waste and on-site borrow pits. Should more substantial quantities of aggregate sheeting be required, Orezone has identified a granitic quarry nearby. During detailed design, AMC recommends that an assessment of all potential local borrow sources is conducted with the aim of minimizing haulage distance while maintaining constant road material supply throughout the mine life.

During the wet season, when heavy rainfall occurs, haulage will be stopped until the rain eases. This will prevent damage and rutting to road surfaces and ensure that road maintenance is kept to a minimum.

AMC recommends that during detailed design, surface haul roads and pit exit points be optimized to reduce construction requirements. The free-dig nature of the oxide mining operations and relatively flat topography mean that changes to road alignments can be easily built into future operations. Where smaller excavations are planned, narrower and/or more temporary haulage routes may be established.

The mining contractor will maintain haul roads using their ancillary fleet to focus on active routes. The contractor ancillary fleet will consist of small excavators, motor graders, dozers, haul trucks and compactors.

16.1.5 Mine Sequence in Restricted Zones

The Restricted Zone pits contain 1.7 Mt of Proven and Probable mineralized material at a gold grade of 0.73 g/t Au.

Mining activities in the Restricted Zone pits will take place during a single dry season of 19 weeks between November to March. AMC has recommended that the Restricted Zone pits are mined after the nearby pits from the other mining blocks are completed in order to minimize any risks of hydraulic connectivity and wall failure between excavations. When mining the Restricted Zone pits, no personnel should be permitted to mine in the nearby pit within a predefined exclusion zone.

The simplified mine sequence for the Restricted Zone pits are:

- Site preparation.
- Load and haul 6 m bench and free digging in 3 m flitches of oxide units only.
- Stockpile ore if needed to maintain high productivity within the pits.
- Set up of temporary waste dumps close to the pit crests for backfilling.
- Backfilling by following the original weathering unit sequence. Additional backfill material requirements to be sourced from nearby pits if required.
- Compacting.
- Revegetating and site closure.

There are three pits in the Nobsin River, three in the MV2 stream, and two pits in the Bomboré River. The majority of the pits can be mined by sharing the oxide fleet used in the main areas. However, the two large pits in the Nobsin River will require excavators with higher productive capacity and larger bucket size in order to complete the excavation and backfilling operations within a single dry season. An additional contractor will be required to mine these two larger pits.

The estimated duration to mine and backfill the Restricted Zone pits is summarized in the following table.

Table 16.3 Estimated Operation Duration Restricted Zone Pits

Mining Block	Mining Area	Load and haul duration (days)	Backfill/Compacting duration (days)
Nobsin	50	25	27
Nobsin	51	59	74
Nobsin	52	60	71
MV2	53	56	61
MV2	54	52	61
MV2	55	53	55
Bomboré	56	44	51
Bomboré	57	61	67

16.1.6 Drill and Blast in Sulphide Material

AMC used the Julius Kruttschnitt Mineral Research Centre (JKMRC) Blast Fragmentation Model to complete the blast fragmentation analysis and target a fragmentation appropriate for the size of the loading equipment buckets and crusher open size setting (OSS):

- Ore fragmentation top size: C120 jaw crusher top size (OSS) X 80% ~ 1.0 m.
- Ore fragmentation P80: top size X 60% ~ 0.6 m.
- Waste fragmentation top size: PC1250 excavator bucket width X 50% ~ 1.2 m.

The Bomboré deposit consists of a highly weathered oxide material near the surface that can be excavated without drill and blast (free-dig by hydraulic excavators). It is assumed that only 30% of the underlying lower-transition material requires drill and blast, and 70% can be loaded following dozer ripping. The sulphide fresh rock is classified as medium strong to strong and requires drill and blast.

All sulphide pits are assumed to be wet, hence emulsion is recommended as the primary bulk explosive type. Emulsion is suitable for wet blasting conditions and the medium to high strength fresh rock. Production blast holes will be drilled on a 6.0 m bench using 127 mm diameter holes, and 1.0 m of subdrill. Blast pattern designs for fresh and transition rock types are presented in Table 16.4.

Atlas Copco D65 FlexiROC down-the-hole production drills are recommended by AMC.

Table 16.4 Blast Pattern Design

Description	Units	Sulphide (Fresh) Ore	Sulphide (Fresh) Waste	Lower Transition (Saprock) Ore	Lower Transition (Saprock) Waste
Bench height	m	6.0	6.0	6.0	6.0
Hole diameter	mm	127	127	127	127
Burden	m	3.4	3.9	6.5	6.8
Spacing	m	3.9	4.5	7.5	7.9
Burden/stiffness ratio		1.76	1.54	0.92	0.88
Spacing/burden ratio		1.15	1.15	1.15	1.16
Charge length	m	5.2	5.2	4	3
Stemming height	m	1.8	1.8	3	4
Powder factor	Kg/m^3	1.06	0.80	0.22	0.15
Charge weight/hole	Kg/hole	84	84	65	49

The ore zones vary in both thickness and dip. Where the ore zone is thick and shallow dipping, hangingwall and footwall dilution will need to be controlled. In steeply dipping zones, the vertical distance to the top of the ore will vary within a blast. Blast boundaries may be cut short, if required, to preserve the ore-waste contact.

Blasting along strike and monitoring of blast movement is recommended. Modelling of blast movement will allow the ore-waste contact post-blast position to be predicted.

Blasting against final walls is likely to require specialist techniques including trim blasting and pre-split of the final pit walls. Trim blasting is expected to be required to buffer final walls against production blasts. Pre-splitting will utilize small 110 mm vertical diameter holes and will need to be 12 m long (double bench) on a 1.0 to 1.5 m spacing. Decoupled or decked explosive charges will be used for pre-split blasting with an uncharged length at the top of the holes of 1.0 to 1.5 m. Pre-split holes will be un-stemmed and have a powder factor ranging between 0.5 – 0.6 kg/m^3.

16.1.7 Grade Control

The Owner's Team will be responsible for grade control activities, mine planning, and supervision and management of the contractor activities on site.

Grade control on site will consist of three key components:

- Advance Reverse Circulation (RC) infill drilling on a 12.5 m grid spacing perpendicular to the orebody strike ensuring one drill hole intersection per SMU. Drill holes will be sampled at 1 m intervals and assayed using the LeachWELL method with 10% QA/QC samples sent for fire assay.

- Multi-element Niton XRF analyses of drill samples and flitch cross sections. Using various indicator elements for example Ca and Fe, Orezone geologists can clearly identify weathering horizons and separate host lithologies.

- Mark-up and spotting. Throughout the deposit, geological contacts dip at predominantly 55° to the East. When a new mining flitch is opened, a cut perpendicular to the strike of the mineralization will provide a face onto which the ore contact can be painted based on the drilling and XRF analyses. To minimize dilution, a spotter at the working face will ensure that dig lines are not exceeded.

AMC recommends that grade control test areas be drilled prior to commencement of operations to refine and calibrate procedures. The construction of the OCR should be used as an opportunity for test drilling, sampling, grade control modelling, and spotting so that refinements can be made prior to commencement of operations.

16.1.8 Pit Dewatering

Pit dewatering quantities will vary from year to year based on the number of and configuration of active pits, with a low of 13,000 m^3 in the first year to a high of 2,226,000 m^3 in 2030. On average, approximately 1 Mm^3 of water will be pumped out of the pits on an annual basis. These quantities are based on historical rainfall data and estimated groundwater inflow volumes.

Ground water estimates were based on the Golder Hydrogeological study completed in April 2013 as part of the Feasibility Level Pit Slope Design Report. Based on the Golder study, there should be minimal inflows of ground water until the pits are excavated below the saprolite/fresh rock boundary, with the majority of inflow occurring at this boundary. As such, groundwater inflows were only considered in pits that excavate below this boundary.

Generally, the mine will have a maximum of 10 dewatering pumps in use at any one time with one spare pump on site. The pump selected was a Sykes XH100 (or equivalent), which is a diesel powered, high head, centrifugal pump. The pumps have sufficient head to pump from the deepest pit with each pump capable of pumping up to 80 L/second of water at 135 m of head; however, actual performance will depend on both elevation gain and line losses for each pit, and will pump approximately 55 L/second on average. Storm data were not considered when sizing the pumps and it was assumed that should a significant storm occur (24-hour 100 year return period) operations would be curtailed while pit dewatering is completed.

Water pumped from the pits will be collected and managed in collection ponds, sediment ponds and the OCR. Water will be pumped to the nearest collection pond on surface with pipe runs varying from 500 m to 4,000 m depending on the location and depth of pit. In pits where mining spans across the wet season, bench floors must be mined at an appropriate slope angle to ensure that water flows effectively to temporary sumps. The ultimate pit design sits predominantly within saprolite material and build-up of water on mining floors can have the potential to cause muddy conditions which can be detrimental to safety and productivity if not managed effectively.

16.1.9 Dust Control

Dust will be generated primarily at digging faces and along sections of haul roads in active use. The mining contractor will be required to have a minimum of two water trucks with at least 25 m^3 capacity in constant service to control dust on site.

AMC recommends testing and costing locally available dust suppression agents during construction for use during mining. A cost benefit analysis will be required to trade-off the cost of additional water versus the use of dust suppression agents.

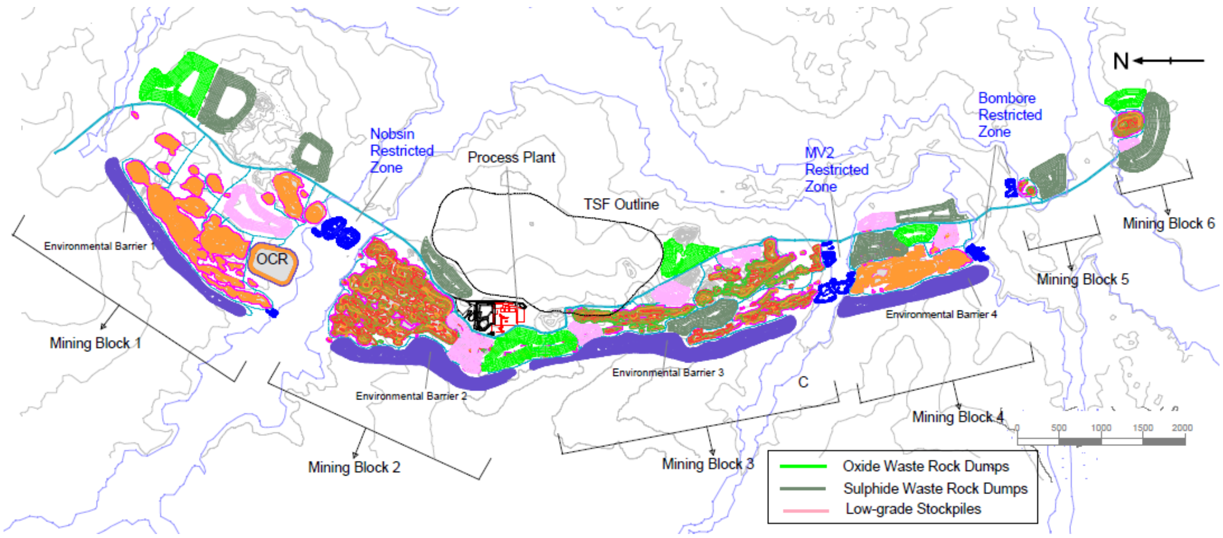
16.2 Waste Dump and Stockpile Design

AMC designed separate waste rock dumps for oxides and sulphides. All of the sulphide waste will be sent to the nearest sulphide WRDs. Testwork is currently underway to determine acid generating and metal leaching potential of fresh rock, but results to date indicate no ARD potential. If further testwork indicates that there is a metal leaching issue or material that may require ARD control from any ongoing or future testwork, it will be dealt with then and within an updated closure plan.

Approximately 53% of oxide waste generated by mining will be hauled to the TSF and used in the phased construction of the TSF. The mining contractor will be responsible for hauling to the TSF footprint. The same mining contractor will likely be responsible to spread and compact the material for construction however a separate independent contractor may be engaged. The remaining oxide waste will be used to build environmental barriers to shield local communities from mining activities or hauled to the other oxide WRDs.

AMC generated designs for multiple low-grade stockpiles, WRDs, and environmental barriers as shown in Figure 16.6.

Figure 16.6 Plan View of WRDs, Low Grade Stockpiles, and Environmental Barriers



Some WRDs and stockpiles have been designed on top of pits due to limited room. The mine schedule accounts for mining and backfilling the pits underneath prior to building the WRDs and stockpiles.

AMC designed the waste dumps and stockpiles based on movement from the selected first pass schedule. The detailed final schedule shows a slightly different distribution of waste between the TSF and the various waste dumps. The stockpile movement is also different by material and mining block. Overall dump and stockpile capacities are adequate. The discrepancies are small and can be managed during detailed design stage.

16.2.1 Design Parameters

As part of the Golder report, Golder made recommendations for WRD and stockpile design parameters. WRDs and stockpiles designed by AMC do not exceed the maximum crest elevations specified in the Golder report (Golder, 2015). AMC recommended to use the same parameters for sulphide WRDs. The design criteria used are shown in Table 16.5.

Table 16.5 Waste Dump and Stockpile Design Criteria

Parameter	Units	Value
Lift Height	m	5
Face Angle	degrees	20.3
Berm Width	m	7.5
Ramp Width	m	14
Ramp Gradient	%	10

AMC has assumed a swell factor of 30% for oxides and 40% for sulphides when converting in-situ volumes to bulk and placed volumes for evaluating the capacity of stockpiles and WRDs. Due to the low density of the oxide material and potential compaction from the mining fleet, a lower swell factor for oxides may be achieved during operation and should be quantified by measurements.

Topsoil stripped within the footprint of WRDs and stockpiles will be stored around their perimeters for rehabilitation. In the case of the environmental barriers, rehabilitation will proceed as soon as they are built, with the topsoil placed onto the graded landforms.

16.2.2 Low-grade Stockpiles

The low-grade stockpiles will be built over the first ten years of production to reach a combined maximum size of 8.9 Mt. The purpose of the stockpiles is to increase initial head grade in the early years of production. Oxide low-grade material will be re-handled and processed at the end of mine life, while sulphide low-grade material will be re-handled as required to meet the process feed requirements. The maximum design capacity of the low-grade stockpiles in Million Loose Cubic Meters (MLCM) is summarized in Table 16.6.

Table 16.6 Low-grade Stockpile Design Capacities

Low-grade Stockpile	Maximum Crest Elevation (m)	Overall Height (m)	Volume (MLCM)
B1 Low-grade Stockpile	300	25	4.2
B2 Low-grade Stockpile	330	35	2.0
B3-1/B3-2/B3-3 Low-grade Stockpiles	315	25	1.7
B4-1/B4-2 Low-grade Stockpiles	285	23	1.2
B6 Low-grade Stockpile	275	15	0.1

The B1 low-grade stockpile will receive low-grade ore from the OCR and all low-grade ore generated in Mining Block 1 north of the Nobsin River.

The B2 low-grade Stockpile situated east of the process plant has been designed with a maximum capacity of 2 MLCM. This stockpile will provide additional environmental screening for the process plant during mining operations. The design backfills several small open pit excavations directly east of the process plant and these volumes are excluded from the stockpile capacity.

The B3 low-grade Stockpiles are intended for the storage of low-grade material mined in Mining Block 3. The B4 and B6 low-grade stockpiles will be used for the southern area of the Project. The B6 low-grade stockpile will receive ore from P17S.

Some stockpiles including (B1, B2, and B6) will receive both oxide and sulphide mineralized material which will be segregated to ensure minimal mixing. Should mixing of materials occur, the following mitigation measures will be in place to ensure minimal impact on processing:

- Oxide ore mixed during reclaiming of sulphide ore will be processed in the sulphide circuit with no significant impact on the process plant.

- Sulphide ore mixed during reclaiming of oxide ore which will consist mainly of large fragments of fresh rock which will be captured by grizzlies and dealt with accordingly.

16.2.3 Waste Management

The design capacity of the environmental barriers and WRDs is summarized in Table 16.7.

Table 16.7 Environmental Barrier and WRD Design Capacities

WRD	Maximum Crest Elevation (m)	Overall Height (m)	Volume (MLCM)
Maga WRD - Oxides	320	45	8.1
Maga North WRD - Sulphides	325	45	6.9
Maga South WRD - Sulphides	305	35	3.5
P8P9 WRD - Oxides	325	40	6.2
P8P9 WRD - Sulphides	300	30	2.0
P11 WRD - Oxides	310	40	2.0
P11 WRD - Sulphides	310	35	2.0
Siga_S WRD - Oxides	285	20	0.9
Siga_S East WRD - Sulphides	280	20	1.1
Siga_S North WRD - Sulphides	300	40	2.3
P16 WRD - Sulphides	300	35	2.3
P17S WRD - Oxides	285	20	0.9
P17S WRD - Sulphides	295	35	4.5
Environmental Barrier 1	295	15	2.5
Environmental Barrier 2	305	20	5.0
Environmental Barrier 3	300	20	6.0
Environmental Barrier 4	290	15	3.4

Environmental barriers reach a maximum height of 20 m (four lifts) and will be immediately graded and re-habilitated to provide a visual and sound barrier between the active mining areas and the local villages. During operations, environmental barriers should be planned and phased in a way that matches the proposed mining areas. As such, priority is given to construction of the environmental barrier over the other oxide WRDs.

During operations, waste may be left in situ on the last bench, if this does not prevent mining of the remaining ore.

16.2.4 ROM Stockpile

The ROM pad is located at the northern end of the process facility and consists of an elevated pad area and a drive-through ore tipping system for the process plant. AMC has designed the ROM pad with a 1.5 week capacity for oxides and 2.5-week capacity for sulphides to allow feed to the mill during any downtime of the mining operations caused by unfavourable weather, and conversely, dumping capacity during processing downtime. Additional surge capacity can be supplied by the nearby low-grade stockpile when required.

16.3 Strategic Mine Plan

Three high-level production schedules were developed to compare various operational scenarios to determine the ultimate strategy for the Project. The inventory was based on Whittle pit optimization shells, with no pit design or ramps included. The Minemax Schedule 6 software package was used to prepare the life of mine schedules.

The following scenarios were assessed:

- Scenario A: Sulphide plant at variable throughput rate of maximum 2.2 Mtpa and balance of total process feed made up by oxides. Maximum total material mined (TMM) 25 Mtpa (ore and waste from pits). Oxides gold feed grade target of 1 g/t during the first 2.5 processing years.

- Scenario B: Sulphide plant at fixed throughput rate of 2.2 Mtpa. Maximum total material mined 25 Mtpa (ore and waste from pits). Oxides gold feed grade target of 1 g/t during the first 2.5 processing years.

- Scenario C: Sulphide plant at fixed throughput rate of 2.2 Mtpa. Maximum total material mined 22 Mtpa (ore and waste from pits). Oxides gold feed grade target of 1 g/t during the first 1.5 processing years.

For the purposes of comparing the initial scenarios an indicative discounted cash flow was generated; this included preliminary operating mining and processing costs. However, the indicative cash-flows do not include capital costs. Gold production and revenues are based on optimization metallurgical recoveries that vary slightly from the final recoveries. The indicative discounted and undiscounted cash flows for the strategic scenarios assessed are shown in Table 16.8.

Table 16.8 Initial Scenarios Results Comparison

Scenario	Description	LOM Indicative Undiscounted Cash flow	LOM Indicative Discounted Cash flow	Indicative Undiscounted Cash flow until end of 2023	Indicative Discounted Cash flow until end of 2023
		M$	M$	M$	M$
A	Sulphide plant at variable throughput rate. Maximum TMM 25 Mtpa. Oxide gold feed grade target of 1 g/t during first 2.5 processing years	797.6	434.2	211.2	159.4
B	Sulphide plant at fixed 2.2 Mtpa throughput rate. Maximum TMM 25 Mtpa. Oxide gold feed grade target of 1 g/t during first 2.5 processing years	797.4	433.8	213.3	161.0
C	Sulphide plant at fixed 2.2 Mtpa throughput rate. Maximum TMM 22 Mtpa. Oxide gold feed grade target of 1 g/t during first 1.5 processing years	797.9	441.5	235.8	181.6

**Indicative cash flows are exclusive of capital costs*

Scenario C was selected as the most attractive development strategy that maximizes project value while remaining practical and in alignment with Orezone's corporate goals. The process feed grade is increased earlier in the schedule by stockpiling and delaying the processing of low-grade ore. Scenario C was used as guiding strategy to develop the detailed schedule.

16.4 Detailed Mine Plan

AMC completed a detailed mine plan using the Minemax Scheduler 6 (Minemax) software. Because the Bomboré deposit is laterally extensive, it is possible to commence mining almost anywhere within the deposit. However, due to variations in mineralization grades and strip ratios, some areas provide better returns than others. Minemax seeks to maximize the discounted operating cash flows while honouring constraints relating to processing and mining inputs. A discount rate of 10% per annum was used in the schedule.

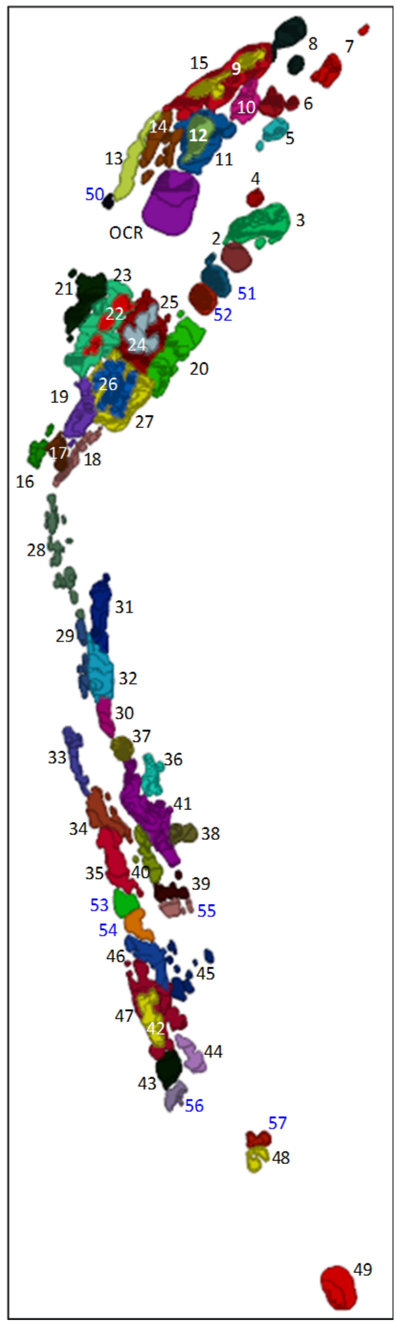
16.4.1 Schedule Inputs

Mining Areas

AMC grouped the pits into 57 mining areas based on geography, tonnage and pit value. The mining areas and their associated mineralized and waste quantities are shown in Figure 16.7.

Figure 16.7 Grouping of Mining Areas for Production Schedule

BLOCK	Pit	Total Tonnes	Waste Tonnes	Mineralized Tonnes	Au grade
		Mt	Mt	Mt	g/t
	1	10.0	7.0	3.0	0.46
	2	1.9	1.4	0.4	0.66
	3	6.8	5.3	1.6	0.72
	4	0.4	0.3	0.1	0.93
	5	1.3	1.1	0.2	0.82
	6	1.6	1.3	0.4	0.85
	7	1.7	1.3	0.3	0.75
Mining Block 1	8	3.1	2.7	0.4	1.12
	9	2.1	1.0	1.1	1.03
	10	2.9	2.0	0.9	0.59
	11	12.7	9.9	2.8	0.84
	12	4.5	3.2	1.4	1.19
	13	2.5	1.5	1.0	0.57
	14	2.4	1.5	0.9	0.56
	15	19.1	14.5	4.6	0.91
	16	0.5	0.3	0.2	0.55
	17	1.6	1.2	0.4	0.77
	18	0.7	0.4	0.3	0.52
	19	4.7	2.4	2.4	0.63
	20	5.8	4.0	1.8	0.53
Mining Block 2	21	3.7	2.6	1.0	0.60
	22	2.3	0.9	1.4	0.75
	23	10.9	6.2	4.7	0.60
	24	3.3	1.6	1.7	0.89
	25	19.8	14.0	5.8	0.87
	26	6.3	3.7	2.6	0.97
	27	17.4	13.0	4.4	0.80
	28	1.6	1.0	0.5	0.69
	29	0.9	0.6	0.3	0.60
	30	1.6	1.2	0.3	0.73
	31	4.1	3.5	0.6	0.98
	32	5.2	3.8	1.4	0.93
	33	1.4	0.7	0.7	0.50
Mining Block 3	34	2.6	1.6	1.0	0.62
	35	4.0	2.4	1.6	0.72
	36	1.5	1.0	0.5	0.72
	37	1.5	1.2	0.3	1.03
	38	1.4	1.0	0.5	0.60
	39	0.8	0.6	0.2	0.64
	40	1.5	0.9	0.6	0.50
	41	10.3	6.3	4.0	0.86
	42	2.5	0.9	1.6	1.00
	43	3.1	2.1	0.9	0.97
Mining Block 4	44	1.3	0.7	0.6	0.69
	45	0.9	0.6	0.3	0.60
	46	2.3	1.3	1.0	0.59
	47	18.3	12.2	6.1	0.93
Mining Block 5	48	0.6	0.4	0.2	0.92
Mining Block 6	49	11.5	10.2	1.3	1.86
	50	0.1	0.1	0.0	0.62
Nobsin	51	1.9	1.6	0.3	0.81
	52	1.9	1.7	0.2	0.77
	53	1.1	0.7	0.4	0.63
MV2	54	1.0	0.8	0.3	0.70
	55	0.5	0.4	0.1	0.64
Bombore	56	0.4	0.3	0.1	0.66
	57	0.6	0.3	0.2	0.91
	TOTAL	**236.2**	**164.4**	**71.8**	**0.80**



Larger pits, for example Block 2, consist of a group of separate pits of varying depths adjacent to one another, separated by ridges of waste. In these areas, the waste ridges between the pits were used to delineate the separate internal mining areas.

Schedule Constraints and Assumptions

The following constraints were applied to the schedule:

- Oxide plant to start processing on June 1, 2021.
- Sulphide plant to start processing on January 1, 2024.
- Mining starts with the OCR pit on April 1, 2020.
- Maximum total material mined (ore and waste) of approximately 22.0 Mtpa.
- Vertical advance rate of average 1 bench per month.
- Maximum number of 20 simultaneously active mining areas.
- Seasonal constraints considered for SigaS, P16, and P17S; access to these blocks is only possible when the riverbed can be traversed as no bridge will be built to accommodate haul trucks during the wet season across the MV2 stream and the Bomboré River along the main trunk road.
- Restricted Zones pits to be mined and backfilled within a single dry season and after nearby pits in the main areas are complete.
- Material for TSF embankment construction is based on staging requirements provided by KP. Minor pits have been created within the footprint of the southern end of the current design for the TSF. It was assumed that the embankments for the TSF will be re-examined in the future in order to eliminate any interaction between these pits and the TSF.
- Some pits in P8P9, P11, and SigaS to be mined and backfilled early in the mine life or when required to allow construction of dumps or stockpiles above them. Backfill material will consist of oxide waste.
- In-pit dumping of sulphide waste material is not allowed.
- No constraints on stockpiling.
- No reclaiming of low-grade mineralized material from year 2035 onwards.

Schedule Targets

The key targets of the schedule are:

- 5.2 Mtpa steady state total processing throughput.
- 2.2 Mtpa steady fixed sulphide plant feed.
- 1 g/t Au target head grade for the first 1.5 years of production.
- Waste mining production profile maintained in line with TSF and environmental barriers construction requirements.

16.4.2 Mine Schedule Summary

Mining operations extend over 13.3 years, excluding the pre-production period. The total annual ex-pit material movement peaks at 22 Mtpa from 2022 to 2024, before dropping to approximately 18.5 Mtpa for the next six years, and reducing thereafter until the end of mine life in 2034.

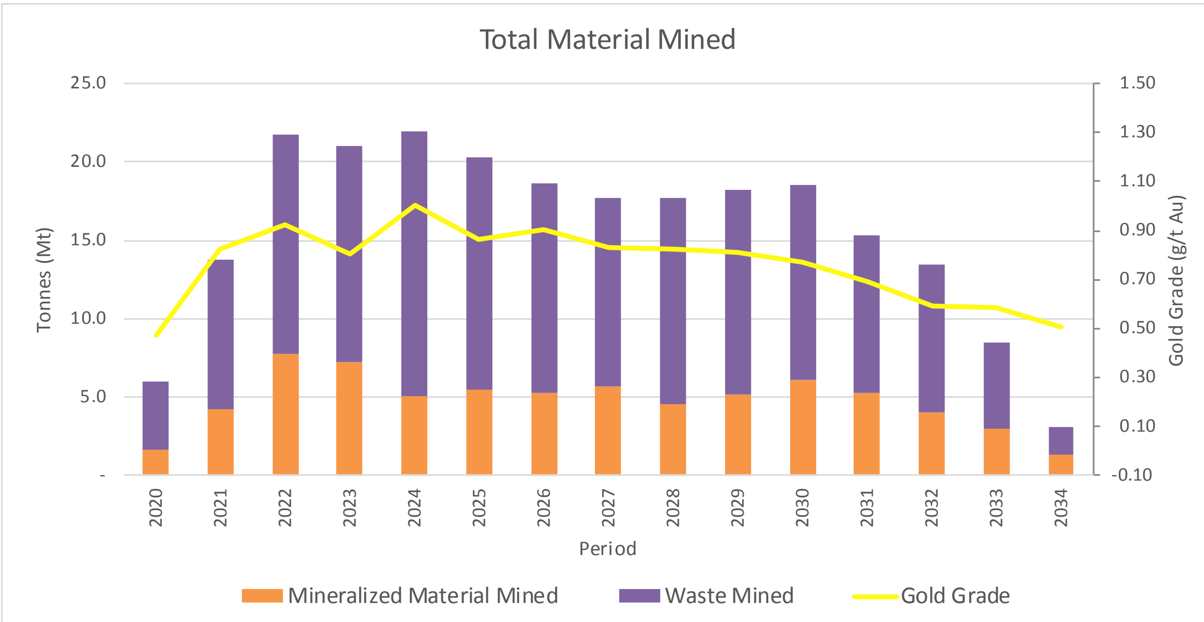
The production schedule is summarized in Table 16.9.

Table 16.9 Annual Mine Production Schedule

Material Mined	Unit	Total	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Total Mineralized Material	Mt	71.8	1.7	4.3	7.7	7.2	5.1	5.5	5.3	5.7	4.5	5.1	6.1	5.3	4.0	3.0	1.4
Gold Grade	g/t Au	0.80	0.47	0.82	0.92	0.81	1.00	0.87	0.90	0.83	0.82	0.81	0.77	0.69	0.60	0.59	0.50
Waste	Mt	164.4	4.3	9.5	14.1	13.8	16.9	14.9	13.4	12.0	13.2	13.1	12.4	10.0	9.5	5.5	1.7
Total Mined	Mt	236.2	6.0	13.8	21.8	21.1	22.0	20.3	18.7	17.7	17.8	18.3	18.5	15.3	13.5	8.5	3.0
Strip Ratio	W:O	2.29	2.58	2.23	1.83	1.92	3.32	2.72	2.55	2.12	2.94	2.57	2.02	1.88	2.39	1.81	1.20
Oxide Mineralized Material	Mt	54.3	1.7	4.3	7.0	6.5	2.7	3.4	2.6	3.3	2.8	3.1	4.3	4.6	3.8	3.0	1.3
Gold Grade	g/t Au	0.68	0.47	0.82	0.87	0.75	0.66	0.65	0.72	0.60	0.70	0.60	0.58	0.59	0.57	0.58	0.50
Oxide – Waste	Mt	121.3	4.3	9.4	12.8	12.2	7.3	9.3	7.4	9.1	8.9	8.8	7.1	8.3	9.2	5.4	1.6
Oxide - Total Mined	Mt	175.6	6.0	13.7	19.8	18.7	10.0	12.8	10.0	12.4	11.7	11.9	11.4	12.9	13.0	8.3	3.0
Sulphide Mineralized Material	Mt	17.6	-	0.0	0.7	0.8	2.4	2.0	2.7	2.3	1.7	2.1	1.9	0.7	0.2	0.1	0.0
Gold Grade	g/t Au	1.19	-	1.37	1.41	1.25	1.38	1.23	1.08	1.16	1.03	1.12	1.21	1.39	1.11	0.92	0.83
Sulphide - Waste	Mt	43.1	-	0.1	1.3	1.6	9.6	5.6	6.0	2.9	4.3	4.4	5.3	1.7	0.3	0.1	0.0
Sulphide - Total Mined	Mt	60.6	-	0.1	2.0	2.4	12.0	7.6	8.7	5.3	6.0	6.4	7.2	2.4	0.5	0.1	0.1
Ore Movement																	
Ore Mine to Process	Mt	54.5	-	2.5	5.2	5.0	3.6	4.0	3.9	4.8	4.0	4.6	4.9	3.8	4.0	3.0	1.4
Gold Grade	g/t Au	0.88	-	1.08	1.01	0.85	1.10	0.98	0.95	0.91	0.88	0.87	0.87	0.80	0.60	0.59	0.50
Ore Mine to Stockpile	Mt	17.3	1.7	1.8	2.5	2.2	1.5	1.5	1.4	0.9	0.5	0.5	1.3	1.5	0.0	0.0	-
Gold Grade	g/t Au	0.57	0.47	0.46	0.74	0.71	0.77	0.57	0.76	0.41	0.38	0.33	0.38	0.41	0.32	0.32	-
Ore Stockpile to Process	Mt	15.6	-	-	-	0.2	1.6	1.2	1.3	0.4	1.2	0.6	0.3	1.4	1.2	2.2	3.8
Gold Grade	Au g/t	0.60	-	-	-	0.56	0.87	1.10	0.81	0.69	0.85	0.70	0.63	0.63	0.49	0.33	0.33
Material Processed																	
Total Process Feed	Mt	70.1	-	2.5	5.2	5.2	5.2	5.2	5.2	5.2	5.2	5.2	5.2	5.2	5.2	5.2	5.2
Gold Grade	g/t Au	0.81	-	1.08	1.01	0.84	1.03	1.01	0.92	0.89	0.87	0.85	0.86	0.76	0.57	0.48	0.37
Oxide Process Feed	Mt	52.5	-	2.5	5.2	5.2	3.2	3.0	3.0	3.0	3.0	3.0	3.0	3.1	5.0	5.1	5.2
Gold Grade	g/t Au	0.69	-	1.08	1.01	0.84	0.68	0.72	0.72	0.66	0.72	0.64	0.66	0.67	0.55	0.47	0.37
Sulphide Process Feed	Mt	17.6	-	-	-	-	2.0	2.2	2.2	2.2	2.2	2.2	2.2	2.1	0.2	0.1	0.0
Gold Grade	g/t Au	1.19	-	-	-	-	1.59	1.41	1.18	1.21	1.09	1.13	1.12	0.88	1.09	0.92	0.83

Figure 16.8 shows the annual production schedule graphically.

Figure 16.8 Annual Total Material Mined



Mining Sequence by Mining Block

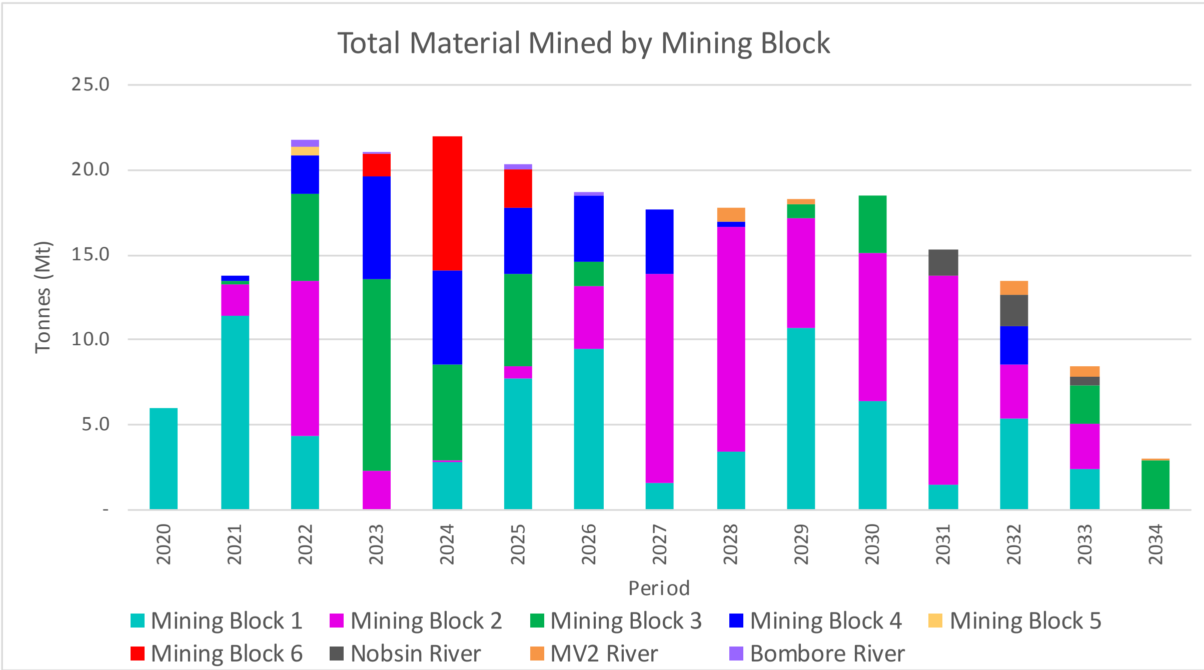
Mining commences with the pre-production development of the OCR in Mining Block 1. The OCR construction is scheduled to commence in April 2020 and will take 14 months to complete. Mining Blocks 1, 2 and to a lesser extent 3 and 4 commence in 2020 followed by the remaining mining areas.

Mining of the sulphide pits in preparation for start-up of the sulphide processing operation starts in 2022.

Mining of the Restricted Zone pits starts in 2022 and is then scheduled to coincide with both seasonal constraints and depletion of adjacent mining areas.

Figure 16.9 summarizes the mining sequence.

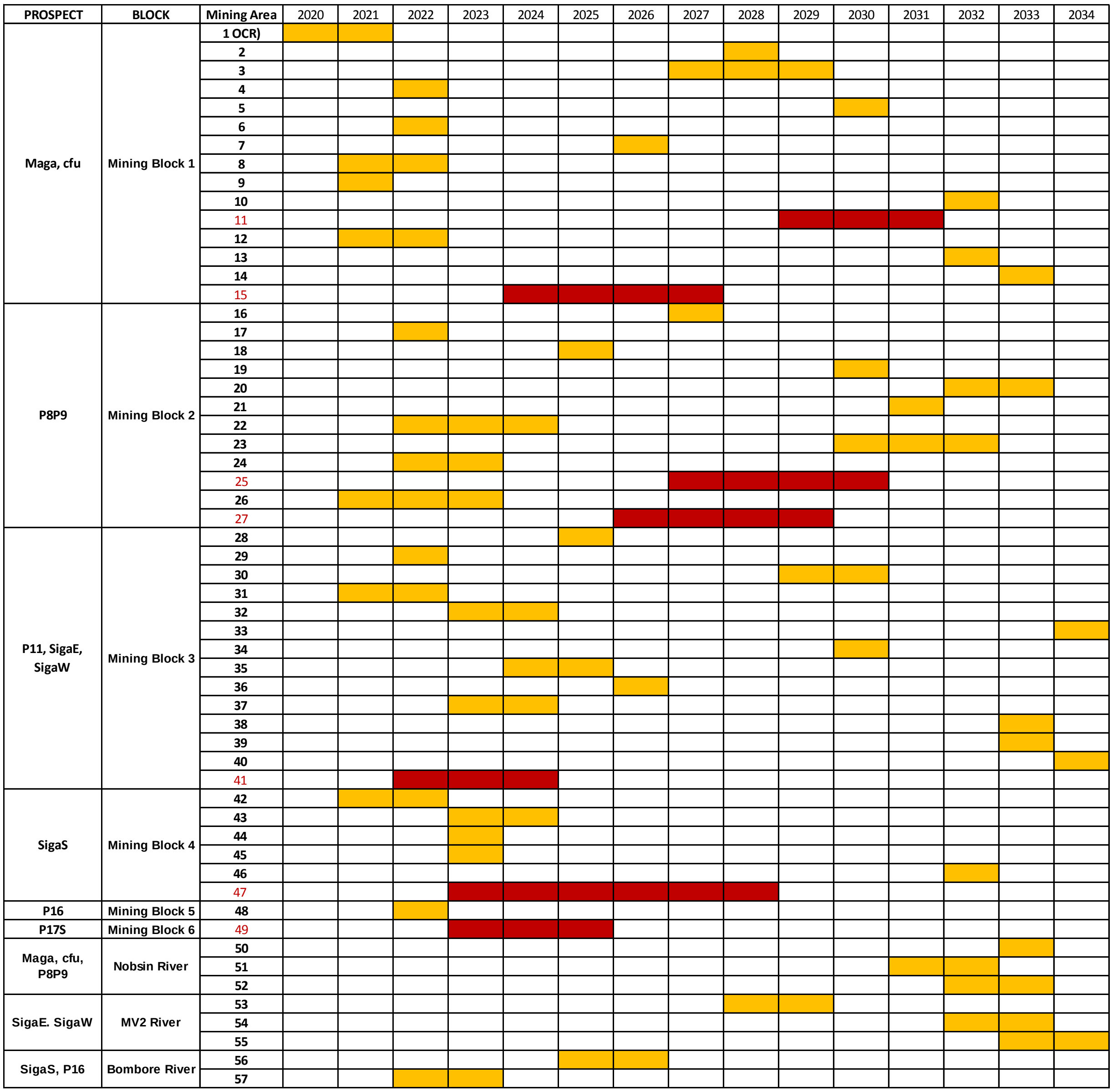
Figure 16.9 Annual Total Material Mined by Mining Block



Mining Sequence by Mining Area

The mining sequence of the 57 mining areas is summarized in Figure 16.10. The mining areas highlighted by red in colour contain more than 1 Mt of sulphide ore.

Figure 16.10 Mining Sequence by Mining Area



16.4.3 Process Feed Schedule

The schedule includes the following oxide process plant ramp-up:

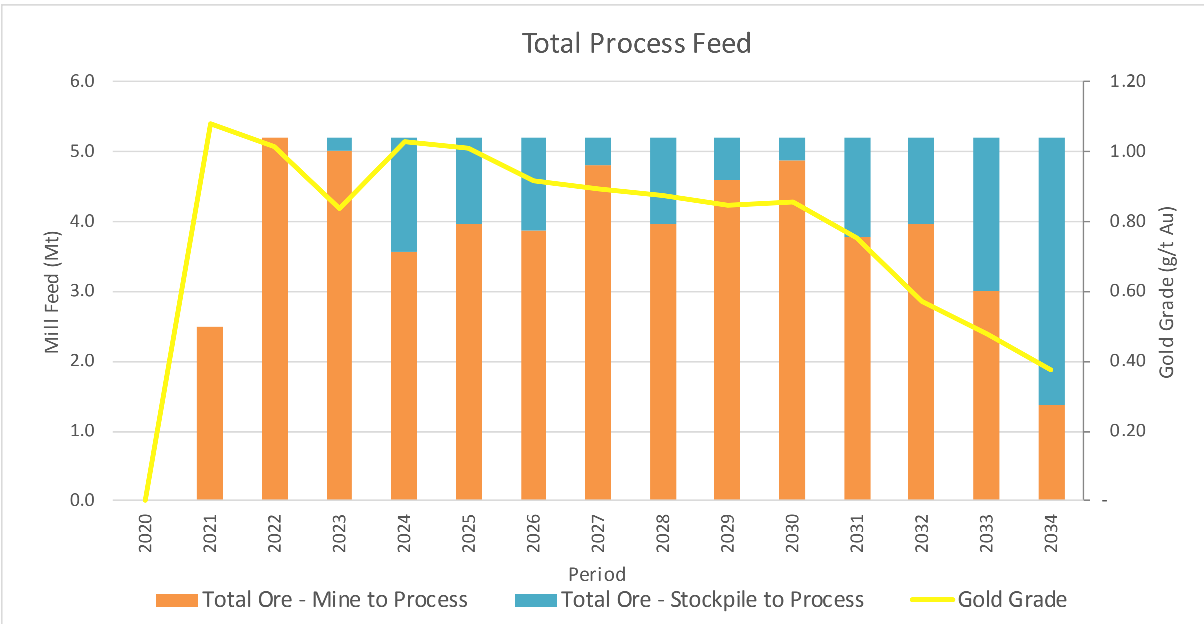
- 40% capacity in June 2021.
- 65% capacity in July 2021.
- 75% capacity in August 2021.

The oxide plant throughput in 2021 is 2.5 Mt, followed by a yearly throughput of 5.2 Mtpa from year 2022.

The sulphide plant throughput during its ramp-up period (January to March 2024) is 0.35 Mt, followed by a maximum sulphide plant throughput of 2.2 Mtpa.

As presented in Figure 16.11, targeted process feed is achieved on a yearly basis from 2022 to 2034. Process feed from 2024 includes ore re-handled from the low-grade stockpiles.

Figure 16.11 Total Process Feed Schedule



The process feed during the payback period consist of 66% of Proven and 34% of Probable Reserves.

Figure 16.12 and Figure 16.13 show the process feed of oxide and sulphide ore respectively.

Figure 16.12 Oxides Process Feed Schedule

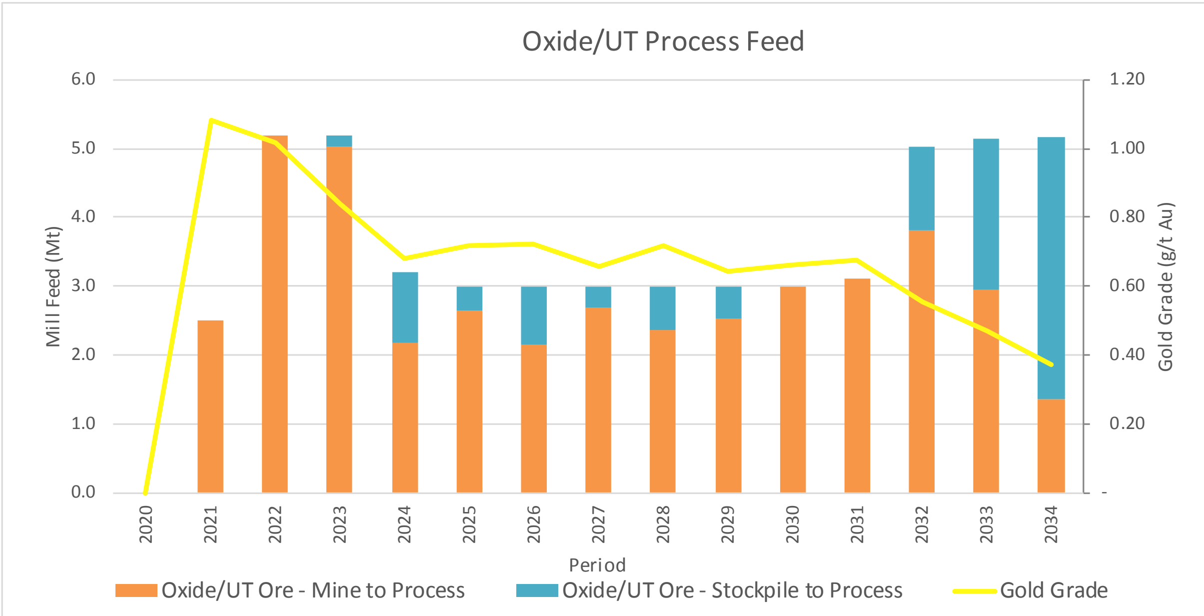
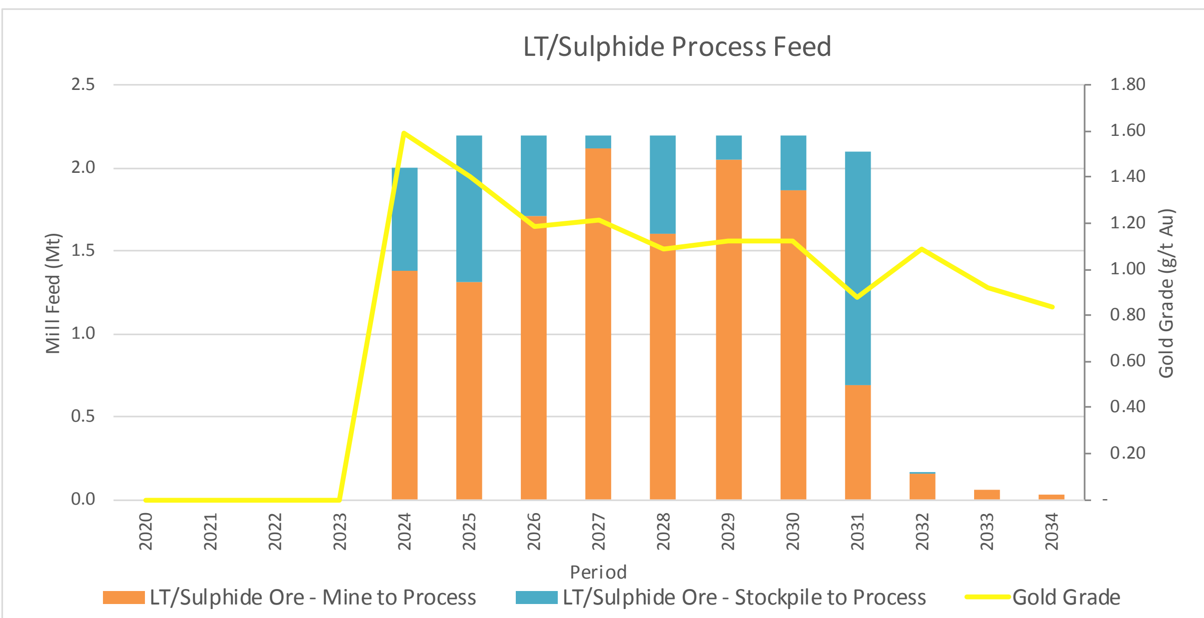


Figure 16.13 Sulphides Process Feed Schedule



The sulphide ore from year 2032 to 2034 (Figure 16.13) will be crushed and processed through the oxide circuit, thereby eliminating the need to operate the sulphide SAG mill.

16.4.4 Stockpiling Strategy

To achieve the target head grade of 1 g/t Au in the first 1.5 years and maximize value of the Project, AMC split the ore into two grade categories for oxides and sulphides as presented in Table 16.10.

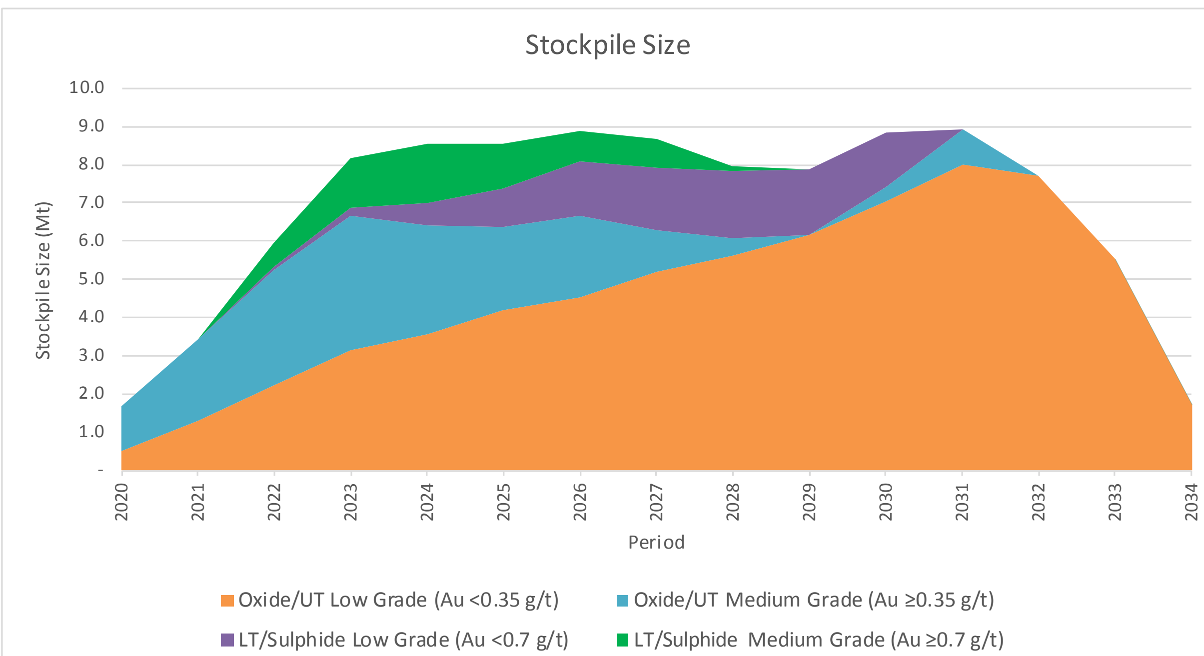
Table 16.10 Stockpiles Grade Bins

	Oxide	Sulphide
Low-grade	< 0.35 g/t	< 0.7 g/t
Medium grade	>= 0.35 g/t	>= 0.7 g/t

Eight stockpiles are located adjacent to mining centres across the site. Low and medium-grade material will be sent to the closest available stockpile and the combined tonnage of these stockpiles reaches 9 Mt by 2026 (Figure 16.14).

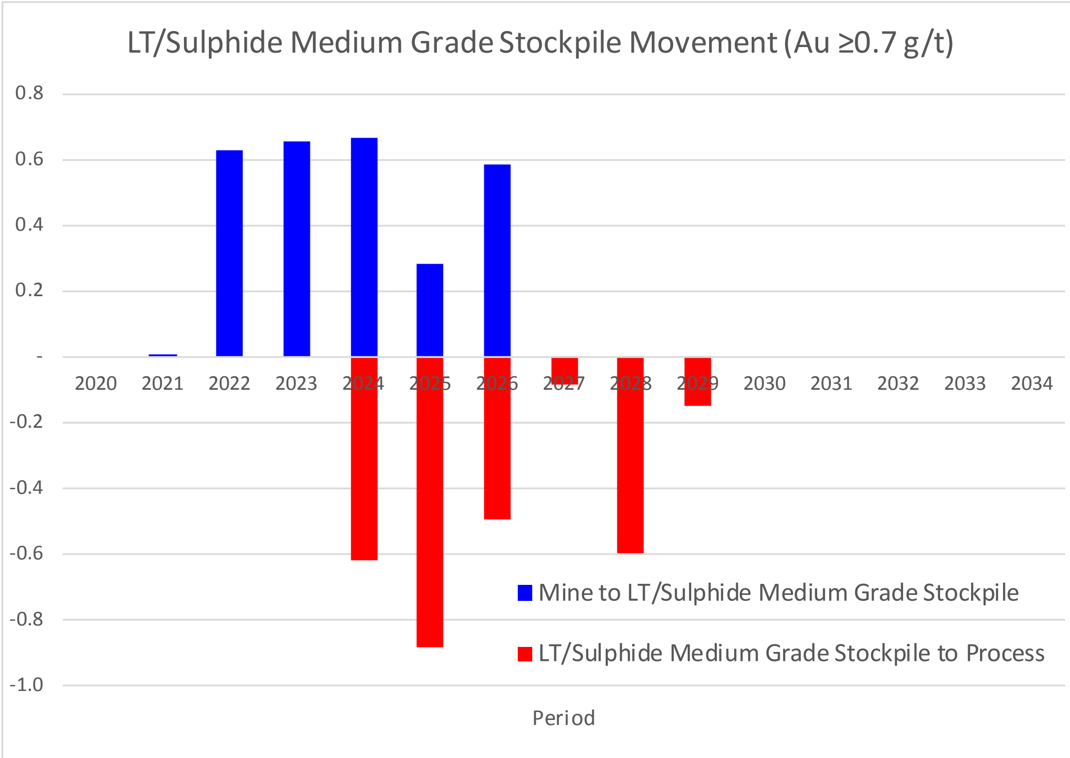
1.7 Mt of mineralized oxide material remains in the oxide low-grade stockpiles at the end of the mine life; as this material is not processed it has been excluded from the Mineral Reserve Estimate.

Figure 16.14 Total Stockpile Size over Life of Mine by Grade Category



The sulphide medium-grade stockpile is built up during 2022 to 2026; no additional sulphide medium-grade material is stockpiled after 2026 (refer to Figure 16.15).

Figure 16.15 Sulphides Medium-grade Stockpile Movement



16.4.5 Waste Movement

In total, 164.4 Mt of waste is mined including 121.3 Mt of oxide waste and 43.1 Mt of sulphide waste. Approximately 64.5 Mt of oxide waste produced will be used in the construction of the TSF. The remainder of the oxide waste is primarily used for construction of the environmental barriers, backfill of pits, and development of the oxide WRDs.

Life-of-mine waste movements are summarized in Table 16.11.

Table 16.11 Life of Mine Waste Movement Schedule

Waste Movement	Unit	Total	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Mine to TSF (Oxides)	Mt	**64.5**	3.0	3.9	5.6	8.6	1.1	2.6	5.5	8.8	5.3	5.7	4.0	5.1	3.8	1.5	-
Mine to Environmental Barrier 1 (Oxides)	Mt	**3.4**	1.4	2.1	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine to Environmental Barrier 2 (Oxides)	Mt	**6.6**	-	0.7	0.2	0.6	0.1	-	-	0.1	0.2	0.2	1.2	1.4	1.2	0.7	-
Mine to Environmental Barrier 3 (Oxides)	Mt	**8.4**	-	0.1	1.5	0.4	1.2	1.7	0.1	-	-	0.5	1.9	-	-	1.1	-
Mine to Environmental Barrier 4 (Oxides)	Mt	**4.4**	-	0.2	0.6	-	2.4	0.1	-	-	-	-	-	-	1.1	-	-
Mine to Oxide WRDs	Mt	**24.4**	-	2.4	3.9	1.0	2.6	3.8	1.3	0.1	2.9	2.1	-	0.5	0.9	1.4	1.5
Mine to Sulphide WRDs	Mt	**43.0**	-	0.1	1.3	1.6	9.6	5.6	6.0	2.9	4.3	4.4	5.3	1.7	0.3	0.1	0.0
Mine to in-pit dumps (Oxides)	Mt	**3.7**	-	-	0.7	1.5	-	1.0	0.4	0.1	-	-	-	-	-	-	-
Mine to Nobsin In-pit Dumps (Oxides)	Mt	**3.3**	-	-	-	-	-	-	-	-	-	-	-	1.4	1.6	0.4	-
Mine to MV2 In-pit Dumps (Oxides)	Mt	**1.9**	-	-	-	-	-	-	-	-	0.6	0.2	-	-	0.7	0.4	0.1
Mine to Bomboré In-pit Dumps (Oxides)	Mt	**0.7**	-	-	0.3	0.1	-	0.2	0.1	-	-	-	-	-	-	-	-
Total Waste		**164.4**	**4.3**	**9.5**	**14.1**	**13.8**	**16.9**	**14.9**	**13.4**	**12.0**	**13.2**	**13.1**	**12.4**	**10.0**	**9.5**	**5.5**	**1.7**

16.4.6 Production Schedule Details

The detailed production schedule was generated monthly for the first two years, followed by quarterly periods for five years, and in yearly increments thereafter. The detailed production schedule is shown in Table 16.12.

Table 16.12 Detailed Production Schedule

		Unit	LOM Total	Apr 2020	May 2020	Jun 2020	Jul 2020	Aug 2020	Sep 2020	Oct 2020	Nov 2020	Dec 2020	Jan 2021	Feb 2021	Mar 2021	Apr 2021	May 2021	Jun 2021	Jul 2021	Aug 2021	Sep 2021	Oct 2021	Nov 2021	Dec 2021
Process Feed	Total Process Feed	Mt	70.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.2	0.3	0.3	0.4	0.4	0.4	0.4
	Gold Grade	g/t Au	0.81	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.09	0.96	1.04	1.01	1.09	1.11	1.21
	Oxides Process Feed	Mt	52.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.2	0.3	0.3	0.4	0.4	0.4	0.4
	Gold Grade	g/t Au	0.69	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.09	0.96	1.04	1.01	1.09	1.11	1.21
	Sulphides Process Feed	Mt	17.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Gold Grade	g/t Au	1.19	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Material Mined	Total Mineralized Material	Mt	71.8	0.1	0.2	0.1	0.2	0.3	0.0	0.2	0.3	0.3	0.3	0.3	0.2	0.2	0.4	0.2	0.3	0.4	0.5	0.5	0.5	0.5
	Gold grade	g/t Au	0.80	0.72	0.45	0.33	0.48	0.45	0.33	0.48	0.46	0.46	0.45	0.45	0.42	0.51	0.45	0.94	0.88	0.99	0.96	1.00	1.02	1.11
	Waste Mined	Mt	164.4	0.3	0.2	0.6	0.5	0.4	0.6	0.6	0.5	0.5	0.5	0.5	0.6	0.7	0.6	0.6	0.6	0.9	0.8	1.1	1.1	1.4
	Total Mined	Mt	236.2	0.4	0.4	0.7	0.7	0.7	0.7	0.8	0.8	0.8	0.8	0.8	0.8	1.0	0.9	0.8	0.9	1.3	1.3	1.6	1.6	1.9
	Strip ratio	W:O	2.29	3.1	1.3	11.9	2.0	1.4	13.7	2.5	2.0	1.9	1.8	1.7	3.1	3.5	1.4	2.7	1.8	2.5	1.7	2.2	2.3	2.7
	Oxides Mineralized Material	Mt	54.3	0.1	0.2	0.1	0.2	0.3	0.0	0.2	0.3	0.3	0.3	0.3	0.2	0.2	0.4	0.2	0.3	0.4	0.5	0.5	0.5	0.5
	Gold Grade	g/t Au	0.68	0.72	0.45	0.33	0.48	0.45	0.33	0.48	0.46	0.46	0.45	0.45	0.42	0.51	0.45	0.94	0.88	0.99	0.96	1.00	1.02	1.10
	Oxides - Waste	Mt	121.3	0.3	0.2	0.6	0.5	0.4	0.6	0.6	0.5	0.5	0.5	0.5	0.6	0.7	0.5	0.6	0.6	0.9	0.8	1.1	1.1	1.4
	Oxides - Total Mined	Mt	175.6	0.4	0.4	0.7	0.7	0.7	0.7	0.8	0.8	0.8	0.8	0.8	0.8	0.9	0.9	0.8	0.9	1.3	1.3	1.6	1.6	1.9
	Sulphides Mineralized Material	Mt	17.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.0
	Gold Grade	g/t Au	1.19	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.37
	Sulphides - Waste	Mt	43.1	-	-	-	-	-	-	-	-	-	-	-	-	0.0	0.0	-	-	-	-	-	0.0	0.0
	Sulphides - Total Mined	Mt	60.6	-	-	-	-	-	-	-	-	-	-	-	-	0.0	0.0	-	-	-	-	-	0.0	0.0
BCM	Mineralized Material BCM	M BCM	36.23	0.06	0.10	0.03	0.14	0.17	0.03	0.14	0.16	0.16	0.16	0.16	0.10	0.11	0.20	0.13	0.19	0.21	0.27	0.29	0.27	0.27
	Waste BCM	M BCM	83.63	0.17	0.13	0.37	0.27	0.24	0.38	0.34	0.31	0.29	0.28	0.27	0.32	0.39	0.29	0.34	0.35	0.54	0.48	0.66	0.65	0.80
	Total BCM	M BCM	119.86	0.23	0.23	0.40	0.41	0.41	0.41	0.48	0.47	0.46	0.44	0.43	0.42	0.49	0.49	0.47	0.54	0.75	0.74	0.95	0.92	1.07
Ore movement	Ore Mine to Process	Mt	54.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.2	0.3	0.3	0.4	0.4	0.4	0.4
	Gold Grade	g/t Au	0.88	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.09	0.96	1.04	1.01	1.09	1.11	1.21
	Ore Mine to Stockpile	Mt	17.3	0.1	0.2	0.1	0.2	0.3	0.0	0.2	0.3	0.3	0.3	0.3	0.2	0.2	0.4	0.0	0.1	0.0	0.0	0.1	0.1	0.1
	Gold Grade	g/t Au	0.57	0.72	0.45	0.33	0.48	0.45	0.33	0.48	0.46	0.46	0.45	0.45	0.42	0.51	0.45	0.41	0.40	0.33	0.33	0.48	0.33	0.63
	Ore Stockpile to Process	Mt	15.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Gold Grade	g/t Au	0.60	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste movement	Mine to TSF (Oxides)	Mt	64.5	-	-	-	0.3	0.4	0.6	0.6	0.5	0.5	0.5	0.5	-	0.6	0.5	0.4	0.2	-	0.2	0.2	0.3	0.4
	Mine to Environmental Barrier 1 (Oxides)	Mt	3.4	0.3	0.2	0.6	0.2	-	-	-	-	-	-	-	0.6	0.1	0.0	0.2	0.3	0.7	-	-	-	-
	Mine to Environmental Barrier 2 (Oxides)	Mt	6.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.2	0.0	0.3	0.3	-
	Mine to Environmental Barrier 3 (Oxides)	Mt	8.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.1
	Mine to Environmental Barrier 4 (Oxides)	Mt	4.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.2
	Mine to Oxide Dumps	Mt	24.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.0	0.0	0.6	0.6	0.5	0.6
	Mine to Sulphide Dumps	Mt	43.0	-	-	-	-	-	-	-	-	-	-	-	-	0.0	0.0	-	-	-	-	-	0.0	0.0
	Mine to in-pit dumps (Oxides)	Mt	3.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

		Unit	LOM Total	Apr 2020	May 2020	Jun 2020	Jul 2020	Aug 2020	Sep 2020	Oct 2020	Nov 2020	Dec 2020	Jan 2021	Feb 2021	Mar 2021	Apr 2021	May 2021	Jun 2021	Jul 2021	Aug 2021	Sep 2021	Oct 2021	Nov 2021	Dec 2021
	Mine to Nobsin In-pit Dumps (Oxides)	Mt	3.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mine to MV2 In-pit Dumps (Oxides)	Mt	1.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mine to Bombore In-pit Dumps (Oxides)	Mt	0.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Total		**164.4**	**0.3**	**0.2**	**0.6**	**0.5**	**0.4**	**0.6**	**0.6**	**0.5**	**0.5**	**0.5**	**0.5**	**0.6**	**0.7**	**0.6**	**0.6**	**0.6**	**0.9**	**0.8**	**1.1**	**1.1**	**1.4**

		Unit	Q1 2022	Q2 2022	Q3 2022	Q4 2022	Q1 2023	Q2 2023	Q3 2023	Q4 2023	Q1 2024	Q2 2024	Q3 2024	Q4 2024	Q1 2025	Q2 2025	Q3 2025	Q4 2025	Q1 2026	Q2 2026	Q3 2026	Q4 2026	2027	2028	2029	2030	2031	2032	2033	2034
Process Feed	Total Process Feed	Mt	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	5.2	5.2	5.2	5.2	5.2	5.2	5.2	5.2
	Gold Grade	g/t Au	0.99	1.09	0.91	1.08	0.91	0.80	0.84	0.79	1.03	1.07	1.00	1.03	1.02	1.04	0.95	1.03	0.98	0.94	0.89	0.87	0.89	0.87	0.85	0.86	0.76	0.57	0.48	0.37
	Oxides Process Feed	Mt	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.0	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	3.0	3.0	3.0	3.0	3.1	5.0	5.1	5.2
	Gold Grade	g/t Au	0.99	1.09	0.91	1.08	0.91	0.80	0.84	0.79	0.71	0.66	0.73	0.61	0.64	0.75	0.71	0.78	0.78	0.77	0.70	0.64	0.66	0.72	0.64	0.66	0.67	0.55	0.47	0.37
	Sulphides Process Feed	Mt	-	-	-	-	-	-	-	-	0.4	0.6	0.6	0.6	0.6	0.6	0.6	0.6	0.6	0.6	0.6	0.6	2.2	2.2	2.2	2.2	2.1	0.2	0.1	0.0
	Gold Grade	g/t au	-	-	-	-	-	-	-	-	1.90	1.62	1.36	1.60	1.54	1.44	1.27	1.37	1.24	1.16	1.16	1.18	1.21	1.09	1.13	1.12	0.88	1.09	0.92	0.83
Material Mined	Total Mineralized Material	Mt	1.8	2.1	1.8	2.0	2.0	1.9	1.8	1.6	1.1	1.4	1.6	0.9	1.1	1.4	1.4	1.6	1.3	1.4	1.3	1.3	5.7	4.5	5.1	6.1	5.3	4.0	3.0	1.4
	Gold grade	g/t Au	0.93	1.00	0.81	0.95	0.86	0.73	0.80	0.84	1.14	1.00	0.93	1.00	0.93	0.90	0.71	0.93	0.85	0.93	0.88	0.95	0.83	0.82	0.81	0.77	0.69	0.60	0.59	0.50
	Waste Mined	Mt	3.5	3.6	3.7	3.3	2.8	3.4	3.4	4.3	4.7	4.5	3.3	4.3	4.4	4.2	2.9	3.3	3.3	3.6	3.1	3.5	12.0	13.2	13.1	12.4	10.0	9.5	5.5	1.7
	Total Mined	Mt	5.3	5.7	5.5	5.3	4.8	5.2	5.2	5.9	5.8	6.0	5.0	5.2	5.5	5.6	4.3	4.9	4.5	5.0	4.4	4.8	17.7	17.8	18.3	18.5	15.3	13.5	8.5	3.0
	Strip ratio	W:O	2.0	1.8	2.0	1.6	1.4	1.8	1.9	2.7	4.5	3.1	2.0	4.6	4.1	3.1	2.2	2.0	2.6	2.6	2.4	2.6	2.1	2.9	2.6	2.0	1.9	2.4	1.8	1.2
	Oxides Mineralized Material	Mt	1.6	1.8	1.8	1.8	1.8	1.8	1.6	1.3	0.6	0.7	0.8	0.5	0.6	0.9	1.0	0.9	0.8	0.7	0.7	0.4	3.3	2.8	3.1	4.3	4.6	3.8	3.0	1.3
	Gold Grade	g/t Au	0.88	0.92	0.79	0.90	0.79	0.69	0.76	0.76	0.77	0.62	0.69	0.59	0.61	0.67	0.62	0.71	0.74	0.78	0.68	0.66	0.60	0.70	0.60	0.58	0.59	0.57	0.58	0.50
	Oxides - Waste	Mt	3.2	3.3	3.5	2.9	2.7	3.3	3.1	3.2	2.1	1.2	2.4	1.6	2.4	2.7	2.4	1.8	2.0	2.0	2.0	1.4	9.1	8.9	8.8	7.1	8.3	9.2	5.4	1.6
	Oxides - Total Mined	Mt	4.8	5.0	5.2	4.7	4.5	5.1	4.7	4.4	2.7	1.9	3.2	2.1	3.0	3.7	3.4	2.7	2.8	2.7	2.7	1.9	12.4	11.7	11.9	11.4	12.9	13.0	8.3	3.0
	Sulphides Mineralized Material	Mt	0.2	0.3	0.1	0.2	0.1	0.1	0.2	0.3	0.5	0.7	0.8	0.4	0.5	0.4	0.4	0.7	0.5	0.7	0.6	0.9	2.3	1.7	2.1	1.9	0.7	0.2	0.1	0.0
	Gold Grade	g/t Au	1.40	1.47	1.13	1.42	1.59	1.68	1.01	1.18	1.55	1.38	1.18	1.54	1.36	1.38	0.96	1.20	1.04	1.07	1.11	1.08	1.16	1.03	1.12	1.21	1.39	1.11	0.92	0.83
	Sulphides - Waste	Mt	0.4	0.4	0.2	0.4	0.1	0.1	0.2	1.1	2.6	3.3	1.0	2.7	2.1	1.5	0.5	1.5	1.3	1.5	1.1	2.0	2.9	4.3	4.4	5.3	1.7	0.3	0.1	0.0
	Sulphides - Total Mined	Mt	0.5	0.6	0.3	0.6	0.3	0.2	0.5	1.4	3.1	4.1	1.8	3.1	2.5	1.9	0.9	2.2	1.8	2.3	1.7	2.9	5.3	6.0	6.4	7.2	2.4	0.5	0.1	0.1
BCM	Mineralized Material BCM	M BCM	1.03	1.07	1.01	1.08	1.08	1.05	0.97	0.81	0.49	0.65	0.71	0.43	0.54	0.66	0.66	0.72	0.59	0.61	0.62	0.57	2.64	2.15	2.47	2.95	2.74	2.18	1.69	0.73
	Waste BCM	M BCM	1.96	1.93	2.03	1.75	1.58	1.92	1.91	2.10	2.06	1.78	1.64	1.80	2.11	1.99	1.51	1.48	1.58	1.70	1.56	1.60	6.27	6.62	6.57	5.73	5.22	5.37	3.09	0.91
	Total BCM	M BCM	2.99	3.00	3.04	2.83	2.66	2.97	2.88	2.91	2.55	2.43	2.35	2.23	2.65	2.66	2.16	2.20	2.18	2.31	2.18	2.16	8.91	8.77	9.04	8.68	7.96	7.55	4.78	1.64
Ore movement	Ore Mine to Process	Mt	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.1	0.7	0.9	1.3	0.7	0.8	1.1	0.9	1.2	0.7	1.1	1.1	0.9	4.8	4.0	4.6	4.9	3.8	4.0	3.0	1.4
	Gold Grade	g/t Au	0.99	1.09	0.91	1.08	0.91	0.80	0.84	0.83	1.31	1.09	0.99	1.13	1.07	1.00	0.78	1.06	0.85	1.00	0.93	1.00	0.91	0.88	0.87	0.87	0.80	0.60	0.59	0.50
	Ore Mine to Stockpile	Mt	0.5	0.8	0.5	0.7	0.7	0.6	0.5	0.5	0.4	0.5	0.4	0.3	0.3	0.2	0.5	0.5	0.5	0.3	0.1	0.4	0.9	0.5	0.5	1.3	1.5	0.0	0.0	-
	Gold Grade	g/t Au	0.77	0.85	0.56	0.73	0.74	0.57	0.68	0.87	0.83	0.83	0.71	0.66	0.60	0.40	0.58	0.61	0.83	0.66	0.43	0.83	0.41	0.38	0.33	0.38	0.41	0.32	0.32	-
	Ore Stockpile to Process	Mt	-	-	-	-	-	-	-	0.2	0.6	0.4	0.0	0.6	0.5	0.2	0.4	0.1	0.6	0.2	0.2	0.4	0.4	1.2	0.6	0.3	1.4	1.2	2.2	3.8
	Gold Grade	g/t Au	-	-	-	-	-	-	-	0.56	0.73	1.00	1.37	0.92	0.96	1.31	1.31	0.76	1.12	0.56	0.57	0.56	0.69	0.85	0.70	0.63	0.63	0.49	0.33	0.33

		Unit	Q1 2022	Q2 2022	Q3 2022	Q4 2022	Q1 2023	Q2 2023	Q3 2023	Q4 2023	Q1 2024	Q2 2024	Q3 2024	Q4 2024	Q1 2025	Q2 2025	Q3 2025	Q4 2025	Q1 2026	Q2 2026	Q3 2026	Q4 2026	2027	2028	2029	2030	2031	2032	2033	2034
Waste movement	Mine to TSF (Oxides)	Mt	1.7	1.6	1.1	1.2	2.0	2.5	2.0	2.1	0.0	0.5	0.6	-	0.3	1.7	0.4	0.3	1.5	1.1	1.5	1.4	8.8	5.3	5.7	4.0	5.1	3.8	1.5	-
	Mine to Environmental Barrier 1 (Oxides)	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mine to Environmental Barrier 2 (Oxides)	Mt	-	-	0.2	-	0.3	0.1	0.1	0.1	0.1	-	-	-	-	-	-	-	-	-	-	-	0.1	0.2	0.2	1.2	1.4	1.2	0.7	-
	Mine to Environmental Barrier 3 (Oxides)	Mt	1.1	-	-	0.4	-	-	0.4	-	0.7	0.3	0.1	0.1	0.5	1.0	0.1	-	-	-	0.1	-	-	-	0.5	1.9	-	-	1.1	-
	Mine to Environmental Barrier 4 (Oxides)	Mt	0.3	0.3	-	-	-	-	-	-	1.0	0.5	0.5	0.5	0.1	-	-	-	-	-	-	-	-	-	-	-	-	1.1	-	-
	Mine to Oxide Dumps	Mt	0.1	1.3	1.6	0.9	-	-	-	1.0	0.4	-	1.2	1.0	1.4	-	1.2	1.1	-	0.9	0.4	0.1	0.1	2.9	2.1	-	0.5	0.9	1.4	1.5
	Mine to Sulphide Dumps	Mt	0.4	0.4	0.2	0.4	0.1	0.1	0.2	1.1	2.6	3.3	1.0	2.7	2.1	1.5	0.5	1.5	1.3	1.5	1.1	2.0	2.9	4.3	4.4	5.3	1.7	0.3	0.1	0.0
	Mine to in-pit dumps (Oxides)	Mt	-	0.1	0.5	0.1	0.3	0.6	0.6	-	-	-	-	-	-	-	0.8	0.2	0.4	-	-	-	0.1	-	-	-	-	-	-	-
	Mine to Nobsin In-pit Dumps (Oxides)	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.4	1.6	0.4	-
	Mine to MV2 In-pit Dumps (Oxides)	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.6	0.2	-	-	0.7	0.4	0.1
	Mine to Bombore In-pit Dumps (Oxides)	Mt	-	-	-	0.3	0.1	-	-	-	-	-	-	-	-	-	-	0.2	0.1	-	-	-	-	-	-	-	-	-	-	-
	Total		**3.5**	**3.6**	**3.7**	**3.3**	**2.8**	**3.4**	**3.4**	**4.3**	**4.7**	**4.5**	**3.3**	**4.3**	**4.4**	**4.2**	**2.9**	**3.3**	**3.3**	**3.6**	**3.1**	**3.5**	**12.0**	**13.2**	**13.1**	**12.4**	**10.0**	**9.5**	**5.5**	**1.7**

16.4.7 Equipment Requirements

AMC estimated equipment requirements based on the production schedule. Table 16.13 summarizes the average yearly productivities of trucks and excavators.

Table 16.13 Truck and Excavator Productivity Inputs

Input	Units	Oxide Fleet	Sulphide Fleet
Shift			
Days/year	days	365	365
Hours/day	hr	24	24
Shift time	hr	12	12
Shifts/day	#	2	2
Effective shift time	mins/hr	55	55
Effective hours/day	hr	22	22
Downtime	days	9	9
Wet Season Downtime Days	days	6	6
Total Downtime	days	15	15
Working days/year	days	350	350
Calendar time	hr	7700	7700
Truck Parameters			
Truck payload	t	27.7	50.0
Truck speed			
In-pit loaded	km/hr	12	12
Ex-pit loaded	km/hr	40	40
Ex-pit empty	km/hr	50	50
In-pit empty	km/hr	35	35
On-bench/on-lift	km/hr	25	25
Truck availability	%	85	85
Truck utilization	%	85	85
Operator efficiency	%	90	90
Productive truck hours	hr	5,007	5,007
Excavator Parameters			
Bucket size	m3	4.5	4
Fill factor	%	95	92
Bucket load	t	5.7	7.2
Passes/truck	#	5	7
Load time/truck	sec	225	280
Waiting/delay time	sec	100	100
Excavator availability	%	80	80
Excavator utilization	%	75	80
Net operating hours/year	hr	5,040	5,376
Productivity/hour	t/hr	307	474
Productivity/year	Mt/yr	1.55	2.55

The schedule accounts for lower excavator productivity during the heavy-rain months as follows:

- Oxide excavator productivity: 115 kt/month from June to September and 136 kt/month during the rest of the year.

- Sulphide excavator productivity: 193 kt/month from June to September and 222 kt/month during the rest of the year.

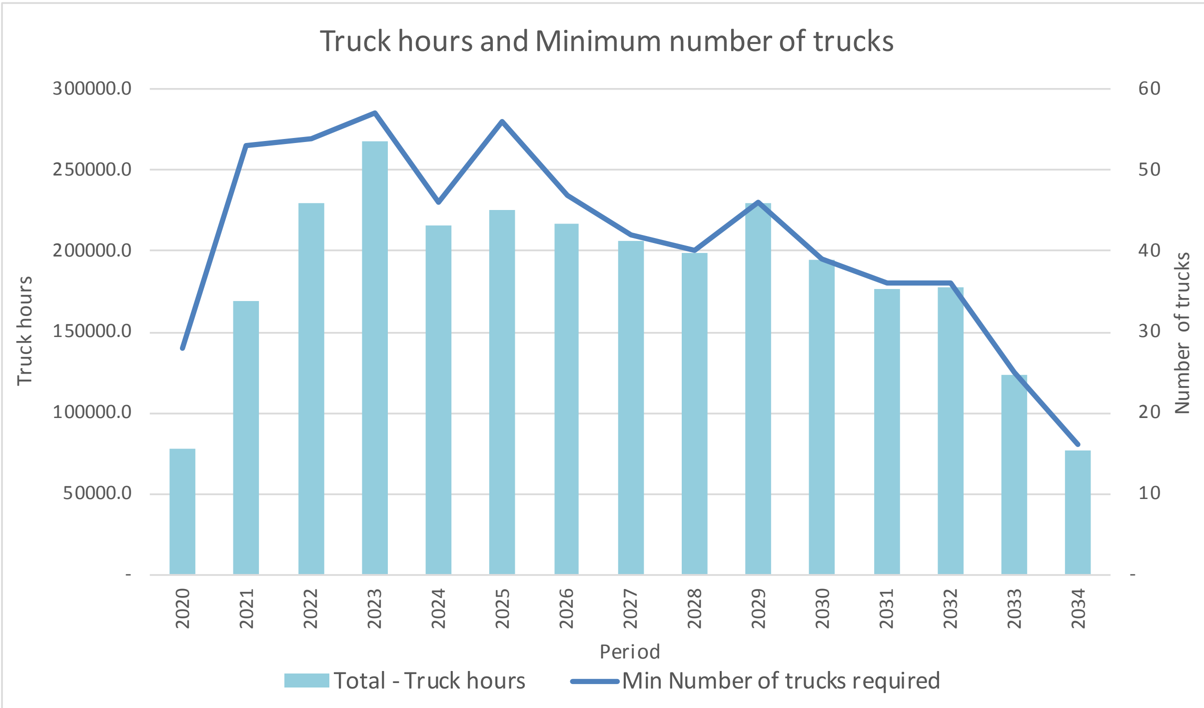
The average haul distances used in the schedule are summarized in Table 16.14.

Table 16.14 Average Haul Distances by Material Movement and Period

	Unit	Average	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Mine to Process	km	4.3	-	4.7	3.5	4.4	6.0	5.7	5.0	4.1	3.5	4.3	3.4	4.0	4.7	3.8	4.7
Mine to Stockpile	km	2.0	1.8	1.7	1.6	2.1	2.4	2.0	1.8	2.1	2.3	1.5	2.3	3.4	1.8	2.9	-
Stockpile to Process	km	3.4	-	-	-	4.2	4.0	3.8	3.6	4.0	4.2	4.1	4.3	4.3	1.8	2.9	2.9
Mine to TSF (oxide)	km	4.7	4.8	5.0	3.8	4.4	5.8	6.7	5.3	3.4	4.0	6.1	4.9	3.1	6.1	6.3	-
Mine to Barriers, WRDs (oxide and sulphide)	km	2.3	2.7	2.4	2.2	1.8	2.1	2.2	2.4	2.9	2.6	2.5	2.7	1.9	1.5	2.2	2.9

The resulting truck hours and number of trucks required are shown in Figure 16.16. The minimum number of trucks was estimated based on 5,000 operating truck hours/year and does not include spares. AMC completed a detailed haulage analysis for each mining area using Haul Infinity.

Figure 16.16 Truck Hours and Minimum Truck Requirements



The mining fleet recommended by AMC is listed in Table 16.15.

Table 16.15 Recommended Mining Fleet

Equipment	Number Required during Oxide only periods	Number Required during mining of both oxides and sulphides
Oxide Excavator e.g. Komatsu PC850	15	8
Sulphide Excavator e.g. Komatsu PC 1250		5
Oxide Haul Truck (30 t Rigid highway Truck) e.g. Sinotruck Howo	53	36
Sulphide Haul Truck (50 t rigid highway truck) e.g. Volvo FMX 10x4		11
Bulldozer e.g. CAT D7 or D8R	5	5
Wheel dozer e.g. CAT 824	3	3
Front-End Loader e.g. Komatsu WA600	2	3
Drills	3	3
Water Truck (25,000l)	3	3
Grader e.g. CAT 140M	4	4
Compactor e.g. BOMAG BW 214	1	3
Lowbed Transport Truck	1	2
Crew Bus (50 person)	3	3
Fuel Truck (15,000l)	1	1
Crane (40t)	1	1
Tyre Vehicle	1	1
Service Vehicle	1	1
Portable Light Towers	20	20
Light Vehicles e.g. Toyota Hilux	6	6
Mine Rescue Truck	1	1

In May 2018, four quotations were received from local contractors with different oxide mining fleets, which specified that truck numbers were in line with AMC's assessment. In April 2019, quotations were received from three local contractors providing costs and fleet specifications for mining the sulphide material. The final specification of equipment will take place during contractor negotiations in detailed engineering.

16.4.8 Fuel Consumption

Fuel requirements for the mining equipment have been estimated based on the mine production schedule and fuel consumption assumptions listed in Table 16.16.

Table 16.16 Parameters for Fuel Consumption

Equipment	Units	Value
Oxide Excavator e.g. Komatsu PC850	L/hr	57.0
Sulphide Excavator e.g. Komatsu PC 1250	L/hr	76.6
Haul Truck (30-50 t Rigid highway Truck)	L/hr	28.0

The annual fuel requirements for mobile equipment are summarized in Table 16.17. Fuel for support equipment has been estimated as 15% of total fuel requirements for excavators and trucks.

Table 16.17 Fuel Requirements for Mining Fleet

Equipment	Unit	Total	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Trucks	Ml	78.0	2.2	4.7	6.4	7.5	6.0	6.3	6.1	5.8	5.6	6.4	5.5	4.9	5.0	3.5	2.2
Excavators	Ml	42.4	1.1	2.6	4.0	3.9	3.8	3.6	3.3	3.2	3.2	3.2	3.3	2.8	2.5	1.6	0.6
Support Equipment	Ml	18.1	0.5	1.1	1.6	1.7	1.5	1.5	1.4	1.3	1.3	1.5	1.3	1.2	1.1	0.8	0.4
Total	**Ml**	**138.5**	**3.8**	**8.4**	**12.0**	**13.1**	**11.3**	**11.4**	**10.7**	**10.3**	**10.0**	**11.1**	**10.0**	**8.9**	**8.6**	**5.8**	**3.1**

16.4.9 Explosives Consumption

Drill and blast consumables are based on the production schedule and are provided in Table 16.18.

It is estimated that the mine will consume between 1,500 – 3,000 t/year of bulk emulsion during peak sulphide production years. It is estimated that 4 to 5 blasts per week will be required during this period.

Table 16.18 Drill and Blast Requirements

Blast Requirements	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Blasted waste movement/day	(t/day)	0	62	2,242	2,845	24,596	13,473	14,625	7,573	8,209	11,023	12,995	3,962	407	81	54
Blasted ore movement/day	(t/day)	0	11	1,221	1,374	6,123	4,816	6,666	6,336	3,357	5,473	4,740	1,403	248	87	47
Waste blasts/week	(#)	0	1	1	1	5	3	3	2	2	2	3	1	1	1	1
Ore blasts/week	(#)	0	1	1	1	2	2	2	2	1	2	2	1	1	1	1
Holes per waste blast	(#)	0	1	43	55	111	99	108	86	83	124	96	87	7	1	1
Holes per ore blast	(#)	0	0	30	35	91	70	98	96	89	82	70	38	6	2	1
Emulsion/waste blast	(kg/blast)	0	26	3,432	4,447	9,270	8,275	9,010	7,178	6,706	10,334	8,010	7,238	502	83	50
Emulsion/ore blast	(kg/blast)	0	7	2,439	2,877	7,598	5,851	8,208	8,060	7,414	6,936	5,896	3,164	465	151	85
Total Emulsion/year	(t/yr)	0	2	305	381	3,200	1,899	2,259	1,585	1,083	1,796	1,863	541	50	12	7
Downhole detonators	(#/yr)	0	33	3,783	4,698	38,228	22,771	27,049	18,875	13,253	21,437	22,301	6,508	643	158	92
Surface detonators	(#/yr)	0	33	3,783	4,698	38,228	22,771	27,049	18,875	13,253	21,437	22,301	6,508	643	158	92
Boosters	(#/yr)	0	33	3,783	4,698	38,228	22,771	27,049	18,875	13,253	21,437	22,301	6,508	643	158	92

Average blast size and frequency based on the yearly production schedule.

Explosives will be stored on site. At least 4 weeks of stocks is recommended at any given time. Bulk emulsion and blast consumables storage requirements are given in Table 16.19.

Table 16.19 Onsite Explosive Storage Requirements

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Bulk Emulsion	(kg)	0	131	23,484	29,298	246,183	146,113	173,784	121,907	83,306	138,159	143,290	41,607	3,866	934	538
Downhole detonators	(#)	0	3	291	361	2,941	1,752	2,081	1,452	1,019	1,649	1,715	501	49	12	7
Surface detonators	(#)	0	3	291	361	2,941	1,752	2,081	1,452	1,019	1,649	1,715	501	49	12	7
Boosters	(#)	0	3	291	361	2,941	1,752	2,081	1,452	1,019	1,649	1,715	501	49	12	7

16.4.10 Surface Haul Road Development

The surface haul roads from the trunk road to pits will be developed as access is required based on the production schedule. Table 16.20 summarizes the haul road requirements.

Table 16.20 Surface Haul Road Schedule

	Unit	Total	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Mining Block 1	km	10.9	4.6	4.0	1.8	-	-	-	0.3	0.2	-	-	-	-	-	-	-
Mining Block 2	km	5.7	-	1.7	3.2	0.6	-	0.1	-	-	-	-	-	-	0.1	-	-
Mining Block 3	km	10.7	-	0.8	5.8	1.3	1.8	0.8	0.2	-	-	-	-	-	-	-	-
Mining Block 4	km	5.7	-	1.3	1.3	2.3	-	0.4	0.4	-	-	-	-	-	-	-	-
Mining Block 5	km	0.7	-	-	0.7	-	-	-	-	-	-	-	-	-	-	-	-
Mining Block 6	km	1.4	-	-	-	1.4	-	-	-	-	-	-	-	-	-	-	-
Total	km	35.0	4.6	7.7	12.9	5.6	1.8	1.3	0.8	0.2	-	-	-	-	0.1	-	-

16.4.11 Mining Personnel

Orezone will maintain an Owner's Team on site which will be responsible for site management, contractor management, grade control and mine planning. The Owner's Team will be expanded from 52 to 60 as the operations move from oxide material mining only to oxide and sulphide mining operations. The required personnel are listed in Table 16.21 and Table 16.22.

Table 16.21 Owner's Team Personnel – Oxide

Position	Expat/National	Count
Mine Management & Technical Services		
Technical Services Manager	Expat	1
Chief Engineer	National	1
Senior Engineer	National	1
Intermediate Engineer	National	3
Junior Engineer	National	2
Chief Surveyor	National	1
Senior Surveyor	National	1
Surveyor/Rodman	National	2
Chief Geologist	National	1
Senior Geologist	National	2
Intermediate Geologist	National	2
Junior Geologist	National	4
Grade Control Technician	National	6
Secretary	National	2
Mine Production		
Mine Superintendent	Expat	1
Mine Shift Foreman	National	4
Leading Hand	National	4
Pump Foreman	National	1
Secretary	National	1
Pump/General Service Crew	National	6
Labourer	National	6
Subtotal Labour		52

Table 16.22 Owner's Team Personnel – Oxide and Sulphide

	Expat/National	Count
Mine Management & Technical Services		
Technical Services Manager	Expat	1
Chief Engineer	National	1
Senior Engineer/Drill & Blast Engineer	National	3
Intermediate Engineer	National	3
Junior Engineer	National	2
Chief Surveyor	National	1
Senior Surveyor	National	1
Surveyor/Rodman	National	6
Chief Geologist	National	1
Senior Geologist	National	2
Intermediate Geologist	National	2
Junior Geologist	National	4
Grade Control Technician	National	8
Secretary	National	2
Mine Production		
Mine Superintendent	Expat	1
Mine Shift Foreman	National	4
Leading Hand	National	4
Pump Foreman	National	1
Secretary	National	1
Pump/General Service Crew	National	6
Labourer	National	6
Subtotal Labour		**60**

The Mine Management & Technical Services team will be responsible for ensuring grade control procedures are completed ahead of mining and generate long, medium and short-term mine plans. The technical staff will provide visual spotting, survey and reconcile mining to the mine plan.

The Mine Production Personnel will be responsible for oversight of contractor in pit mining activities as well as providing ancillary support and management of pit de-watering.

Mining contractors will provide onsite administrative and management personnel along with required equipment operators, mechanics, welders, electricians, and labourers sourced from the local communities.

AMC estimated manning requirements based on equipment numbers at peak requirement for the oxide operation and oxide and sulphide operations and the following assumptions:

- Four work crews working a 7 days on 7 days off roster and 12-hour shifts for operators.
- 5 days on 2 days off (residential) for national management staff.

The contractor manning estimate is provided in Table 16.23.

Table 16.23 Contactor Manning Estimate

Mining	**Oxide**	**Oxide and Sulphide**
Excavator (4.5 m^3 bucket) e.g. Komatsu PC850	60	32
Excavator eg PC 1250	0	20
Haul Truck (30 t Rigid highway Truck) e.g. 30 t Sinotruck Howo	212	144
Haul Truck (50 t rigid highway truck)	0	44
Bulldozer e.g. CAT D7 or D8R	20	20
Wheel dozer e.g. CAT 824	6	6
Front End Loader e.g. Komatsu WA600	8	12
Water Truck (25,000l)	12	12
Grader e.g. CAT 140M	16	16
Compactor e.g. BOMAG BW 214	2	6
Lowbed Transport Truck		
Crew Bus (50 person)	12	12
Fuel Truck (15,000l)	2	2
Clerk	4	4
Shot crew and supervision		6
Drillers	12	12
Dewatering crew	8	8
Pit dispatcher	4	4
General worker	8	8
Mining supervisors and leading hands	8	8
Maintenance		
Maintenance Manager	1	1
Clerk	4	4
Maintenance Superintendent	1	1
Maintenance Warehouse Supervisor	1	1
Store-person	8	8
Maintenance Planner	3	3
HV Electrician	8	8
Maintenance supervisor	4	4

Mining	Oxide	Oxide and Sulphide
Tyre Fitter	8	8
Welder	12	12
Fitter	134	122
Contractor G&A		
Project Manager	1	1
Assistant Project Manager	1	1
Trainer	2	2
HSE	5	5
Total	587	557

16.5 Mineral Reserve Estimate and Mining Risks & Opportunities

In AMC's opinion, the key risks to the Mineral Reserve Estimate are:

- Potential for greater than expected inflows of water from either surface or groundwater sources.
- Potential for grade control drilling to alter interpretations of mineralization.

The key opportunities are:

- Potential of design optimization within free dig pits.
- Potential production schedule flexibility with large number of pits available.

The risks and opportunities identified by AMC are highlighted in the following sections:

16.5.1 Risks

- Resource block model does not reconcile with grade control drilling.
  - Mitigation: Prior to commencement of operations, AMC recommends that grade control drilling is conducted in test areas to reconcile against the Mineral Resource and Reserve Estimates. The OCR construction should be used as a test case area to trial the proposed grade control procedures and make any required adjustments prior to the commencement of operations.
- Instability in saprolite slopes due to high pore pressures.
  - Mitigation: Golder has recommended installing Vibrating Wire Piezometers (VWPs) where there is potential for high pore pressure in saprolite slopes. If high pore pressures are encountered, de-watering drill holes may be required.

- Higher than expected drill and blast requirements.
  - Mitigation: Currently, detailed multi-element analysis by the Orezone team can accurately pinpoint the upper to lower transition contact. With the size of the Project, if non-free-dig material is unexpectedly encountered, the mine plan has the flexibility to be adjusted to move to a different mining area, allowing the sulphide mining fleet to continue mining into the fresh rock with conventional drill and blast.
- Excessive ore losses when mining narrower ore bodies in fresh rock.
  - Mitigation: In addition to a well-planned grade control programme, AMC recommends that Orezone investigate the use of blast movement analysis equipment to maintain selectivity within the fresh rock units.
- Muddy conditions during the wet season leading to loss of productivity.
  - Mitigation: Rain episodes during the wet season typically occur as a single event during the day and AMC has built in a 1 hour/day loss of production time into the mine schedule. Haulage will be halted during rain events to prevent damage to mining areas and roads. The mining contractor will maintain active haul roads with appropriate ancillary fleet.
- Flooding of pits due to heavy rainfall event during the wet season.
  - Mitigation: There is flexibility in the mine plan to switch mining areas and allow time to pump the flooded workings should the event occur. Mine planning must be cognizant of the wet season and dry season to minimize disruption to the operations. Previously mined-out areas may be allowed to flood to provide temporary water storage for the mining operations.
- Flooding due to intersecting flood plains.
  - Mitigation: AMC recommends that a monitoring programme is set up ahead of mining operations in the vicinity of flood plains to understand their potential impact on the current mine design and alter the mine plan if required. Mining within the floodplains (Restricted Zones) will be during the dry season only and appropriate buffers will be maintained adjacent to working pit areas.
- Inability to supply key project technical personnel.
  - Mitigation: Orezone already has key geological staff trained with a firm understanding of the lithologies, weathering horizons and style of mineralization. There are numerous similar projects in West Africa. In addition, because the Project is located approximately 85 km from the capital Ouagadougou, this will be attractive to technical personnel.

16.5.2 Opportunities

- Optimization of surface haul roads.

    - The topography is generally flat without dense vegetation hence changes can be easily incorporated into the surface haul road network to minimize road construction requirements. For some smaller satellite excavations, temporary haul road solutions may be adopted.

- Optimization of in-pit ramps.

    - When mining in free-dig material, in-pit ramps can be readily re-located to shorten haul distances. When the contractor fleet is finalized, ramps may be narrowed to save on construction and maintenance costs.

17.0 RECOVERY METHODS

17.1 Process Design

The process plant design is based on a robust metallurgical flowsheet developed for optimum recovery while minimizing initial capital expenditure and life of mine operating costs. The flowsheet is based on unit operations including crushing, milling, Carbon-in-Leach (CIL) leaching, Zadra elution, gold electrowinning and carbon regeneration that are well proven in the industry.

Process plant feed comprises three main ore types – oxide, transition and sulphide (fresh or unweathered) material. Oxide and upper transition ores are mined by 'free digging' methods with no drill and blast required in the pits, while lower transition and sulphide ores will be mined by drill and blasting as required.

The process plant design has been based on a nominal capacity of 5.2 Mtpa. Initial plant feed will consist of the soft oxide and upper transition ore types only processed through a grinding and CIL circuit designed for the softer ores. For these materials crushing is not required and the material has low grind energy requirements.

After the third year, plant feed will include more competent lower transition and fresh ores and these will be crushed and ground through a separate crushing and grinding circuit feeding a leach circuit designed to provide the additional leach residence time required by these ores. The partially leached sulphide slurry will then be combined with the oxide feed to the CIL circuit displacing approximately 2.2 Mtpa of the lower grade oxide ores which will be processed later in the mine life.

For simplicity, the portion of the plant required for processing only the oxide and upper transition ore will be referred to as the "oxide plant" or "oxide circuit", and the portion of the plant required for the inclusion of lower transition and fresh ore will be referred to as the "sulphide plant" or "sulphide circuit".

The plant design is considered appropriate for a project with a 14-year process plant operating life. The key criteria for selection of equipment type are suitability for duty, reliability and ease of maintenance with price then being a major criterion for selection between vendors of broadly similar equipment. The plant layout provides ease of access to all equipment for operating and maintenance requirements while maintaining a layout that will facilitate construction progress in multiple areas concurrently.

The key design criteria for the plant are:

- June 2021 to December 2023 – Plant throughput of 5.2 Mtpa with an ore blend of 85% w/w oxide and 15% w/w upper transition.
- Oxide grinding and CIL circuit availability of 91.3%.
- From January 2024 – Combined plant throughput of 5.2 Mtpa, of which 3.0 Mtpa is the oxide and upper transition ore blend, and 2.2 Mtpa is lower transition and sulphide ore blend introduced through the new sulphide circuit.

- Sulphide crushing plant availability of 70% (design minimum).

- Sulphide grinding and leach circuit availability of 91.3%, supported by the incorporation of surge capacity and standby equipment where required.

- Both circuits will be provided with sufficient automated plant control to minimize the need for continuous operator interface and allow manual override and control if and when required.

Study design documents have been prepared by incorporating engineering design criteria and key metallurgical design criteria derived from the results of the metallurgical testwork carried out on Bomboré ores. The metallurgical testwork programs are summarized in Section 13 of this Technical Report.

17.1.1 Selected Process Flowsheet

The oxide plant will be designed for a 5.2 Mtpa throughput in order to accommodate the initial years of mine schedule where only oxide and upper transition ore will be processed. The sulphide plant portion will be added after the third year to allow the treatment of lower transition and fresh ore. The sulphide plant will process 2.2 Mtpa of sulphide ore, with oxide throughput reduced to 3.0 Mtpa to maintain an overall plant throughput of 5.2 Mtpa. A life of mine milling and gold production schedule is provided in Section 24 of this Technical Report.

As the oxide and upper transition material is fine grained generally friable and essentially free of quartz the oxide circuit does not include a crusher. The trucks transporting the oxide plant feed will drive across a static grizzly while rear-dumping the ore. The grizzly will be kept clear, as necessary, by a front-end loader. The saprolitic ore will be broken further by chains on the ROM bin discharge apron feeder and will be fed, by conveyor, into a ball mill for slurrying and grinding the small coarse fraction. Similar flowsheets are successfully used elsewhere in West Africa and South and Central America for fine and friable saprolitic material.

The treatment plant design incorporates the following process unit operations:

Oxide Plant

- ROM ore fed through a static grizzly to a surge bin.

- Apron feeder and conveyor feed to the milling circuit.

- A single stage ball mill, in closed circuit with hydrocyclones, to produce a P_{80} grind size of 125 µm.

- A hydrocyclone pack with overflow slurry density of 40% w/w solids for direct feed to the leach tanks.

- A leaching circuit with one leach and seven CIL tanks to achieve the required 24 hours of residence time for optimum leach recovery.

- A tailings thickener for cyanide and water recovery.

- Loaded carbon acid wash and pressure Zadra elution circuit with gold electrowinning and recovery to doré.

- Carbon regeneration kiln to remove organic foulants from the carbon and reactivate the adsorption sites on the activated carbon.

Sulphide Plant

- Primary crushing with a jaw crusher to produce a P_{80} of 105 mm.

- Mill feed surge bin that overflows to an approximately 4,900 t stockpile to provide 18 hours of surge capacity.

- The grinding circuit is a SSAG type, which consists of a closed-circuit single stage SAG mill with pebble recycling to produce a final P_{80} of 75 µm. Provision has been made to install a pebble crusher in the future should, additional throughput capacity be desired.

- A hydrocyclone pack with an overflow slurry density of 25% w/w solids to maintain efficient particle size separation.

- A pre-leach thickener to increase leach slurry density, which in turns also minimizes leach tank volume and reduce overall reagent consumptions.

- A leach circuit with a pre-oxygenation tank followed by three leach tanks to provide 24 hours of residence time for optimum recovery. Partially leached slurry is pumped to the oxide plant for further processing in its CIL circuit providing an overall leach duration of 48 hours for the sulphide ores.

Process block flow diagrams depicting the unit operations incorporated in the oxide, sulphide and combined circuits are presented in Figures 17.1, 17.2 and 17.3. Plan and isometric views of the process plant are provided in Figures 17.4 and 17.5.

Figure 17.1 Overall Process Flow Diagram for Oxide Circuit

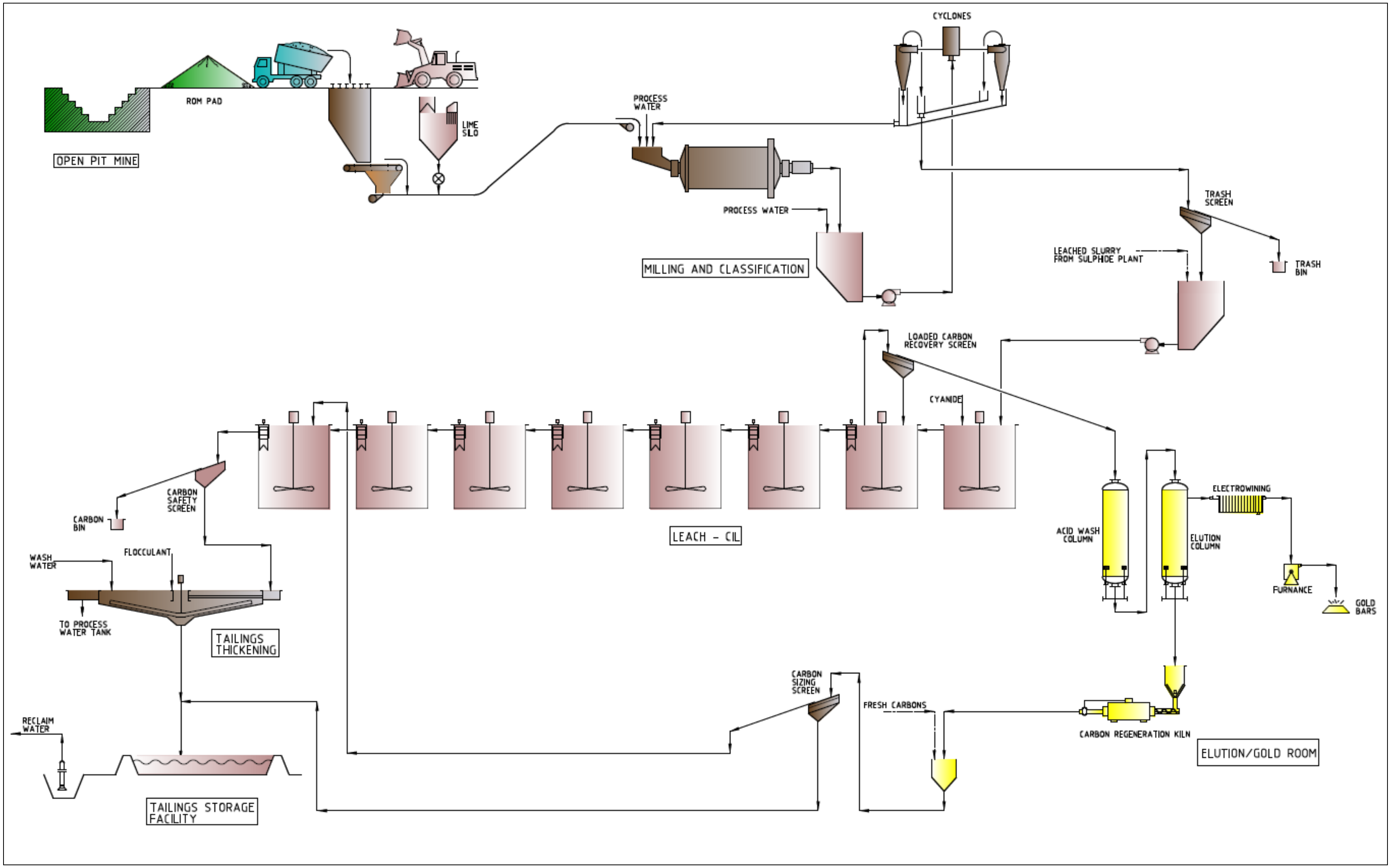


Figure 17.2 Overall Process Flow Diagram for Sulphide Circuit

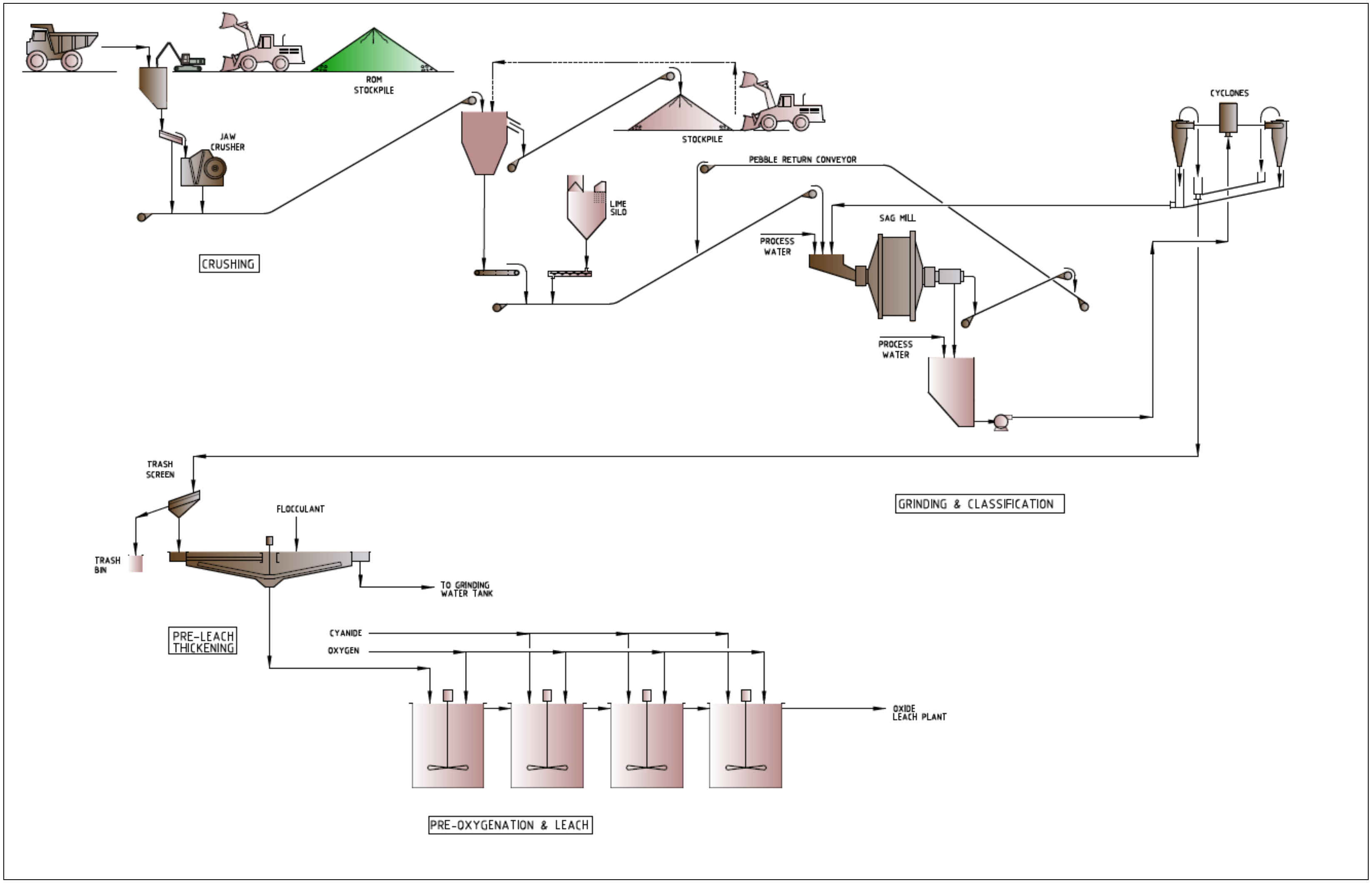


Figure 17.3 Process Flow Diagram for Combined Circuits

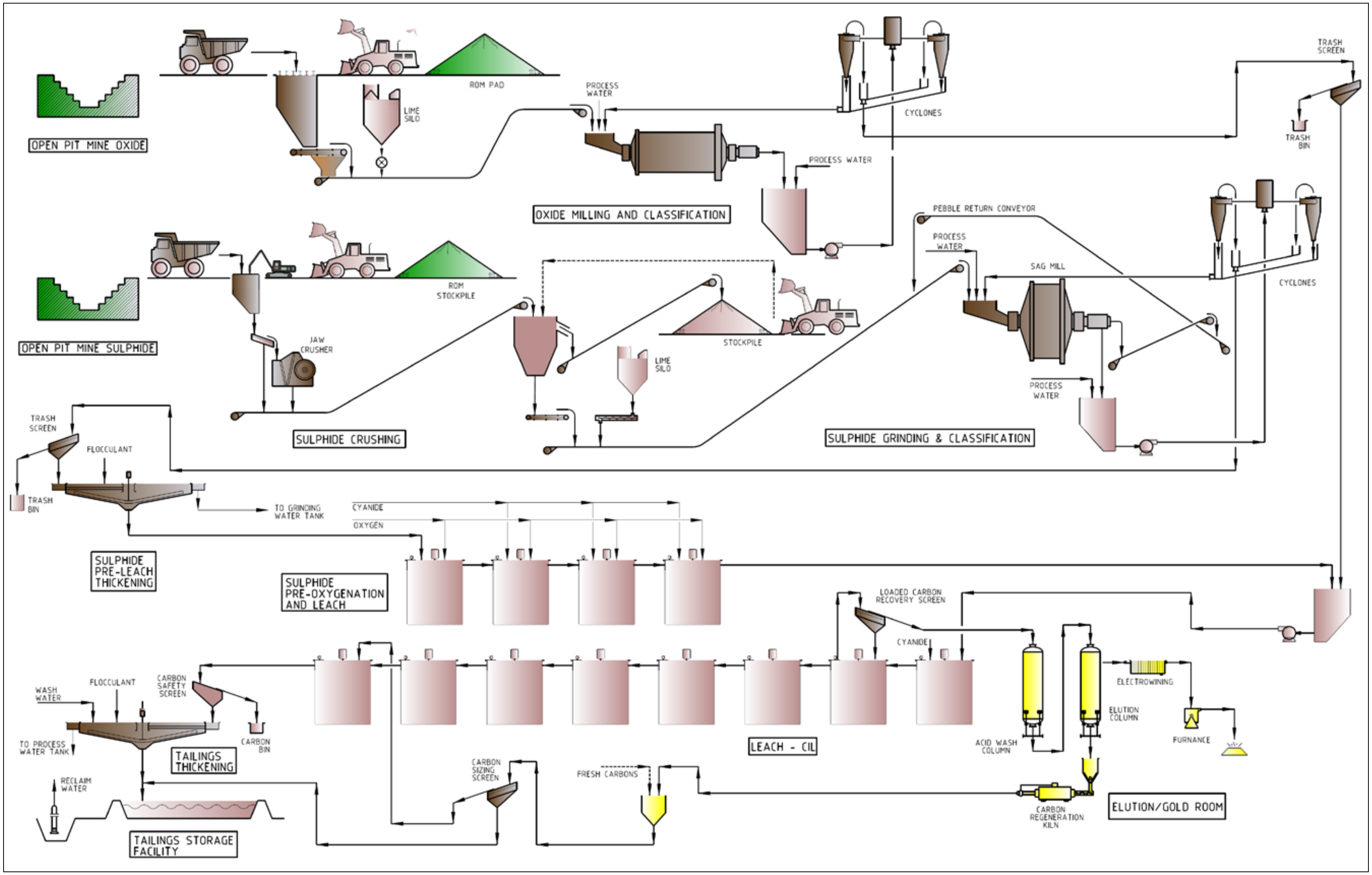


Figure 17.4 Combined Circuits Plan View

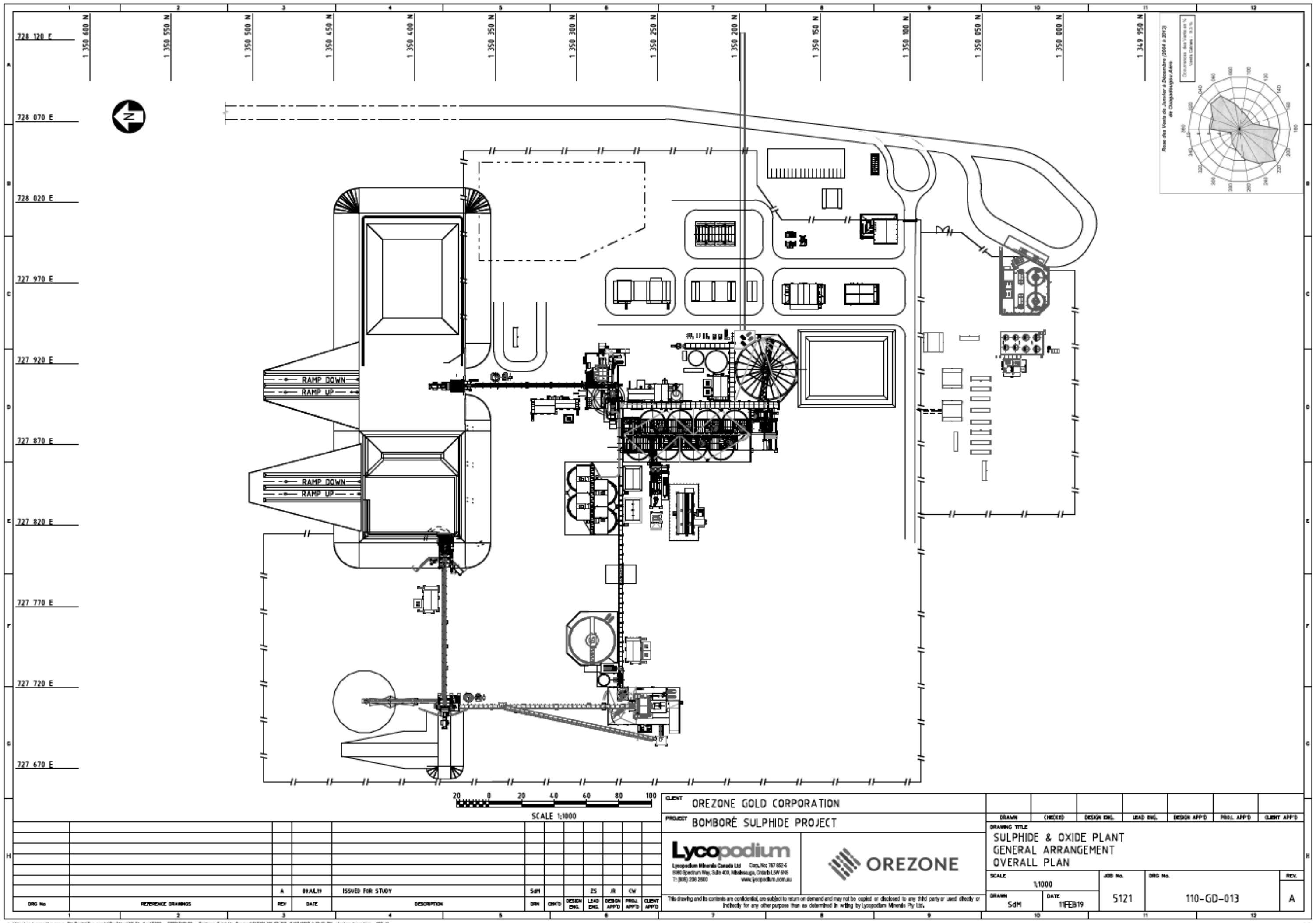
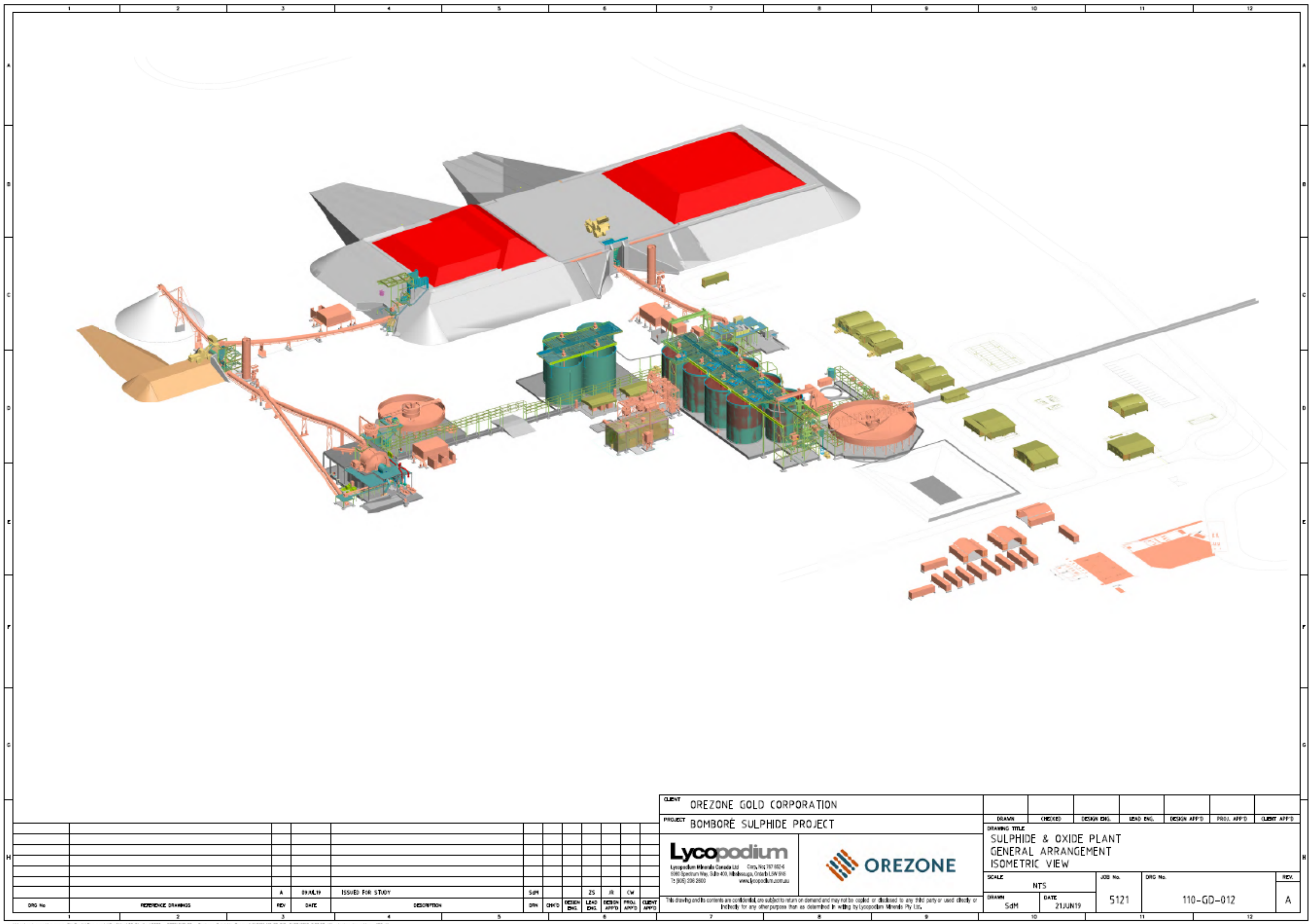


Figure 17.5 Combined Circuits Isometric View



17.1.2 Key Process Design Criteria

Key process design criteria for the oxide plant are summarized in Table 17.1.

Table 17.1 Summary of Key Process Design Criteria for Oxide Plant

	Units	Design	Source
Plant Throughput	tpa	5,200,000	Orezone
	tpd	14,247	Calculated
Design Ore Blend - Oxide	% w/w	85	Mine Schedule
- Upper Transition	% w/w	15	Mine Schedule
Design Head Grade - Gold	g/t Au	1.0	Orezone
Overall Gold Recovery[1]	% w/w	92.0	Lycopodium
Ore Specific Density	t/m^3	2.8	Testwork
Ore Bulk Density	t/m^3	1.65	Orezone
Angle of Repose	degrees	37.0	Lycopodium
Plant Availability	%	91.3	Lycopodium
Crushing Work Index (CWi, average)	kWh/t	7.7	Testwork
Bond Ball Mill Work Index (BWi, average)	kWh/t	4.8	Testwork
Bond Abrasion Index (Ai, average)		0.0314	Testwork
Grind Size (P_{80})	µm	125	Lycopodium
CIL Circuit Residence Time	hrs	24	Testwork
CIL Slurry Density	% w/w solids	40.4	Lycopodium
Number of Leach Tanks		1	Lycopodium
Number of CIL Tanks (Stages)		7	Lycopodium
Air Addition Rate	$Nm^3/h/m^3$	0.15	Lycopodium
Tails Thickener Solids Loading	$t/m^2.h$	0.60	Testwork
Sodium Cyanide Addition	kg/t ore	0.28	Lycopodium
Lime Addition[2]	kg/t ore	2.07	Lycopodium
Sodium Hydroxide Addition	kg/t ore	0.033	Lycopodium
Elution Circuit Type		Pressure Zadra	Lycopodium
Elution Circuit Size	t	10.0	Lycopodium
Frequency of Elution	strips/wk	8.4	Lycopodium
Kiln Capacity	kg/h	600	Lycopodium

1. *At design head grade of 1.00 g/t Au including 0.08 g/t Au total losses.*
2. *Lime addition based on 90% w/w CaO content of supplied quicklime.*

Key process design criteria for the sulphide plant are summarized in Table 17.2.

Table 17.2 Summary of Key Process Design Criteria for Sulphide Plant

	Units	Design	Source
Plant Throughput	tpa	2,200,000	Orezone
	tpd	6,027	Calculated
Ore Blend - Lower Transition	% w/w	24	Mine Schedule
- Fresh	% w/w	76	Mine schedule
Design Head Grade - Gold	g/t Au	1.25	Orezone
Overall Gold Recovery[1]	%	84.6	Lycopodium
Ore Specific Density	t/m^3	2.8	Testwork
Ore Bulk Density	t/m^3	1.65	Orezone
Angle of Repose	degrees	37.0	Lycopodium
Crushing Plant Availability	%	70.0	Lycopodium
Plant Availability	%	91.3	Lycopodium
Crushing Work Index (CWi, average)	kWh/t	15.1	Testwork
Bond Ball Mill Work Index (BWi, average)	kWh/t	14.5	Testwork
Rod Mill Work index (RWi, average)	kWh/t	16.2	Testwork
Bond Abrasion Index (Ai, average)		0.258	Testwork
Grind Size (P_{80})	µm	75	Testwork
Pre-leach Thickener Solids Loading	$t/m^2.h$	0.5	Orezone
Leach Circuit Residence Time	hrs	24	Testwork
Leach Slurry Density	% w/w solids	49.7	Lycopodium
Number of Pre-oxygenation Tanks		1	Testwork
Number of Leach Tanks		3	Lycopodium
Oxygen Addition in Leach	Nm^3/t ore	0.80	Lycopodium
Sodium Cyanide Addition	kg NaCN/t ore	0.78	Testwork
Lime Addition[2]	kg/t	1.03	Testwork

Notes:

1. *At design head grade of 1.25 g/t Au including 0.19 g/t Au total losses.*
2. *Lime addition based on 90% w/w CaO.*

17.2 Oxide Plant Process Description

17.2.1 Run of Mine Ore Receipt and Mill Feed

ROM oxide and upper transition ore from the open pits will be transported to the plant by 30 t to 50 t capacity rear dump tip trucks. The trucks used are anticipated to be triple axle road-legal trucks and not mine haul trucks. The trucks will tip directly into the feed bin by driving over a horizontal grizzly above the bin and tipping while moving across the grizzly. Surplus ore will be stockpiled on the ROM pad and fed to the bin by front-end loader (FEL) when truck deliveries are interrupted. The FEL will also be used to clear any build-up of material from the grizzly should this occur. The ore stockpile will facilitate ore blending to ensure a uniform feedstock to the plant.

The feed bin will have a live capacity of approximately 140 t (equivalent to 13 minutes' plant feed to the ball mill). A static grizzly (200 mm slot), mounted above the feed bin, will prevent the ingress of oversize material. Ore will be withdrawn from the feed bin, by a variable speed apron feeder, discharging directly onto the mill feed conveyor.

The mill feed conveyor will be fitted with a weightometer, to monitor and control feed to the mill by adjusting the apron feeder variable speed drive.

Quicklime will be delivered in bulk, stored in a lime storage silo and added directly onto the mill feed belt conveyor, using a variable speed lime screw feeder. The lime storage silo will have a capacity of 70 t, which is slightly over 2 days of consumption. To mitigate the risk of lime shortages a small inventory of bagged lime will be retained on site for use in emergencies. Lime addition will be controlled by a pH meter in the leach circuit and by the mill feed rate as measured by the conveyor weightometer.

Spillage generated within the conveying area will be recovered and transported to the ROM pad for feeding back into the plant.

17.2.2 Grinding and Classification

The grinding circuit consists of a single, variable speed, ball mill with a 3.2 MW variable speed drive. The mill is 5.19 m in diameter and has a 7.44 m equivalent grinding length (EGL). The ball mill will operate in closed circuit with hydrocyclones. Pebbles will be rejected from the mill and placed in a scats bunker. It is anticipated that pebble rejection will be minimal and the geological drilling suggests that any quartz pebbles will have a low gold assay. Should it be found that the pebbles contain enough gold to make treatment a viable option they will be stockpiled and processed through the sulphide circuit. The pebbles are gold barren, they will be used for stabilizing slopes and landscaping.

The mill has been designed to be capable of achieving the required grind size with a mill feed blend of 85% w/w oxide and 15% w/w upper transition ore, which is similar to the life of mine average blend. The installed variable speed drive will be used to control the mill power draw to achieve the required leach feed size when treating different blends. The mill will operate with a nominal 25% by volume ball charge.

Process water will be added to the ball mill feed box, to wash the new ore into the mill and to adjust the mill pulp density. The slurry will discharge the mill through a 22 mm trommel screen to the cyclone feed pump box.

Grinding media (80 mm balls) will be added to the mill through the mill media feed chute using a ball kibble.

Undersize from the ball mill trommel screen will gravitate to the cyclone feed pumpbox from where it will be pumped to the classifying hydrocyclones by the cyclone feed pumps (1 duty, 1 standby). Process water will be added to the pumpbox to control the hydrocyclone performance and overflow density to the leach circuit. Cyclone underflow will flow by gravity back to the ball mill feed box for further grinding. The hydrocyclone overflow will gravitate, via a trash screen, to the leach feed pumpbox where the slurry is then pumped to the leach feed distribution launder. The trash removed (wood chips, etc.) will be discharged to a trash bin for disposal. A "Y" piece connection will be installed on the pipeline to the leach feed distribution launder to provide a sample point location.

The leach feed pumpbox will receive the leached slurry from the sulphide plant when the expansion is complete.

Fine spillage within the grinding circuit sump area will be managed by area sump pumps. Coarse spillage will be removed and recycled using a Bobcat or similar equipment.

17.2.3 CIL Circuit

Trash screen underflow will be pumped from the leach feed pumpbox to the leach feed distribution launder and distributed to the pre-CIL leach tank. If the pre-CIL leach tank is offline, the slurry can be diverted to the first CIL tank, via an internal dart plug distribution system.

The CIL circuit consists of one pre-CIL leach tank and seven CIL tanks, each mechanically agitated and operating in series. The tanks are each 15.8 m diameter by 18.6 m high providing a 24-hour leach residence time at the design feed rate of 1,190 m^3/h at 40.4% w/w solids. The first CIL tank will operate with a carbon concentration of 15 g/L and subsequent CIL tanks will operate with a carbon concentration of 10 g/L.

Sodium cyanide as a 20% w/w solution will be added to the circuit from the cyanide dosing pumps. The primary cyanide dosing point will be at the first pre-CIL leach tank and first CIL tank, with further addition points located at the second and third CIL tanks. The operating pH of the circuit will be maintained above 10.3 by the quicklime addition to the mill feed conveyor to maintain the protective alkalinity of the circuit and prevent the generation of gaseous hydrogen cyanide.

To aid with gold dissolution, low pressure air will be added to the circuit to maintain oxygen saturation in the slurry. Air will be supplied from low pressure air compressors and distributed down the agitator shafts.

Slurry will flow sequentially through the leach/CIL circuit tanks driven by the up-pumping impeller that is part of each inter-stage screen, and the activated carbon will be retained in each of the CIL tanks by the inter-stage screen. Carbon will be advanced through the CIL circuit, counter current to the slurry flow, using recessed impeller vertical pumps.

On a daily basis, a complete batch of loaded carbon from the first CIL tank, will be pumped with slurry to feed the loaded carbon screen, from which washed carbon will gravitate to the acid wash vessel. Undersize from the loaded carbon screen will gravitate to CIL tank #1 and can bypass to the pre- CIL tank.

To replace the recovered loaded carbon, regenerated carbon (or fresh carbon) will be pumped to CIL tank #7, from the carbon regeneration circuit.

Slurry discharging from the last CIL tank will gravitate to the carbon safety screen then to the tailings thickener.

Should the pre-CIL leach tank or any CIL tank be off-line for any reason, it will be bypassed using pneumatically actuated gate valves located within the CIL inter-stage launders, diverting slurry to the next downstream CIL tank.

The leach/CIL area will be serviced by four sump pumps. Sump pumps #1 and #2 are in close proximity to CIL tanks #2 and #3, hence will return the spillage back to those tanks. Sump pumps #3 and #4 are in close proximity to the back end of the CIL circuit, hence will return the spillage back to CIL tanks #6 and #7, respectively. The leach/CIL bund area will overflow in the case of emergencies to the event pond. The volume of the bunded area plus the event pond is sufficient to contain the contents of the largest vessel in the area, plus rainfall from a severe storm event, without overflowing to the environment.

Auxiliary equipment within the leach area will include:

- A cyanide analyzer for controlling cyanide addition rates.
- Hydrogen cyanide (HCN) monitors to warn personnel in the event of above-threshold levels of gaseous cyanide.
- A pH analyser to control the quicklime addition to the grinding circuit.

17.2.4 Desorption and Carbon Regeneration

A 10t pressure Zadra elution circuit was selected for gold recovery from the activated carbon.

The desorption circuit will have separate acid wash and elution columns allowing a strip to be completed within 15 hours. At a carbon gold loading of ~1,200 g/t and silver loading of ~675 g/t, the required peak daily carbon movement will be 12 t. A 10 t pressure Zadra plant, performing 8.4 strips per week would be equivalent to advancing 12 t of carbon per day.

Acid Wash

A cold acid wash sequence is required to remove accumulated calcium scale on the carbon surface. This process improves elution efficiency and has the beneficial effect of reducing the risk of calcium-magnesium slagging within carbon during the regeneration process. The acid wash column fill sequence will be initiated once the carbon recovery pump in CIL tank #1 starts pumping to the loaded carbon screen. Carbon will gravitate from the loaded carbon recovery screen directly into the acid wash column, with the underflow slurry from this screen gravitating back into the pre-CIL leach tank. Once the acid wash column is filled to the required level, the carbon fill sequence will be stopped.

The acid wash cycle will utilize a 3% w/w hydrochloric acid solution. Hydrochloric acid (32% w/w) will be diluted to 3% w/w by injecting a measured amount of acid into raw water as it fills the acid wash column. The carbon will be allowed to soak in the dilute acid for a period of half an hour.

Upon completion of the acid soak, the acid rinse cycle will be initiated; loaded carbon will be rinsed with water to displace acid solution and contaminants. Four bed volumes (4 BV) of water, at 2 BV/h, will be pumped through the column. Displaced solution from both the acid rinse and wash steps will pass through the acid wash discharge strainer before discharging to the carbon safety screen underpan.

The acid wash sequence will conclude with carbon being transferred to the elution column.

Elution

A 1% w/v caustic soda (NaOH) and 0.2% w/v sodium cyanide (NaCN) solution (barren eluate) from the strip solution surge tank will be pre-heated to 95°C using a diesel fired solution heater. Once pre-heated, the solution will be diverted through the heat recovery exchanger and fed to the elution column to commence stripping gold and silver from the loaded carbon. During this process the eluate will be further heated, under pressure, to 135°C.

Eluate will flow up through the carbon bed and out of the top of the column, passing through the recovery heat exchanger, and then fed to the electrowinning cells for gold and silver recovery. Barren eluate exiting the electrowinning cells will return to the strip solution tank for re-use in that or subsequent elution cycles.

A total of 15 bed volumes of eluate strip solution will be cycled through the column and electrowinning cells. 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 total strip solution which will be removed as a bleed from the barren solution return circuit, discharging to the leach feed distribution launder. Upon completion of the cool down sequence, the carbon will be transferred to the carbon regeneration kiln de-watering screen, by the stripped carbon transfer pump.

Carbon Regeneration

From the elution column the stripped carbon will be transferred 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 feed box.

Carbon will be withdrawn from the kiln feed hopper, by the kiln screw feeder, and fed directly to the carbon regeneration kiln, at a rate of 600 kg/h. The carbon will be heated within the diesel-fired, horizontal rotary kiln up to 750°C, to remove volatile organic foulants from the carbon surface and restore the carbon activity.

Re-activated carbon exiting the kiln will discharge directly to the carbon quench vessel, where it will be submerged in water and rapidly cooled. From the quench tank, carbon will be pumped by the barren carbon transfer pump to the carbon sizing screen. Sizing screen oversize will gravitate to CIL tank #7 with a bypass option to CIL tank #6. Sizing screen undersize will be discarded to the carbon safety screen feed box. Fresh carbon will be added to the CIL circuit via the carbon quench tank.

17.2.5 Electrowinning and Gold Room

Gold and silver recovery, from the pregnant eluate, will be achieved by electrowinning onto stainless steel cathodes. The electrowinning circuit will consist of two double-chamber electrowinning cell/rectifier combinations.

Once the elution pre-heating cycle has been completed, the electrowinning sequence will be initiated by diverting pregnant eluate solution to the electrowinning cells. The electrowinning cell discharge or barren eluate will be returned to the strip solution tank for re-use.

Upon completion of the electrowinning cycle, the cell covers will be removed and gold and silver sludge will be washed off the cathodes and the bottom of the cell with a hand-held high pressure washer. The gold and silver bearing sludge draining from the cell will then be filtered and dried.

Dried sludge will be mixed with a prescribed flux mixture (silica, sodium nitre, borax and soda ash), prior to charging into the diesel-fired gold furnace to produce slag and doré ingots. The doré ingots will be cleaned, assayed, stamped and stored in a secure vault ready for dispatch. The furnace slag produced will periodically be returned to the grinding circuit, via the ball mill feed box.

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

17.2.6 Tailings Disposal

Slurry from the CIL circuit will gravitate to the carbon safety screen. The carbon safety screen will recover most of the undersize carbon exiting the CIL circuit through the last inter-stage screen. The safety screen will also serve to prevent carbon loss to tailings in the event of a defective inter-stage screen.

The safety screen oversize will report to a fine carbon bin while the undersize will gravitate to the tailings thickener. An automatic slurry sampler, installed on the carbon safety screen feed, will collect a representative sample of the CIL tail stream. This sample will be assayed and the result will be used for circuit monitoring and metallurgical accounting.

Decant water from the TSF, and process water make-up, will be added to the tailings thickener feed box to 'wash' the CIL tailings and optimize the recovery of cyanide from the tailings stream and reduce the WAD (weak acid dissociable) cyanide level in the tailings to below 50 ppm.

Tailings thickener overflow will flow to the process water tank to be reused as plant process water. Tailings thickener underflow slurry will be pumped to the lined TSF for permanent storage. Due to the viscous nature of this slurry, dilution to 42% will be targeted for pumping to the TSF.

Residual cyanide in the plant tailings will degrade naturally through hydrolysis and UV irradiation in the TSF.

17.2.7 Event Pond

The process plant is designed to operate with zero discharge of process solutions to the local environment. To ensure compliance, the plant has been provided with a lined event pond designed to contain any foreseeable spillage event. The event pond combined with the bunded concrete areas within the plant perimeter, is designed to contain the run-off from a one in a 100-year storm event occurring simultaneously with the catastrophic failure of the largest slurry containing vessel within the plant site. Solution accumulating in the event pond will be returned periodically to the tailings thickener circuit.

17.2.8 Reagents Mixing and Storage

The major reagents utilized within the process plant are:

- Quicklime (90% w/w CaO content) for pH control.
- Sodium cyanide (NaCN) for gold dissolution and desorption.
- Caustic soda (NaOH) for desorption.
- Hydrochloric acid (HCl) for carbon acid washing.
- Flocculant for thickening.

In addition, fluxes (silica, nitre, borax and soda ash) will be required for smelting charge preparation. Antiscalant will also be used as required to reduce scaling in the process water distribution, carbon wash and stripping circuits. Lastly, sulphamic acid will be used to de-scale the elution heat exchangers as required.

Quicklime

Quicklime will be delivered to site in 30 t bulk tankers. Bulk tankers will be pneumatically off-loaded, using a blower, directly to the 70-t capacity lime silo. Quicklime will be withdrawn from the silo by a variable speed screw feeder and deposited directly onto the mill feed belt conveyor. A small inventory of bulk bags of lime will be retained on site for use in the case bulk deliveries are delayed. When using bagged lime, the bags will be discharged via the transfer hopper and to the silo by the blower and rotary valve.

Sodium Cyanide (NaCN)

Sodium cyanide will be delivered by the full container load (20 t) as double-bagged briquettes in standard 1,000 kg plywood boxes. Cyanide will not be removed from the shipping containers until it arrives on site and the containers can be unloaded in a controlled environment. Site stocks of cyanide will be stored in an ICMC compliant manner.

The cyanide bulk bag will be lifted by the reagents area hoist from the plywood box to a bag breaker above the agitated cyanide mixing tank, which will have been previously partially filled with process water and a small amount of caustic solution to provide a high pH environment. After dissolution of the cyanide briquettes the mixing tank will be topped up with process water to achieve a 20% w/v cyanide concentration.

During operations cyanide will be drawn from the mixing tank and dosed to the leach circuit as required. When mixing a fresh batch of cyanide solution, the cyanide dosing pumps will draw directly from the cyanide storage tank.

The sodium cyanide and caustic soda mixing and storage areas are provided with a common bunded area serviced by a common sump pump. Any spillage generated within this area will be pumped to the leach feed distribution launder.

Sodium Hydroxide (NaOH)

Sodium hydroxide (caustic soda or caustic) will be delivered to site as pearls in 25 kg bags. The bag will be lifted to the bag breaker mounted above the manual rotary feeder.

Caustic will be fed slowly by the caustic rotary feeder to the caustic mixing tank which will have been previously filled with sufficient water to prevent localised heat generation during dissolution. Raw water will be topped up to the mixing tank to achieve a solution with the desired caustic concentration (20% w/v). The mixing tank will be mechanically agitated to assist with caustic dissolution. Common dosing pumps will be used to deliver caustic solution to the acid wash column strip solution tank and cyanide mixing tank.

Hydrochloric Acid (HCl)

Hydrochloric acid (32% w/w) will be delivered to site in 1,000 L intermediate bulk containers (IBC). The acid metering pump will supply acid, from the bulk containers to the acid wash column where in-line dilution with water will result in a 3% w/w solution feeding the acid wash vessel.

The hydrochloric acid storage and sump area will be protected with an acid-resistant liner and any spillage will be transferred to the carbon safety screen underpan by the area's dedicated sump pump.

Flocculant

Flocculant powder will be delivered to site in bulk bags and mixed in a proprietary mixing system, comprising a storage hopper, screw feeder, raw water booster pump, wetting head, mixing tank and flocculant transfer pump. The flocculant plant will mix flocculant powder with raw water to achieve the required storage concentration (0.25% w/w).

From the storage tank, flocculant will be distributed to the tailings thickener (via an in-line mixer) by the flocculant dosing pump. Additional water is added to the in-line mixer to dilute the flocculant to 0.025% w/w prior to dosing to the tailings thickener feed, and also to the pre-leach thickener feed when the sulphide plant is in operation.

The flocculant area will be serviced by a sump pump. Any spillage generated within this area will be pumped to the carbon safety screen underpan.

Antiscalant

Antiscalant will be delivered to the plant in intermediate bulk containers (IBC). Metering pumps will distribute antiscalant directly from the IBC to the required dosing points.

Fluxes

The following fluxes will be delivered to the plant in 25 kg bags and used in the gold room: Borax ($Na_2B_4O_7 \cdot 10H_2O$), Sodium Nitrate ($NaNO_3$), Sodium Carbonate (Na_2CO_3) and Silica (SiO_2).

Grinding Media

The oxide plant ball mill will require grinding media in the 80 mm diameter size range. The grinding media will be delivered to the plant in 205 L drums.

Activated Carbon

Activated carbon will be delivered to the plant in 500 kg bulk bags for making up carbon in the CIL circuit. An inventory of 1 leach tank worth of carbon will be kept on site.

17.2.9 Water Services

To the extent possible, the process plant will re-use process water recovered from the tailings thickener and TSF to meet process needs. Water previously uncontaminated by process chemicals (raw water) will only be used for make-up for the process water circuit to the minimum extent as necessary and for applications where quality water with low dissolved solids is required.

Raw Water

Raw water will be harvested from the Nobsin River during the wet season and stored in the OCR until needed. Where the water quality allows this will be supplemented by water captured by the site water management system. Three OCR pumps (two duty, one standby) will be available to deliver water to a raw water tank and to the plant's process water tank (as make-up if required). These pumps will provide the necessary capacity and flexibility to meet the varying water demand between wet and dry seasons.

The raw water tank has a capacity of approximately 630 m^3 of which 430 m^3 is fire water reserve. This provides a 4-hour supply of raw water for all uses other than thickener wash and process water make-up, and a minimum 2-hour supply of fire water at the rated capacity of the fire pumps.

Raw water from the raw water tank will be reticulated through the plant by the raw water pumps for various uses including:

- Eluate make-up.
- Carbon regeneration and quenching.
- Flocculant mixing and dilution.
- Reagent mixing.
- Potable water treatment plant.
- Grinding mill cooling water.
- Loaded carbon recovery screen.

Fire Water

Firewater will be supplied from the plant raw water tank, via a dedicated suction manifold. Fire water will be provided by a conventional fire water pump skid comprising an electric and a diesel fire water pump, and a jockey pump to maintain system pressure.

Gland Seal Water

Gland seal water will be supplied to slurry pumps from the raw water tank by dedicated low pressure gland water pumps (duty and standby).

Potable Water

A small quantity of raw water will be treated for potable water purposes (ablutions and site safety showers). The potable water treatment plant will filter, chlorine dose and UV sterilize the water.

Process Water

Process water is water that has come into contact with process slurries containing chemicals such as lime and cyanide. Under no circumstances will process water be released to the environment. Process water will be continuously recovered from plant tailings and re-used to slurry fresh feed to the milling circuit. However, during normal operations process water will be lost by:

- Entrainment in the tailings in the TSF.
- Evaporation from the TSF decant pond.

Recovered process water will predominantly consist of tailings thickener overflow and water reclaimed from the TSF. Process water will be recovered to a 1,025 m^3 process water tank, which provides approximately 1.0 hour of surge capacity.

From the process water tank, process water will be distributed around the plant by the process water pumps, with offtakes supplied for the main user points, namely:

- Milling area (mill feed chute, cyclone feed dilution and trommel spray water).
- CIL circuit.
- Cyanide mixing.

17.2.10 Plant Air Services

High Pressure Air

Plant air at 700 kPag will be provided by two high pressure air compressors, operating in a lead-lag configuration. The entire high pressure air supply will be dried to avoid the need for a duplicate instrument air system. Dried air will be distributed to the required plant areas for use in air actuated valves, hose points for tools and other general applications.

Low Pressure Air

Low pressure air for providing oxygen to the CIL circuit will be supplied by three air blowers (two duty, one standby) and distributed to the leach/CIL tanks and injected into each tank down the agitator shafts.

17.3 Sulphide Plant Process Description

17.3.1 Ore Receiving and Crushing

ROM ore from the open pit will be transported to the plant by 50 t capacity rear dump trucks with direct tip into the ROM bin. A ROM stockpile will be available to blend material based on grade and hardness. Ore will be reclaimed from the stockpile to the ROM bin by a front-end loader.

A static grizzly (600 mm x 600 mm), mounted above the ROM bin, will prevent the ingress of oversize material. A mobile rock breaker, supplied by the mining contractor, will be used to break oversize material retained on the static grizzly. Ore will be drawn from the ROM bin by a variable speed vibrating grizzly feeder to feed the jaw crusher operating in an open circuit. The feeder undersize will report directly to the crusher discharge conveyor to combine with the primary crushed ore sent to the surge bin.

The crusher discharge conveyor will be fitted with a weightometer to monitor and control the crushing area throughput by adjusting the output of the vibrating grizzly feeder variable speed drive.

The crushing circuit will be serviced by a single dust collection system comprising a series of extraction hoods, ducting and a bag house. Dust collected from this system will be discharged onto the crusher discharge conveyor.

A static tramp metal magnet will be installed at the discharge end of the primary crusher discharge conveyor. Tramp metal will be manually removed from the magnet when necessary and stored in a tramp bin.

Any coarse spillage generated, within the crushing area, will be recovered and transported to the mill feed bin. An area sump pump will pump drainage and any other water and fines accumulation in this area to the reclaim area sump.

Auxiliary equipment for the crushing circuit will include:

- Crushing area control station.
- Primary crusher maintenance hoist.
- Primary crusher lube pack.

17.3.2 Crushed Ore Storage and Mill Feed

The surge bin (or mill feed bin) will have a live capacity of 91 t to provide approximately 20 minutes of plant feed at the instantaneous feed rate to the SAG mill. The surge 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 4,900 t (providing 18 hours of plant feed). Crushed ore will be reclaimed from the stockpile to the ore bin via a front-end loader.

Crushed ore will be drawn from the surge bin by a variable speed apron feeder and discharged onto the SAG mill feed conveyor. The SAG mill feed conveyor will be fitted with a weightometer in order to control the speed of the apron feeder and also to monitor mill feed tonnage.

Quicklime will be added directly to the SAG mill feed conveyor via a variable speed rotary lime feeder. The lime silo will have a storage capacity of 70 t which is equivalent to 7 days of storage. To mitigate the risk of lime shortages, a small inventory of 1 t quicklime bags will be kept in storage.

Any spillage generated within the surge bin area will be captured in the reclaim area sump. Diluted slurries from this sump will be pumped to the cyclone feed pumpbox while coarse solids will be manually transported to the surge bin with an FEL.

Water sprays will be used to suppress dust generated in the SAG mill apron feeder discharge chute and at the crushed ore stockpile.

17.3.3 Grinding and Classification

The grinding circuit will be a SSAG circuit comprising a single variable speed SAG mill. The SAG mill will operate in closed circuit with hydrocyclones while pebbles will be removed by a trommel screen and recycled back to the SAG feed conveyor via a return conveyor to the surge bin. The ground product (cyclone overflow) will have a targeted P_{80} of 75 µm. To achieve the required product size when treating ore at the 85th percentile of hardness, a dual pinion 9.14 m x 6.25 m SAG mill (9.0 MW) will be required.

Crushed ore, reclaimed from the surge 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 adjust the in-mill pulp density

Grinding media (a mix of 125 mm and smaller balls) will be added to the SAG mill feed chute as required to maintain the required ball charge in the mill. The mill will have a variable speed drive.

SAG mill trommel screen oversize will be recycled back to the SAG Mill feed conveyor via a pebble return conveyor. Undersize from the discharge screen will flow by gravity to the cyclone feed pumpbox prior to being pumped to the classification cyclone cluster by a single variable speed cyclone feed pump. A spare pump will be supplied but not installed. The classification cyclone cluster overflow will flow by gravity, via a trash screen, to the pre-leach thickener feed distribution box. 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.

The classification cyclone cluster will consist of 17 installed 15” cyclones and one spare port 15 of the 17 installed cyclones will be operating, while the remaining two will be installed as spares to allow in-line maintenance of the cyclones.

Spillage within the grinding circuit will be managed through a slanted floor draining to a central sump fitted with a sump pump. Slurry from this sump will be discharged into the cyclone feed pumpbox. During flooding events the excess water will flow via trenches to the oxide plant and eventually to the event pond.

17.3.4 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. This stream will be sampled for metallurgical accounting ahead of the flocculant addition.

Overflow from the 29 m diameter pre-leach thickener will flow by gravity to the grinding water tank. Underflow from the pre-leach thickener, at 50% w/w solids, will be pumped by dedicated thickener underflow pumps to the leach circuit feed distribution box.

The pre-leach thickener area will be serviced by a dedicated sump pump. Spillage and wash down collected by the sump pump will be returned to the pre-leach thickener distribution box. Excess water will overflow from this bunded area via the pipe rack bunded area to the event pond at the oxide plant.

17.3.5 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 and leach circuit to oxidize cyanide consuming species and to improve the leach kinetics by maintaining a high dissolved oxygen level through the subsequent leach.

The leach circuit will consist of three mechanically agitated leach tanks operating in series. Their combined live volume provides a residence time of 24 hours at a slurry feed rate of 376 m^3/h. The physical dimensions of the leach tanks at 15.8 m diameter by 18.6 m high are the same as the CIL circuit tanks.

Cyanide, for gold dissolution, will be added to the leach circuit by cyanide dosing pumps from the oxide plant cyanide storage tank. The primary cyanide dosing point will be the first leach tank, with further addition points located down the leach train. The cyanide dosage to the sulphide leach circuit will be in excess of the minimum required as this will improve leach kinetics and the surplus cyanide will be transferred to the oxide leach circuit reducing dosage rates at that point.

The operating pH of the leach circuit will be maintained above 10.3 to maintain the protective alkalinity of the circuit and prevent the loss of cyanide in the form of gaseous hydrogen cyanide. Protective alkalinity will be maintained by the addition of quicklime to the SAG mill feed conveyor.

Leached slurry from the sulphide plant will flow by gravity to the oxide plant leach feed pumpbox to be further treated in the oxide leach circuit.

Should a leach tank be off-line for maintenance, it will be possible to bypass that tank. The ability to bypass tanks will be made possible by the installation of two pneumatic gates located within the leach inter-stage launders. One gate will divert slurry to the following leach tank while the second gate will allow slurry diversion to the subsequent leach tank.

The leach area will be serviced by an area sump pump. The sump pump will return spillage to the leach feed distribution box. The leach bund area will overflow to the event pond in the case of emergencies.

17.3.6 Reagents Mixing and Storage

The major reagents utilized within the process plant will include:

- Quicklime (90% w/w CaO) for pH control.
- Sodium cyanide (NaCN) for gold dissolution and desorption (at the oxide plant).
- Flocculant for thickening.

Lime

Quicklime will be delivered to site in 30-t bulk tankers. Bulk tankers will be pneumatically off-loaded, using a blower, directly to the 70-t capacity lime silo. Quicklime will be withdrawn from the silo by a variable speed screw feeder and deposited directly onto the SAG mill feed conveyor. A small inventory of seventy 1-t bulk bags of lime will be retained on site for use in the case bulk deliveries are delayed. When using bagged lime, the bags will be discharged via the transfer hopper and to the silo by the blower and rotary valve.

Sodium Cyanide (NaCN)

Sodium cyanide solution will be supplied to the sulphide plant leach train by an expansion of the oxide plant's cyanide reagent mixing and distribution system. At the expected nominal addition rate of 0.78 kg/t and at a 20% w/v solution strength, the sulphide plant will be fed 1,100 L/h of cyanide solution.

Flocculant

Flocculant will be supplied to the pre-leach thickener from the oxide plant's flocculant system. At the nominal consumption rate of 22 g/t, the sulphide plant will consume flocculant solution (at 0.25% w/v strength) at a rate of 2.4 m^3/h.

Grinding Media

The sulphide plant SAG mill will require grinding media in the 105 mm to 125 mm diameter size range. The grinding media will be delivered to the plant in 205 L drums.

17.3.7 Water Services

The sulphide plant will require grinding water with process water make-up for the SSAG circuit, raw water for dust suppression and SAG mill cooling, and gland water for sealing slurry pumps.

Grinding Water

Grinding water will predominantly be supplied from pre-leach thickener overflow with make-up from the oxide plant's process water system. Grinding water will be stored in a 261 m^3 grinding water tank, which will provide more than 20 minutes of surge capacity. The total nominal reticulation of grinding water is estimated to be 788 m^3/h, of which 245 m^3/h will be sourced from the oxide plant's process water system.

Grinding water will be distributed via headers by duty and standby configuration single stage water pumps with offtakes supplied for the following predominant users:

- Grinding (in-mill dilution, trommel spray water and cyclone feed density control/dilution).
- Flocculant dilution and trash screen sprays.

Raw Water

Raw water will be supplied to the sulphide plant by the oxide plant's raw water distribution system. This water will be used for dust suppression and SAG mill cooling at nominal consumption rate of 10.8 m^3/h.

Gland Seal Water

Low pressure gland seal water will be supplied from the oxide plant at a nominal rate of approximately 10 m^3/h to the sulphide plant. Gland seal water will be required for the cyclone feed pumps, pre-leach thickener underflow pumps, and grinding water pumps.

Potable and Fire Water

Potable and fire water will be sourced from the oxide plant systems.

17.3.8 Air and Oxygen Services

Plant air will be provided by the plant air system at the oxide plant. An estimated nominal rate of 500 Nm^3/h at a delivery pressure of 750 kPag will be required.

Oxygen for the pre-oxygenation tank and leach tanks will be supplied from a PSA or similar type package supplied oxygen system.

17.4 Water, Power and Reagent Consumption

17.4.1 Water Consumption

Oxide Plant

A water balance for the oxide plant has been completed by Lycopodium and incorporated into the site wide water balance undertaken by KP.

Irrespective of season (dry or wet), the process plant requires an average of 50 m^3/h of raw water for applications for which process water is unsuitable.

Up to 545 m^3/h of decant return water will be recycled from the TSF to the process plant as process water make-up. This quantity will likely only be available during the wet season when rainfall on the TSF, and site run-off stored on the TSF, increases the size of the decant pond. The shortfall will be made up by pumping raw water from the OCR.

At the end of the wet season, when the Nobsin River is no longer topping up the OCR level, decant water from the TSF will continue to be used in preference to raw water make-up to conserve water in the OCR for use later in the dry season.

Sulphide Plant Expansion

Raw water to the sulphide plant will be supplied from the oxide plant raw water pumps. Raw water consumption is estimated to be 20.8 m^3/h which includes water for dust suppression, SAG mill cooling and gland water. A further nominal rate of 3.5 m^3/h will enter the plant as reagent solution from the oxide plant.

Water from the pre-leach thickener overflow stream will be recycled within the process plant to reduce external water requirements. The sulphide plant will produce a leached slurry and a substantial volume of make-up water will be required to balance the flowrate of pregnant leach solution exiting with this slurry. Process water from the oxide plant will be used as the primary source of make-up water and will be required at a nominal rate of approximately 245 m^3/h.

17.4.2 Energy Consumption

Electrical power for the site and camp will be provided by on-site power generation (refer to Section 18).

The average site power demand with the stand-alone oxide plant in operation is estimated to be 6.581 MW with an annual power consumption of 57.6 GWh (an average of 11.1 kWh/t). With the oxide and sulphide circuits operating simultaneously the site average power demand increases to 14.406 MW with an annual power consumption of 126.2 GWh (an average of 24.2 kWh/t).

17.4.3 Reagent and Consumable Consumption

Tables 17.3 and 17.4 provide a summary of major reagent and consumable usage for the oxide plant and the incremental increase post commissioning the sulphide plant, respectively.

Table 17.3 Major Reagents and Consumables for 5.2 Mtpa Oxide

Reagent / Consumable	Annual Consumption
Ball Mill Grinding Media	468 t
Quicklime (90% w/w CaO)	9,672 t
Sodium Cyanide	1,430 t
Activated Carbon	131 t
Sodium Hydroxide (Caustic)	186 t
Hydrochloric Acid	463 t
Flocculant	52 t
Distillate Diesel Oil DDO (elution heaters)	2,186 m^3

Table 17.4 Incremental Reagent and Consumable Consumption Post Sulphide Plant Commissioning

Reagent/Consumable	Annual Consumption
SAG Mill Grinding Media	2,002 t
Quicklime (90% w/w CaO)	2,970 t
Sodium Cyanide	880 t
Flocculant	48.4 t

Flocculant and sodium cyanide for the sulphide plant will be sourced from the oxide plant. The only dedicated reagent system within the sulphide plant will be the lime storage and dispensing system on the SAG mill feed conveyor.

17.5 Plant Control System

17.5.1 General Overview

A single control system will service both the sulphide and oxide plants. The general control philosophy for the plant will be one with a moderate level of automation and central control facilities to allow critical process 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 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).

The process control system that will be adopted for the plant will be a programmable logic controller (PLC) and SCADA based system. The PCS will control the process interlocks and control loops for non-packaged equipment. Control loop set-point changes for non-packaged equipment will be made at the OIT.

17.6 Sampling and Assaying

Titration facilities and an on-line analyser unit will be provided to monitor cyanide concentration in CIL process liquors. Automatic samplers taking shift composite samples from the mill cyclone overflow and CIL tailings streams will provide the primary gold balance for both the oxide and sulphide plants. These samples will be filtered and dried on site with splits then sent for assay in Ouagadougou.

Manual sampling of slurry and carbon in the leach circuit will be used to monitor the leach profile and provide end of month gold in inventory measurement for metallurgical accounting. Manual sampling of loaded and barren carbon and the pregnant and barren eluate streams will monitor the performance of the elution and electrowinning circuits respectively.

A basic metallurgical laboratory will be established on site with the facility to undertake simple bottle roll leach testing. This will be used to monitor the metallurgical properties of pre-production mining samples to ensure that plant performance can be predicted in advance.

Site laboratory facilities will be established to provide array services for process control and metallurgical balance purposes. Fire assays can be supplemented by LeachWell type rapid cyanide soluble gold assays if required.

18.0 PROJECT INFRASTRUCTURE

The overall site at the commencement of plant commissioning and prior to decommissioning and rehabilitation are shown in Figures 18.1 and 18.2 respectively.

The overall site major facilities include mine open pits, process plant, TSF, mine services, fuel storage and distribution, waste dumps, and access road, camp and relocation areas. Power is provided by an on-site power plant. The site will be fenced to clearly delineate the mine area and deter access by unauthorized persons and prevent grazing animal access.

Figure 18.1 Project Site – Plant Commissioning

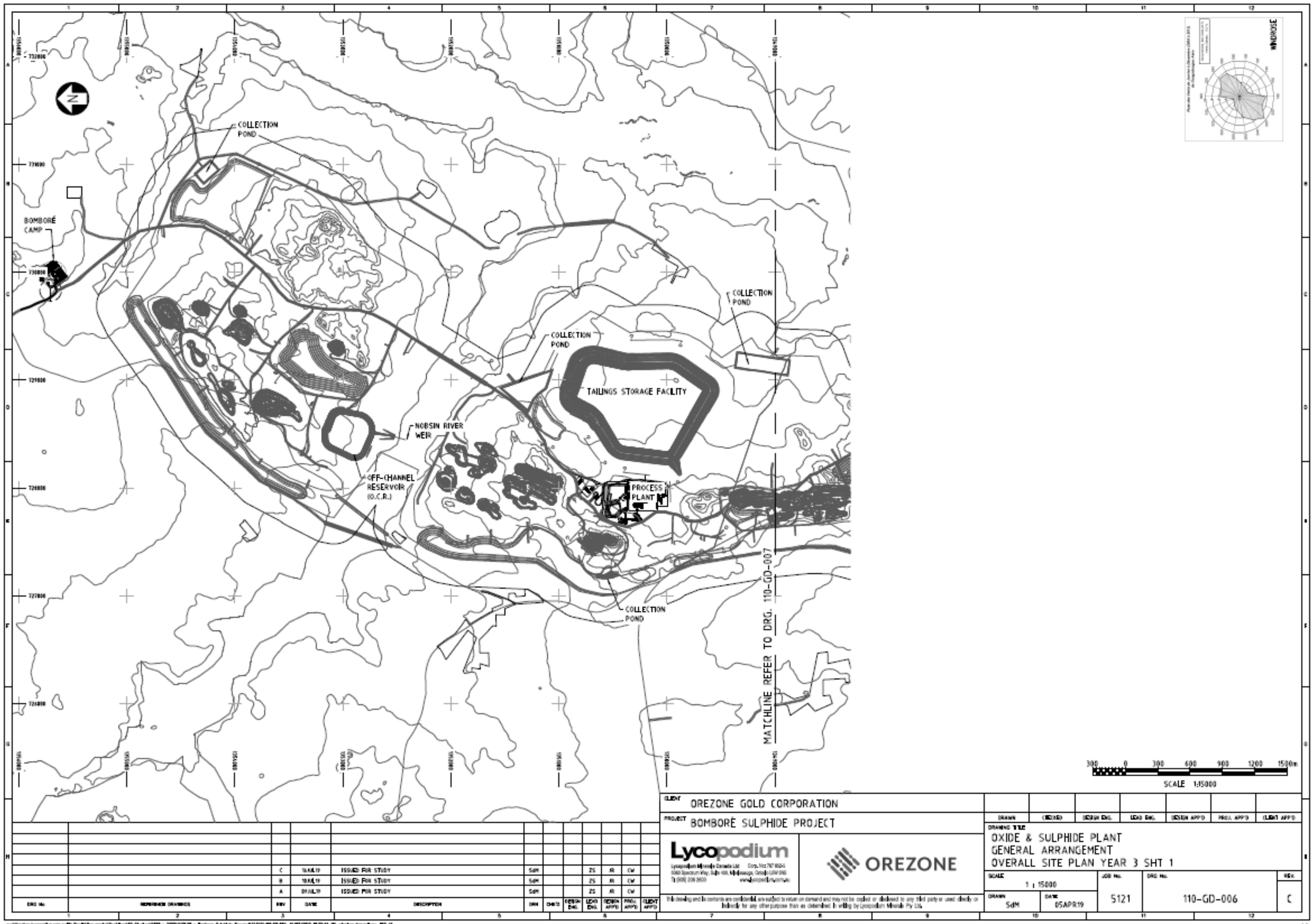
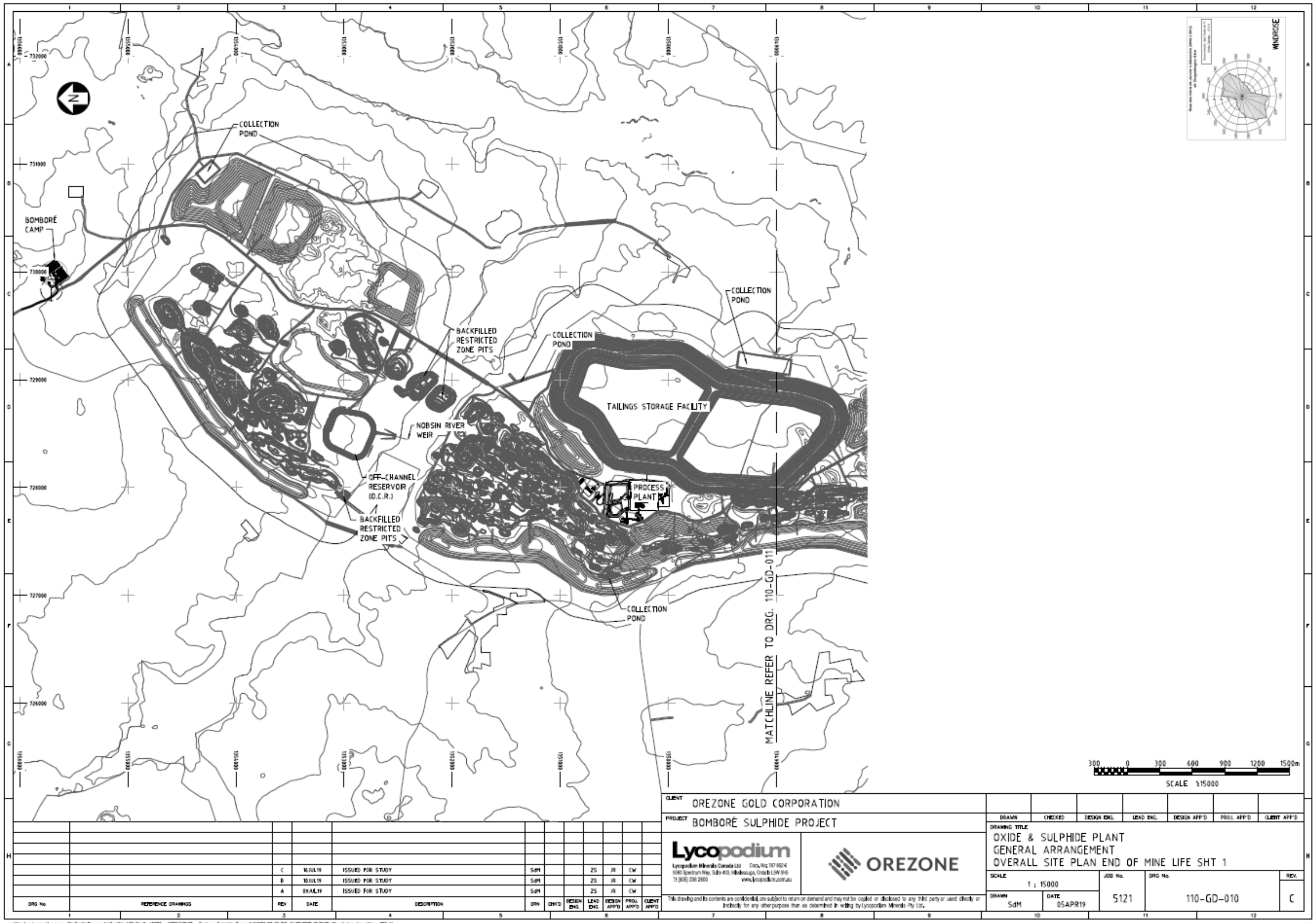
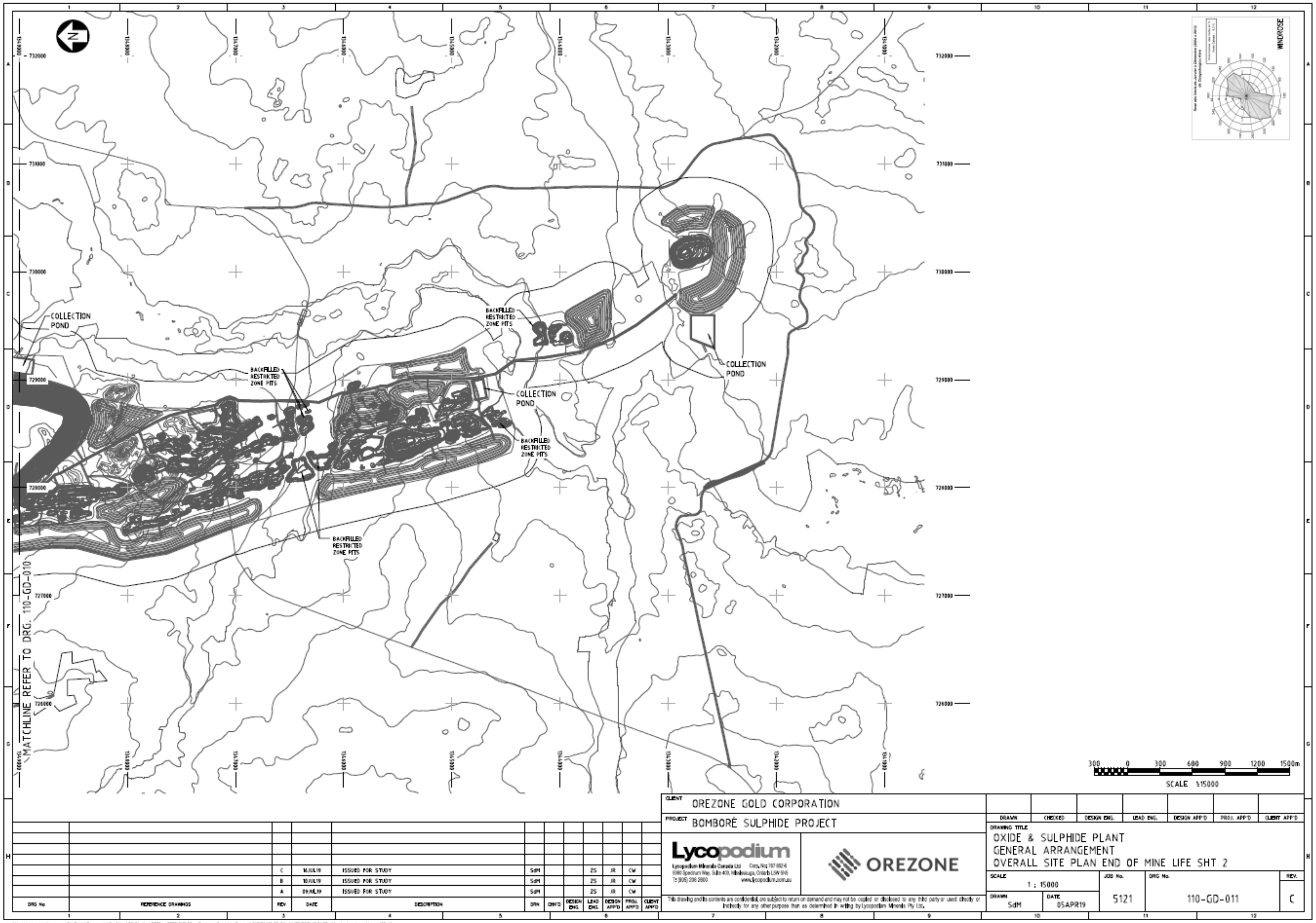


Figure 18.2 Project Site – Prior to Decommissioning




COLLECTION POND
BACKFILLED RESTRICTED ZONE PITS
COLLECTION POND
COLLECTION POND
MATCHLINE REFER TO DRG. 110-GD-010
WINDROSE
300 0 300 600 900 1200 1500m
SCALE 1:15000
CLIENT OREZONE GOLD CORPORATION
PROJECT BOMBORÉ SULPHIDE PROJECT
Lycopodium
OREZONE
DRAWING TITLE
OXIDE & SULPHIDE PLANT
GENERAL ARRANGEMENT
OVERALL SITE PLAN END OF MINE LIFE SHT 2
SCALE 1 : 15000
DRAWN SdM
DATE 05APR19
JOB No. 5121
DRG No. 110-GD-011
REV. C
C 16JUL19 ISSUED FOR STUDY
B 10JUL19 ISSUED FOR STUDY
A 09JUL19 ISSUED FOR STUDY

18.1 Site Access

Site access will be from the north using the upgraded 5 km public road that connects the site to national highway N4 and passes the existing Bomboré camp. Close to the camp public roads branch off around the eastern and western perimeters of the Project site. A private Project road will service the Project entering through a manned checkpoint providing access control to the site. This road extends to the process plant situated close to the mid-point of the site and provides easy access for personnel and material movements.

An agricultural style wire fence will be installed around the entire site to prevent grazing animal access and provide a clear indicator to deter entry of unauthorized personnel.

Monitored high-security fencing will surround the process plant and associated infrastructure. Access into the fenced area will be adjacent to a manned gatehouse with separate personnel and vehicle access. Only authorized personnel will be allowed entry to the process plant area and vehicles and personnel entering and leaving the area will be subject to security checks.

18.2 Accommodation

The existing camp at the northern end of the Project area provides accommodation and office facilities for the exploration teams and Orezone support personnel currently at site. The camp is fully enclosed within a double 2 m chain linked fence with razor wire, which is secured and patrolled by armed security personnel.

The main camp buildings are constructed from locally produced red bricks with terrazzo floors. The office is a 480 m^2 building with reception, conference rooms and compartmentalized office facilities. There are separate buildings housing the Environment and Security departments. The camp can currently accommodate 116 persons in a combination of single and shared rooms, both with and without en-suite bathroom facilities. Potable quality water for cooking and ablution purposes is provided from a bore adjacent to the camp with chlorine dosed manually into the storage tank. Containerized water is provided for drinking at carious water stations. A kitchen/dining room operated by a contract service provider serves meals for camp residents and guests.

Recently completed additions to the camp, consists of two accommodation blocks, one block with 18 bedrooms with private bathrooms, including 2 executive suites, a second block of 20 rooms with private bathrooms. There is enough space within the existing camp boundary fence to install several additional accommodation blocks or other facilities should they be required in the future.

In addition, upgrades to expand the kitchen and dining room have been completed in anticipation of serving more camp residents and daily workers during construction and operations.

Planned additions included in the project capital estimate are:

- Extending the site-wide power supply to the camp with the existing generators then being used as back-up units.

- Potable water treatment plant incorporating filtration, automated chlorination and UV sterilization.
- Sewage treatment plant.
- A recreation room/exercise facility.
- A fully upgraded clinic.

Site accommodation, at the expanded Bomboré camp, will be provided for the owner’s team, EPCM contractor’s team, and vendor representatives and in exceptional circumstances, for senior contractor employees.

The Project site is within a thirty-minute drive from the regional town of Mogtédo, with a population in excess of 15,000. The town is developing rapidly with many substantial multi story concrete block buildings established or under construction.

Most of the semi-skilled and unskilled labour required for project development and operations will be sourced from Mogtédo and surrounding villages. As the town has the capacity to provide rented rooms and leased accommodation for the contractor’s skilled workforce, the contractors will make their own accommodation arrangements with local businesses. Contractors will also arrange for bussing their employees to and from site and for providing a midday meal.

Contractor costs are based on the above assumptions for accommodation and transport.

18.3 Mine Service Area

Mining operations will be undertaken by a contractor. As the oxide ore and waste are free digging and the pits small and generally shallow, it is planned that mine operations will initially be undertaken with a fleet of small excavators, 30 t to 50 t capacity road-legal dump trucks (likely Chinese built or equivalent) and a modest support fleet. A more conventional contract mining fleet will be mobilized for the later sulphide mining when drill and blast operations are required.

Power and water will be supplied to an area north of the plant site and adjacent to the diesel fuel facility, where the mining contractor will be required to establish the mine service area. The mine service area will provide offices, meals and ablution facilities for the contractor personnel plus workshop/warehouse facilities for servicing the mining fleet.

18.4 Site Buildings

With many administration functions centred in the Orezone office in Ouagadougou there will be a minimal requirement for site office space. This will be met by the existing office block at the camp.

Plant site buildings will be ‘fit for purpose’ reflecting the cost-conscious approach to project development.

The following buildings will comprise arched polyethylene covers between modified shipping containers with simple concrete slab floors:

- Reagent storage (including a fenced and gated section for cyanide storage).
- Plant warehouse.
- Plant workshop.
- Safety/emergency response facility.

The following buildings will be erected from modified shipping containers or alternatively purchased as pre-fabricated modules:

- Assay and metallurgical laboratory.
- Plant mess.
- Plant office.
- Plant ablutions.
- Site bus shelter.
- Plant site security gatehouse/change-house.

Any need for additional satellite offices/guard posts or similar will be met by using portable converted shipping containers similar to those used as site construction offices or by construction of suitable concrete block buildings.

18.5 Power Supply

Power will be provided by a site power station operating under a 'build, own, operate' (BOO) contract arrangement with an independent power provider (IPP).

A containerized HFO power station will be provided and considered the most 'fit for purpose' with adequate operating flexibility at a low over-the-fence power cost. The power station will be an 'n+2' configuration (n units running with 2 on standby) with sufficient installed power to meet the power demand surge when starting the mill.

The electrical system is based on 11 kV distribution and 415 V working voltage.

During the initial oxide treatment phase the annual average electrical load on site is estimated to be 6.6 MW with a peak demand of 8.6 MW. Annual power consumed is estimated to be 54.9 GWh.

The sulphide treatment plant will add a further 8.42 MW of annual average electrical load (9.27 MW peak).

When treating sulphides, however, the oxide load will reduce as the oxide circuit will be operating at a lower throughput. The combined annual average electrical load is estimated to be 14.18 MW. The average annual energy consumption with both the oxide and sulphide circuits in operation is estimated to be 124.2 GWh.

11 kV overhead power lines will distribute power across the site, stepped down at point of use with pole top transformers, kiosks or conventional transformers and MCCs as appropriate.

18.6 Potable Water

Two vendor-packaged modular potable water treatment plants including filtration, ultra-violet sterilization and chlorination will be installed at the accommodation camp and process plant for reticulation to the camp, site buildings, ablutions, safety showers and other potable water outlets. The plants will be installed with additional capacity to meet peak demand and to provide potable water during construction.

18.7 Sewage & Waste Management

Grey water and effluent from the accommodation camp will drain to a vendor package sewage treatment plant located adjacent to the camp for treatment.

Treated effluent will be discharged into leach drains. Treatment plant sludge will be suitable for direct landfill burial in unlined pits.

The process plant, mine services area and other remote facilities will use septic tanks for collection of sewage. These will be emptied as required and the contents transported to the camp sewage treatment plant for treatment.

Site solid waste is currently sorted into bins and removed by a contractor for recycle or disposal at the municipal facilities in Ouagadougou.

Wastes will continue to be sorted and reused or recycled as far as the limited access to recycling facilities in Burkina Faso allows.

Inert solid wastes will be deposited into suitable landfill sites at the toe of the waste dumps and promptly covered to deter unauthorised access and re-use. Materials such as cyanide packaging will be cleaned and buried, under supervision, on site beneath mine waste to prevent unauthorized use of the packaging. On site incineration is not permitted.

Putrescible waste from the kitchen and general site refuse bins will continue to be transported to the municipal facilities for disposal or, if quantities dictate an alternative solution, a lined site disposal facility will be permitted and constructed.

Waste lubricating oils will be returned to the supplier for recycling.

18.8 Communications

Internal communications and IT services shall be via a site-wide fibre optic network.

A local mobile phone provider will be contracted to upgrade existing facilities on site and provide a link into the local, national and international telecommunication network (voice and data, recently completed).

A radio network will be established with dedicated operational, security and emergency channels.

18.9 Fuel & Lubricant Supply

On site fuel and lubricant storage and dispensing facilities will be supplied, installed and administered by the selected fuel distributor in exchange for long-term supply contracts. Payment for the facilities will be through a small mark-up on the price of the fuel supplied.

Initially the fuel depot will have a minimum storage capacity of 14 days of HFO and diesel. With the proximity of the Project to Ouagadougou, this is considered an adequate site fuel reserve. The requirement to expand the fuel depot capacity for the sulphide phase will be assessed based on the future supply conditions/reliability.

During construction a temporary fuel depot will be established using ‘bullet’-type tanks leased from the fuel supplier. The temporary depot will be within a bund constructed in accordance with appropriate international standards to contain fuel spills and will have an oil/water separation system for draining rainwater.

18.10 Site Security

Site security is based on concentric lines of fencing/access control.

The entire Project area will be enclosed within a patrolled agricultural-type stock fence line to prevent animal access and discourage casual entry by unauthorized persons. The main point of entry will be where the main access road enters the site. This point of entry will be provided with a gate and manned security post.

The process plant will be enclosed by a 2 m chain link security fence topped by razor wire and monitored by closed circuit cameras. The fence line will be provided with perimeter lighting. Entry will be via a single monitored security post and will be strictly controlled using swipe cards and turnstiles. Exit from the plant area will be subject to a search of vehicles and toolboxes and ‘pat down’ and/or metal detector search of all persons.

Access to the goldroom within the plant will be restricted and strictly controlled. Video surveillance will be installed and entry points will be monitored and alarmed. A minimum of two authorized staff must be in the goldroom while work is in progress with a third person monitoring the surveillance systems.

The accommodation camp is currently fenced with a manned entry gate to prevent unauthorized access. Security personnel contracted to Orezone are supplemented by an armed detachment from the National security forces.

18.11 Ouagadougou Facilities

In Ouagadougou, Orezone owns and operates a fully functional office and warehouse facility with auxiliary power, water and redundant internet connectivity.

The Ouagadougou facility is sufficient to serve as a management and logistics base for the Bomboré operation. Administration functions and services such as procurement, accounting and government relations will be based out of the Ouagadougou office reducing the burden on site facilities and accommodation.

18.12 Tailings Storage Facility

Tailings from the Project process plant will be deposited in a fully lined tailings storage facility (TSF) that will be stage-developed at a site immediately east of the process plant. The tailings will be pumped as a slurry and placed hydraulically into the facility in a controlled manner from a series of strategically positioned drop bar pipes around much of the perimeter to build a consolidated and stable deposit. Bleed water will be recovered from a small supernatant pond via a permanent decant tower and continuously recycled to the mill for re-use.

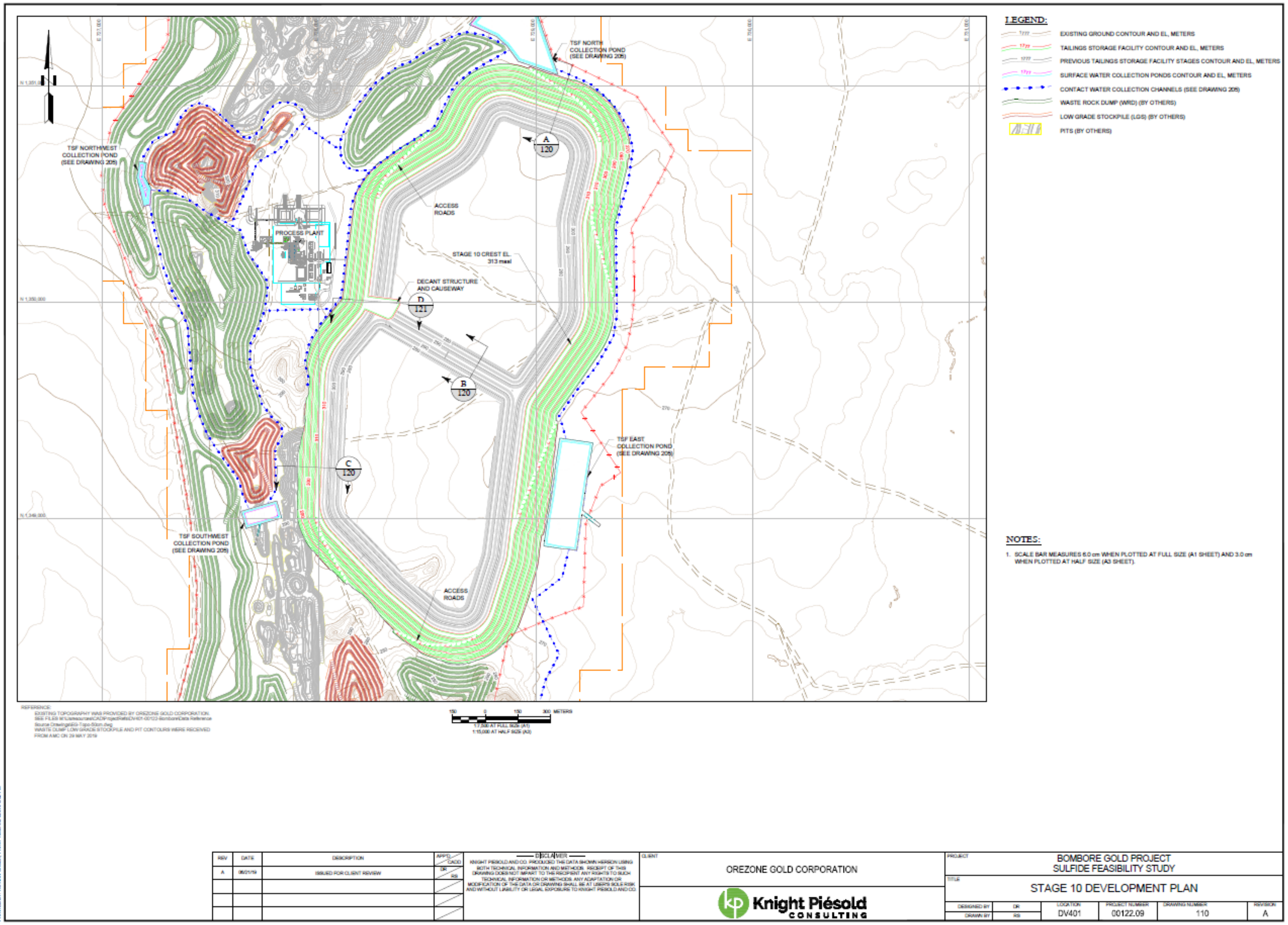
The TSF occupies a larger area than that envisioned in the 2018 FS. This has been made possible by the relocation of low-grade stockpiles and waste dumps. The larger TSF footprint has reduced life of mine costs by reducing the quantity of fill required to construct the TSF embankments.

The site is quite flat, and thus the facility will be developed within dams wrapping entirely around an internal basin. The facility will have a northern half that will be developed first to store tailings from the initial three years of operations and a southern half that will be developed to store tailings from years 4 to 6. After this, the two halves will be combined to store tailings from year 7 and onwards. The objective of splitting the facility initially into halves is to reduce the up-front capital by reducing the area of the initial basin and its liner. The overall design has been completed for storage of approximately 73 Mt of tailings to be produced at a rate of 5.2 Mtpa over a period of approximately 14 years (vs. current reserves of 70.1 t).

Most of the fill for TSF construction will be obtained from the mining operations as oxide mine waste. It will be placed and compacted in controlled, thin, horizontal lifts to form dense and stable wrap-around dams. Mine waste used as embankment fill will be entirely from the oxide and upper transition zones of saprolite and saprock. Therefore, the material will be relatively fine grained (gravel, sand and silt sizes) and will be geochemically benign. Termed “embankment fill” in the dams, the material will be received and placed largely on a continuous basis throughout the life of the mine, and the staging plan has been coordinated with the mining plan to account for this use of non-sulphide waste. The remaining mine waste will be used to construct visual barriers along the western side of the Project site and placed in waste dumps.

The TSF has been designed to a high level of standard for security, safety, stability and environmental protection. Canadian Dam Safety (CDA) Guidelines have been followed for dam safety, and the principles of the Mining Association of Canada’s (MAC) guidelines for successful overall tailings management have also been followed where applicable.

Figure 18.3 Tailings Storage Facility Layout (Stage 10 – Final)



The basin areas will be fully lined with a 300 mm thick low permeability soil bedding layer overlain by a 1.5 mm (60 mil) HDPE geomembrane similar to the inside facing on the dams described above. The low permeability soil layer and geomembrane will be continuous with the soil layer and geomembrane on the dams to form a continuous and fully lined impoundment. These two layers will also be extended entirely over the internal dividing dam between the two basin halves for complete environmental protection.

A summary of the facility staging giving the stage number, the year when operations will begin in that stage, the dam crest elevation at the top of the stage, the maximum tailings elevation at the end of the stage and the tonnage capacity in that stage is provided in Table 18.1. Of note, the crests at the top of each stage will be at a constant elevation around the facility.

Table 18.1 TSF Staging Summary

Stage	Initial Year of Operation of the Stage	Dam Crest Elevation (Completed by May) (m)			Maximum Tailings Elevation (September) (m)			Tailings Capacity (Mt)*		
		North Area	South Area	Combined Area	North Area	South Area	Combined Area	North Areas	South Area	Combined Area*
1	2021	282	-	-	-	-	-	-	-	-
2	2022	289	-	-	281	-	-	5.2	-	-
3	2023	296	-	-	288	-	-	10.4	-	-
4	2024	-	283	-	294	-	-	15.6	-	-
5	2025	-	290	-	-	282	-	-	5.2	-
6	2026	-	296	-	-	289	-	-	10.4	-
7	2027	-	-	300	-	295	-	-	15.6	-
	2028	-	-		-	-	296	-	-	36.4
8	2030	-	-	304	-	-	299	-	-	41.6
	2031	-	-		-	-	302	-	-	46.8
9	2032	-	-	309	-	-	303	-	-	52.0
	2033	-	-		-	-	305	-	-	57.2
10	2034	-	-	313	-	-	308	-	-	62.4
	2035	-	-				310			67.6
	2036	-	-		-	-	312	-	-	72.8

** The storage capacity may exceed 73 Mt if the actual in situ tailings density is greater than that assumed for this study*

A summary of the years of tailings filling in each stage, the dry density used for modelling the filling in each stage, the minimum and maximum tailings elevations at the end of each stage, the maximum water level in each stage, the crest elevation of the dams in each stage. A list of comments relating to which half of the basin is being used is provided in Table 18.2.

Table 18.2 Staged Development of Tailings Storage Facility

Stage #	Years of Construction	Year(s) of Tailings Filling	Planned Tailings Dry Density (t/m^3)	Cumulative Tailings Production at End of Stage (M t)	Min. Tailings Elev. End of Stage (m)(1)	Max. Tailings Elev. (m)(1)	Max. Water Level Elev. of 72-Hour PMP (m)(3)	Embankment Crest Elev. (m)(2)	Comments
1	1	1	1.1	5.2	278.1	281	281.0	282	North half of TSF
2	1	2	1.1	10.4	284.2	288	287.8	289	North half of TSF
3	1	3	1.1	15.6	290.1	295	293.7	296	North half of TSF
4	1	4	1.1	20.8	284	282	282.0	283	South half of TSF
5	1	5	1.1	26.0	284.4	289	288.7	290	South half of TSF
6	1	6	1.1	31.2	290.1	295	294.5	296	South half of TSF
7	1	7, 8	1.15	41.6	293.8	299	298.8	300	Both halves of TSF
8	2	9 , 10	1.2	52.0	297.4	303	303.0	304	Both halves of TSF
9	2	11, 12,	1.21	62.4	302.2	308	307.6	309	Both halves of TSF
10	2	13, 14	1.21	72.8	306.4	312	311.5	313	Both halves of TSF

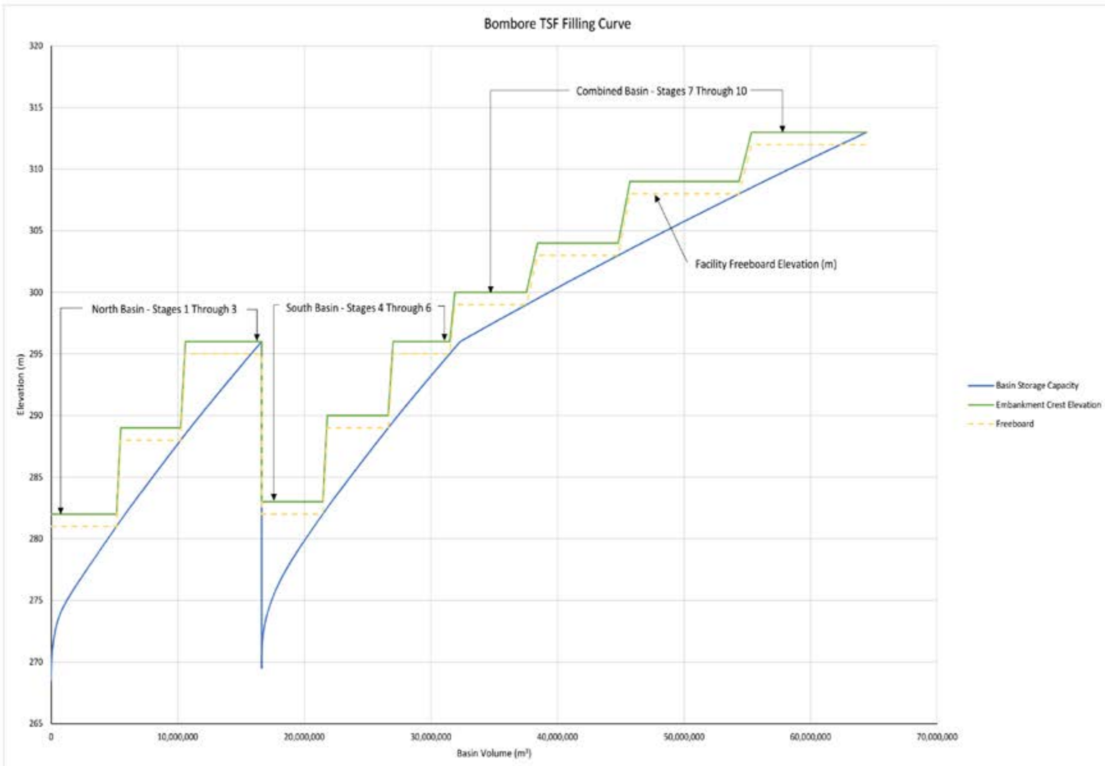
Notes:

1. *Tailings will be deposited from the North, East and South sides of the TSF making the maximum tailings elevation on the East side and the minimum tailings elevation on the West side for all stages.*
2. *The TSF embankment crest will be at a constant elevation around the facility and set at 1 m above the maximum tailings elevation on the East side for all stages.*
3. *The storage volume capacity above the tailings on the west side of the TSF will be sufficient to contain the average normal operating pond plus the 72-hour PMP at all times.*

Supernatant water will be temporarily stored and managed above the low points of the tailings deposits. Sufficient water storage capacity will be provided above the tailings in each stage of development to contain the average operating pond volume (determined on a monthly basis from the water balance) plus the 72-hour PMP volume (stage to stage volumes described above) plus freeboard. Tailings deposit and pond modelling show that the pond limits for the PMP will not rise above the top of the tailings beach around the north, east and south sides of the facility. Once those beaches are established and thus the 1 m of dam height above the final tailings beach level described above will ensure that the freeboard above the PMF pond will exceed 1 m at all times.

Figure 18.4 presents the design filling curve for the TSF. The maximum tailings level at any time is represented on the vertical axis while time (years) and dry tonnes of tailings stored in the facility are represented on the horizontal axis. The two-plotted lines show (1) the maximum tailings level at any time and (2) the crest to account for the designed added freeboard at any time. As described earlier, the maximum design water pond level that includes for the average operational pond plus the 72-hour PMP will be below the maximum tailings level once the tailings beach is established, and thus the water pond is not shown on the drawings. The stepped line represents the staged raising sequence for the dams and their crest elevations and can be seen to remain above the 1 m freeboard curve at all times. This confirms the adequacy of the facility-staging plan.

Figure 18.4 Tailings Storage Facility Design Filling Curve



The tailings slurry will be pumped from the tailings pump station, located at the process plant, to the TSF by two separate pipeline header pipes, each with the capacity to transport the full slurry flow. Two header pipes are required to facilitate their progressive raising as the dams are raised. This process is required since the headers cross the width of the fill on the western dam and this fill will be raised progressively and continuously over the life of the facility. Typically, one pipe will be in operation while the other is being lifted and raised, and the raising will be done in maximum 3 m vertical increments in an alternating manner.

The distribution pipeline around the facility will be installed at the inside crest of each of the dams and will be raised up onto the next crest after that staged raise has been completed. Each new staged raise will be positioned to leave a bench around the inside crest of the previous stage that will support the pipe and an access road until the pipe is raised. When it is raised it will be done in a segmented manner in defined lengths that will be isolated using isolation valves along the pipeline length. These valves will also be used to control the points of deposition and allow for drainage of pipeline segments.

The deposition pipeline will deliver the tailings to multiple HDPE drop-bars around the facility. These drop-bars will be spaced 100 m apart. One single drop-bar will be always operated at a time to avoid solids settling in the distribution pipeline. The drop bar will be sacrificed with each staged raise of the distribution pipeline.

The distribution pipeline and drop-bar design allows for complete drainage of the pipeline and specifically:

- The delivery pipe header will gravity-drain into appropriately sized tailings dump pit at the lowest point of the pipeline alignment.
- The distribution pipeline will gravity drain into the TSF.

Water collected in the TSF will be collected for use in the process via a permanent decant tower and pump and pipe system. The system will initially consist of a single decant located in the southwest corner of Cell 1. The decant will be constructed in a vertical manner using a poorly graded decant rock with a D50 of 100 mm surrounding a 1.5 m slotted decant pipe wrapped in a non-woven filter fabric and a geogrid. The pipe will be constructed out of 1-meter high concrete box culvert sections to be cast on site. The concrete blocks will be 1.5 m^2 and will have four 75 mm holes, one per side. The concrete pipe sections will be fabricated with a bell-and-spigot connection to facilitate stacking of the segments. The decant rock will have plan view dimensions of 10 m by 10 m and will be raised vertically concurrently with the Cell 1 dam raises. A 1 m wide soil filter will be placed around the perimeter of the rockfill to prevent piping of tailings and embankment fill into the decant tower. Water collected in the decant will be pumped to the process plant. The pump will be accessible using a winching system which can be mounted to a steel frame at the top of the decant pipe. The decant tower will be accessed by a 10 m wide causeway constructed from dam fill material connecting the west embankment to the decant tower.

A second decant tower, identical to the primary decant tower described above, will be constructed parallel to the Cell 1 decant in Cell 2 south of the divider berm. This decant tower will be required once deposition commences in Cell 2 and will be used to transfer water to the primary decant in Cell 1 during the operation of Stages 4 through 6. Once Cell 1 and Cell 2 come together in Stage 7, the Cell 2 decant will continue to be raised for the life of the facility serving as a backup decant tower for redundancy.

18.13 Site Water Management

18.13.1 Site Water Balance

The climate in the region of the Project is classified as semi-arid. The average annual rainfall depth is approximately 800 mm, with 85% of this depth occurring in the wet season (June through September). Rainfall during the dry season (October through May) typically only occurs during the months of April, May, and October. The minimum and maximum annual rainfall depths contained in the 54-year historical record are approximately 560 mm and 1,240 mm, respectively.

On average, rainfall occurs 33% of the days in the wet season and 4% of the days in the dry season. When rainfall occurs, the average daily rainfall depths during the wet and dry seasons are approximately 16 mm and 4 mm, respectively. The following storm event depths were estimated from the Mogtédo Station daily data:

- 2-year/24-hour: 70 mm.
- 100-year/24-hour: 140 mm.
- PMP/24-hour: 410 mm.
- PMP/72-hour: 596 mm.

The temperatures range from an average low of 14°C in January to an average high of 43°C in April. The average annual potential evaporation depth is approximately 2,980 mm, with higher rates occurring in the dry season than the wet season. A full climatological analysis was performed by KP to support the water balance.

The operational water management strategy is to utilize water captured within the mine limits to the maximum practicable extent in an efficient manner. This strategy includes significant water storage, recirculation, and reuse efforts. The following summarizes the water management strategy:

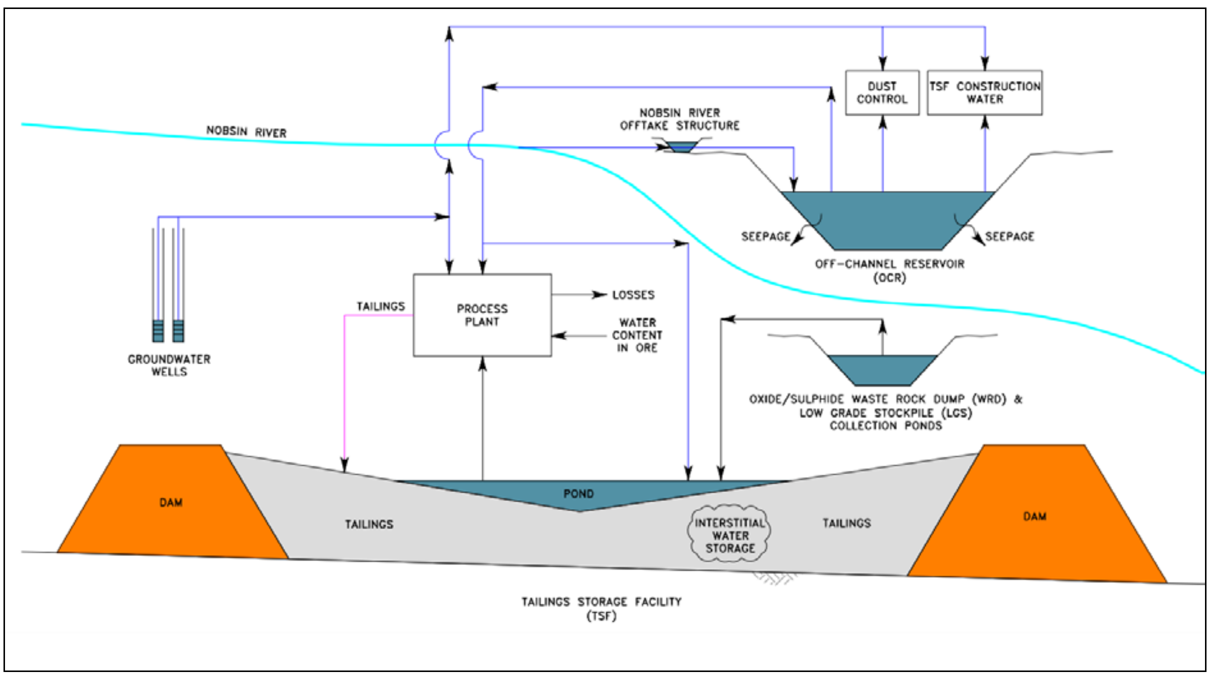
- Raw (i.e.-fresh) water from the Nobsin River will be harvested each wet season and stored in the Off-Channel Reservoir (OCR) for year-round use. The amount of water that will be harvested each year will be, per Orezone, a minor portion of the Nobsin River streamflow and will not negatively impact downstream users. The OCR will serve as the main raw water supply source for the Project. Specifically, it will supply water to the process plant, dust control, and TSF dam construction water (i.e., moisture conditioning of the dam fill) demands.
- Storm water run-off from the waste rock dumps (WRDs) and low-grade stockpiles (LGSs) will be collected in diversion channels and collection ponds. The water captured in the collection ponds will be pumped to the TSF or OCR (dependent on water quality) for use in the process.
- The water volumes that will accumulate in the TSF supernatant pond from direct precipitation, supernatant water liberated from the tailings, the WRD/LGS collection ponds, and the OCR will be reclaimed via the decant system and will be a primary source of process water to the process plant.
- When the reclaim water quantities from the TSF supernatant pond are insufficient to meet the process plant water demand, additional water will be sourced as-needed from the OCR as presented in the site wide water balance.
- On a limited basis prior to the dry seasons some water may be pumped from the OCR to the TSF for temporary storage and ultimate use in the process plant. The objective will be to add further temporary storage capacity to the water management system as a greater volume of water can be harvested from the Nobsin River each wet season. In addition, this transfer capability will be advantageous in the event the OCR needs to be drained for maintenance. This was accounted for in the water balance modelling in the months when the water harvesting was occurring in the first ten years of operation.

- Water yield from groundwater wells can also be utilized as necessary for process plant, dust control, and TSF dam construction water makeup. This is the last source of makeup water for the Project.

Localized, field-fit best management practice (BMP) sediment control structures are recommended for the areas that are not captured by the infrastructure designed as part of this Technical Report. Specifically, BMPs should be constructed prior to the full reclamation of the environmental barriers and prior to the commencement of excavation of the pits where stormwater run-off would flow outside of the mine boundary. Typical BMPs include silt fences, hay bales, temporary sediment ponds, leaky rockfill berms, etc. These temporary structures are included instead of permanent collection ponds because (1) the pits will be excavated quite rapidly and thus, the rainfall run-off will be quickly contained within the excavation footprint and (2) the majority of the year is extremely dry, and run-off will generally be negligible during the long dry season. The BMPs will be more important during the shorter wet season.

KP developed a probabilistic water balance model to predict water quantities in the mine circuit from June 2020 through December 2034, which includes the production timeframe from June 2021 through December 2034. As presented in the operational water balance schematic on Figure 18.5, the mine water circuit considered in the analyses consists of the OCR, oxide and sulphide WRDs and LGSs, process plant, TSF, and groundwater wells.

Figure 18.5 Operations Water Balance Schematic



The following summarizes the key findings of the water balance analyses:

- The TSF will have sufficient capacity to satisfy the water storage design criteria – average pond volume per month of operation plus the rainfall volume generated from the 72-hour PMP storm event plus 1 m minimum freeboard. This is illustrated on Figures 18.6 and 18.7 for TSF Stages 1 through 3 and Stages 4 through 10, respectively, which show the simulated pond elevations from the water balance.

- The maximum deterministic pond volumes and the 1% chance of exceedance pond volumes, if ever achieved, are predicted to encroach into the 1 m freeboard depth at certain times but are not projected to overtop the embankment.

- The TSF pond volumes will increase over time due to increasing contributing areas to the WRD and LGS collection ponds over time, and thus increasing contributing flows to the TSF over time. The TSF pond volume will be most affected by the collection ponds because the water detained in the collection ponds will be pumped to the TSF for ultimate storage and use in the process. Other factors that will affect the TSF pond volumes include the TSF configuration (i.e., north/south/combined basins and associated catchment and pond surface areas), the tailings slurry solids content, and the tailings unit weights.

- The OCR will require a total storage volume of 3.3 Mm^3, which includes 3.0 Mm^3 of water storage and an assumed 0.3 Mm^3 for sediment accumulations. The OCR water storage volume will be filled during each wet season for average conditions based on an assumed river flow capture efficiency to the OCR of 50%. There is a relatively low chance that the total impounded water volume would be exhausted by the end of the dry season and only in extreme dry conditions.

- The OCR water volume will be most affected by the losses to seepage, which were estimated based on the preliminary seepage analysis performed by KP. Variations or modifications to the seepage losses assumed herein will affect the required size of the OCR, the water transfer philosophy to the TSF, and the process plant water shortfall predictions.

- Minimal flows will generally be required from the groundwater wells for average conditions, but extreme dry conditions would require the full water yield from the wells.

- For average and wetter conditions, the combined process plant, dust control, and TSF construction water demand will be fully met via water from the OCR and TSF.

- During extreme dry conditions, limited additional outside source makeup water may be required for the combined water demand of the process plant, dust control, and TSF construction water. The minimal shortfalls were simulated at the end of the dry season in four years of the total mine life. They would occur only in extreme dry conditions and, if realized, a deterministic maximum flow rate of approximately 180 m^3/h would be required from external sources. On average, the required flow rate would be less than 5 m^3/h. The average and maximum cumulative makeup volumes over the life of the mine would equate to approximately 8,000 m^3 and 140,000 m^3, respectively.

Figure 18.6 TSF Pond Elevations, Stages 1 through 3

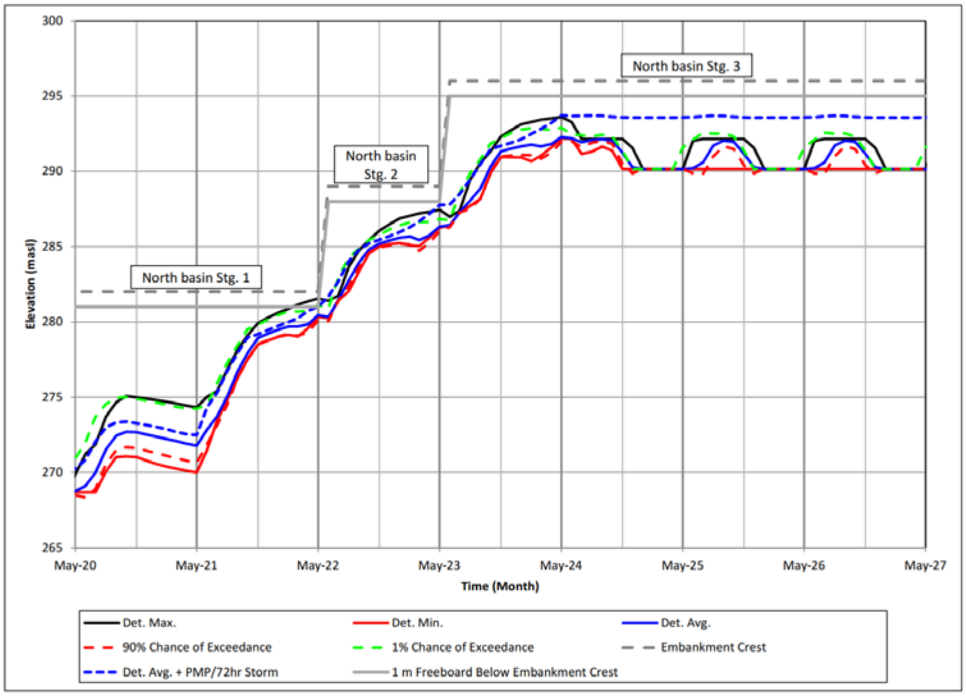
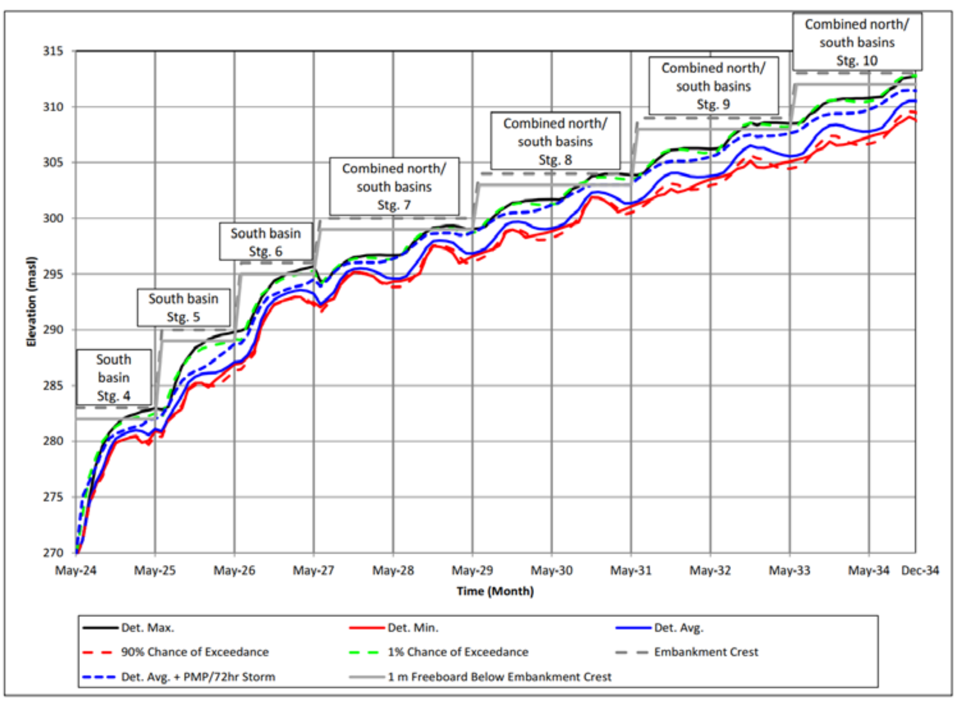


Figure 18.7 TSF Pond Elevations, Stages 4 through 10



18.13.2 Off-Channel Reservoir

The OCR will be located north of, and adjacent to, the Nobsin River. It will require a total storage volume of 3.3 Mm^3, which includes 3.0 Mm^3 of water storage, and an assumed 0.3 Mm^3 for sediment accumulations. The water storage volume was selected by Orezone to reduce the risk of water shortfalls based on KP's water balance analyses. The maximum elevation of water storage within the facility will be approximately 265 m above mean sea level (masl), per Orezone, which is coincident with the highest elevation of the offtake channel invert (at the flow-control gate location). The capacity of the OCR above this elevation is therefore not usable for water storage.

Water losses from the OCR to groundwater may potentially limit its water storage capacity and effectiveness. Therefore, a hydrogeological investigation was conducted by KP to estimate the groundwater interactions with the OCR. Four piezometers and three monitoring wells were installed across the OCR area with aquifer testing performed in each to estimate hydraulic conductivity of the aquifer. Results from this aquifer testing were utilized in a numerical groundwater flow model of the OCR area. The model was developed and used to simulate a 14-year period of OCR operation. The model simulated groundwater fluctuations caused by infiltration of precipitation and from the nearby Nobsin River. Furthermore, groundwater interactions (seepage loss to groundwater from the OCR or groundwater seepage into the OCR) with the OCR were shown to be most significant during the first several years of operations.

The model indicated a peak water loss to groundwater of approximately 16,000 m^3 per month during the initial year of operation of the OCR. Inflow into the OCR from groundwater also peaked during the initial year at approximately 15,000 m^3 per month. As this initial year showed the highest potential for losses to or gains from groundwater, simulated losses and gains from groundwater (during this initial year) were used to develop a conservative relationship between OCR stage and losses to/gains from groundwater. This relationship was utilized in the water balance.

18.13.3 Nobsin River Offtake Infrastructure to OCR and Flood Protection Levees

The Nobsin River offtake system to transfer water to the OCR will consist of a weir within the Nobsin River to create backwater and direct river flows into the offtake channel to the OCR. A flow-control gate within the channel is designed to prevent over-filling of the OCR and for maintenance purposes. Orezone will design and construct a backup pumping system to transfer water from the Nobsin River to the OCR/process plant/TSF in the event the offtake system fills with sediment and cannot operate as designed, or if maintenance of the OCR is required. Three flood protection levees will be required to prevent the ingress/encroachment of Nobsin River flows to the OCR and associated infrastructure.

The weir and offtake channel were designed to withstand (erosion protection) the 100-year flood. The capacity of the offtake system was designed such that the OCR water storage volume of 3.0 Mm^3 could be filled in one month during the wet season assuming a river flow capture efficiency to the OCR of 50% (i.e., 50% of the total simulated river flow was transferred to the OCR as-needed). The offtake channel invert elevation at the location of the flow-control gate will be 265 masl. Thus, the impounded water within the OCR once full will reach the location of the flow-control gate. The weir will be constructed of high-cube sea shipping storage containers, per Orezone. The offtake channel will be lined with concrete up to the down-chute to the bottom of the OCR, which will be lined with 1.5 mm HDPE. The flow-control gate will be constructed of concrete and steel.

The levees were designed with sufficient heights, extents, and erosion protection measures to prevent river flooding of the OCR and associated infrastructure in the 100-year flood, per Orezone. The levees will be constructed of compacted fill from the OCR excavation and will be protected with riprap. However, some of the pits are located well within the natural floodplains and cannot be adequately protected from flooding via levees (restricted zone pits). The pits planned within the natural river floodplains should be completely mined within a single dry season for safety purposes. Other levee-type structures will be designed by others as part of the standard construction practices for the pits and roads, per Orezone and Lycopodium.

The resettlement areas provided by Orezone will not be affected by the 100-year floodplain.

18.13.4 Closure Concepts

The OCR and associated Nobsin River offtake infrastructure will remain operational for the local population's use post-closure. The steel plates in the flow-control gate will be converted to lighter stop-logs for ease of use. The HDPE-lined portion of the offtake channel will be converted to riprap lining and may be realigned to the location of the ramp for better long-term stability. KP does not envisage that other modifications to the infrastructure will be required.

The diversion channels, culvert crossings, collection ponds and associated emergency spillways will likely not be required once the WRDs and LGSs are reclaimed to natural ground conditions and deemed chemically stable. Once the facilities are classified as landforms, and not mining structures, based on stability and water quality considerations, the channels, culverts, and ponds should be removed. They will not provide a benefit to the local population since the OCR will remain operational for their use.

18.13.5 Site Water Quality

A preliminary operational water quality estimation for the Project was developed. The site wide water quality model was developed to identify if the runoff, seepage, or discharge from the waste facilities would have water quality unsuitable for use in the process plant or direct discharge to surface water or groundwater. The water quality standards used in the assessment are for reference only.

Mass flow rates into the TSF from the site wide water balance were coupled with the concentrations of Constituents of Concern (CoCs) in the runoff and seepage from the Oxide and Sulphide Waste Rock Dumps (WRDs), Low Grade (LG) Stockpiles, and the tailings supernatant to estimate combined water chemistry using a mass balance approach. A separate model for the Sulphide WRDs was developed to estimate seepage and runoff water quality using the preliminary results of the Humidity Cell Tests (HCTs) on samples collected from each pit. The latest model uses data from the first eleven weeks of a forty-week test. The results of the Sulphide WRD model were incorporated into the site wide water quality model. For the water quality model, KP modelled the Sulphide WRD as a single standalone facility containing all sulphide waste rock generated by mining operations. The Oxide WRD and LG Stockpiles were modelled similarly.

The water quality model presents the average concentrations of the CoCs from operational years 2021 to 2034, and compares the concentrations to the International Finance Corporation (IFC) and World Health Organization (WHO) water quality guidelines. The TSF supernatant pond, which will be pumped to the Processing Plant for use is estimated to have a neutral pH (7.1) but exceeds both the IFC and WHO guidelines for arsenic (0.54 mg/L) and copper (3.9 mg/L). The supernatant pond also exceeds the IFC discharge guidelines for iron (55 mg/L), zinc (4.1 mg/L) and cyanide (65 mg/L). The CoCs that are predicted to exceed the WHO drinking water guidelines are cadmium (0.05 mg/L), chromium (0.08 mg/L) and lead (0.17 mg/l). The model predicts that the seepage and runoff of the Oxide and Sulphide WRDs and the Low Grade Stockpiles exceed both the IFC and WHO guidelines for arsenic.

The water quality of the TSF supernatant pumped from the plant to the TSF is the controlling input to the site wide water quality model results. However, the chemistry of the supernatant is not anticipated to impact surface water or groundwater in the Project area, as it will be stored in the fully lined TSF.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Product Sales

No formal market studies have been undertaken.

Gold doré bars will be sold to one of a number of recognized refineries servicing the West African gold mining industry. Commercial terms offered by the refineries are similar with variations between refineries of no consequence to the economic outcome of the Project. There is no indication of the presence of penalty elements that may impact the price or render the product unsalable. The doré will be transferred to the custody of the refinery 'at the goldroom door' with responsibility for in-country transport and export assumed by the refinery.

There are no material contacts in place as of the effective date of this Technical Report. Refining contracts are typically put in place with well recognized international refineries and sales are made on spot gold prices. The ability to get a contract in place for the sale of doré prior to start of production is not seen as a risk to the project.

19.2 Major Operations Contracts

It is anticipated that the following major contracts will be established to support operations:

- Mining contract(s).
- TSF construction (possibly using the same mining contractor).
- Power supply.
- Fuel supply.
- Camp management & catering.
- Site security.
- Bullion transport and refining.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The approach developed by Orezone throughout the various environmental and social studies that have been conducted since 2009, especially in the context of the Environmental and Social Impact Assessment (ESIA), emphasized stakeholder concerns and integrated the environmental and social aspects into the initial stages of the Project design. This approach maximized the Project's integration into the environment and has minimized its negative impacts, thus increasing the environmental and social acceptability of the Project. In addition, this approach allowed better consideration of the social aspects arising from the resettlement of households that will be required due to an eventual mining project.

A list of all permits received for the oxide portion of the Project and those currently in process for the sulphide expansion are listed in Section 20.2.5 below.

20.2 Regulatory and International Standards Requirements

Burkina Faso has a regulatory framework for environmental and social management. The relevant policies, laws and regulations of Burkina Faso were all considered during the implementation of the ESIA.

20.2.1 Policies and Strategies for Environmental Protection

Since the early 1990s, Burkina Faso has developed numerous policies and strategies for the protection of the environment and management of natural resources. A declaration of Mining Policy was formulated in 1995 that highlighted the importance of the private sector as an engine of economic development. In May 2013, a Mining Sectoral policy was also adopted covering the period 2014-2025. Other policies on environmental protection include:

- Plan National De Développement Économique et Social[1] (PNDES).
- Politique Nationale en matière d'Exploitation des Ressources Minières[2].
- Politique sectorielle des mines[3] (POSEM) 2015-2022.
- Stratégie de Développement Rural[4] (SDR) 2015.
- Politique Nationale en matière d'Environnement[5] (PNE).
- Politique Nationale de Développement Durable[6] (PNDD).

[1] National Plan of Social and Economic Development
[2] National Policy on the Exploitation of Mining Resources
[3] Mining Sector Policy
[4] Rural Development Strategy
[5] National Policy on Environmental Matters

- Plan d'Environnement de Développement Durable[7] (PEDD).
- Politique Nationale de Sécurisation Foncière en Milieu Rural[8] (PNSFMR).
- Programme d'Action Nationale d'Adaptation à la variabilité et aux changements climatiques[9] (PANA).
- Politique Nationale Genre[10] (PNG).

20.2.2 Legal Framework

The legal framework with respect to environmental and social aspects related to economic activities is supported by many laws and decrees, including:

- Environmental Code.
- Mining Code.
- Forest Code.
- Public Health Code.
- General Local Authorities Code.
- Act on Rural Land Tenure.
- Act on Agrarian and Land Reorganization.
- Law on Water Management.
- Act on Pastoralism.

Other relevant regulations include:

- Decree No. 20151187 / PRES / TRANS / PM / MERH / MATD / MME / MS / MARHA / MRA / MICA / MHU / MIDT / MCT on conditions and procedures for carrying out and validating the strategic environmental assessment as well as the environmental and social impact study or notice.
- Decree No. 2007-853/PRES/PM/MCE/MECV/MATD dated 26 December 2007 on specific environmental regulations for the exercise of mining in Burkina Faso.
- Decree No. 2006-590/PRES/PM/MAHRH/MECV/MRA dated 6 December 2006 on the protection of aquatic ecosystems.

[6] National Sustainable Development Policy
[7] Environmental Plan for Sustainable Development Program
[8] National Policy on Rural Land
[9] National Action Program for Adaptation to Climate Variability and Change
[10] National Gender Policy

- Decree No. 2006-588/PRES/PM/MAHRH/MECV/MPAD/MFB/MS dated 6 December 2006 determining the perimeters of protection for water bodies and streams.

- Decree No. 2001-342/PRES/PM/MEE dated 17 July 2001 on the scope, content and procedure for Environmental Impact Assessment Study and Environmental Impact Instruction.

- Decree No. 2001-185/PRES/PM/MEE dated 7 May 2001 on setting standards for discharges of pollutants into the air, water, and soil.

- Decree No. 2015-1187/PRES-TRANS/PM/MERH/MATD/MME/MS/MARHASA/MRA/MHU/MIDT/MCT dated October 22, 2015 on conditions and procedures relevant to the realisation and validation of the strategic environmental assessment and the environmental and social impact notice.

- Decree No. 2015-1200 /PRES-TRANS/PM/MERH/MME/MICA/MS/MIDT/MCT dated October 28, 2017 on the terms and conditions of environmental audit.

- Decree No. 2015-1205/PRES-TRANS/PM/MEF/MARHASA/MS/MRA/MICA/MME/MIDT/MATD dated October 28, 2015 on setting standards for discharges of used waters.

20.2.3 Mining Code

The Mining Code (Law N°036-2015/CNT pertaining to the Mining Code of Burkina Faso) is administered by the *Ministère des Mines et des Carrières*[11] (MMC) and provides the legal framework for the mining industry in the country. The state owns title to all mineral rights and these rights are acquired through a map-based system by direct application to the MMC.

The current mining code was modified following a resolution from the *Commission Nationale de la Transition*[12] (CNT) (Act 2015/CNT pertaining to the Mining Code of Burkina Faso) to review the Mining Code that has been in force since 2003. By abstraction of taxation, royalties and operational aspects, this review includes: the promotion of Companies' social responsibility and local community rights; the creation of a *Fonds Minier de Développement Local* (Decree N°2017-0024 Decree No. 2 on the organization, operation and methods of collection of the Local Development Mining Fund); the need to include a training plan for National corporate executives in the Feasibility Study (FS) and a plan for their promotion; preference will be given to local suppliers, contractors and to national employees whenever possible.

There are three types of mining permits and three types of authorizations according to the Mining Code:

Mining Permits

- Exploration Permit.
- Industrial Operating Permit.
- Operating Permit for Semi Mechanized Mining.

[11] Ministry of Mines and Quarries
[12] National Transitional Commission

Authorizations

- Prospecting Authorization.
- Traditional Artisanal Mining Authorization.
- Quarrying Exploration and Operation Authorization.

Details pertaining to the above permits can be found in Section 20.2.5.

The Mining Code guarantees a stable tax and custom regime for the life of any mine developed, except for mining duties, land taxes and royalties. The Mining Code also states that no new taxes can be imposed except for mining duties, taxes and royalties. However, the title holder can benefit from any reductions of tax rates during the life of the operating license. The holder of a permit or authorization is subject to the payment of fixed duties and proportional fees including a surface fee and a proportional royalty with the amount, the base, the rate and the methods of recovery determined by regulation.

20.2.4 Institutional Framework

The main institutional stakeholders in the environment include:

- Ministère de l'Environnement de l'Économie Verte et du Changement Climatique[13] (MEEVCC).
- Bureau National des Évaluations Environnementales[14] (BUNEE): this organization is part of MEEVCC and has the mandate to promote, regulate, and manage the environmental assessment process of the country. BUNEE holds sessions to review the terms of reference submitted by the project promoter. It formulates an opinion on the admissibility of studies and makes recommendations to MEEVCC on the environmental acceptability of projects.
- Comité technique sur les Évaluations Environnementales[15] (COTEVE): this organization was created by Decree No. 2006-025/MECV/CAB in 19 May 2006 establishing the powers, composition, and functioning of COTEVE. COTEVE is the technical and scientific framework to examine and analyse research reports and notices of environmental impacts presented by the project promoters to MEDD and MEEVCC.
- Direction Générale de Préservation de l'Environnement (DGPE).
- Direction Nationale des Eaux et Forêts (DNEF).
- Laboratoire d'Analyse de la Qualité de l'Environnement (LAQE).
- Ministère des Mines et des Carrières (MMC).

[13] Ministry of Environment, Green Economy and Climate Change
[14] National Office of Environmental Assessments
[15] Technical committee for environmental assessments

- Direction Générale des Mines et de la Géologie[16] (DGMG).
- Bureau des Mines et de la Géologie du Burkina[17] (BUMIGEB).
- Chambre des Mines du Burkina Faso[18] (CMB).
- Commission Nationale des Mines[19] (CNM).
- Ministère de l'énergie[20] (ME).

Other Ministries and Departments involved:

- Ministry of Infrastructure.
- Ministry of Territorial Administration and Decentralization.
- Department of Health.
- Department of Agriculture and Food Security.
- Department of Water, Hydraulic and Sanitation.
- Ministry of Animal Resources and Fisheries.
- Ministry of Social Action and National Solidarity.

20.2.5 Required Permits

The application for an Industrial Operating permit requires an ESIA that must first be accepted by MEEVCC. The ESIA must be supported by a Feasibility Study (FS) and must include a Resettlement Action Plan (RAP) that has been accepted by all stakeholders if the project requires the exploration of land held by any resident. Once in production, the holder of an Industrial Operating Permit is required[21] to open, under his name, a fiduciary account called "*Fonds de préservation et de réhabilitation de l'environnement minier*[22]" at the *Banque Centrale des États de l'Afrique de l'ouest*[23] (BCEAO). This account must be funded annually on January 1st, by an amount equal to the total rehabilitation budget presented in the ESIA, divided by the number of years of production to cover the costs of mine reclamation, closure and rehabilitation.

[16] General Management of Mines and Geology
[17] Bureau of Mines and Geology of Burkina Faso

[18] Chamber of Mines of Burkina Faso
[19] National Commission of Mines
[20] Ministry of Energy
[21] Decree No. 2007-845/PRES/PM/MCE/MEF.
[22] Fund for the Preservation and the Rehabilitation of the Mining Environment.
[23] Central Bank of West African States.

In 2016, Orezone received the Industrial Operating Permit following the delivery and acceptance by the authorities of the ESIA and RAP.

In February 2019, Orezone signed the mining convention with the State of Burkina Faso. The purpose of the mining convention is to clarify the rights and obligations of the parties and to guarantee Orezone stability, including taxation and foreign exchange regulation. The mining convention is not a substitute for the law but specifies the provisions of the law. It is valid for the initial duration of the operating license and is thereafter renewable for one or more periods of five years at the request of Orezone.

The required permits and administrative procedures are presented in Table 20.1.

Table 20.1 List of Required Permits and Authorizations

Permit / Authorization	Main Requirements	Timeframe	Costs

Environmental Compliance Certificate	• Delivered by the Minister of the MEEVCC based on a compliant ESIA Report to Société Orezone Inc. SARL. on May 12, 2016 by “Arrêté n°2016-0295/MEEVCC/CAB”:
	• Submission of the ToR of the ESIA to the BUNEE
	• Validation of the ToR
	• Submission of the draft version of the ESIA to the BUNEE
	• Public Inquiry
	• COTEVE Session
	• Final ESIA Report approved by the BUNEE
		Completed on the 2015 application area.	• Cost for review and validation of the ToR: 500,000 FCFA.
			• Fees for the public inquiry and COTEVE session:
			  • Public inquiry: 6,666,000 FCFA
			  • Session: 2,870,500 FCFA.
			• Cost for file processing, Project value of 50 billion FCFA and more:
			  • Flat Fee: 25 million FCFA.
			  • Proportional rights: 0.02% of the total investment cost.
			  • Total: 56,344,805 FCFA

Environmental Compliance Certificate	• To be delivered by the Minister of the MEEVCC based on a compliant ESIA Report
	• Submission of the ToR of the ESIA to the BUNEE, (2019-05-01)
	• Validation of the ToR (2019-06-18) by the BUNEE
	• Submission of the draft version of the ESIA to the BUNEE
	• Public Inquiry
	• COTEVE Session
	• Final ESIA Report approved by the BUNEE
		On-going for the sulphide expansion and the restricted zones.	• Cost for review and validation of the ToR: 500,000 FCFA.
			• Fees for the public inquiry and COTEVE session:
			  • Public inquiry: 6,666,000 FCFA
			  • Session: 2,870,500 FCFA.
			• Cost for file processing, Project value of 50 billion FCFA and more:
			  • Flat Fee: 10 million FCFA.
			  • Proportional rights: 0.04% of the total investment cost.
			  • Total: 22,760,000 FCFA

Permit / Authorization	Main Requirements	Timeframe	Costs

Industrial Operating Permit	• Delivered to Société Orezone Bomboré SA. on December 30, 2016 by "Décret n°2016-1266/ PRES/PM/MEMC/MINEFID/MEEVCC"	Completed for the oxide reserves outside of the restricted zones.
		
		On-going for oxide reserves within the restricted zones and for the sulphide expansion as contemplated in the current study.
			• Costs for the National Commission of Mines session: 2,500,000 FCFA.
			• Fixed duties on mineral titles: 5 million FCFA.
			• Proportional rights (area taxes):
			  • First five years: 7.5 million FCFA/ km^2/y.
			  • Year 6 to10: 10 million FCFA/ km^2/y.
			  • From the year 11 onward: 15 million FCFA/ km^2/y.
			• Cost of proportional royalty on gold production: 5% of turnover if gold is higher than US$ 1,300, 4% if US$1,000-1,300 and 3% if less than US$1,000.

Environmental and Social Management Plan Protocol	• Protocol to be signed between Orezone Bomboré SA and the BUNEE with respect to the monitoring of the ESMP by the BUNEE	30 days before the beginning of the construction.	• Environmental inspection visit: typically, 2 visits per annum, by BUNEE: 5,000,000 FCFA per annum. • Environmental audit visits: every 3 years, by independent consultant and submitted to BUNEE, about 1 million FCFA per visit. • ESMP monitoring program: protocol to be agreed between Orezone and BUNEE.
Authorization for the Management of raw water	• Application to the Directorate of Legislation and regulations of the Ministry in charge of water. • Application to the Nakanbé Water Agency.	30 days before the beginning of the construction. Water consumption must be reported on a quarterly basis.	• Tax of raw water for mining and industrial purposes: 125 FCFA/m^3
Authorization for the Collection of raw water for civil work	• Application to the Nakanbé Water Agency • However, tax must be paid.	30 days before the beginning of the construction. Water consumption must be reported on a quarterly basis.	• 10 FCFA/m^3 for every m^3 backfill placed. • 20 FCFA/m^3 for every m^3 of concrete poured
Authorization for Road infrastructures	• Authorization from the Ministry of Transport and the Ministry of Environment only required outside of the Mining Lease or for infrastructure that was not included in the ESIA.	-	• Depending on the infrastructure and technical studies that have been conducted.
Authorization for Hydraulic work or dam	• Authorization from the Ministry in charge of water based on technical studies.	-	• Depending on the work and technical studies that have been conducted.

20.3 Baseline Studies

20.3.1 Baseline Studies Conducted

In July 2009, an Environmental Baseline Study was completed for the Project by the *Bureau d'Études des Géosciences des Énergies et de l'Environnement* (BEGE) (BEGE, 2009a) located in Ouagadougou. The study focused on collecting data from existing sources and new field studies to establish an appropriate baseline for measuring the overall environmental and social impacts of the Project. In September 2009, an Environmental Assessment Report was also completed by BEGE (BEGE, 2009b) to conceptually assess the environmental and community impacts related to the possible development of the Project within the Bomboré 1 permit.

At the end of 2010, Orezone commissioned G Mining Services Inc. (GMSI) to prepare a National Instrument 43-101 (NI 43-101) Compliant Preliminary Economic Assessment (PEA) for the Project and this PEA was delivered in June 2011.

In 2011, Orezone commissioned SOCREGE to conduct a socio-economic study of the Project, and BEGE to conduct an environmental impact study based on the carbon-in-leach (CIL) project description delivered by GMSI in June 2011. The Terms of Reference (ToR) for both studies were validated as required by the BUNEE, which classified the Project as a Category A project that was subject to an ESIA in accordance with Article 9 of Decree No. 2001-342/PRES/PM/MEE. The ToR established by Orezone stipulated that both the SOCREGE and BEGE studies had to meet the Equator Principle requirements and the relevant International Finance Corporation (IFC) standards.

SOCREGE delivered an interim report in January 2012 (SOCREGE, 2012) on their demographic and socio-economic studies within the 2011 GMSI Project footprint.

In June 2012, Orezone commissioned GMSI to prepare a NI 43-101 compliant feasibility study for the Project based on an oxide milling CIL scenario. The 2012 FS engineering and geotechnical investigations led to a new proposed site layout, which required an expansion of the ESIA study area to be covered by both SOCREGE and BEGE. This area included the surface mine, infrastructures and access roads but excluded potential resettlement sites.

SOCREGE delivered a second interim report in January 2013 on their demographic and socio-economic studies within the expanded Project footprint. BEGE delivered an interim report in July 2012 on their environmental baseline studies also within the 2011 GMSI Project footprint and another interim report in June 2013 on their botanical and archaeological studies within the expanded footprint. In April 2013, Orezone commissioned Cabinet Archi Consult to conduct a financial evaluation of the replacement cost of the buildings potentially impacted by the Project development and to produce the architectural plans and site layouts for the resettlement sites and buildings. As of December 2013, detailed baseline studies were completed over a study area that covered 83 km^2, and baseline studies, discussed below, were in progress in adjacent areas favorable to host the relocated population.

In 2014, BEGE delivered a report setting out the results of the environmental baseline studies conducted since 2009 on the 2013 GMSI Project footprint. New ToR were prepared and sent to BUNEE in July 2014 based on the changes to the Project. In parallel, updated environmental and social baseline studies were conducted by BEGE, SOCREGE and Cabinet Archi Consult. This baseline characterization of the physical, biological and human components was done through different field missions that occurred during both the 2014 wet and dry seasons and until February 2015 for the human components. The data collected was used to describe the initial conditions of the natural environment and has been considered in the Project's design, ESIA and RAP.

In May 2015, Orezone applied for an Industrial Operating Permit to the Ministry of Mines and coincidently submitted to the Ministry of Environment a preliminary version of the ESIA for the Project together with the preliminary RAP. The BUNEE conducted the Public Enquiry in November 2015.

In January 2016 Orezone presented the ESIA together with the RAP and FS to the Burkina Faso authorities (COTEVE). The ESIA and RAP were reviewed by Burkina Faso authorities and Orezone addressed all the comments with final revised versions of both documents submitted to the Ministry of Environment in April 2, 2016 The Minister of Environment delivered to Orezone Inc. s.a.r.l. on May 12, 2016 a favorable opinion about the Project by way of Arrêté n°2016-0295/MEEVCC/CAB.

In January 2019, Orezone commissioned SOCREGE to update the socio-economic study of the Project, and BEGE to update the environmental impact study of the Project by including additional areas (P17S area located in the south and 3 areas in rivers floodplains) as well as the sulphide expansion phase of the Project.

20.3.2 Description of the Main Environmental and Social Components

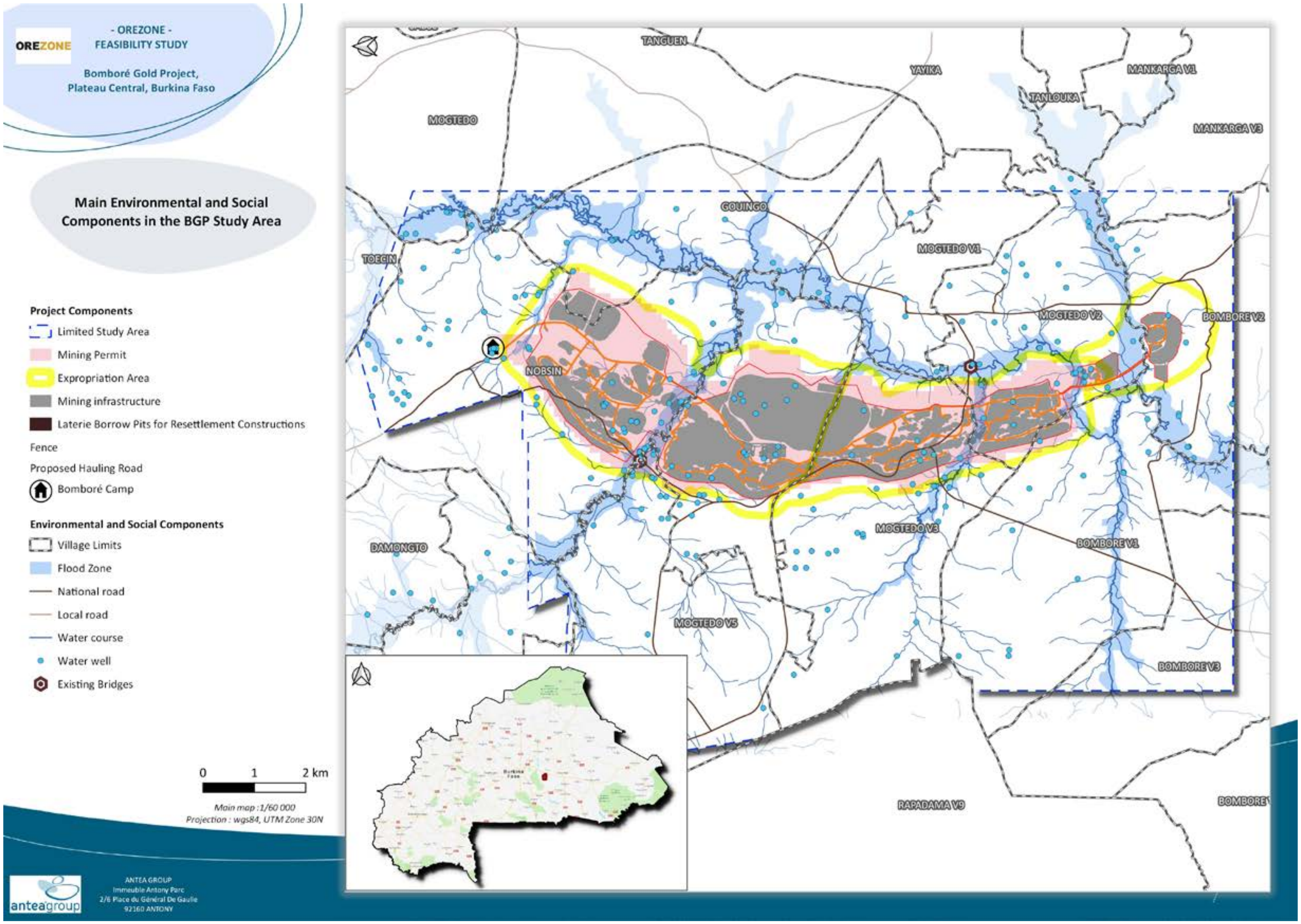
The Project site is within the Sahelian climate zone and has a semi-arid type of climate that is subject to a wet season from May to October and a dry season from November to April. The average annual rainfall varies from 700 mm to 900 mm. Monthly minimum and maximum temperatures recorded by Orezone on the property between October 2011 and April 2015 ranged from 18°C in December-January for the average monthly minimum and 40°C in April for the average monthly maximum. Monthly minimum and maximum humidity levels during the same period ranged from 6% in the dry season (measured in February) to 96% in the wet season (measured in August and September) with high evaporation and evapotranspiration rates. Daily sunshine is nine hours per day on average. Prevailing winds during the wet season are from west direction, shifting to the dry Harmattan from east direction during the dry season.

The land is generally flat with a few scattered hills whose altitude reaches up to 344 m; these hills consist of Birimian rocks typically overlain by ferruginous duricrust formations. The Project site is in a low-risk seismic zone. Surface water and stream flow is confined to the wet season up to November. Siltation of any river-based water storage facilities is extremely rapid due to the torrential nature of the rainfall. The main water courses include the Bomboré River and the Nobsin River, a Bomboré tributary.

Field hydrogeological drilling results indicate that the water table is present within the first 40 m below surface and most of the groundwater occurs within the first 80 m below surface. Groundwater is generally of good potable quality. Except for arsenic (As) levels exceeding World Health Organization (WHO) drinking water quality standards in some of the boreholes (as well in some surface water samples), almost all samples tested were within the WHO drinking water quality standards and the Effluent Guidelines set by IFC and Burkina Faso for the mining sector. The As level should not be an issue as the water will be collected in the open pits through the operation, reused in one of the processes or sent to the sedimentation ponds where it will be monitored and if necessary treated before discharge.

Eight morpho-pedological units were identified and ascribed to four major units with respect to their agroforestry potential, from totally improper for agriculture (Unit A: 12% of the Project area), to marginally apt for pluvial crops (Unit B: 15% of the Project area), apt to moderately apt for pluvial crops, (Unit C: 57% of the Project area) and apt to moderately apt for pluvial crops, pluvial rice and orchards (Unit D: 16% of the Project area). Eroded areas are numerous and are caused by run-off water, wind and human activities (grazing, small scale mining).

Figure 20.1 Main Environmental and Social Components in the Study Area of the Project



Deforestation is widespread over the permit area. Vegetation in uncultivated areas is mainly composed of savannah with denser vegetation, shrubby vegetation and riverine forests, growing only along the watercourses and the draining system. The main vegetation types comprised inside the study area can be divided into four main types, i.e. cultivated parcels (42% of the Project area), savannah (37% of the Project area), riverine forest and riparian formations (15% of the Project area, essentially along the Bomboré and Nobsin Rivers and tributaries) and barren lands (6% of the Project area).

Due to human pressures and degradation of the natural environment, terrestrial fauna is generally scarce, whilst avian fauna is diversified. Hare, hedgehog, squirrel, rat, wild cats, varan, land turtles, frogs, toads and adders have been observed. Bird species observed include pigeons, turtledoves, guinea fowls, partridges, herons, ducks, coucals and brown vultures. Fish species including carp, catfish, sardine, tilapia and lungfish have been recorded inside the hydrological network. No terrestrial species has a conservation status at either national or international level, except for one bird species, the Hooded Vulture (*Necrosyrtes monachus*), that is Endangered according to the IUCN red list.

In addition to degradation of vegetation and animal species, the artisanal mining activities have caused a significant change in the soil and surface water quality. Digging of tunnels and accumulation of rocky material, creating small water tanks, the use of chemical products for the treatment of ore and waste water discharges, are activities that modified the environmental conditions in sectors in which these activities are practiced. These changes resulted in soil erosion and water pollution.

On the administrative and human terms, the land ownership within the Project area is peculiar due to the coexistence of the traditional and modern land tenure schemes. In the traditional system, families have inherited or have been temporarily granted the right to use the land but there is no private ownership of the land.

In 1974, a state company named *l'Autorité pour l'Aménagement des Vallées de la Volta*[1] (AAVV) was created with the mandate to create new villages and the required infrastructure within parts of the Volta River catchment that were previously affected by onchocerciasis, or river blindness, and trypanosomiasis, or sleeping sickness. The main objective of this voluntary resettlement program was to reduce the demographic pressure within the Mossi plateau by colonizing more fertile lands along the Volta Rivers. The Mogtédo area, where the Project is located, was one of the areas selected for the Aménagement des Vallées des Volta[2] (AVV) resettlement program. From 2003 to 2006, a project named the *Plan foncier rural du Ganzourgou*[3] ("PFR-G"), financed by the *Agence Française de Développement*[4] ("AFD"), built a database of the land ownership within the AVV area, proceeded with the survey of the individual and communal plots, created a land register and delivered a deed title to the land owners. About 75% of the Project is within the PFR-G area, and about 58% of the area occupied by fields and communal parcels has been registered.

[1]Authority for the Development of the Volta Valleys.
[3] Development of the Volta Valleys.
[3]Ganzourgou Rural Land Plan.
[4]French Agency of Development.

The total population in the Project area is approximately 7,700 inhabitants. Women represent on average 52% of the population and each household has an average of seven people. Youth (age 0-20) account for 60% of the population. Added to this is the population of the two artisanal mining villages (Kagtanga and Sanam Yaar), which have a combined population of approximately 4,600 inhabitants. The proportion of men in these two villages is 56%.

There are primary schools within the Project area located in the villages of Nobsin, Mogtédo V3 and Mogtédo V4. Only about 16% of the community is literate. Health facilities are very basic with only two public health clinics in Nobsin and Mogtedo V3 that are within the Project area. More than 35% of the population suffers from malaria. The number of new AIDS cases has declined steadily since 2010 with 72 cases in 2012 as compared to 108 in 2010.

Agriculture is very important in the Project area but yields are low and declining due to the intensive nature of the activity; sorghum, corn and millet account for the bulk of the production. Most of the producers (86%) are using seeds from their own crop and organic manure as fertilizer but 63% also buy some chemical fertilizers. Most households practice animal husbandry; most own livestock consisting of poultry, cattle, sheep and goats. The collection of non-ligneous forest products is also a source of food and medicinal plants. Depending on location, almost three quarters of households practice artisanal mining, which is a valuable source of income. Crafts, hunting and tourism are marginal activities.

Most households have no toilets with latrines. Approximately 96% of wastes are disposed in dumps and/or buried. Several wells are present in the Project area and include some large diameter traditional wells and new drillings. There is no access to the electricity network (SONABEL) in the Project footprint with only a few households having private equipment (solar panels or generator) that is used mainly for lighting, to recharge mobile phones and to watch national or satellite television.

Inventories carried out in 2012, 2014 and 2019 identified 156 archaeological sites and 195 ethnographic sites (graves and sacred sites) in the Project area. Some objects were collected by Orezone and stored for their preservation, including former furnaces and some artifacts. With the appropriate ceremonies, it will be possible to move or abandon all of these sites that are sacred places, an exercise that has already been essentially completed.

20.4 Community Information and Consultation Program

The stakeholder information and consultation process is an integral part of the ESIA. To date, Orezone has put in place mechanisms and communication tools so that all those involved in, or affected by, the Project can freely express themselves. The information collected during these consultations has helped identify issues, risks, benefits, and opportunities for the Project to avoid, minimize, or offset negative impacts and enhance the positive ones.

As part of the stakeholder information and consultation process, a Stakeholder Engagement Plan (SEP) was developed. Information about the Project was transmitted by information sheets meetings with administrative authorities, technical services, as well as representatives of the surrounding villages and public radio broadcasts.

Many initiatives have been undertaken by Orezone to inform and consult with affected communities as part of its exploration activities and preparation of the Project. These initiatives include:

- Establishment of a permanent team for environmental and community relations.
- Adoption of a Stakeholders Engagement Plan.
- Establishment of a grievance mechanism procedure.
- Adoption and implementation of a Sustainable Community Development Program.
- Several ad hoc meetings with authorities and other stakeholders.

A Provincial Compensation and Resettlement Committee of the people affected by the Project was set up by Order No. 2013-010/MATS/RPCL/PGNZ/HC-ZRG dated May 28, 2013 and was officially activated on April 4, 2014. The first public meeting was held in July 2014 to discuss issues related to resettlement. In addition, Orezone has established, at early stages of the study, a community information and consultation mechanism, which has been implemented throughout the ESIA process. The main concerns raised during the communication activities included:

- Disturbance of subsistence activities.
- Compensation to be supplied to traditional landowners.
- Air, water, and soil degradation.
- Disruption of sacred sites.
- Promotion of women.
- Access to jobs and training.
- Influx of foreign workers and spread of disease.
- Road safety and accident prevention.
- Closure plan and the safe take-over of the land by the local communities after the mine closure.
- Control and transparency during the implementation of social and environmental compensation measures.

Orezone considered these concerns expressed by stakeholders and incorporated specifications to optimize the Project design to avoid and manage any of these constraints. These actions led to a more balanced approach between the financial objectives of Orezone and the preservation and conservation of the environmental and social components, which are part of sustainable development.

20.5 Project Impacts, Risk Analysis, Environmental and Social Management Plan

20.5.1 Project Impacts

The methodology used to identify and analyze the environmental impacts is based on an approach recognized by international funding agencies. This approach identifies the direct interactions between the Project activities considered impact sources and physical, biological, and human components. These interactions are customized according to project-specific phases (pre-construction, construction, operation, and closure). All interactions identified are then analyzed based on three criteria (intensity, extent and duration) to obtain a global indicator, of the absolute importance of the impact. The importance of the impact is then qualified as; minor, medium, or major.

Most of the impacts on the physical environment are of low or medium absolute importance given the predicted disturbances on air, soil, surface, and groundwater during the construction and operational phases, subject to proper implementation of the mitigation measures. The mining installation and operations were designed for zero water discharge, which is a clear advantage in terms of minimal impact. Protection of the natural groundwater from process water containing cyanide and from sulphide waste rock dumps run-off water containing leachable metals was properly planned.

Impacts on the biological components are mostly minor since these components are poorly represented in the Project area. Impacts on the human components have an importance ranging from minor to major depending on the particular issues.

The most significant impact caused by the Project will be the resettlement of the population currently living on the Project site. Although Orezone has assumed the expropriation of the whole area within the 500 m buffer zone because of the development of the Project, the land could remain accessible for farming activities in the outer 250 m portion of this buffer zone potentially reducing the loss of the AVV fields and communal parcels by more than 30%.

The economic impact of the Project at local, regional, and national levels is extremely positive. Beginning with the construction phase, direct and indirect jobs will be created, resulting in tangible economic benefits for both local and regional communities. The Project will create hundreds of skilled and unskilled direct and indirect jobs, most of them awarded to Burkinabe workers. This job creation will increase household incomes and improve living conditions. In addition, the procurement of goods and services required for the construction, operation, and closure of the mine will bring significant economic benefits to local and regional businesses, the majority in terms of supplying food and/or various products and services.

The revenues generated by the mine operation will increase Burkina Faso's internal revenue through taxes and royalties charged by the local and regional authorities, revenues that should have a beneficial impact at the local and regional levels through increased investments in social and health services, and local infrastructure.

In addition, Orezone supports several social programs for displaced households and in a broader context, local and regional communities. Small scale artisanal mining activities currently exist in the Project area. Orezone has identified within its exploration's licenses two economically viable deposits that miners from the two expropriated villages, Sanam Yaar and Kagtanga, could apply to the state for legal artisanal mining permits without objection from Orezone. Orezone would retain right of first refusal should the permits be abandoned or sold in the future.

20.5.2 Risk Analysis

A Preliminary Risk Analysis was conducted to assess the environmental risks of the Project. Like any other heavy industrial activities, the Project may unintentionally experience critical issues like spills, emissions and fires that could have a direct negative impact on the surrounding environment. The causes and consequences of each of these situations were determined and detailed preventive and emergency implementation measures were identified. The criteria considered for this risk assessment consider the severity of events, as well as the consequences and the likelihood of an occurrence.

An analysis of the Project's facilities and consumables to be used on the mine site revealed several involving risks. The main environmental risks associated with the Project are as follows:

- Fire.
- Explosion.
- Degradation of walls and ramps in the pits and waste dump areas, berms and retention structures.
- Spills or leaks of hazardous materials.
- Toxic emissions.
- Natural disasters.
- Insurrection of the population.

To minimize the level of risk related to both personnel and the environment, health and safety and security measures have been identified. In addition, an Emergency Response Plan (ERP) will be implemented at the earliest stages of the operational phase of the Project. The main objective will be the management of those risks, which cannot be eliminated by the protection measures already in place so that the ERP will immediately be initiated if any such incident or accident occurs. The intent of the ERP is to define emergencies that could reasonably occur, and the measures of prevention, preparedness, response, and repairs required for such situations, including staff training.

20.5.3 Environmental and Social Management Plan

The Environmental and Social Management Plan (ESMP) presents all the environmental and social management measures to be implemented as part of the Project including all the operational aspects. The ESMP covers all project phases and covers the avoiding, minimizing, enhancing, or compensating of the various anticipated negative impacts by reducing them to an acceptable level for all stakeholders.

The ESMP identifies the objectives to comply with the regulations in Burkina Faso and international best practices in the mining sector. The ESMP also includes environmental monitoring programs and environmental and social follow-up, providing the basis for assessing the effectiveness of management measures to be implemented by Orezone. The ESMP includes several measures to strengthen the capacity of the stakeholders concerned by the application of environmental and social management measures.

Management measures are to be implemented at the earliest stages of the construction phase. Some measures will last throughout the operations at the mine site and others will last beyond the closure and rehabilitation phase of the Project. The planned management measures for the physical, biological, and human components include the following:

- Protection of soils.
- Control of run-off water, restrictions during heavy rain periods, respecting buffer zones along watercourses, etc.
- Implementation of restrictions regarding cutting trees, limits for working areas, etc.
- Reduction of noise and dust emissions.
- Control of traffic speed, access roads, the use and maintenance of equipment (fuel and lubricant tanks, vehicles and motorized equipment, etc.).
- Management of human resources, logistics, mobilization and demobilization of personnel and contractors.
- Management of the arrival of unwanted ‘opportunistic’ populations in the area i.e. people expecting jobs and commercial opportunities related to construction and exploitation of the mine.
- Maximization of job opportunities for the local workforce, of supplies of goods and services by local stakeholders, and of women’s benefits and management of unrealistic expectations.
- Population and workers awareness to the risks of transmitting HIV/AIDS and other endemic diseases.
- Precise location and protection of worship and sacred sites.

Some measures implemented during previous project phases concerning soil, surface water, groundwater, ambient noise, population and social cohesion, economy, and infrastructure, etc. will be maintained during the operational phase. Several additional measures will include the following:

- Monitoring of the mine tailings site in compliance with the applicable regulations and requirements.
- Management of waste rock dumps and progressive re-vegetation to minimize wind erosion.
- Management of water, hazardous materials, wastes, traffic, maintenance of vehicles, etc.

Mining will be carried out according to best practices and with specific attention to occupational health and safety.

Finally, various management measures are planned for the closure phase and include the following:

- Dismantling of infrastructure and facilities, except for structures that will be kept in place and handled over to the local authorities without compromising the integrity and security of places and people.
- Site rehabilitation and re-vegetation.
- Restoration of livelihood conditions for neighbouring populations and workers.

Access roads, power lines and other infrastructures built for mining will be left in place, as necessary, for use by communities at the end of mine life. Restricted areas may be defined within the permit to protect the environment, the natural habitat, archaeological sites or public interest infrastructures.

A monitoring program will be implemented during the construction phase and will be conducted by Orezone on an ongoing basis. The program will ensure compliance with the commitments agreed to as part of the ESIA and environmental obligations, as well as compliance with the proposed management measures and with laws, regulations and other environmental considerations included in the contractors' technical specifications. These measures to implement will be included in the contractors' technical specifications according to their respective activities.

Although the Project area includes habitats heavily modified by human activities, including degraded critical habitats, it supports some special-status species in terms of biodiversity. The Project's environmental acceptability by the National Authorities as well by the regional and local communities is related to the consideration of these biodiversity issues.

The environmental and social follow-up program to be implemented will:

- Monitor changes for certain sensitive environmental components.
- Compare current conditions with pre-Project initial conditions to identify trends or impacts that may result from Project activities or natural events.

The main elements planned as part of the Project's follow-up monitoring activities include:

- Surface and ground water quality.
- Ambient air quality.
- Ambient noise.
- Status of the flora and effectiveness of re-vegetation.
- Fauna.
- Local and regional economy.
- Gender.
- Social cohesion.

Regarding water quality, the monitoring will determine if arsenic is leaching from the weathered mining wastes and if it is present in the process water. The geochemical studies conducted to date suggest that arsenic leaching will be minimal. Additional geochemical characterization will be performed at early construction phase to refine the existing geochemical model.

20.6 Resettlement

20.6.1 People and Activities Affected by the Project

The resettlement of many people (about 731 households or about 5,095 people) from seven traditional villages, as well as two artisan gold processing sites (about 1,360 households or about 3,100 people) and the expropriation of a large area of agricultural land (about 656 ha) represents a complex activity that will require an immediate and important focused effort. The processing infrastructure is in the northern area of the Project where about 60% of the gold resources are located. This area will have to be cleared prior to the start of any major construction activities. This will require the initial (Stage 1) resettlement of approximately 410 households from traditional villages and the expropriation of approximately 915 households from the Sanam Yaar artisanal gold processing site. The subsequent resettlement (Stage 2) of approximately 250 farming households and the expropriation of 450 households from the Kagtanga artisanal gold processing site, all from the southern area of the Project, could occur after the initial Phase 1 resettlement as this area will not be immediately affected by the mine construction.

Orezone has successfully completed the expropriation and the settlement of the compensations to the households from the Sanam Yaar and Kagtanga artisan gold processing sites and construction of the Phase 1 resettlement sites is in progress.

The reports issued by SOCREGE, BEGE and Cabinet Archi Consult, complemented by Orezone field validation and updated inventories as of June 2019 were used as a baseline for the Resettlement Action Plan that is included in the new ESIA that will cover mining of oxides in the restricted zones and the future sulphide expansion Phase of the Project. These reports contain an inventory conducted on the households and their properties per village within the study area, in addition to an overall inventory of the existing public infrastructures.

20.6.2 Scope of Resettlement

The RAP scope is based on the following assumptions:

- The area impacted is based on the site layout with a 500 m buffer zone to be expropriated around the mining infrastructure (pits, pads, ponds, haul roads, etc.).
- Census data and property inventories as of March 2018 for Phase 1, May 2018 for Phase 2 and June 2019 for Phase 3.
- Staged development of the Project, i.e. Stage 1 in the North during the construction period (Year-2), to be followed by Stage 2 in the South (Year -1). The Stage 3 in the South and Southeast will be conducted during Year 2.
- Best-practice strategy to meet the national and IFC standards. The budget includes the replacement of all private houses, a latrine for each household, all public infrastructures upgraded to the national standards, plus financial compensation for granaries, sheds, ovens, parks, shops, roosts and similar small infrastructures.
- Cash compensation for expropriated farmlands at market value plus crop compensations over five years were included in the estimate. Crop compensations are basedon the value of a basket of harvested products, as established from the average yields and market values compiled by the local branch of the Ministère de l'Agriculture, des Ressources Hydrauliques, de l'Assainissement et de la Sécurité Alimentaire (MARHAS)[5] for the area, and the surface area of the parcel, as recorded during the field study. No compensation for farmland in the 250 m to 500 m buffer zone is planned as these farmlands will remain active.

20.6.3 Staged Resettlement

The resettlement of villages is progressive and divided into stages as a function of the mining plan (Cf. Figure 20.2).

Stage 1 of the resettlement program covers an area of 1,833 ha and will result in the physical and economic displacement of approximately 410 households (2,415 people) and 392 ha of farmland, specifically portions of the villages of Nobsin, Goingo, Mogtédo V4 and Mogtédo V5. The village of Sanam Yaar will also be moved: about 915 households and 1,700 people. The program has commenced since this area is required for Project construction.

[5] Ministry of Agriculture, Water Resources, Sanitation and Food Security

Stage 2 covers an area of 1,055 ha where approximately 250 households (1,500 people) will have to be relocated and 646 ha of farmland compensated, specifically for the villages of Mogtédo V3, Mogtédo V2 and Bomboré V1. The village of Kagtanga will also be moved: about 450 households and 1,400 people. Stage 2 of the resettlement program will be initiated immediately after Stage 1 and is expected to be completed before the end of 2020.

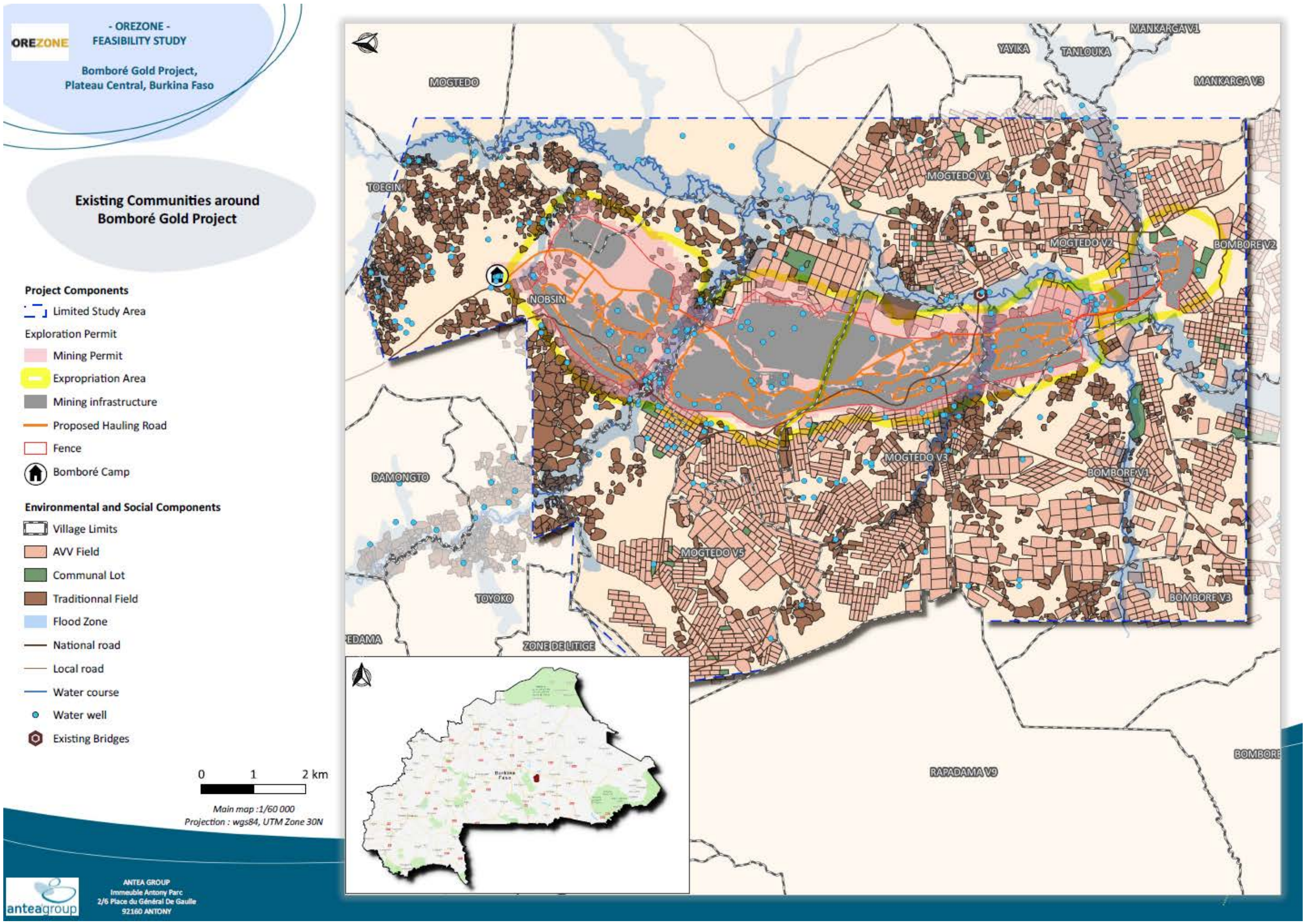
Stage 3 covers an area of 180 ha where approximately 111 households in the vicinity of P17S area will have to be relocated and 140 ha of farmland compensated, specifically for the villages of Mogtédo V2 and Bomboré V2. Stage 3 of the resettlement program will be initiated immediately at year 3, prior to start-up of the sulphide plant.

The RAP budget is presented in Table 20.2.

Table 20.2 Resettlement Action Plan Budget (US$)

Area	Capex	G&A	Total
Private Buildings	$15,689 889	$0	$15,689 889
Public Buildings	$1,143 613	$0	$1,143 613
Land Expropriation	$1,047 786	$0	$1,047 786
Crop Compensation	$261 102	$1,760 776	$2,021 878
Tree compensation	$1,721 298	$0	$1,721 298
Bridge and Bypass Roads	$1 903 318	$0	$1,903 318
Misc. Allowances	$1,339 713	$0	$1,339 713
Community livelihood compensation	$397 254	$1,638 897	$2,036 151
RAP implementation	$1,083 105	$301 515	$1,384 620
Total	**$24,587 078**	**$3,701 188**	**$28,288 266**

Figure 20.2 Existing Communities and Proposed Resettlement Sites for the Project



20.7 Acid Rock Drainage and Metal Leaching

Samples of waste rock, potential construction materials and tailings were subjected to laboratory geochemical tests to assess their potential to generate ARD and to leach metals (ML).

The results of the ARD assessment were compared to the evaluation criteria presented in the Global Acid Rock Drainage (GARD) Guide (INAP, 2009), a reference document on best practices related to mine waste characterization and ARD prediction, prevention and mitigation measures. Results of the metal leaching tests and process water chemistry were compared to the applicable effluent discharge guideline values specified in Section 2.0 of the IFC/World Bank Group EHS Guidelines for Mining (IFC, 2007).

20.7.1 Waste Rock and Construction Materials

The ARD/ML assessment of the waste rock focused on dominant rock types and weathering zones (Oxide and Transition) from the CFU, KT, Maga, P8P9, P11, P16, P17, Siga East, West, and South prospects. Three separate sampling and analytical programs were initiated in 2011 (McClelland), 2012 (Golder) and 2019 (KP), a total of 141 rock samples were collected from exploration drill cores from within the open pit outline while the laterite was collected from a nearby borrow source. The results of this ARD/ML programs to date are presented below.

Most of laterite, oxide (saprolite), and transition (saprock) units demonstrate little potential to generate ARD (non-PAG) and are not expected to leach metals at concentrations above the Burkina Faso nor the IFC effluent guidelines. Therefore, laterite, saprolite, and saprock material is considered suitable as potential construction material for most rock types.

However, saprock from the mafic intrusive (MI3) unit at P8P9 and Siga South, as well as the meta-sandstone (S3) unit at P8P9 and Siga South & East as well the meta-pelite (S4) indicated some potential to generate ARD. In addition, the meta-pelite (S4) unit indicate Arsenic leach potential.

Additional geochemical investigation will be performed to increase the representativeness of the geochemical database and to improve confidence in the design of the waste management plan to prevent ARD and leaching of metals.

As part of the long-term management of these waste materials, they will not remain exposed to the atmosphere (i.e., will not be placed near the top or edge of a waste rock pile) to avoid “hot spots” of high ARD potential. Drainage from waste piles containing these materials will be redirected to the TSF.

20.7.2 CIL Tailings

The tested tailings samples were sourced from both a non-acid generating (S at 0.1%) and sulphide head sample (S at 1.78%) master composite samples that were subjected to bench-scale laboratory testing with the carbon-in-leach. The Bomboré proposed gold extraction scenario is 100% CIL circuits.

Tailings solids and process water subjected to geochemical testing were prepared during the metallurgical testing conducted by McClelland) on an oxide Master Composite ("MC") sample containing 50% medium-grade and 50% high-grade materials. The oxide MC sample was subjected to a cyanidation process, after which a portion underwent cyanide destruction. The solids used for static testing were obtained from a portion of the materials subjected to cyanide destruction (CY142 and CY149), while the process water chemistry comes from decanted water both cyanided but not treated and cyanided with cyanide destruction (CY149).

The oxide MC tailings sample has a low potential to generate ARD as demonstrated by neutral kinetic test results, however arsenic exceeded the Burkina Faso EDC. The process water reports total and WAD cyanide concentrations of 1.9 mg/L each, which is above the IFC effluent guidelines (1 and 0.5 mg/L, respectively) and the Burkina Faso EDC total CN guideline (0.1 mg/L). In addition, the arsenic and copper concentrations all exceed their applicable IFC effluent guidelines while arsenic and copper exceeded the Burkina Faso EDC. However, since the Bomboré TSF will be zero discharge, effluent values do not apply and there will be no issues with effluent quality under dry conditions.

Cyanide destruction units are generally not included in the process when the TSF is lined and is zero discharge. The Project TSF will be lined and is planned to be a zero-discharge facility. Natural degradation will occur with the current design during the dry months, however in the rainy months the cyanide concentrations will increase, and the rainy period is the time issues with the TSF could occur resulting in unplanned discharges to the environment. Therefore, it is recommended to increase monitoring of the facility to ensure that the local Burkinabé do not access the water. In the future if it is determined that a reduction in tailings cyanide level is required, a cyanide destruction circuit can easily be added in a short time.

20.8 Waste Disposal and Sanitary Wastewater

20.8.1 Solid Waste

Solid waste generally includes bags, pallets, empty drums, worn out parts, liners, and other supply packaging. Suppliers will be requested to recycle used materials. For all non-recyclable wastes, a solid waste disposal site is to be created at a suitably enclosed area, restricted to prevent animal access, and located to avoid contamination of water and vegetation. This site is to be operated as a landfill site. On-site waste incineration is not permitted.

Domestic waste (glass, metal, paper and plastics) will be separated and stored in special containers for recycling by qualified companies in Ouagadougou.

20.8.2 Hazardous Waste

Hazardous wastes, which will primarily include waste oils, packaging for process reagents and laboratory chemicals, will be disposed of in a safe and environmentally sound manner. The supplier will recycle waste oils, while most reagents and chemicals that require disposal will be disposed of in the lined TSF. Empty sodium cyanide boxes and inner bags will be cleaned and recycled by suppliers. No onsite incineration is permitted.

Spills of hazardous materials on site will be given the highest operating priority and will generally include the excavation of contaminated soils, neutralization of the affected site, disposal and/or neutralization of the impacted soils on site. The mining equipment on site will be immediately available for use in the event of a spill.

Biomedical waste will be safely stored at the site clinic and transported to a hospital in Ouagadougou to be safely disposed.

20.8.3 Sanitary Wastewater

At the Bomboré camp, a sewage piping network will be provided to collect sewage from the various lavatories, showers and laundry facilities and sent to a modular bacterial digester wastewater treatment system to process both black and grey waters. This unit will be designed not to contaminate groundwater or existing watercourses.

In the process area, satellite workshops and small office buildings, a standard septic tank and soak-away drain-field system will be installed for each respective building, with sludge trucked to the sewage treatment plant for further treatment.

20.9 Closure, Decommissioning and Reclamation

The closure, decommissioning and reclamation costs of the Project of US$ 15.5M (before TSF related closure costs of US$ 2.4M was included in the financial analysis for these closure activities related to the environmental and social aspects.

The Closure and Rehabilitation Plan includes work to be conducted from the closure of the mine, at the end of operation activities, as well as progressive rehabilitation work.

The goal is to return the site to a satisfactory state as quickly as possible in terms of:

- Reducing the risks for health and safety.
- Controlling erosion.
- Limiting maintenance and monitoring.
- Developing a compatible profile with the future uses of the site, primarily for the plant site.

The main objectives of the Closure and Rehabilitation Plan include restoring ecosystems and take-over and recovery of land uses. This plan includes:

- Dismantling and removal of plant equipment, machinery and infrastructure (except for structures that will be kept in place and handed over to the local authority without compromising the integrity and security of places and people).

- Progressive rehabilitation to allow rapid recovery of the vegetation cover and the early recovery of the ecosystem.
- Sustainability of rehabilitation work and control of water and wind erosion.
- Take-over and recovery of land uses.
- Maximization of material and equipment recovery.
- Site rehabilitation as part of a participatory approach involving interested communities.
- Implementation of a post-closure monitoring program.

In addition, a waste rock dump development program will be implemented and will notably include the development of agricultural plots. All structures that can be used by communities will be maintained, except for all facilities that may constitute a risk to people or the environment.

A breakdown of the closure and remediation costs is presented in Table 20.3.

Table 20.3 Cost Breakdown for Closure, Decommissioning and Reclamation (US$) (Excludes TSF Related Closure Costs)

ID	Description	Cost
1. Environmental and Social Management Plan		
1.1	Environmental Management	**$521,300**
1.2	Environmental and Social Clauses for Contractor Specifications	**$22,200**
1.3	Waste Management Plan	**$100,000**
1.4	Environmental Monitoring Program During Deconstruction	**$150,000**
1.5	Environmental & Social Monitoring Program (During closure)	**$347,100**
1.6	Building Plan	**$100,000**
	Sub Total	**$1,240,600**
1.7	Management Fees	**$224,988**
1.8	Contingency	**$494,974**
	TOTAL Management Plan	**$1,960,562**
2. Dismantlement and Rehabilitation		
2.1	General Mine Cleaning	**$427,910**
2.2	Mine Pit Management (Security Fencing)	**$45,372**
2.3	General Dismantling	**$0**
2.4	Site Grading	**$6,652,674**
2.5	Seeding & Reforestation of Dumps & Tailings	**$2,094,343**
2.6	Seeding Labour	**$84,292**
2.7	Post Closure Monitoring Over 5 Years (babysitting, sampling, capacity building of stakeholders)	**$758,693**
2.8	Supporting Employees and Suppliers	**$97,900**
	Sub Total	**$10,161,182**
2.9	Management Fees (10%)	**$1,126,591**
2.10	Contingency (20%)	**$2,257,555**
	TOTAL Site Reclamation & Closure	**$13,545,328**
3. Grand Total		**$15,505,890**

Note: calculation errors are due to rounding

21.0 CAPITAL AND OPERATING COSTS

21.1 Introduction

The following exchange rates have been used in the compilation of the estimate:

- 1 US$ = 1.27 AUD.
- 1 US$ = 0.84 EUR.
- 1 US$ = 550 CFA.
- 1 US$ = 0.716 GBP
- 1 US$ = 1.27 CAD.

21.2 Initial Capital Costs

The overall study 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 Technical Report.

The various elements of the Project estimate have been subject to internal peer review by Lycopodium and have been reviewed with Orezone for scope and accuracy.

All costs are expressed in US$ unless otherwise stated and based on 2Q 2019 pricing. The estimate is deemed to have an accuracy of ±15%. Summary mine capital costs (developed by AMC) are included in the estimate tables below.

KP provided quantities for the TSF and for the surface water management systems (refer to Section 18), including the Nobsin River weir and facilities associated with the OCR and sediment control structures. Rates, indirect costs and contingency were then applied by Lycopodium to derive the capital estimate.

Orezone compiled the Owner's costs estimate incorporating inputs from other consultants. The Orezone estimate was reviewed by Lycopodium for completeness and accuracy and benchmarked against other projects in the region.

The Project Capital Costs in Table 21.1 exclude process operating costs associated with plant operations prior to achieving commercial production on October 1, 2021. The table also excludes the value of gold produced in that period and costs such as bullion transport and refining costs and government royalties associated with this gold production and sales. These additional capitalized expenses and the pre-commercial production gold revenue are addressed in the Project Economic Model.

Table 21.1 Project Capital Costs to 1 October 2021 (US$, 2Q 2019, ±15%)

Project Capital Area	US$ M
Construction In-directs	9.9
Treatment Plant	38.7
Reagents & Plant Services	12.8
Mining Infrastructure	0.8
Site Infrastructure	21.3
Management Costs (EPCM)	11.2
Resettlement Action Plan (RAP)	20.8
Owner's Costs[1]	26.1
Subtotal	**141.7**
Contingency	11.3
Subtotal	**153.0**
Mine Costs (2020/2021)	23.9
Total	**176.9**

[1]excludes $0.9M in opening stock of consumables reclassified to working capital in the economic analysis in Section 22.

The capital cost estimate in Table 21.1 includes:

- Owner's costs (excluding the RAP expenditure) and other costs during the period.
- RAP expenditures.
- Process facilities.
- Mining infrastructure.
- Site infrastructure.
- Stage 1 of the TSF.
- Initial surface water management facilities.
- Installation costs, EPCM costs and contractor distributable costs.
- Site earthworks and site roads and tracks.
- Project contingency.

Exclusions include the following:

- Project sunk costs (including the site access road upgrade, camp upgrade and RAP costs classified as sunk costs).
- Import duties and taxes on the basis that the Project will be exempt.

- Sulphide plant expansion costs.
- Escalation.

21.2.1 Mining

The mining costs for the period up to the declaration of commercial production is listed in Table 21.2. The costs were developed and modelled in accordance with the mining methods details in Section 16 of this Technical Report.

Table 21.2 Mining Capital Costs (US$, 2Q 2019)

Mining Costs	Total US$ ('000)	2020 US$ ('000)	2021 US$ ('000)
OCR Construction and Mine Development (Mine Costs)	23,897	10,276	13,621
Surface Trunk Road (Mining infrastructure)	200	200	
Surface Haul Roads (Mining infrastructure)	181	181	
Pit Dewatering (Mining infrastructure)	409	409	
Total Mining Costs	**24,687**	**11,066**	**13,621**

21.2.2 Process Plant and Infrastructure (Oxide)

The process plant and infrastructure capital estimate for the Oxide plant summarized by discipline is provided in Table 21.3.

Table 21.3 Oxide Plant Capital Estimate (Excluding OCR & Mine Development) Summary by Discipline (US$, 2Q 2019, ±15%)

Discipline	Supply Cost US$ ('000)	Freight Cost US$ ('000)	Installation Cost US$ ('000)	Contingency US$ ('000)	Total US$ ('000)
General	1,810	0	147	258	2,215
Earthworks	4,706	60	9,099	1,339	15,204
Concrete	2,770	0	3,443	696	6,910
Steelwork	2,202	220	740	331	3,492
Platework	4,341	122	2,674	695	7,832
Mechanical	20,671	2,880	1,724	2,253	27,528
Piping	8,577	604	4,326	1,764	15,271
Electrical	6,239	749	1,337	1,055	9,380
Instrumentation & Control	1,538	221	401	210	2,370
Buildings	2,263	348	320	325	3,256
Mining Infrastructure	790	0	0	0	790
Owners Costs[1]	44,916	180	0	2,397	47,493
Management Costs (EPCM)	338	0	10,895	0	11,233
Grand Total	**101,162**	**5,384**	**35,105**	**11,323**	**152,975**

[1]*excludes $0.9M in opening stock of consumables reclassified to working capital in the economic analysis in Section 22.*

21.2.3 Process Plant and Infrastructure (Sulphide)

The process plant and infrastructure capital estimate for the Sulphide plant summarized by discipline is provided in Table 21.4

Table 21.4 Sulphide Plant Capital Estimate Summary by Discipline (US$, 2Q 2019, ±15%)

Discipline	Supply Cost US$ ('000)	Freight Cost US$ ('000)	Installation Cost US$ ('000)	Contingency US$ ('000)	Total US$ ('000)
General	895	0	18	137	1,049
Earthworks	0	0	0	0	0
Concrete	2,700	0	3,362	718	6,780
Steelwork	1,465	144	517	233	2,359
Platework	2,664	121	1,439	422	4,646
Mechanical	16,632	2,334	1,230	2,023	22,220
Piping	2,060	212	1,046	488	3,806
Electrical	2,803	325	641	467	4,235
Instrumentation & Control	1,076	140	349	187	1,752
Buildings	389	51	91	62	592
Owners Costs	8,707	257	0	455	9,420
Management Costs (EPCM)	221	0	6,131	0	6,351
Grand Total	**39,613**	**3,586**	**14,822**	**5,191**	**63,211**

[1] *excludes $1.4M in opening stock of consumables reclassified to working capital in the economic analysis*

21.2.4 Sustaining Capital

The Sustaining Capital Costs estimate for all areas is summarized in Table 21.5

21.3 Mine Closure and Salvage Costs

Closure and Salvage Costs are summarized in Table 21.6.

Table 21.5 Sustaining Capital (US$, 2Q 2019, ±15%)

	Sustaining Capital Costs	Sustaining Total Cost (US$)	Year 2021	Year 2022	Year 2023	Year 2024	Year 2025	Year 2026	Year 2027	Year 2028	Year 2029	Year 2030	Year 2031	Year 2032	Year 2033
	TSF Stage		2	3	4	5	6	7	8	8	9	9	10	10	10
	Infrastructure														
1	Second Stage Tails Pump	239,115		239,115											
2	High Pressure Gland Water Pump	72,762		72,762											
3	TSF	52,833,643	3,681,409	4,940,917	7,461,905	2,816,560	3,404,527	6,570,155	2,742,044	3,894,966	3,834,844	5,226,089	3,395,254	4,864,974	
4	TSF Pipeline and Valves	5,139,674	371,602	428,124	1,556,539	342,849	279,767	642,796	252,497	252,497	244,431	244,431	262,070	262,070	
5	Surface Water Management	1,389,503	567,051	438,701			178,701		68,350		68,350		68,350		
	Mining														
6	Pit Dewatering Capital Costs	3,414,644	1,286,677	1,089,328	171,917	133,578	133,823				10,502	223,156		266,485	99,178
7	Surface Haul Road	1,124,151	300,950	473,439	222,834	71,489	23,145	22,899	6,017					3,378	
8	Main Access Road	530,086	436,891	41,326	51,869										
	G&A														
9	General & Admin items (vehicles, etc.)	1,500,000	100,000	200,000	100,000	200,000	100,000	200,000	100,000	200,000	100,000	100,000	100,000		
	Total	**66,243,578**	**6,744,579**	**7,923,712**	**9,565,063**	**3,564,477**	**4,119,963**	**7,435,850**	**3,168,908**	**4,347,463**	**4,258,127**	**5,793,676**	**3,825,674**	**5,396,907**	**99,178**

Table 21.6 Closure and Salvage Costs

Salvage and Closure Costs	Closure Year (US$M)
Salvage value (end of mine life)	
General Dismantling Cost	2.36
Salvage Value of Mechanical Equipment	-7.94
Closure Costs	
TSF Closure	2.37
Environmental/Social Management Plan/Rehabilitation	15.51
Total	**12.30**

21.4 Operating Costs

21.4.1 Overall

The Project operating cost estimate is built-up from three components:

- The mine operating costs developed by AMC (refer to Section 21.4.2).
- The process plant operating costs developed by Lycopodium (refer to Section 21.4.3).
- The general and administration (G&A) operating costs developed by Orezone and Lycopodium (refer to Section 21.4.4).

The estimated life-of-mine operating cost per tonne of ore treated and per ounce of gold produced is summarized in Table 21.7.

Table 21.7 Life-of-Mine Operating Costs per Tonne and per Gold Ounce (US$, 2Q 2019)

Cost Components	Total Cost ($M)	$/Tonne Processed	$/oz Au
Mining	386.3	5.51	242
Processing	456.9	6.52	286
G&A	139.4	1.99	87
Refining & Bullion Transport	2.4	0.03	1
Government Royalties & Dev Tax	103.9	1.48	65
Total Cash Cost	**1,089.0**	**15.53**	**681**

21.4.2 Mining

The total mining cost for the Life-of-Mine is based on the schedules and strategy outlined in Section 16 and is shown in Table 21.8.

Table 21.8 Mining Costs for Life-of-Mine (US$, 2Q 2019)

Mining Costs	Average Mining Cost ($/t Processed)	Total Mining Cost ($M)	Pre-Production ($M)	Year 1 ($M)	Year 2 ($M)	Year 3 ($M)	Year 4 ($M)	Year 5 ($M)	Year 6 ($M)	Year 7 ($M)	Year 8 ($M)	Year 9 ($M)	Year 10 ($M)	Year 11 ($M)	Year 12 ($M)	Year 13 ($M)	Year 14 ($M)
Grade control costs	0.08	5.90	0.39	0.61	0.65	0.45	0.38	0.44	0.43	0.38	0.39	0.47	0.46	0.37	0.28	0.16	0.03
Owner's team labour costs	0.47	32.78	3.45	2.14	2.18	2.26	2.26	2.26	2.26	2.26	2.26	2.26	2.26	2.18	2.15	2.08	0.51
Owner's team consumables cost	0.02	1.15	0.10	0.07	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.07	0.02
Pumping cost	0.01	0.95	0.01	0.02	0.05	0.09	0.09	0.09	0.12	0.12	0.08	0.09	0.07	0.06	0.04	0.01	0.00
Direct mining costs	4.80	336.56	19.95	29.93	28.24	31.34	28.92	28.10	26.59	25.50	27.27	27.09	22.34	20.18	14.02	6.06	1.02
Stockpile reclaiming cost	0.13	8.99	0.00	0.00	0.00	0.77	1.15	0.65	0.42	0.67	0.49	0.28	0.84	0.63	0.94	1.68	0.47
Total Combined Mining Costs	**5.51**	**386.34**	**23.90**	**32.78**	**31.19**	**35.00**	**32.88**	**31.62**	**29.91**	**29.02**	**30.58**	**30.28**	**26.06**	**23.50**	**17.51**	**10.06**	**2.05**

21.4.3 Process Plant

For the Feasibility Study, the process plant operating costs were developed for both the initial period with only the oxide circuit in operation and for the periods following the commissioning of the sulphide circuit when 2.2 Mtpa of oxide ore will be displaced by a similar quantity of sulphide ore. The operating cost by year was then estimated and is provided in Table 21.9.

The process operating costs for the Project have been developed in accordance with typical industry standards applicable to gold ore processing plants. The cost estimates are expressed in 2Q 2019 US$ with an accuracy of ±15%.

The process plant operating cost by ore type is shown in Table 21.9.

Table 21.9 Process Operating Cost by Ore Type (US$, ±15%, 2Q 2019)

	Average Treatment Cost $/t	Oxide Treatment Cost $/t	Sulphide Treatment Cost $/t
Treating 5.2 Mtpa oxide ore only	**4.71**	**4.71**	**N/A**
Treating 3.0 Mtpa oxide + 2.2 Mtpa sulphide	**7.74**	**4.92**	**11.56**

The quantities and cost data on which the estimates were based were compiled from a variety of sources including:

- Metallurgical testwork.
- Consumables prices from reputable suppliers.
- Lycopodium database and experience with recent projects in the region.
- Input from Orezone.
- First principle calculations.

Qualifications and Exclusions

The process operating cost estimate includes all direct costs associated with the Project to allow production of a leached slurry that will be transferred from the sulphide plant to the leaching section of the oxide plant. Each cost estimate is presented with the following exclusions:

- All taxes and import duties.
- Any impact of foreign exchange rate fluctuations.
- Any escalation beyond the date of the estimate.

- Bullion transportation, refining, marketing and insurance costs on the basis that these are included in the financial model.
- Contingency.

Cost Categories

The operating cost estimate was estimated according to the following categories:

1. Operating consumables: oxide US$2.23/t, combined circuits US$3.12/t.
2. Plant maintenance: oxide US$0.21/t, combined circuits US$0.33/t.
3. Power: oxide US$1.65/t, combined circuits US$3.73/t.
4. Laboratory and assay costs: oxide US$0.03/t, combined circuits US$0.03/t.
5. Labour (operation and maintenance): oxide US$0.58/t in the Project early years, combined circuits US$0.52/t.

For the purpose of this Technical Report the costs relating to the periods when 2.2 Mtpa of sulphide ores are co-treated with 3.0 Mtpa of oxides are described below. Similar methodology was used to estimate the oxide operating cost in the early Project years.

Operating Consumables

The consumables category covers all wear parts, consumable materials and reagents. Annual costs and cost per tonne rates for consumables and reagents are summarized in Table 21.10.

Comminution consumables (crusher liners, mill liners and grinding media) were evaluated based on comminution testwork performed on various samples representing the Bomboré ore body. Crusher liner, SAG mill liner, ball mill liner and steel ball consumption costs are based on calculations and vendor supplied data.

Laboratory testwork results were used to determine reagent consumption rates. In the absence of testwork data, reagent consumption rates for minor items were based on first principle calculations, Lycopodium experience and/or generally accepted practice within the industry.

Table 21.10 Consumables Cost Summary

Area	Consumables Cost US$/yr	Consumables Cost US$/t
Primary Crushing	225,913	0.04
Grinding	4,828,010	0.93
Leach and CIL	8,392,019	1.61
Thickening	342,264	0.07
Acid Wash and Elution	2,092,840	0.40
Miscellaneous	337,018	0.06
Total	**16,218,064**	**3.12**

Maintenance

Maintenance materials and spares costs were factored from the installed mechanical equipment costs with due allowance for the low abrasion index of the oxide ores and the moderate abrasion index of the sulphide ores. Overall the factor averages to 2.8% of the installed mechanical equipment cost for the process plant areas.

Crusher and mill wear parts (liners) as well as cyclone parts and screen panels are included in the consumables cost section. Maintenance labour costs are included in the labour section below.

The Process Plant Maintenance cost is summaries in Table 21.11.

Table 21.11 Process Plant Maintenance Cost

Area	Maintenance Cost US$/yr	Maintenance Cost US$/t
Process Plant	1,226,150	0.24
Reagents & Services	179,200	0.03
Mobile equipment	214,530	0.04
Maintenance - General	70,000	0.01
Miscellaneous	45,040	0.01
Total	**1,734,920**	**0.33**

Power

The process plant electricity consumption estimate is based on the installed power for all driven equipment 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 21.12.

The unit cost of power of $0.156/kWh, based on an independent power provider operating a heavy fuel oil power station, was provided by Orezone based on published information from other operations in the region.

Table 21.12 Site Power Cost by Area

Area	Sulphide Plant Average Load (kW)	Oxide Plant Average Load (kW)	Combined Average Load (kW)	Annual Power Consumption (kWh)	Total Annual Power Cost (US$)	Cost, US$/t
Area 120 - Crushing	184	67	251	2,198,760	343,007	0.07
Area 130 - Reclaim & Milling	7,571	2,139	9,710	85,059,600	13,269,298	2.55
Area 140 - Screening & Thickening	97	399	496	4,344,960	677,813	0.13
Area 160 - Leaching & CIL	918	627	1,545	13,534,200	2,111,335	0.41
Area 170 - Elution	7	10	17	148,920	23,232	0.00
Area 180 - Gold Room	17	23	40	350,400	54,663	0.01
Area 210 - Reagents	36	13	49	429,240	66,961	0.01
Area 230 - Plant Water Services	479	336	815	7,139,400	1,113,746	0.21
Area 250 - Air Services	374	171	545	4,774,200	744,775	0.14
Area 260 - Fuel Dispensing	0	1	1	8,760	1,367	0.00
Area 270 - Electrical Services - Lighting	177	88	265	2,321,400	362,138	0.07
Area 340 - TSF	52	71	123	1,077,480	168,087	0.03
Area 350 - Plant Buildings	34	47	81	709,560	110,691	0.02
Area 310 - Environmental	0	1	1	8,760	1,367	0.00
Area 380 - Camp and Camp Services	71	97	168	1,471,680	229,582	0.04
Area 440 - Mine Facilities	32	43	75	657,000	102,492	0.02
TOTAL	**10,049**	**4,133**	**14,182**	**124,234,320**	**19,380,554**	**3.73**

Plant Laboratory

The total estimated annual laboratory cost, excluding labour, is US$165,247 (or US$0.03/t) for the oxide and sulphide plants combined.

Process Plant Labour

Labour rates including base salaries, benefits, bonuses and overhead burdens were provided by Orezone based on their existing site complement and information from other operations in the region. These rates were reviewed by Lycopodium and benchmarked against recent study data.

The labour organizational chart was developed by Lycopodium with input from Orezone. The operating roster is based on a four panel 12-hour shift rotation. It was determined that there was an initial requirement for 104 operating and maintenance personnel, rising to 120 upon commissioning of the sulphide circuit. Burkina Faso has had an established mining industry for a number of years and it is expected that experienced personnel will be attracted to the operation with it being in close proximity to Ouagadougou. Allowance was made in the estimates for reduced expatriate numbers as nationals are trained to occupy more senior roles.

The estimated annual process plant labour cost once the sulphide circuit is in operation is US$2.7M or US$0.52/t.

Processing Plant Operating Cost by Year

As input to the financial analysis, operating costs were estimated on an annual basis based on the process plant feed predicted by the mine plan. The estimated operating costs by year are shown in Table 21.13.

21.4.4 G & A Costs

The G&A costs were estimated based on Orezone's experience of operating in Burkina Faso for several years, gazetted rates for land tax and similar costs, rates and quotations from reputable service providers such as Orezone's current catering contractor and insurance providers, and a build-up of the expected G&A organization chart from first principles. G&A costs were estimated to average US$10.5M annually, or approximately US$2.02/t ore over the LOM, not including preproduction.

These costs include insurance, permitting, office supplies, general management, accounting, communications informational technology environmental and social management, human resources, purchasing and procurement, health and safety, security, international travel and camp operating costs. In most cases, these services represent fixed costs for the site with some exceptions such as camp and transportation costs of employees.

The G&A costs exclude certain costs such as the transport and refining of gold and royalty payments which are included in the economic analysis. Environmental rehabilitation costs, which are treated as separate line items in the financial model, are included in the sustaining capital.

The G&A costs have been summarized in table 21.14.

Table 21.13 Processing Plant Operating Costs by Year (US$, 2Q 2019)

Cost Centre	Units	LOM	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14
Oxide Ore (variable)																	-
Consumables	US$M	116.2	2.7	11.6	11.6	8.3	6.6	6.6	6.6	6.6	6.6	6.6	6.8	10.0	11.4	11.5	2.9
Plant Maintenance	US$M	13.5	0.3	1.1	1.1	1.0	0.9	0.9	0.9	0.9	0.9	0.9	0.9	1.3	1.1	1.1	0.3
Power	US$M	93.2	2.0	8.6	8.6	6.8	5.6	5.6	5.6	5.6	5.6	5.6	5.8	8.6	8.4	8.5	2.1
Attributable to Oxide Ore	**US$M**	**222.9**	**4.9**	**21.2**	**21.3**	**16.0**	**13.1**	**13.1**	**13.1**	**13.1**	**13.1**	**13.1**	**13.5**	**19.9**	**20.9**	**21.1**	**5.3**
Sulphide Ore (variable)						-	-	-	-	-	-	-	-	-	-	-	-
Consumables	US$M	76.8	-	-	-	6.3	9.6	9.6	9.6	9.6	9.6	9.6	9.3	2.8	0.4	0.2	0.0
Plant Maintenance	US$M	6.7	-	-	-	0.6	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.2	0.0	0.0	0.0
Power	US$M	109.8	-	-	-	9.1	13.7	13.7	13.7	13.7	13.7	13.7	13.3	4.1	0.6	0.3	0.1
Attributable To Sulphide	**US$M**	**193.3**	**-**	**-**	**-**	**16.0**	**24.2**	**24.2**	**24.2**	**24.2**	**24.2**	**24.2**	**23.4**	**7.2**	**1.1**	**0.5**	**0.1**
Laboratory	US$M	2.3	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0
Labour (Plant O&M)	US$M	38.4	1.0	3.0	3.0	3.4	2.7	2.7	2.7	2.7	2.8	2.8	2.8	2.8	2.6	2.6	0.7
Total	**US$M**	**456.9**	**6.0**	**24.4**	**24.5**	**35.5**	**40.2**	**40.2**	**40.2**	**40.2**	**40.3**	**40.3**	**39.8**	**30.0**	**24.7**	**24.4**	**6.1**
	US$/t	**6.52**	**4.96**	**4.71**	**4.71**	**6.84**	**7.74**	**7.74**	**7.74**	**7.74**	**7.75**	**7.75**	**7.65**	**5.78**	**4.76**	**4.69**	**4.68**

Table 21.14 G&A Summary Cost by Year (US$, 2Q 2019)

G&A Department	LOM	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
	US$	US$	US$	US$	US$	US$	US$	US$	US$	US$	US$	US$	US$	US$	US$
General Administration	**19,129,189**	1,351,895	1,351,895	1,351,895	1,351,895	1,351,895	1,351,895	1,351,895	1,380,847	1,380,847	1,380,847	1,380,847	1,380,847	1,380,847	1,380,847
Guest House (Ouagadougou)	**1,595,203**	113,296	113,296	113,296	113,296	113,296	113,296	113,296	114,590	114,590	114,590	114,590	114,590	114,590	114,590
Catering	**7,402,023**	528,716	528,716	528,716	528,716	528,716	528,716	528,716	528,716	528,716	528,716	528,716	528,716	528,716	528,716
Camp	**2,393,567**	169,039	169,039	169,039	169,039	169,039	169,039	169,039	172,899	172,899	172,899	172,899	172,899	172,899	172,899
Maintenance	**2,120,449**	150,429	150,429	150,429	150,429	150,429	150,429	150,429	152,492	152,492	152,492	152,492	152,492	152,492	152,492
Light Vehicle	**4,252,161**	302,095	302,095	302,095	302,095	302,095	302,095	302,095	305,357	305,357	305,357	305,357	305,357	305,357	305,357
Power House	**210,000**	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000	15,000
Community Relations/CSR	**4,764,562**	336,762	336,762	336,762	336,762	336,762	336,762	336,762	343,890	343,890	343,890	343,890	343,890	343,890	343,890
Finance	**11,478,312**	810,342	810,342	810,342	810,342	810,342	810,342	810,342	829,417	829,417	829,417	829,417	829,417	829,417	829,417
Health & Safety	**8,970,235**	635,151	635,151	635,151	635,151	635,151	635,151	635,151	646,311	646,311	646,311	646,311	646,311	646,311	646,311
Environment	**14,968,568**	1,243,964	940,433	1,098,614	1,098,614	1,146,848	1,146,848	988,667	998,317	1,156,498	998,317	1,156,498	998,317	998,317	998,317
Human Resources	**4,921,910**	347,882	347,882	347,882	347,882	347,882	347,882	347,882	355,248	355,248	355,248	355,248	355,248	355,248	355,248
Information Technology	**9,296,498**	661,148	661,148	661,148	661,148	661,148	661,148	661,148	666,923	666,923	666,923	666,923	666,923	666,923	666,923
Supply Chain Management	**5,913,040**	417,445	417,445	417,445	417,445	417,445	417,445	417,445	427,275	427,275	427,275	427,275	427,275	427,275	427,275
Security	**15,589,524**	1,109,312	1,109,312	1,109,312	1,109,312	1,109,312	1,109,312	1,109,312	1,117,763	1,117,763	1,117,763	1,117,763	1,117,763	1,117,763	1,117,763
Insurance	**12,095,000**	780,000	780,000	855,000	880,000	880,000	880,000	880,000	880,000	880,000	880,000	880,000	880,000	880,000	880,000
Permits and Taxes	**18,468,295**	1,136,591	1,165,000	1,165,000	1,167,386	1,174,545	1,345,000	1,401,818	1,401,818	1,406,591	1,420,909	1,420,909	1,420,909	1,420,909	1,420,909
RAP	**3,701,188**	532,884	546,521	546,521	549,248	322,719	179,445	179,445	120,629	120,629	120,629	120,629	120,629	120,629	120,629
G&A Total	**147,269,724**	**10,641,951**	**10,380,466**	**10,613,647**	**10,643,761**	**10,472,625**	**10,499,806**	**10,398,442**	**10,457,491**	**10,620,445**	**10,476,582**	**10,634,763**	**10,476,582**	**10,476,582**	**10,476,582**
x % of year operating		100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	25%
G&A Cost for Financial Model	**139,412,288**	**10,641,951**	**10,380,466**	**10,613,647**	**10,643,761**	**10,472,625**	**10,499,806**	**10,398,442**	**10,457,491**	**10,620,445**	**10,476,582**	**10,634,763**	**10,476,582**	**10,476,582**	**2,619,145**

22.0 ECONOMIC ANALYSIS

22.1 Introduction

An economic assessment of the Bomboré Gold Project has been conducted using a pre and after-tax cash flow model prepared by Lycopodium on behalf of Orezone.

Input data were provided from a variety of sources, including the various consultants' contributions to this Technical Report, pricing obtained from external suppliers and contractors, and exchange rates and Project specific financial data such as the expected Project taxation regime received from Orezone. The assessment was based upon:

- Capital cost estimates and expenditure schedules prepared by Lycopodium and AMC with input from KP in terms of TSF starter dam and site water management systems.
- Mine schedule and mining operation cost estimates based on contract mining operations, as developed by AMC for the study.
- Process operating and general and administrative cost estimates prepared by Lycopodium, with contributions from Orezone.
- Metallurgical performance characterized by testwork completed for the study.
- Sustaining capital cost estimates for the mine development provided by AMC, and TSF stage development provided by KP.
- Owner's capital cost estimates prepared by Lycopodium and Orezone.
- Royalty, tax, discount rates and other model inputs provided by Orezone.
- Closure costs estimated by Antea.
- Salvage costs estimated by Lycopodium.
- The cash flow analysis excludes any effects due to inflation and all dollars are expressed in real US$ as at Q3 2019.
- Only cash flows after June 30, 2019 are taken into consideration. All costs prior to July 1, 2019 are considered sunk and have been excluded from the cash flow model.
- The forecast gold price of $1,300/oz for the life of mine was agreed with Orezone.
- The cash flow analysis is based on full equity funding and any cost of borrowing is excluded.

The cash flow model reports:

- All costs in real US$ exclusive of escalation or inflation.
- A net present value (NPV) at a 5% discount rate.

- An internal rate-of-return (IRR) based on pre and post-tax net cash flows.

Table 22.1 presents a summary of the production information on which the cash flow model is based.

The life-of-mine capital cost for the project is estimated at $303.2 M, with an initial project construction cost of $153 M. Expansion capital is estimated at $63.2 M in Project year 2.

Table 22.2 shows the Project cash flow summary. At a gold price of $1,300/oz, the Project is estimated to have an after-tax IRR of 43.8% and a payback period of 1.5 years. At a discount rate of 5%, the after-tax NPV is estimated at $361 M. The project economics are summarized in Table 22.3.

There are 1.7 Mt of low-grade mineralized oxide material above cut-off grade remaining in stockpiles at end of mine life that are not included in the Mineral Reserves Estimate. The cost of these unprocessed low-grade stockpiles has been treated as a cash outflow as unrecovered working capital at end of mine life.

Table 22.1 Production Summary

	Value
Ore processed	70.1 Mt
Total tonnes mined	236.2 Mt
Average head grade	0.81 g/t Au
Contained gold in material	1.8 Moz
Total gold produced	1.6 Moz
Average gold recovery	87.2%
Production life (processing)	13+ years
Nominal annual processing rate	5.2 Mtpa

Table 22.2 Net Profit after Tax Summary (LOM Summary)

	$M	$/Ore t Processed	$/oz Au
Revenue (99.93% payable)	**$2,078**	**$29.64**	**$1,299**
Mine Operating Cost	$386.3	$5.51	$241.5
Processing Cost	$456.9	$6.52	$285.7
G&A Cost	$139.4	$1.99	$87.2
Refining & Transport Costs	$2.40	$0.03	$1.5
Government Royalties	103.9	$1.48	$65.0
Total Cash Cost	**$1,089**	**$15.53**	**$680.8**
EBITDA	**$997.5**	**$14.23**	**$623.6**
Initial Capital	$153.0	$2.18	$95.6
Expansion Capital	$63.2	$0.90	$39.5
Sustaining Capital	$66.2	$0.94	$41.4
Rehabilitation & Closure (net of salvage)	$12.3	$0.18	$7.7
Total Capital Costs	**$294.7**	**$4.20**	**$184.3**
Gross Profit before tax	**$694.3**	**$9.90**	**$434.0**
Corporate Tax Payable	$187.2	$2.67	$117.0
Net Profit after tax	**$507.1**	**$7.23**	**$317.0**

Table 22.3 Financial Summary

	Value
Revenue from gold (99.93% payable)	$2,078M
Adjusted Operating Costs (AOC)	$681/oz Au
Initial Capital	$153M
Expansion Capital	$63.2M
Sustaining capital	$66.2M
Closure costs/salvage	$12.3M
Pre-tax economics:	
IRR	61.9%
NPV (5%)	$513M
Payback	1.5 Years
After-tax economics:	
IRR	43.8%
NPV (5%)	$361M
Payback	2.5 Years

22.2 Project Total Upfront Costs

The Total Upfront Costs are shown in Table 22.4.

Table 22.4 Total Upfront Costs

	$ M
Process Plant	$51.4
Infrastructure	$21.3
Mining (Haul Roads & Pit Dewatering)	$0.8
Construction In-directs	$9.9
EPCM	$11.2
Resettlement Action Plan	$20.8
Owner's Costs	$26.1
Subtotal	**$141.7**
Contingency	$11.3
Total Initial Construction Costs	**$153.0**
Working Capital	$24.9
Pre-production Operating Costs	$8. 4
Total Upfront Costs Before Sales	**$186.3**
Pre-production Gold Sales	-$47.6
Total Upfront Costs	**$138.7**

The Total Upfront Costs represent the Project capital estimate plus capitalized costs incurred to achieve commercial production (on October 1, 2021) less the value of gold recovered during the pre-commercial production period (June to September 2021 inclusive). For further details on the Project capital costs refer to Section 21.

22.3 Project Expansion Capital Costs

The Total Project Expansion Capital Costs are shown in Table 22.5.

Table 22.5 Total Project Expansion Capital Costs

	$ M
Process Plant	$36.2
Infrastructure	$1.1
Construction In-directs	$5.4
EPCM	$6.4
Resettlement Action Plan	$3.7
Owner's Costs	$5.23
Subtotal	**$58.0**
Working Capital	$1.42
Contingency	$5.2
Total Capital Costs	**$63.2**

The Total Project Expansion Capital Costs represents the capital estimate plus capitalized costs incurred to install production facilities for the sulphide ore. There is no pre-commercial production period as the plant will be in operation. For further details on the Project capital costs, refer to Section 21.

22.4 Assumptions and Qualifications

General

- The cash flow model has been based on a 2-year project development period beginning on July 1, 2019 with gold production commencing in June 2021. The model has considered only cash flows from project 'go-ahead'. Any previous expenditure (sunk costs) have not been carried forward or included in the model.
- Annual mined tonnage and head grade have been based on the mining schedule as presented in Section 16 and process plant throughput and production rates as presented in Section 17.
- The mining, processing and administration costs are based on the operating cost estimates presented in Section 21.
- Gold recovery is based on testwork and interpretation presented in Section 13.
- The capital costs are based on the estimates presented in Section 21.
- Closure costs of $17.9 M have been included.
- An estimated asset residual sale value of $5.6 M has been included.

- The cash flow model assumes full equity funding.
- No provision has been made for interest on the cost of capital.
- No provision has been made for corporate head office costs during construction and operations.
- No provision has been made for escalation or inflation.
- The NPV calculation is based on cash flows occurring mid-year.

Working Capital

- Working capital has been calculated and included in the cash flow model.
- Working capital includes opening stocks.
- 18% TVA (value added tax) has been applied to operating expenses and remains in working capital until refunded from the tax authorities.

Gold Price

- A gold price of $1,300/oz has been applied in the cash flow model.

Refining Terms

- Gold is assumed to be payable at 99.93%.
- Refining costs have been included at $0.5/oz.
- Transport costs inclusive of insurance have been included at $1.0/oz.

Government Royalties and Taxes

- The Government of Burkina Faso benefits from:
  - A 10% free-carried interest.
  - Sales royalties (4% NSR between $1,000 and $1,300 Au).
  - A local Development Mining Fund tax (1% NSR).
- The treatment of depreciation and corporate taxes are based on Orezone’s understanding of current Burkina Faso tax laws.
- Provision has been made for corporate income tax at 27.5% of taxable income as advised by Orezone.

22.5 Sensitivity Analysis

The Project value was assessed by undertaking sensitivity analyses on the gold price, gold recoveries, operating costs and capital costs. The Project is most sensitive to changes in the gold price and then operating costs. The results of all pre-tax sensitivity analyses are presented in Tables 22.6 and 22.7 and in Figures 22.1 and 22.2. The results of all after tax sensitivity analyses are presented in Tables 22.8 and 22.9 and in Figures 22.3 and 22.4.

Table 22.6 NPV Sensitivity Analysis (Pre-tax)

	Lower	Base Case	Higher
Gold Price (±10%)	$360,680,618	$513,489,065	$648,603,902
CAPEX (±10%)	$539,823,342	$513,489,065	$487,154,788
OPEX (±10%)	$587,896,719	$513,489,065	$439,081,411
Recovery (±2%)	$482,964,495	$513,489,065	$544,013,635

Table 22.7 IRR Sensitivity Analysis (Pre-tax)

	Lower	Base Case	Higher
Gold Price (±10%)	45%	62%	77%
CAPEX (±10%)	73%	62%	53%
OPEX (±10%)	69%	62%	55%
Recovery (±2%)	59%	62%	65%

Figure 22.1 NPV Sensitivity Analysis (Pre-tax)

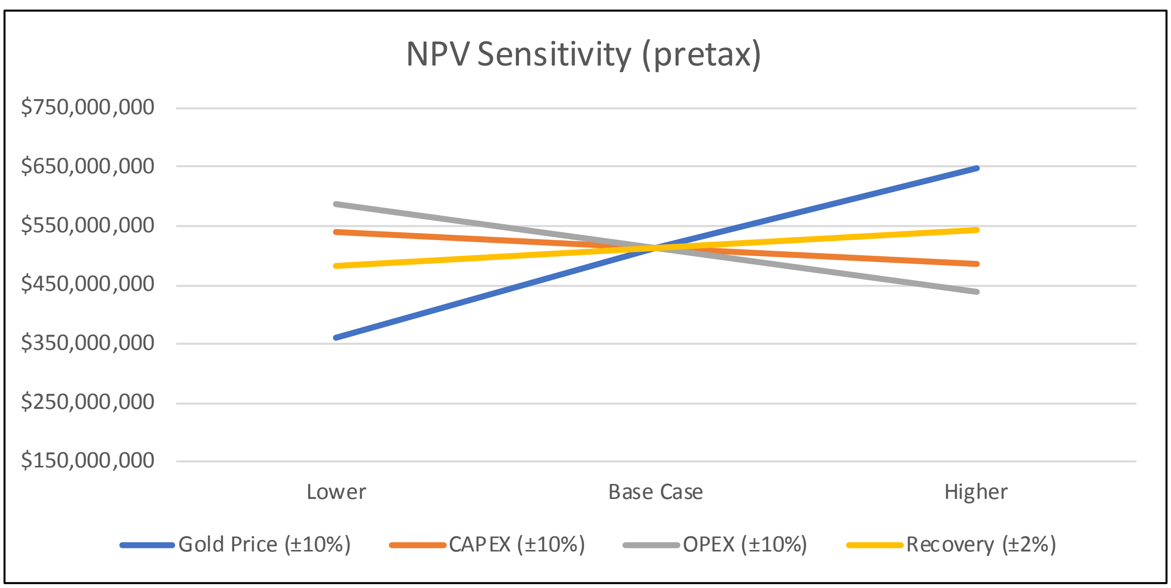


Figure 22.2 IRR Sensitivity Analysis (Pre-tax)

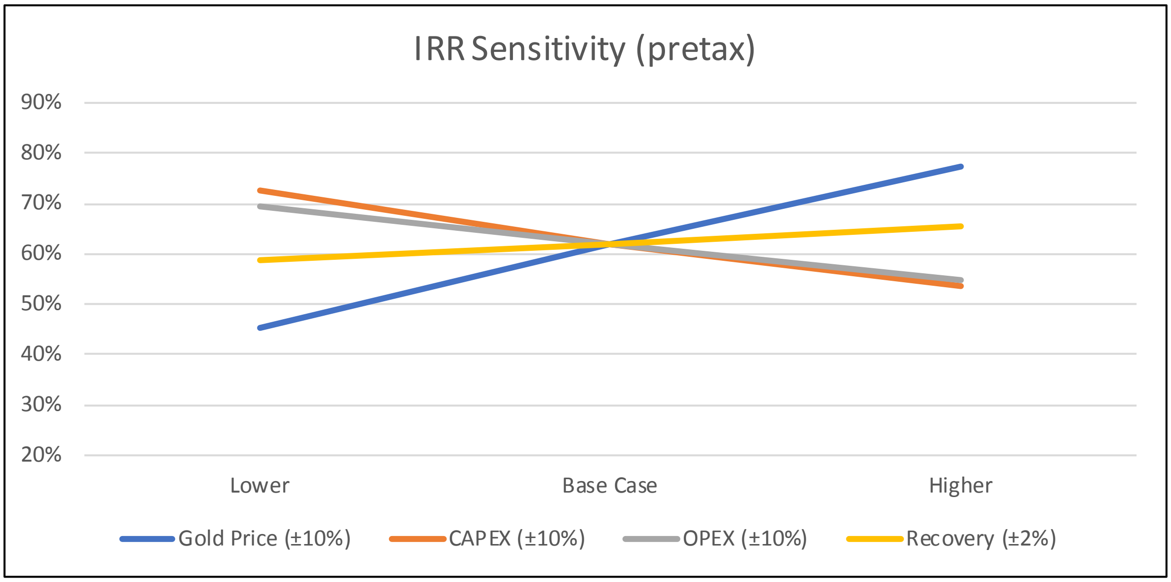


Table 22.8 NPV Sensitivity Analysis (After tax)

	Lower	Base Case	Higher
Gold Price (±10%)	$247,608,018	$360,990,394	$460,311,497
CAPEX (±10%)	$382,696,303	$360,990,394	$338,341,461
OPEX (±10%)	$416,990,735	$360,990,394	$304,279,760
Recovery (±2%)	$338,341,461	$360,990,394	$383,638,558

Table 22.9 IRR Sensitivity Analysis (After tax)

	Lower	Base Case	Higher
Gold Price (±10%)	32%	44%	55%
CAPEX (±10%)	52%	44%	38%
OPEX (±10%)	49%	44%	38%
Recovery (±2%)	41%	44%	46%

Figure 22.3 NPV Sensitivity Analysis (After tax)

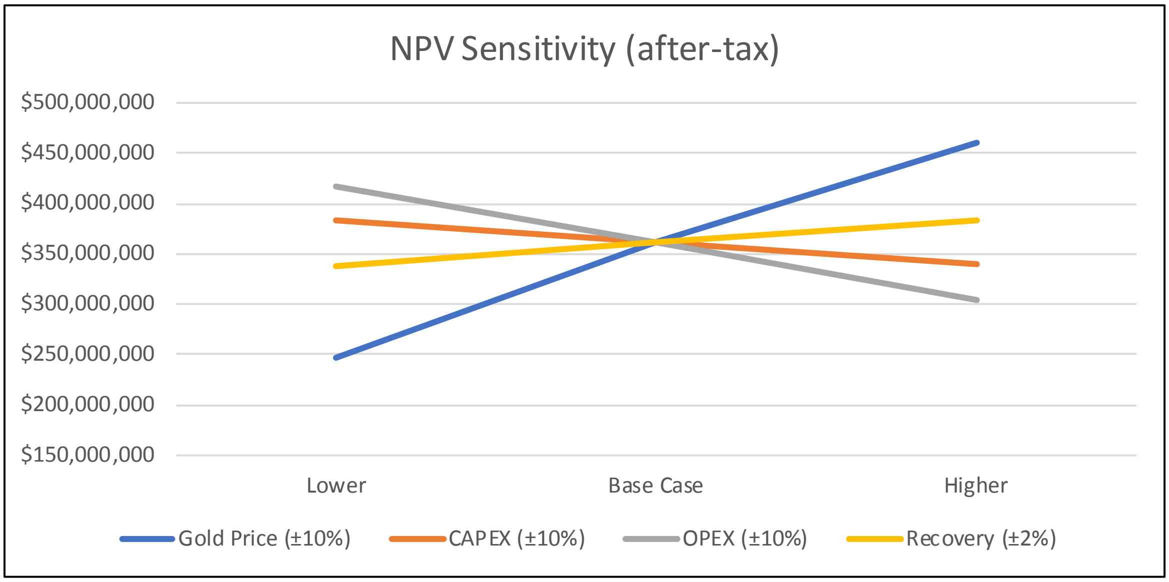
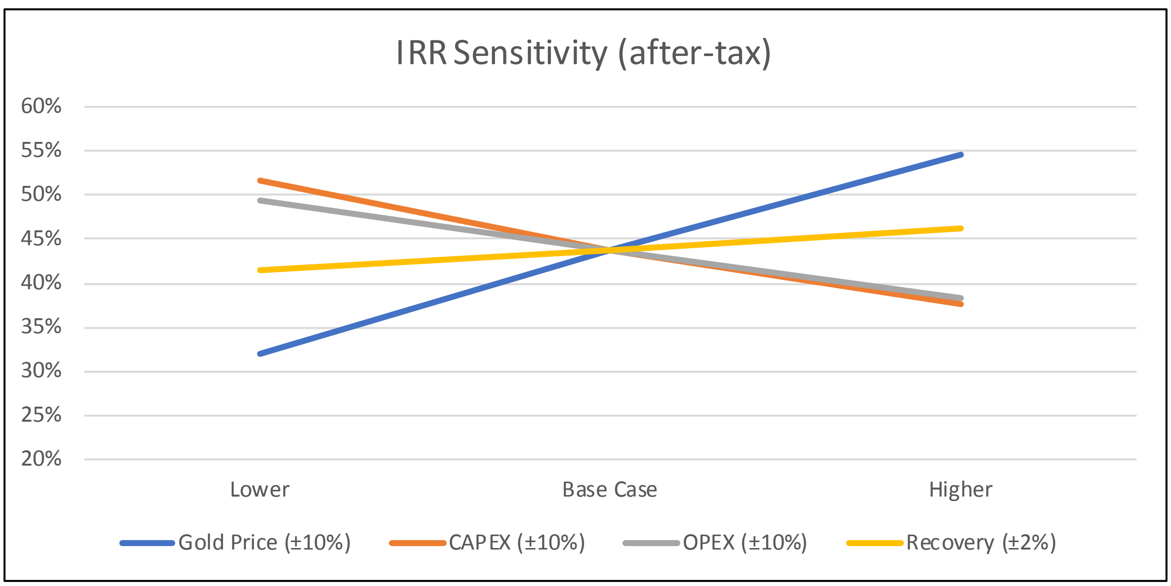


Figure 22.4 IRR Sensitivity Analysis (After tax)



23.0 ADJACENT PROPERTIES

The Qualified Persons for this Technical Report have been unable to verify the below noted information and note that the information is not necessarily indicative of the mineralization on the Property that is the subject of this Technical Report.

23.1 West African Resources Boulsa Project

Parts of the land tenements related to WAF (ASX: WAF) Boulsa Project are in close proximity to the Bomboré Project.

Within the WAF tenements lying within a 15 km radius of Bomboré multiple mineralized prospects have been identified and are at various stages of early exploration. In particular, WAF have previously reported mineralization at their Moktedu prospect approximately 2 km northeast of the Bomboré tenements.

23.2 West African Resources Sanbrado Gold Project

In April 2019 WAF reported Mineral Reserves and Resources with an effective date of 25 March 2019 at their Sanbrado Project approximately 10 km to the southeast of the Bomboré tenements. Probable Reserves for the open pit total 19.5 Mt at an average grade of 1.6 g/t Au for 1.004 Moz and for the underground total 2.0 Mt at an average grade of 10.2 g/t Au for 0.646 Moz. These reserves are based on Indicated Resources totalling 39.40 Mt at an average grade of 1.9 g/t Au for 2.405 Moz. WAF resource statement also shows Inferred Resources totalling 15.65 Mt at an average grade of 1.3 g/t Au for 0.684 Moz. The project is currently under construction.

23.3 B2GOLD Toega Gold Project

In February 2018 B2GOLD (TSX: BTO) reported Inferred Resources, with an effective date of 8 January 2018, totalling 17.5 Mt at an average grade of 2.01 g/t Au for 1.13 Moz at their Toega Project approximately 7 km to the south of the Bomboré tenements.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation and Schedule

The initial strategy for implementation of the Project early works is driven by Orezone's intent to self-manage the majority of construction for the resettlement, access road upgrade, initial camp upgrade and other similar activities required to prepare the site for mobilization of the bulk earthworks contractor.

Significant progress has been made with the owner managed early works with the access road upgrade and camp upgrade complete and work being well advanced on the initial phase of the RAP. In addition, progress has been made on the initial phase of Front-End Engineering and Design (FEED), with the design of the oxide plant being well advanced, project controls and many of the various Project plans completed or drafted, and procurement of long lead items having progressed to the stage where firm pricing has been obtained and used to update the oxide capital estimate for the 2019 DFS.

Orezone will prepare, tender and award the mining contract to enable the successful contractor to mobilize and commence the OCR excavation works. It is likely that the mining contractor will be awarded elements of site bulk earthworks, particularly those associated with the raising of the TSF starter embankments.

Design of the TSF must also be completed early in the schedule and a contract for its construction awarded to generally coincide with the OCR excavation, as excavated OCR material is to be immediately transported and utilized in the construction of the TSF embankment.

A freight forwarder will be engaged prior to the end of 2019 to manage shipping and logistics requirements.

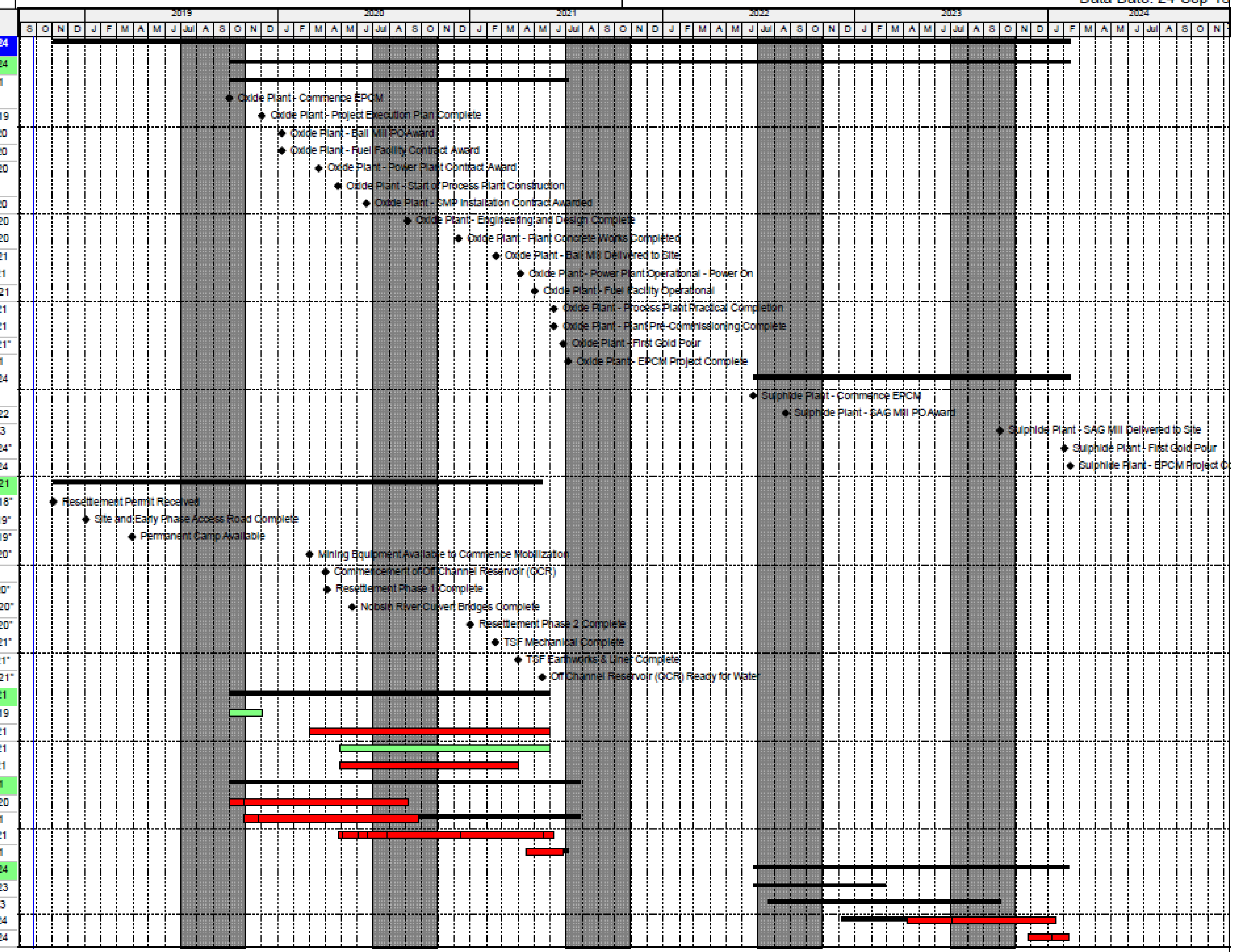
Construction of the plant will be completed using horizontal packages. Works in the plant area are anticipated to commence in April 2020, leading to first gold pour in June 2021.

A Level 1 schedule is provided in Figure 24.1.

Figure 24.1 Oxide and Sulphide Combined Level 1 Schedule

Bombore EPCM Oxide / Sulphide | Page 1 of 1 | Date: 07-Aug-19 | Data Date: 24-Sep-18

Activity ID	Activity Name	Original Duration	Start	Finish
Bombore EPCM Oxide / Sulphide		1376	01-Nov-18	12-Feb-24
Project Milestones		1139	01-Oct-19	12-Feb-24
Oxide Plant		459	01-Oct-19	05-Jul-21
500.EP0020	Oxide Plant - Commence EPCM	0	01-Oct-19*	
500.PM.0040	Oxide Plant - Project Execution Plan Complete	0		02-Dec-19
500.PM.0080	Oxide Plant - Ball Mill PO Award	0		09-Jan-20
500.PM.1020	Oxide Plant - Fuel Facility Contract Award	0		09-Jan-20
500.PM.0100	Oxide Plant - Power Plant Contract Award	0		19-Mar-20
500.PM.1040	Oxide Plant - Start of Process Plant Construction	0	24-Apr-20	
500.PM.0110	Oxide Plant - SMP Installation Contract Awarded	0		17-Jun-20
500.PM.0160	Oxide Plant - Engineering and Design Complete	0		03-Sep-20
500.PM.0130	Oxide Plant - Plant Concrete Works Completed	0		09-Dec-20
500.PM.0140	Oxide Plant - Ball Mill Delivered to Site	0		18-Feb-21
500.PM.0990	Oxide Plant - Power Plant Operational - Power On	0		05-Apr-21
500.PM.1030	Oxide Plant - Fuel Facility Operational	0		03-May-21
500.PM.0980	Oxide Plant - Process Plant Practical Completion	0		08-Jun-21
500.PM.0150	Oxide Plant - Plant Pre-Commissioning Complete	0		08-Jun-21
500.PM.1000	Oxide Plant - First Gold Pour	0		25-Jun-21*
500.PM.1010	Oxide Plant - EPCM Project Complete	0		05-Jul-21
Sulphide Plant		429	21-Jun-22	12-Feb-24
500.EP0025	Sulphide Plant - Commence EPCM	0	21-Jun-22	
500.PM.1060	Sulphide Plant - SAG Mill PO Award	0		22-Aug-22
500.PM.1070	Sulphide Plant - SAG Mill Delivered to Site	0		02-Oct-23
500.PM.1080	Sulphide Plant - First Gold Pour	0		01-Feb-24*
500.PM.1050	Sulphide Plant - EPCM Project Complete	0		12-Feb-24
Client and Interface Milestones		662	01-Nov-18	17-May-21
805.CM.0010	Resettlement Permit Received	0		01-Nov-18*
805.CM.0020	Site and Early Phase Access Road Complete	0		02-Jan-19*
805.CM.0030	Permanent Camp Available	0		30-Mar-19*
805.CM.0050	Mining Equipment Available to Commence Mobilization	0		02-Mar-20*
805.CM.0040	Commencement of Off Channel Reservoir (OCR)	0	01-Apr-20*	
805.CM.0060	Resettlement Phase 1 Complete	0		04-Apr-20*
805.CM.0070	Nobsin River Culvert Bridges Complete	0		21-May-20*
805.CM.0100	Resettlement Phase 2 Complete	0		31-Dec-20*
805.CM.0110	TSF Mechanical Complete	0		17-Feb-21*
805.CM.0090	TSF Earthworks & Liner Complete	0		01-Apr-21*
805.CM.0080	Off Channel Reservoir (OCR) Ready for Water	0		17-May-21*
Client		436	01-Oct-19	01-Jun-21
920.PP.1010	Project Execution Plan	45	01-Oct-19	02-Dec-19
920.PP.1050	Infrastructure	327	02-Mar-20	01-Jun-21
920.PP.1060	Mine Developement	287	27-Apr-20	01-Jun-21
920.PP.1070	Tailings Storage Facility (TSF) - Stage 1	244	27-Apr-20	01-Apr-21
Process Plant - Oxide		478	01-Oct-19	29-Jul-21
Engineering & Drafting		235	01-Oct-19	03-Sep-20
Procurement & Contracts		450	29-Oct-19	29-Jul-21
Construction		440	24-Apr-20	08-Jun-21
Commissioning / Start up		85	16-Apr-21	05-Jul-21
Process Plant - Sulphide		429	21-Jun-22	10-Feb-24
Engineering & Drafting		180	21-Jun-22	27-Feb-23
Procurement & Contracts		315	19-Jul-22	02-Oct-23
Construction		435	06-Dec-22	15-Jan-24
Commissioning / Start up		85	23-Nov-23	10-Feb-24



Actual Work | Remaining Work | Critical Remaining Work | Milestone | Summary

OREZONE | Bombore Project | EPCM Schedule | Date | Revision | Checked | Approved | Lycopodium

Source: *Lycopodium Minerals Canada*

Key dates in the Project schedule are as follows:

- Access Road & Camp Upgrades Complete
- Oxide Plant – Re-commence EPCM October 2019
- Commence OCR Design Q1 2020
- Award Oxide Ball Mill January 2020
- Award Fuel Facility Contract Q1 2020
- Commence Mine Development Q2 2020
- Commence TSF Stage 1 ` Q2 2020
- Award Power Supply Contract Q2 2020
- Resettlement Phase 1 Complete April 2020
- Oxide Plant – Commence Site Works April 2020
- Nobsin River Culvert Bridges Complete May 2020
- OCR Ready for Water May 2021
- TSF Stage 1 Complete May 2021
- Resettlement Phase 2 Complete December 2020
- Oxide Power Plant On April 2021
- Oxide Plant Practical Completion June 2021
- First Gold Pour June 2021
- Achieve Commercial Production October 2021
- Sulphide Plant – Commence EPCM June 2022
- Award Sulphide SAG Mill August 2022
- SAG Mill Delivered to Site September 2023
- Sulphide Commissioning Commences November 2023
- First Gold Attributable to Sulphide January 2024

Critical early activities on which the Project schedule depends have been identified as follows:

- Complete Phase 1 RAP.
- Preparation for OCR mining.
- Mobilization of the mining contractor.
- Detailed engineering for the oxide plant, TSF and early infrastructure.
- Award of long lead procurement items.
- Award early earthworks and OCR construction contracts.
- Complete OCR and TSF for 2021 wet season.

24.2 Operational Readiness

Orezone will establish an experienced operational management team on site well in advance of commissioning.

Comprehensive policies are currently being developed and induction, training and operating procedures are being implemented as part of an overall Operational Readiness Plan to ensure the transition from exploration to Project development to operations is managed in a safe and effective manner.

The current operating cost estimate is based on a preliminary organizational structure that is appropriate for an operation of this scale and type in the region.

The headcount by project year broken down by department and Expatriate/National numbers is shown in Table 24.1.

Table 24.1 Annual Headcount

	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
Mining (contract mining used)														
Ex-pats	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Nationals	50	50	58	58	58	58	58	58	58	58	58	58	58	58
Processing														
Ex-pats	8	8	9	6	6	6	6	6	6	6	6	6	6	6
Nationals	96	96	112	114	114	114	114	114	114	114	114	102	102	102
G&A														
Ex-pats	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Nationals	102	102	102	102	102	102	102	102	102	102	102	102	102	102
Headcount total	**263**	**263**	**288**	**287**	**287**	**287**	**287**	**287**	**287**	**287**	**287**	**275**	**275**	**275**

Note: Excluding contractors

24.2.1 Recruitment

Orezone is committed to providing preferential employment opportunities to Burkinabé nationals, based on identifying individuals with the necessary qualifications, experience and skills. This process has already commenced with the hiring of Burkinabé nationals in several senior positions at site and at the Ouagadougou office. The development of the local and regional mining industry in recent years has led to the growth of a pool of experienced labour, some of whom will be attracted to the Projects location within easy driving distance of Ouagadougou. Initially, however, it is anticipated that expatriate staff will fill a number of senior management, key supervisory and training roles. A succession plan will be developed to transition Burkinabé nationals into those roles.

Expatriate positions will be advertised internationally and regionally as appropriate which recognizes the growing trend of African expatriates that are now a large part of the international mining workforce. It is anticipated (and reflected in the annual manning predictions) that international expatriates will be replaced by regional expatriates and/or Burkinabé nationals on a progressive basis.

Allowance has been made to employ senior Burkinabé nationals on a single status ‘drive in, drive out’ basis from Ouagadougou. This will be limited to mostly operational support as some service departments will work from Orezone’s Ouagadougou office. Employment opportunities will be offered to local residents where feasible and appropriate.

24.3 Annual and Life-of-Mine Production

Life of mine (LOM) ore milled will be 70.1 Mt over a 14-year period treating predominantly run of mine ore followed by the treatment of low-grade stockpiles once the pits are exhausted.

Mill feed grade will be significantly higher in the early years as lower grade run-of-mine material will be stockpiled for processing at the end of mine life. Annual tonnes mined, ore milled tonnes and grade and gold production are shown in Table 24.2.

Table 24.2 Annual and LOM Production

Year	Total ore tonnes processed (Mt)	Gold grade (g/t)	Recoveries (%)	Gold Production ('000 oz)
Pre-prod.	1.21	1.02	92.3	37
1	5.19	1.03	92.3	159
2	5.2	0.91	91.2	139
3	5.2	0.97	88.7	144
4	5.2	1.01	88.7	150
5	5.2	0.96	87.2	140
6	5.2	0.89	85.0	126
7	5.2	0.88	86.0	126
8	5.2	0.85	85.4	122
9	5.2	0.85	85.3	122
10	5.2	0.78	85.8	112
11	5.2	0.62	85.8	89
12	5.2	0.50	83.9	70
13	5.2	0.40	80.1	54
14	1.3	0.37	78.7	12
Life of Mine	**70.1**	**0.81**	**87.2**	**1600**

24.4 Closure

Disturbed areas and dump slopes will be stabilised and revegetated on an ongoing basis to reduce sediment loads in run-off, to minimize the sites visual impact and to reduce the quantum of rehabilitation work required at closure.

The transfer at closure of buildings and infrastructure that have ongoing value to the community will be considered on a case by case basis and specific commitments in this regard are within the ESIA, otherwise buildings and facilities will be dismantled, foundations removed and buried, and sites stabilized and revegetated as appropriate.

In particular, it is considered likely that the OCR and the abandoned pit receiving run-off from the rehabilitated tailings storage facility will provide future sources of water for agricultural activities in the local area, reducing the dependence on agriculture linked to the annual wet season.

Appropriate drainage will be established on the TSF to ensure long-term stability and the surface capped and revegetated.

The Economic Modelling of the Project includes allowances for salvage values and closure costs. The TSF closure cost was derived from KP's estimated closure scope of work and rates derived from contractor proposals. Lycopodium provided an estimate of the salvage and scrap value of plant, equipment and transportable buildings based on a residual percentage of their purchase price. The balance of site rehabilitation costs was estimated by Orezone and other consultants participating in the study.

25.0 INTERPRETATION AND CONCLUSIONS

Based on the work undertaken, as summarized in this Technical Report, and the conclusions listed below from the individual Qualified Persons, the Bomboré Project is a viable development opportunity centred around the initial mining and processing of the oxide and upper transition zones of the mineralization material on the Bomboré tenements followed by the supplemental mining and processing of higher grade lower transition and sulphide material after the staged sulphide expansion to the processing plant.

25.1 Geology and Mineral Resources

The following conclusions for geology and Mineral Resources are from RPA:

- The Bomboré gold deposit is a large structurally-controlled orogenic gold deposit similar to deposits found elsewhere in late Proterozoic Birimian terranes of West Africa.

- Drilling has outlined mineralization with three-dimensional continuity, and size and grades that can potentially be extracted economically.

- Orezone's protocols for drilling, sampling, analysis, security, and database management meet industry standard practices.

- The drill hole database was verified by Orezone, RPA, and other consultants and is suitable for Mineral Resource estimation work.

- In RPA's opinion, the 2017 Mineral Resource estimation work is in accordance with the CIM (2014) definitions and the results are reasonable.

- At a cut-off grade of 0.2 g/t Au for oxide and transition material and 0.38 g/t Au for fresh material, M&I Mineral Resources are estimated to total 229.4 Mt at an average grade of 0.69 g/t Au for 5.1 million ounces of contained gold. At the same cut-off grades, Inferred Mineral Resources are estimated to total an additional 53.3 Mt at an average grade of 0.65 g/t Au for 1.1 million ounces of contained gold.

- The updated Mineral Resource estimate has a slightly lower average grade compared to the September 7, 2016 Mineral Resource estimate due to the addition of 391 new low-grade mineralized wireframes and a new unconstrained domain ("third domain") of selected assays above 0.20 g/t Au.

- Data from 497 new holes totalling 30,760 m, located within the resource area but outside P17S, were received after the resource database was finalized for the January 5, 2017 resource statement. RPA reviewed the results and is of the opinion that the resource model is still appropriate to be used as the basis for this Technical Report and that the effective date remains at January 5, 2017.

- A number of mineralized drill hole intervals, though included within the database, remain beyond the limits of the mineralization wireframes. No tonnage or grade estimates were provided for these intersections at this stage.

25.2 Mining

The Bomboré mine will be developed as an open pit operation mining oxide and sulphide material from 60 separate pits of variable size and depth across a mineralized zone approximately 12.2 km long and 3 km wide. The oxides include the regolith, upper saprolite, lower saprolite, and upper transition weathering units. The oxide material can be readily excavated in situ (free-dig material). The sulphides include lower transition and fresh rock, which will require a varying degree of drill and blast prior to being loaded onto trucks.

The key project life of mine (LOM) highlights are:

- 236.2 Mt total material mined:
  - 71.8 Mt of mineralized material:
    - 70.1 Mt of ore at 0.81 g/t Au mined and processed, including 52.5 Mt of oxides at 0.69 g/t Au and 17.6 Mt of sulphides at 1.19 g/t Au.
    - 1.7 Mt of mineralized low-grade material remaining on stockpiles and not processed at the end of the mine life.
    - 164.4 Mt waste.
    - 2.34 strip ratio.
    - 13.3-year mine life.
- Pre-production mining of 1.5 years, including excavation of the Off-Channel Reservoir (OCR) for water storage and supply.
- Total production:
  - 54.5 Mt at 0.88 g/t Au ROM ore.
  - 15.6 Mt at 0.60 g/t Au low-grade ore rehandled from stockpiles.
  - 1.6 Moz Au produced.

The Mineral Reserve Estimate is based on the updated Mineral Resource Estimate (MRE) prepared by RPA with an effective date of January 5, 2017 and incorporates the oxide material within the previously excluded “Restricted Zones” and takes into account all drilling completed to December 31, 2018 on the P17S deposit.

This Technical Report considered all available Measured and Indicated material in the MRE within the oxide, transition, and sulphide horizons. AMC developed mine models by applying modifying factors to the resource block models using Datamine's™ Studio OP software (Datamine). Pit optimizations were conducted on the mine models using Gemcom's Whittle™ 4.X software (Whittle). The pit optimization was then used as basis for producing practical mine designs. The mine block models were evaluated against the mine designs to provide the Mineral Reserve Estimate.

The weathered saprolite and upper transition (UT) horizons, which reach a thickness of up to 90 m across the site, can be excavated without the need for prior blasting (free-dig material).

AMC assumed that 70% of the Lower Transition (LT) material will require ripping prior to being loaded onto the haul trucks, while the remaining 30% will have to be blasted.

The sulphide material below the weathered horizons requires drill-and-blast.

The pit optimization included both oxide and sulphide horizons, with inputs varied depending on the proposed mining method. Inferred Mineral Resources were treated as waste, and only Measured and Indicated Mineral Resources were considered as feed to the processing plant.

The gold price of US$ 1,250/oz and associated off-site charges were provided by Orezone in calculating the Mineral Reserve Estimate. Royalties are applied to the totality of the gold produced.

The mine will be contractor operated with mining of oxides undertaken with 4.5 m^3 hydraulic excavators (i.e. Komatsu PC850) and 30-50 t highway dump trucks. The sulphides will be mined using a separate fleet (i.e. Komatsu PC1250 and 50 t Volvo FMX rigid body trucks) to account for the increased density, abrasion, and hardness of the material. An owner's team on site will be responsible for contract management, grade control and mine planning activities.

25.3 Tailings Disposal and Site Water Management

The following conclusions for tailings disposal and site management are from KP:

- The large majority of materials used in the TSF dams will be mine waste, although some select materials for filters and drains may be sourced elsewhere.
- By utilizing sub-aerial rotational tailings deposition methods, the actual tailings density achieved is expected to exceed the design tailings density thus creating more capacity within the TSF or will allow the dams to be lowered in elevation to achieve the same storage tonnage.

- The TSF has been designed with sufficient capacity to satisfy the water storage design criteria, comprising of the average supernatant pond volume per month of operation plus the precipitation volume generated from the 72-hour PMP storm event plus 1.0 m minimum freeboard. The TSF supernatant pond volumes will increase over time due to increasing contributing areas to the WRD and LGS collection ponds that will have their water pumped to the TSF for ultimate use in the process. Other factors that could affect the TSF supernatant pond volumes include any changes to the tailings slurry solids content and the in-storage tailings unit weights. Diligent monitoring and management of the TSF supernatant pond will be critical throughout the operational life.

- The OCR will be a benefit to the local population as a long-term water resource after mine operations cease.

25.4 Metallurgy and Process

The following conclusions for metallurgical testwork with regards to CIL process are from Lycopodium:

- Oxide, transition and sulphide ores at Bomboré are readily amenable to CIL whole ore cyanidation.

- Oxide plant: Gold recoveries are predicted to be over 90% for head grades over 0.80 g/t Au, high 80%'s for head grades of 0.55 g/t Au to 0.80 g/t Au, and low 80%'s for head grades of 0.4 g/t Au to 0.55 g/t Au.

- Sulphide plant: Gold recoveries are predicted to be over 80% for head grades over 0.70 g/t Au, and stay in the high 70%'s even for lower head grades.

- Optimum grind size for the oxide plant was determined to be P_{80} of 125 µm based on grind size and recovery relationship.

- Optimum grind size for the sulphide plant was selected to be P_{80} of 75 µm however, this requires additional testwork to finalize it.

- Leach extraction rates are essentially complete within 24 hours based on the observed leach kinetics.

- Oxygen addition is beneficial for sulphide ore leaching.

- Cyanide consumption rates are expected to be low, averaging about 0.19 kg/t NaCN for the oxide ore and about 0.37 kg/t NaCN for the sulphide ore.

- Lime consumption rates are expected to be moderate, averaging about 1.86 kg/t CaO for the oxide ore and about 1.35 kg/t CaO for the sulphide ore.

Sufficient testwork and engineering studies have been completed to support the development of a process plant based on CIL technology to economically recover gold from the Bomboré ores. There are, however, some opportunities to further optimize the sulphide circuit and it is recommended these be investigated, by way of a modest testwork program, prior to commencing detail design.

25.5 Environmental and Permitting

The Mining Code of Burkina Faso guarantees a stable fiscal regime for the life of any mine developed. It also guarantees stabilization of financial and customs regulations and rates during the period of operation to reflect the rates in place at the date of the approved Industrial Operating Permit. The Mining Code also states that no new taxes can be imposed except for mining duties, taxes and royalties.

In 2016, Orezone received the Industrial Operating Permit following the delivery and acceptance by the authorities of the ESIA and RAP. Once in production, a mining permit holder is required to open under his name a fiduciary account named Fonds de préservation et de réhabilitation de l'environnement minier at the Banque Centrale de États de l'Afrique de l'ouest (BCEAO). This account must be funded annually on January 1st by an amount equal to the total rehabilitation budget divided by the number of years of production to cover the costs of mine reclamation, closure and rehabilitation.

In February 2019, Orezone signed the mining convention with the State of Burkina Faso. The purpose of the mining convention is to clarify the rights and obligations of the parties and to guarantee Orezone stability, including taxation and foreign exchange regulation. The mining convention is not a substitute for the law but specifies the provisions of the law. It is valid for the initial duration of the operating license and is thereafter renewable for one or more periods of five years at the request of Orezone.

All laterite, oxide, and transition units demonstrate little potential to generate acid; however, in the Siga South and the P8P9 prospects the transition zone meta-sediments report variable ARD potential and a minor portion of this unit could have the potential to generate ARD. The laterite, oxide and transition units are not expected to leach metals at concentrations above IFC effluent guidelines, although the possible discharge of arsenic should be considered. Geochemical studies conducted to date suggest that arsenic leaching will be minimal.

No terrestrial species has a conservation status at either national or international level, except for one bird species, the Hooded Vulture (*Necrosyrtes monachus*), that is Endangered according to the IUCN red list.

25.6 Social, Community and Resettlement

The resettlement of many people (about 731 households or about 5,095 people) from seven traditional villages, as well as two artisan gold processing sites (about 1,360 households or about 3,100 people) and the expropriation of a large area of agricultural land (about 656 ha) represents a complex activity that will require an immediate and important focused effort. The processing infrastructure is in the northern area of the Project where about 60% of the gold resources are located. This area will have to be cleared prior to the start of any major construction activities. This will require the initial (Stage 1) resettlement of approximately 410 households from traditional villages and the expropriation of approximately 915 households from the Sanam Yaar artisanal gold processing site. The subsequent resettlement (Stage 2) of approximately 250 farming households and the expropriation of 450 households from the Kagtanga artisanal gold processing site, all from the southern area of the Project, could occur after the initial Phase 1 resettlement as this area will not be immediately affected by the mine construction.

Orezone has successfully completed the expropriation and the settlement of the compensations to the households from the Sanam Yaar and Kagtanga artisan gold processing sites and construction of the Phase 1 resettlement sites is in progress.

It is important to note that Orezone personnel have direct experience with resettlement on a large scale at a previous project in Burkina Faso.

26.0 RECOMMENDATIONS

The Property hosts a significant gold deposit with estimated Mineral Resources and Mineral Reserves. Orezone has undertaken considerable exploration, front-end engineering, and development work. The near-term primary objective is to further advance engineering work on the Project as discussed herein. Investigation of the Project optimizations noted in this Technical Report should also be considered. The overall recommendations are discussed below.

26.1 Mineral Resource Model

- Incorporation of the in-fill drilling completed after the current resource model cut-off date, to further confirm the continuity of the gold grades in the high-grade pods contained within the resource pit shell is warranted, in particular in the South model area. The information gained from this work will improve the variogram models in these areas and will improve the accuracy and level of confidence of the local grade estimate.

- The use of a single grade threshold (of 0.2 g/t Au) for creation of mineralization envelopes would simplify the estimation process. RPA recommends that Orezone test this approach by re-running the grade estimate without the 0.45 g/t Au envelopes.

- Complete a detailed study to determine the optimal grade control drill hole spacing. The selection of the test areas for these studies should be synchronized with the proposed mine production schedule to focus on the initial production period.

26.1.1 Proposed Exploration Programme and Budget

RPA has reviewed and concurs with Orezone's proposed budgets. The recommended Phase I program, to be initiated as soon as operationally practical, consists mainly of resource modelling. The budget for this program is US$320,000.

Details of the recommended Phase I program can be found in Table 26.1.

Table 26.1 Proposed Phase I Budget

Tenement	Program Phase I	Quantity	Budget (US$)
Mining Lease	Resource Model Update		150,000
	Restricted Zones/Sulphides ESIA		100,000
Toéyoko	P13 Maiden Resource Model		35,000
Bomboré II	KT Maiden Resource Model		35,000
TOTAL Phase I			**320,000**

Contingent upon the Phase I program results, a Phase II program consists of resource expansion RC and core drilling on the mining permit, high-resolution resistivity surveys on regional targets, and resource expansion RC and core drilling on the satellite deposits and regional targets. The budget for this Phase II exploration program is US$1,192,000 (Table 26.2).

Table 26.2 Proposed Phase II Exploration Budget

Tenement	Program Phase II (Contingent on positive Phase I results)	Quantity	Budget (US$)
Mining Lease	Oxide Resource Expansion RC Drilling	5,000 m	250,000
	Sulphide Resource Expansion Core Drilling	2,500 m	313,000
Toéyoko	P13-Resistivity Survey	121 km	58,000
	P13-Resource Expansion RC Drilling	2,500 m	125,000
	P13-Resource Expansion Core Drilling	750 m	90,000
	BV1-Resistivity Survey	22 km	19,000
	BV1-Scout RC Drilling	500 m	27,000
Bomboré II	KT-Resistivity Survey	23 km	20,000
	KT-Resource Expansion RC Drilling	2,500 m	125,000
	KT-Resource Expansion Core Drilling	750 m	90,000
Bomboré III	P17S-NE-Resource Expansion Core Drilling	500 m	75,000
TOTAL Phase II			**1,192,000**

26.2 Mining

AMC recommends the following steps moving from FS into Front-End Engineering and Design:

- Grade control drilling conducted in test areas to reconcile against the Mineral Resource and Mineral Reserve Estimates. The OCR construction should be used as a test case area to trial the proposed grade control procedures and make any required adjustments prior to the commencement of operations.

- Continued negotiations with local and regional mining contractors as detailed mine planning progresses.

- Optimization of in-pit and ex-pit haul road designs to minimise haulage distances and construction requirements.

- Further investigations and definition of local borrow source materials required for haul road construction.

- Investigations into locally sourced dust suppression agents and associated trade-off versus usage of water.

- Further investigations into pit dewatering to improve pumping requirement calculations.

- Further investigation and field tests into the free digging potential and ripping requirements of transition material.

26.3 Tailings Disposal and Site Water Management

It is recommended, as indicated in Section 25.7.1 that the opportunity to slightly realign the southwest portion of the tailings dam be investigated to avoid the requirement to backfill a shallow pit that extends into the footprint of the dam in this area. A realignment of the TSF such that it will maintain a 20 m buffer from the pit will remove this pit backfilling requirement for a slight crest height increase.

A dam break assessment for the final TSF dam configuration should be completed.

26.4 Metallurgy and Process

Front-End Engineering and Design for the oxide process plant is substantially complete while detailed design is scheduled to commence in Q4 2019. It is recommended that work recommence early in the fourth quarter of 2019 if the current implementation schedule is to be achieved.

It is recommended that during plant operations:

- Natural cyanide attenuation (free and WAD) be monitored in the TSF.

- 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.

26.5 Environmental and Permitting

The Environmental and Social Management Plan (ESMP) to support the sulphide expansion and "Restricted Zones" will need to be approved by the Bureau National des Évaluations Environnementales (BUNEE). The required work on the ESMP has been substantially completed by Orezone and it is expected that Orezone will submit the final ESMP to BUNEE in Q4 2019.

26.6 Overall Recommendation

It is recommended that Orezone commence implementation of the Project in line with the preliminary implementation plan and schedule developed during the FS, thus committing to the capital expenditure presented in Section 21.

Initial work will include:

- Complete execution of Phase 1 of the RAP (in progress).

- Appointment of a lead EPCM or EPC Engineer.

- Further development of the FS schedule and budgets into detailed control tools for executing the project (in progress).

- Additional metallurgical testwork on the sulphides and Lower Transition to determine optimal recoveries.

- Finalization of Front-End Engineering and Design across the Project scope and commencement of detailed design.

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