**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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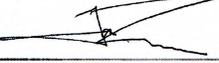Jean-Sebastien Houle, P.Eng., Project Director, WSP Canada Inc.

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## DATE & SIGNATURE PAGE

Project Name:            Bomboré Gold Project

Title of Report:         NI 43-101 Technical Report

Location:                Burkina Faso

Effective Date of Report:   9 July 2018

**[SIGNED]**                                                   **DATE**

| | | |
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## 1.0    SUMMARY

### 1.1    Introduction

In 2017 Orezone Gold Corporation (Orezone or The Company) after detailed review of all historical work, determined that the most viable option for the Bomboré Gold Project (BGP or the Project) would be to focus on the free digging oxide resources with a simple Carbon in Leach (CIL) gold recovery circuit. Orezone subsequently engaged several leading consulting firms to undertake a Feasibility Study (FS) to define and cost this preferred development option and prepare a FS Report and a corresponding Canadian National Instrument 43-101 (NI 43-101) compliant technical report (Technical Report). The principal contributors to the study are shown in Table 1.1.

**Table 1.1          Study Contributors**

| Contributor | Scope |
|---|---|
| Orezone Gold Corporation (Orezone) | Project history, metallurgical testwork, resettlement |
| Lycopodium Minerals Canada Limited (Lycopodium) | Metallurgy 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 |
| WSP Global Inc. (WSP) | Environment, permitting, community relations |

The study on which this NI 43-101 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 paved road, national highway N4 from Ouagadougou. The project area is approximately 11 km north to south and approximately 3.7 km east to west.

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**
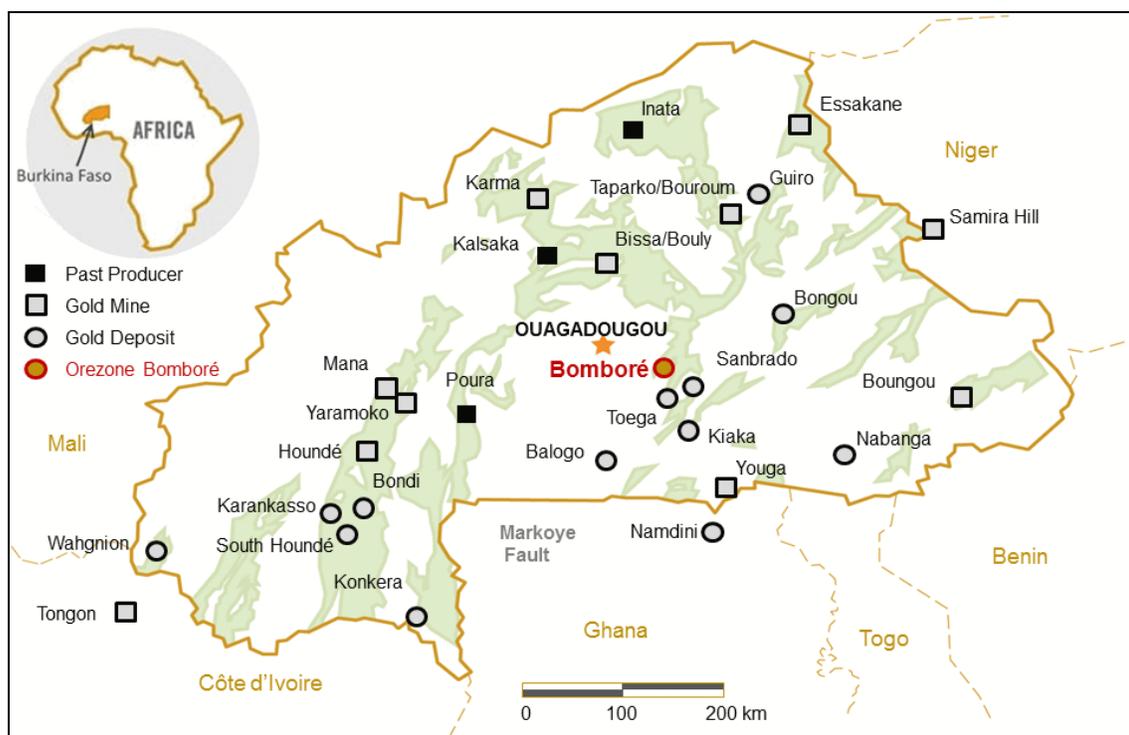


The Property consists of one Industrial Operating Permit (the Bomboré Mining Permit) of 2,500 ha, surrounded by four Mining Exploration Permits: the Toéyoko Exploration Permit of 4,669 ha, the Bomboré II Exploration Permit of 1,815 ha, the Bomboré III Exploration Permit of 4,810 ha and the Bomboré IV Exploration Permit of 1,235 ha (Figure 1.2).

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 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 final 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, with one deposit on the Toéyoko Exploration Permit (P17S) and one 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 produced 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**
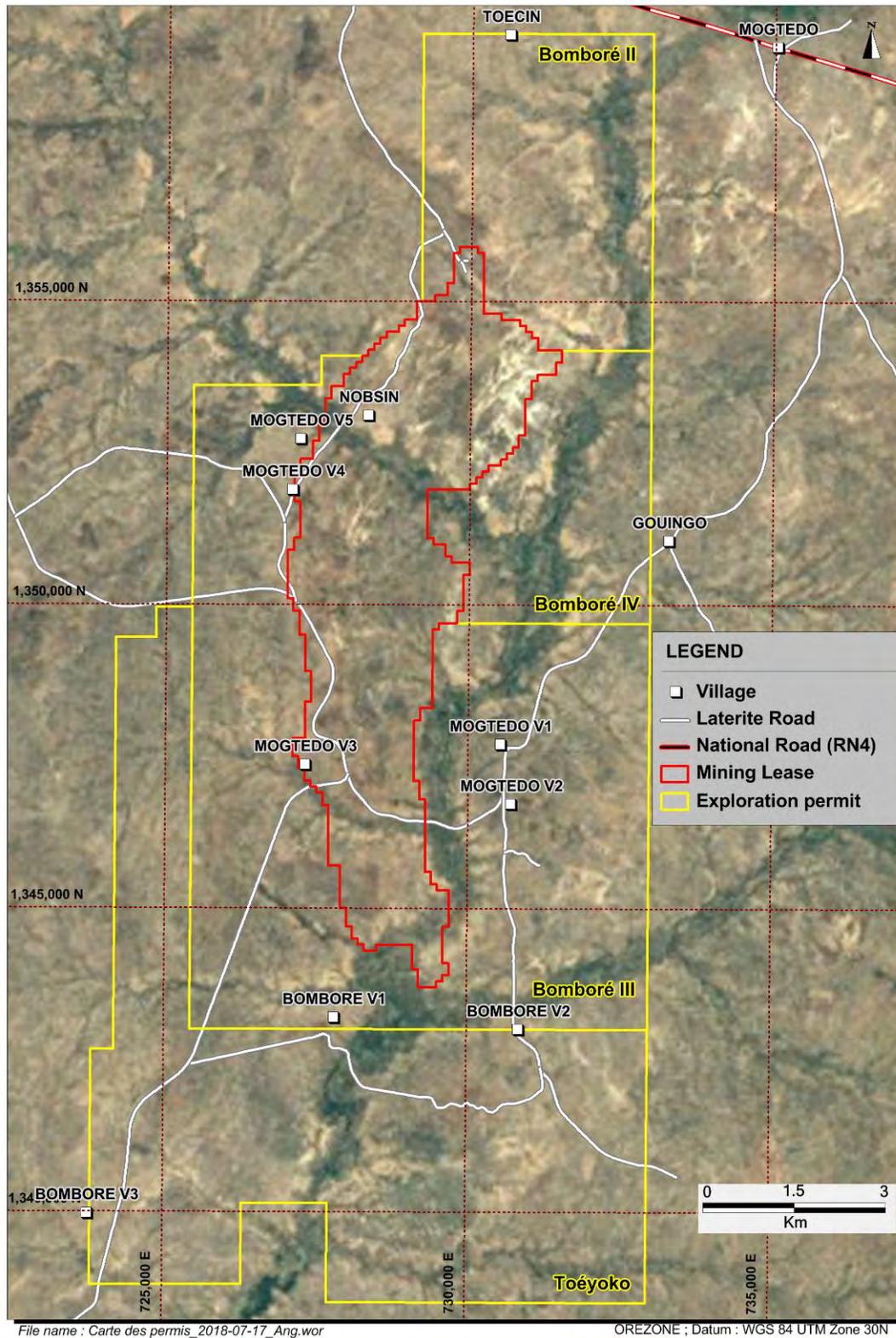
## 1.3     Geology and Mineralization

The Project lies in a 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.  The Project 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 (peridotitic) 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 on the west of the property.

The gold mineralization on the Property is hosted in the Bomboré Shear Zone (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 Bomboré gold mineralization trend is defined by a gold-in-soil anomaly in excess of 0.1 g Au/t, as well as by the presence of numerous gold showings and orpaillage (artisanal mining) sites.  The Bomboré gold-in-soil anomaly measures 14 km in length, is several hundreds of metres in width, and occurs within the BSZ.

## 1.4     Exploration Status

The BGP 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 metres), 261 RC (19,501 metres) and 1,000 rotary air blast (34,249 metres) boreholes on the BGP 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 2014, Orezone completed 1,025 core holes for approximately 156,000 m, 4,703 RC holes for approximately 284,000 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 the 2018 FS and that the effective date should remain at January 5, 2017.

## 1.5    Mineral Resources

The estimation of Mineral Resources was carried out initially by Orezone which included the development of the geological model, block model and the grade estimation.  RPA carried out audits, updated and classified the initial model and reported the updated Mineral Resource estimate.  Mineral Resources are estimated at variable cut-off grades depending on weathering layer and geographic location.  At a cut-off grade of 0.2 g Au/t oxide and transition material and 0.38 g Au/t for fresh layers, Measured and Indicated (M&I) Mineral Resources are estimated to be 218.1 Mt at an average grade of 0.68 g Au/t for a total of 4.8 million ounces of contained gold.  Using the same cut-off grades, Inferred Mineral Resources are estimated to total an additional 48.2 Mt at an average grade of 0.64 g Au/t for 994,000 ounces of contained gold.  The Mineral Resource estimate has an effective date of January 5, 2017.  RPA reviewed the results of drilling on the Property since that date and is of the opinion that the current resource model is still appropriate to be used as the basis for the 2018 FS described herein.

The gold mineralization at the Project has a strike length of 12 km and has been split into four separate block model areas 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, and P17 areas.  Together, the North and South block models contain the majority of the Mineral Resources.

The methodology included estimating the grade in two principal grade domains, a higher grade +0.45 g Au/t domain (the core of mineralization) and a lower grade 0.2 g Au/t to 0.45 g Au/t domain (the lower grade halo around the core).  Gold grades were estimated using the ordinary kriging (OK) interpolation algorithm for the North, South, and P16 model areas.  The gold grades for the P17 model area were estimated using the inverse distance squared ($ID^2$) interpolation algorithm.  The gold grades inside the January 2017 new low-grade mineralized wireframe models for the North and South areas were estimated using the OK interpolation algorithm; there were no new low-grade wireframe domain models in the P16 and P17 model areas.  The grade of each domain (or envelope) was estimated using only the composited assays that occur within each envelope and thus there was a hard boundary between each domain.

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 criteria to constrain the Mineral Resource report included the exclusion of several non-permitted areas related to flood plains, environmentally sensitive areas, and mineralized areas being set aside for the benefit of local artisanal miners.

The January 5, 2017 resource model was initially prepared by Orezone and was subsequently audited, updated, classified, and accepted by RPA as part of the Mineral Resource estimation (Table 1.2).  The estimate conforms to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) 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**

| | Measured | | | Indicated | | | Measured + Indicated | | | Inferred | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Tonnes | Grade | Gold | Tonnes | Grade | Gold | Tonnes | Grade | Gold | Tonnes | Grade | Gold |
| Material Type | Mt | g Au/t | koz | Mt | g Au/t | koz | Mt | g Au/t | koz | Mt | g Au/t | koz |
| Oxide+Tran HG | 16.9 | 0.94 | 513 | 36.5 | 0.83 | 974 | 53.4 | 0.87 | 1,487 | 4.8 | 0.77 | 117 |
| Oxide+Tran LG | 18.5 | 0.33 | 196 | 50.1 | 0.33 | 531 | 68.6 | 0.33 | 727 | 16.4 | 0.29 | 151 |
| **Total Oxide+Tran** | **35.4** | **0.62** | **709** | **86.7** | **0.54** | **1,505** | **122.0** | **0.56** | **2,214** | **21.2** | **0.39** | **268** |
| Fresh HG | 2.3 | 1.18 | 87 | 68.7 | 0.96 | 2,121 | 71.0 | 0.97 | 2,208 | 20.1 | 0.97 | 630 |
| Fresh LG | 0.8 | 0.43 | 11 | 24.2 | 0.43 | 337 | 25.0 | 0.43 | 348 | 6.9 | 0.43 | 96 |
| **Total Fresh** | **3.1** | **0.99** | **97** | **93.0** | **0.82** | **2,458** | **96.0** | **0.83** | **2,556** | **27.0** | **0.84** | **726** |
| **Total HG** | **19.2** | **0.97** | **600** | **105.3** | **0.91** | **3,095** | **124.5** | **0.92** | **3,695** | **24.9** | **0.93** | **747** |
| **Total LG** | **19.2** | **0.33** | **206** | **74.4** | **0.36** | **868** | **93.6** | **0.36** | **1,075** | **23.3** | **0.33** | **246** |
| **Total HG+LG** | **38.4** | **0.65** | **806** | **179.6** | **0.69** | **3,964** | **218.1** | **0.68** | **4,770** | **48.2** | **0.64** | **994** |

*Notes:*

1. *CIM (2014) definitions were followed for Mineral Resources.*

2. *HG indicates high grade resources above the higher cut-offs (0.45 g Au/t for oxide and transition, and 0.5 g Au/t for fresh), LG indicates low grade material between the breakeven cut-off grades (0.2 g Au/t for oxide and transition, and 0.38 g Au/t for fresh) and HG cut-offs.*

3. *Mineral Resources are estimated at variable cut-off grades depending on weathering layer and location; cut-off grades are approximately 0.2 g Au/t for oxide and transition material, and 0.38 g Au/t for fresh material.*

4. *Mineral Resources are estimated using a long-term gold price of US$1,400 per ounce.*

5. *A minimum mining width of approximately 3 m was used.*

6. *Bulk density varies by material type.*

7. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*

8. *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 Mineral Resource estimate.

## 1.6     Mineral Reserves

The Reserve estimate is based on the 2017 Mineral Resource estimate prepared by RPA. The Mineral Resource consists of three separate block models:

- North model including the KT, 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.

AMC developed Mineral Reserve models by applying modifying factors to the resource block models using Datamine's Studio OP software (Datamine). Diluted models were optimized using Gemcom's Whittle 4.X software (Whittle). The pit optimization was then used as a basis for producing practical mine designs in Datamine. The reserve block models were evaluated against the mine designs to provide the Mineral Reserve Estimate.

This study considered only the weathered saprolite and upper transition horizons which reach a thickness of up to 90 m across the site; these material types can be excavated without the need for prior blasting (free dig material).

AMC constrained the pit optimization to free dig material only. In addition, Inferred Mineral Resources were treated as waste, and only Measured and Indicated Mineral Resources were considered as feed to the processing plant.

**Table 1.3          Mineral Reserve Estimate – July 9, 2018**

| Classification | Units | Block 1 | Block 2 | Block 3 | Block 4 | Block 5 | Total |
|---|---|---|---|---|---|---|---|
| **Summary of Mineral Reserves** | | | | | | | |
| **Cut-off Grade** | g Au/t | 0.29 | 0.28 | 0.29 | 0.30 | 0.28 | |
| **Proven:** | | | | | | | |
| Ore | 000 t | 8,038 | 13,308 | 0 | 0 | 192 | **21,539** |
| Gold Grade | g Au/t | 0.71 | 0.66 | 0.00 | 0.00 | 0.85 | **0.68** |
| Contained Gold | 000 oz Au | 182 | 284 | 0 | 0 | 5 | **472** |
| **Probable:** | | | | | | | |
| Ore | 000 t | 7,684 | 11,859 | 9,398 | 5,526 | 2 | **34,468** |
| Gold Grade | g Au/t | 0.61 | 0.54 | 0.67 | 0.68 | 0.57 | **0.61** |
| Contained Gold | 000 oz Au | 151 | 205 | 201 | 120 | 0 | **678** |
| **Proven & Probable:** | | | | | | | |
| **Ore** | **000 t** | **15,722** | **25,167** | **9,398** | **5,526** | **194** | **56,007** |
| **Gold Grade** | **g Au/t** | **0.66** | **0.60** | **0.67** | **0.68** | **0.85** | **0.64** |
| **Contained Gold** | **000 oz Au** | **333** | **489** | **201** | **120** | **5** | **1,149** |

- *Mineral Reserves have been estimated in accordance with the CIM Definition Standards.*

- *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.*

- *Mineral Reserves are estimated at an average long-term gold price of US$ 1,250/oz.*

- *Mineral Reserves are reported effective July 9, 2018*

## 1.7 Mining

The Bomboré mine will be a free dig open pit mining operation consisting of over 60 separate pits of variable size and depth across a mineralized zone 11 km long and 2.5 km wide. The production schedule is based on the Mineral Reserve Estimate summarized below and described in Section 15. Mine operations are planned to span over 12 years. Run of mine (ROM) ore together with reclaimed ore from stockpiles will be the source of material to the plant for the first 11 years followed by processing low-grade stockpiles exclusively for the final 2 years.

The key project life-of-mine highlights are:

- 149.8Mt total material mined:

    - 56.0Mt of ore

    - Diluted ore grade 0.64 g Au/t

    - 1.1 million in situ ounces Au

    - 93.8Mt waste

    - 1.68 strip ratio

    - 13 year mine life

- Pre-production of 1 year for excavation of Off-Channel Reservoir (OCR) for water storage and supply of waste for Tailings Storage Facility (TSF) construction.

Mining of ore and waste will be by contractor with an owner's team on site responsible for site management, grade control and mine planning activities. Mining will be undertaken with 3-6 m$^3$ excavators and several 30-50 tonne highway dump trucks.

ROM ore will be hauled to the process plant and low-grade ore hauled to one of three low-grade stockpiles. Approximately 82% of waste produced will be used for TSF embankment construction with the remainder being hauled to two waste rock dumps (WRD) and four environmental barriers.

## 1.8 Metallurgy

Extensive testwork, at different laboratories, has been conducted for the Project with the first test program initiated in 1997 and the latest completed in 2018. The test programs were conducted on drill core composites, RC cuttings, and rotary air blast (RAB) drill samples. RAB testwork data was not used in the process design. The samples tested were sourced from multiple drill holes considered to be representative of the material that will be processed through the mine life.

The following conclusions can be made from the metallurgical testwork with regards to a Carbon in Leach (CIL) process:

- Oxide and transition ores at Bomboré are readily amenable to whole ore cyanidation.

- Gold recoveries are over 90% for head grades over 0.75 g Au/t and dropping to just over 80% for head grades below 0.5 g Au/t.

- Optimum grind size was determined to be $P_{80}$ 125 µm based on grind size and recovery relationship.

- Leach extraction is essentially complete within 24 hours based on the observed leach kinetics.

- Cyanide consumption levels are expected to be low, averaging about 0.19 kg NaCN/t ore.

- Lime consumption levels are expected to be moderate, averaging at 2 kg/t ore for 90% CaO quicklime.

Key process design criteria are shown in Table 1.4.

**Table 1.4          Key Process Design Criteria**

| Criteria | Units | Design | Notes / Source |
|---|---|---|---|
| Plant Throughput | tpa | 4,500,000 | Agreed design throughput |
| Ore Blend Per Mine Plan - Oxide | % | 85 | Mine plan |
|                          - Transition | % | 15 | Mine plan |
| Head Grade        - Gold (Design) | g Au/t | 1.0 | Agreed design parameter |
|                 - Gold (LOM average) | g Au/t | 0.64 | Mine plan |
|                 - Silver | g Ag/t | 0.97 | Agreed design parameter |
| Gold Extraction Estimation at 1 g Au/t | | | |
|         - Upper Oxide | % | 92.7 | Extraction equation |
|         - Lower Oxide | % | 92.5 | Extraction equation |
|         - Upper Transition | % | 92.2 | Extraction equation |
|         - Per Mine Plan Ore Blend | % | 92.5 | Calculated |
| Silver Extraction Estimation | % | 50.3 | Avg. McClelland test result |
| Overall Recovery Estimation at 1 g Au/t | | | |
|         - Gold | % | 92.2 | Calculated |
|         - Silver | % | 49.8 | Calculated |
| Ore Specific Gravity | t/m³ | 2.8 | Testwork |
| Bond Ball Mill Work Index (BWi, blend) | kWh/t | 5.3 | Testwork |
| Bond Abrasion Index (Ai) | | 0.035 | Testwork |
| Grind Size $P_{80}$ | µm | 125 | Agreed design parameter |
| CIL Circuit Residence Time | hrs | 24 | Based on fast leach kinetics in testwork |
| CIL Slurry Density (for saprolitic ore) | % solids | 40% | Lycopodium experience |
| Sodium Cyanide Consumption | kg NaCN/t | 0.28 | Calculated |
|     - Consumption per Blended Ore | kg NaCN/t | 0.19 | Testwork |
|     - NaCN Loss to Tails | kg NaCN/t | 0.17 | Calculated based on 60 ppm CN⁻ in Tails |
|     - Recycled NaCN in Process Water | kg NaCN/t | 0.08 | Calculated |
| Lime Consumption (90% purity) | kg/t | 2.07 | Testwork/Calculated |

## 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 operating costs.

The treatment plant design incorporates the following unit process operations:

- 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 overflow slurry density suitable for direct feed to the leach tanks.

- A leaching circuit with one (1) leach and six (6) CIL tanks to achieve the required 24 hours of residence time for optimum leach recovery. A pre-CIL leach tank is used to reduce capital cost and raise the gold tenor of the leach solution prior to loading on the carbon in the first CIL tank.

- A tailings thickener for cyanide and process water recovery and to provide an optimum slurry density for disposal to the TSF to minimize water losses by evaporation on the dam.

- 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.

A block flowsheet and process plant layout is provided in Figure 1.3 and 1.4.

**Figure 1.3** **Overall Process Block Flow Diagram**

**Figure 1.4          Process Plant Plan**
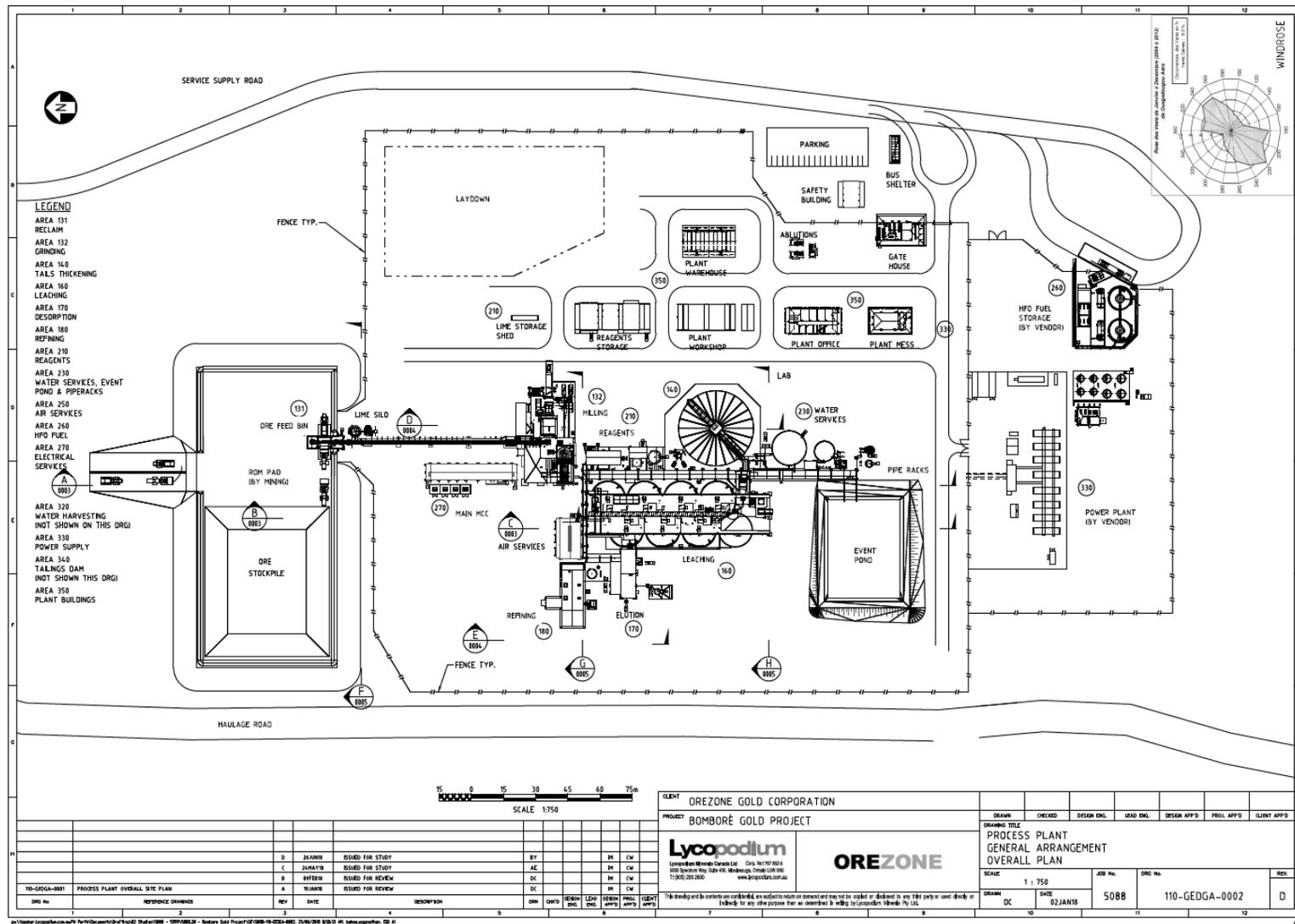
The process plant is designed to operate with zero discharge of process solutions to the environment. To ensure compliance the plant includes a lined event pond designed to contain any foreseeable spillage event. The event pond, combined with the bunded concrete areas within the plant perimeter, is designed 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 1.5 and 1.6 below.

**Figure 1.5          Site Layout**
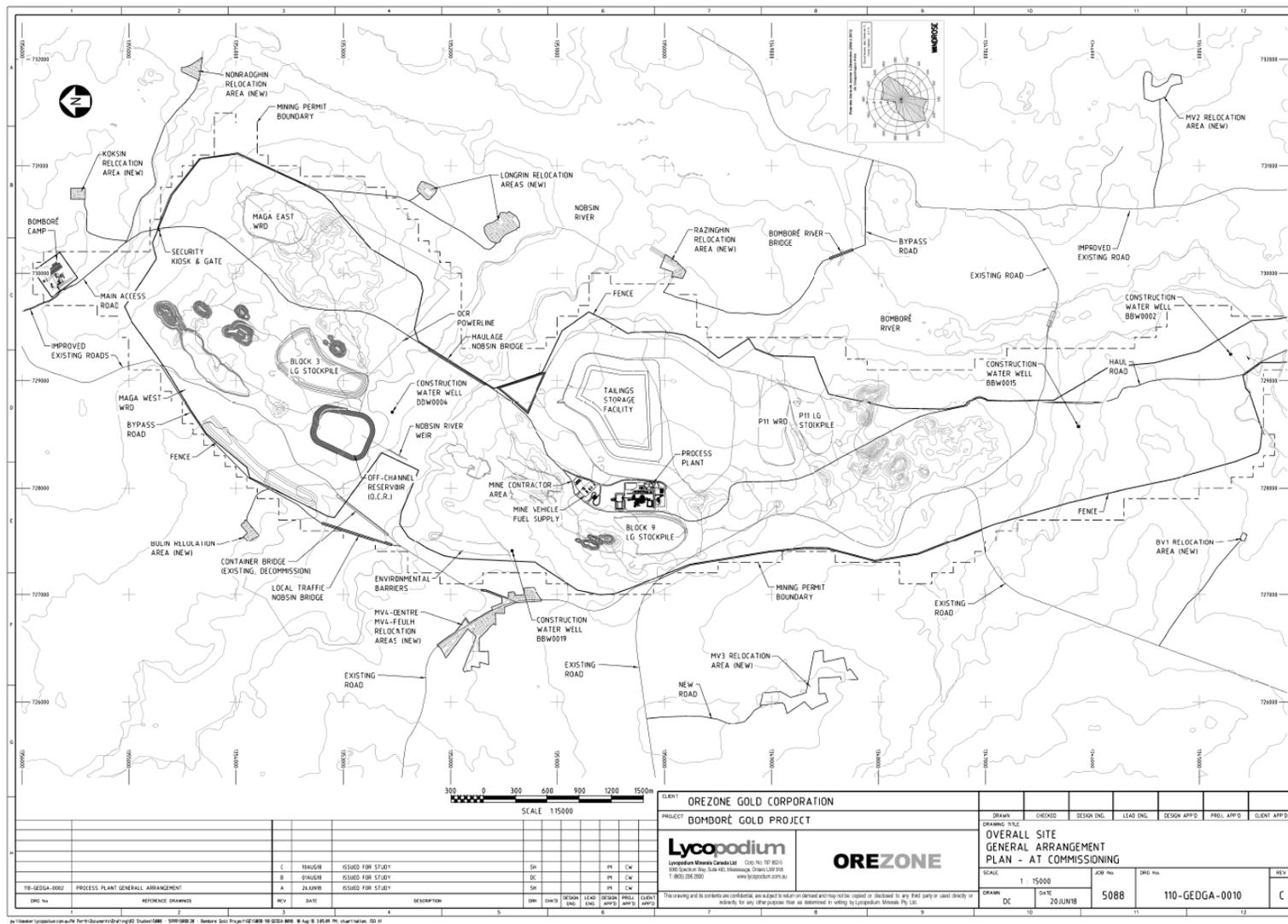
**Figure 1.6        Site Layout End of Mine Life**
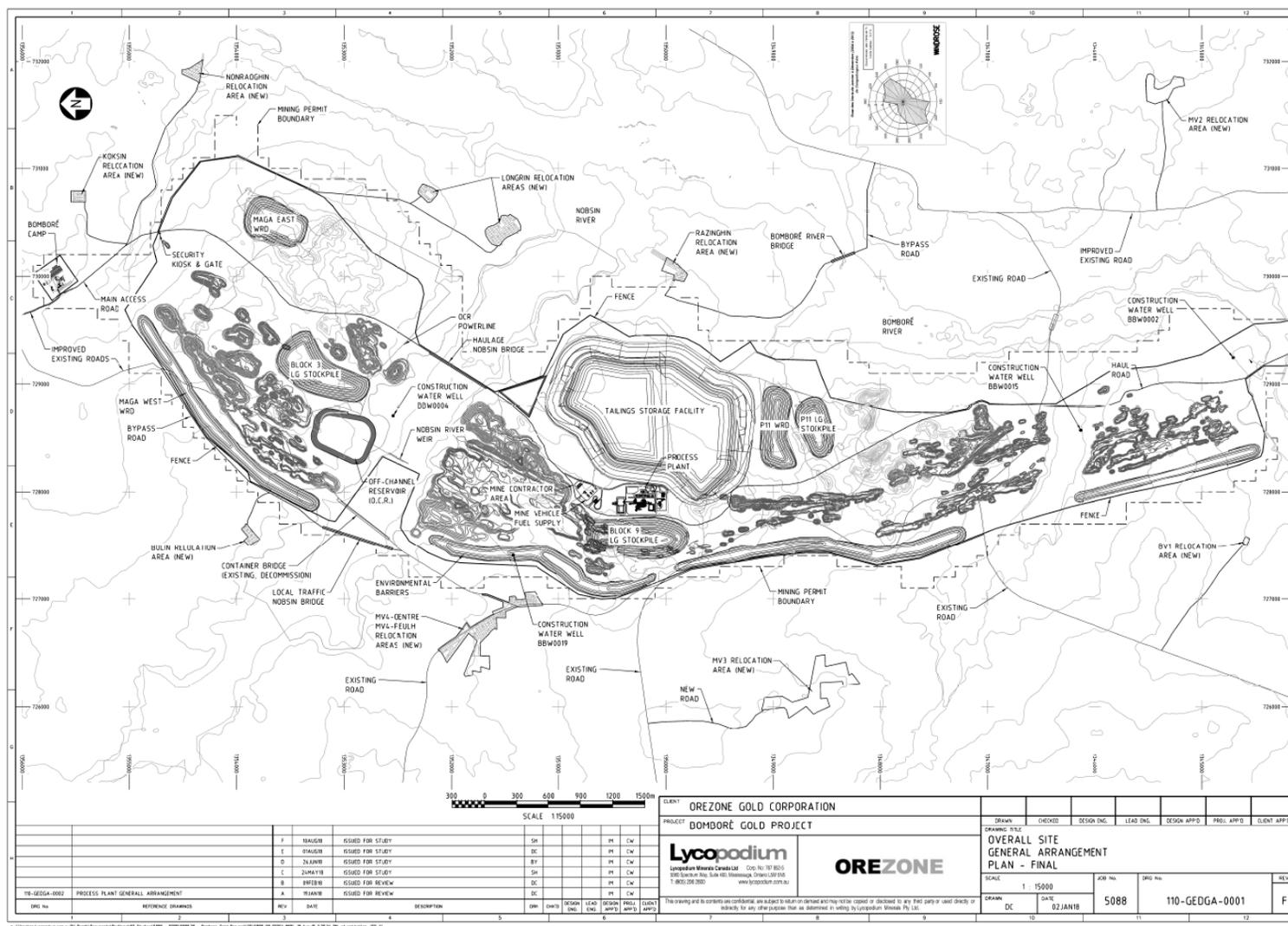
The existing camp at the northern end of the Project area provides accommodation and office facilities for the exploration teams and Orezone support personnel currently at site and will be expanded to accommodate owners and EPCM engineers staff during construction and non-local staff during operations. 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 be required to make their own accommodation arrangements with local businesses.  Contractors will also be required to make arrangements for bussing their employees to and from site 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 using heavy fuel oil (HFO) and operating under a 'build, own, operate' (BOO) contract arrangement with an independent power provider (IPP).  Expressions of interest with budget pricing have been received from three IPP companies all experienced in providing similar services.  The average electrical load on site is estimated to be 5.45 MW with a peak demand of 7.45 MW.  Annual energy consumption is estimated to be 46.0 GWh per year.

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 small 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) conducted from 2014-2015, has 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 are required.

The application for an Industrial Operating permit requires an ESIA that must first be accepted by the Ministry of the Environment and Durable Development (MEDD). The ESIA must be supported by a FS and include a Resettlement Action Plan (RAP) that has been accepted by all stakeholders. Once in production, the holder of an Industrial Operating Permit is required to open, under his name, a fiduciary account called "*Fonds de préservation et de réhabilitation de l'environnement minier*" at the *Banque Centrale des 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 presented in the ESIA, divided by the number of years of production to cover the costs of mine reclamation, closure and rehabilitation.

In 2016, Orezone received the Industrial Operating Permit following the delivery and acceptance by the authorities of the ESIA and RAP, both supported by the 2015 FS.

### 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 as well as all the operational aspects.  The ESMP covers all project phases and allows avoiding, minimizing, enhancing, or compensating the various anticipated impacts by reducing them to an acceptable level for all stakeholders.

The ESMP identifies the objectives needed 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 requirements, providing the basis for assessing the effectiveness of management measures to be implemented by Orezone.  The ESMP includes many measures to strengthen the capacity of the stakeholders impacted by the application of environmental and social management measures.

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

- Control 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.

### 1.11.2    Resettlement Action Plan (RAP)

The resettlement of local people (about 655 households or about 3,915 people) from seven traditional villages, as well as two artisanal 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 Project 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 about 410 households from traditional villages and about 915 households from the Sanam Yaar artisanal gold processing site.  The subsequent resettlement (Stage 2) of about 250 farming households and 450 households from the Kagtanga artisanal gold processing site, all from the southern area of the Project, will occur after the initial Stage 1 resettlement as this area will not be immediately affected by the mine construction.

### 1.11.3    Closure and Reclamation

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 without compromising the integrity and security of places and people.

- Site rehabilitation and re-vegetation.

- Restoration of livelihood conditions for neighbouring populations and workers.

The estimated cost of closure, decommissioning and reclamation for the Project is US$14.5M and this is included in the financial analysis.  Access roads, power lines and other infrastructures built for the Project will be left in place 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.

## 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 1.5 exclude process operating costs associated with plant operations in the last quarter of 2020 (4Q 2020) prior to achieving commercial production on January 1, 2021. The table also excludes the value of gold produced in 4Q 2020 and costs such as bullion transport and refining costs and government royalties associated with this gold production and sales.

**Table 1.5          Project Capital Costs to January 1 2021 (US$, 2Q2018, ±15%)**

| Project Capital Area | US$ M |
|---|---|
| Construction In-directs | 13.2 |
| Process Plant | 34.7 |
| Reagents & Plant Services | 10.6 |
| Infrastructure | 16.2 |
| Management Costs (EPCM) | 11.6 |
| RAP | 24.3 |
| Owner's Costs[1] | 22.4 |
| **Subtotal** | **133.0** |
| Contingency | 10.6 |
| Sub-total | **143.6** |
| Mine costs (2019/2020) | 37.5 |
| Total | **181.1** |

[1]includes $1.0M in opening stock of consumables reclassified to working capital in the economic analysis in Section 22.

Exclusions include the following:

- Project sunk costs.

- Import duties and taxes on the basis that the Project will be exempt.

- Escalation.

### 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 1.6.

Table 1.6          Life of Mine Operating Costs per Tonne and per Gold Ounce (US$, 2Q2018 +/-15%)

|  | Total Cost ($M) | $/Tonne Processed | $/Ounce Au |
|---|---|---|---|
| Mining | 257.0 | 4.59 | 251 |
| Processing | 275.8 | 4.92 | 269 |
| G&A | 93.6 | 1.67 | 92 |
| Refining & Bullion Transport | 1.5 | 0.03 | 2 |
| Government Royalties & Dev Tax | 65.2 | 1.17 | 64 |
| **Total Cash Cost** | **693.1** | **12.38** | **677** |

## 1.13    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. The model was structured using an EXCEL workbook.  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 $1,275/oz are summarized in Tables 1.7, 1.8 and 1.9 below.

**Table 1.7          Production Summary**

|  | Value |
|---|---|
| Ore processed | 56.0 Mt |
| Total tonnes mined | 149.8 Mt |
| Average head grade | 0.64 g Au/t |
| Contained gold in ore | 1.1 Moz |
| Total gold produced | 1.0 Moz |
| Average gold recovery | 89.1% |
| Production life (processing) | 13 years |
| Nominal annual processing rate | 4.5 Mtpa |

**Table 1.8          Net Profit after Tax Summary (LOM Summary)**

|  | $M | $/Ore Processed | $/oz Au |
|---|---|---|---|
| Revenue (99.93% payable) | $1,305 | $23.31 | $1274 |
| Mine Operating Cost | $257.0 | $4.59 | $251 |
| Processing Cost | $275.8 | $4.92 | $269 |
| G&A Cost | $93.6 | $1.67 | $91 |
| Refining & Transport Costs | $1.5 | $0.03 | $2 |
| Government Royalties | $65.2 | $1.17 | $64 |
| **Total Cash Cost** | **$693.1** | **$12.38** | **$677** |
| **EBITDA** | **$611.9** | **$10.93** | **$597** |
| Initial Capital | $143.8 | $2.57 | $140 |
| Sustaining Capital | $58.9 | $1.05 | $58 |
| Rehabilitation & Closure (net of salvage) | $12.2 | $0.22 | $12 |
| **Gross Profit before tax** | **$396.9** | **$7.09** | **$388** |
| Corporate Tax Payable | $104.4 | $1.86 | $102 |
| **Net Profit after tax** | **$292.6** | **$5.22** | **$286** |

**Table 1.9          Financial Summary**

| | Value |
|---|---|
| Revenue from gold (99.93% payable) | $1,305M |
| Adjusted Operating Costs (AOC) | $677/oz Au |
| Initial Capital | $143.8M |
| Sustaining capital | $58.9M |
| Closure costs/salvage | $12.2M |
| Pre-tax economics: | |
|    IRR | 59.3% |
|    NPV (5%) | $315M |
|    Payback | 1.3 years |
| After-tax economics: | |
|    IRR | 42.4% |
|    NPV (5%) | $223M |
|    Payback | 1.7 years |

### 1.13.1    Project Upfront Capital Costs

The Total Upfront Costs is reproduced in Table 1.10.

**Table 1.10          Total Upfront Costs**

| | $ M |
|---|---|
| Process Plant | $45.3 |
| Infrastructure | $16.2 |
| Mining (Haul Roads & Pit Dewatering) | $1.2 |
| Construction In-directs | $13.2 |
| Management Costs (EPCM) | $11.6 |
| RAP | $24.3 |
| Owner's Costs (less Opening Stocks) | $21.5 |
| **Subtotal** | **$133.3** |
| Contingency | $10.5 |
| **Total Initial Construction Costs** | **$143.8** |
| Working Capital | $33.7 |
| Pre-production Operating Costs | $8.5 |
| Pre-production Gold Sales | -$31.2 |
| **Total Upfront Costs** | **$154.8** |

The Total Upfront Costs represent the project capital estimate plus capitalized costs incurred to achieve commercial production (on January 1 2021) less the value of gold recovered during the pre-commercial production period (October to December 2020 inclusive).

### 1.13.2    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 1.7 and 1.8.

**Figure 1.7        NPV Sensitivity Analysis (Pre-tax)**
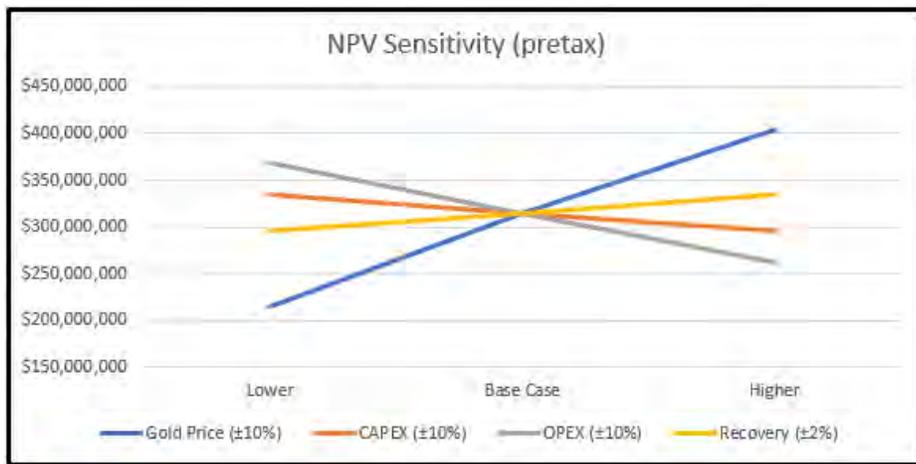


**Figure 1.8        IRR Sensitivity Analysis (Pre-tax)**
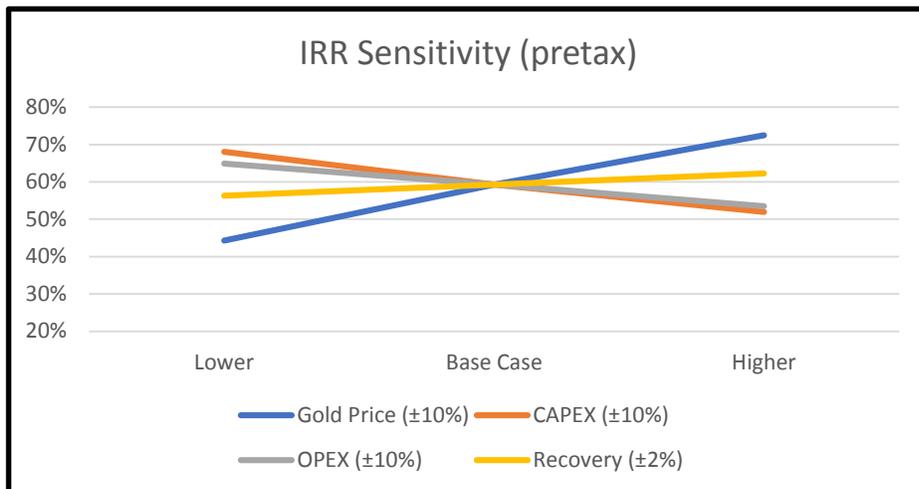
## 1.14    Conclusions and Recommendations

Based on the work undertaken, as summarized in this Technical Report, and the individual Qualified Persons conclusions listed in Section 25, the FS has identified a viable and attractive development opportunity centred around the mining and processing of the oxide and upper transition elements of the mineralization on the Bomboré tenements.

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

The opportunity, as outlined in Section 25, exists for reducing capital costs with a minor redesign of the TSF and potentially further optimizing the processing schedule by adding an additional CIL tank and debottlenecking the process plant design.

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:

- Commence execution of Phase 1 of the RAP.

- Appointment of a lead EPCM Engineer.

- Preliminary site works to prepare for contractor mobilization.

- Development of the FS schedule and budgets into detailed control tools for executing the project.

- Commencement of Front End Engineering and Design across the Project scope.

## 2.0    INTRODUCTION

## 2.1    Terms of Reference and Purpose of this Report

This Technical Report was prepared by Lycopodium for Orezone to summarize the results of a FS of the Bomboré Gold Project. The 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 by paved road, national highway N4 from Ouagadougou.

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

The Project was the subject of a 2013 Mineral Resource estimate update, a 2014 Preliminary Economic Assessment (PEA), a 2015 FS, and a 2017 Mineral Resource update.  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 September 7, 2016 Mineral Resource estimate was subsequently updated, with the results disclosed in a News 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 |
| Neil Lincoln | VP Business Development & Studies | Lycopodium Minerals Canada Ltd | Yes | N/A | P.Eng | 1.1, 1.2, 1.8-1.10, 1.12-1.14, 2, 3.1, 3.3, 4-6, 13, 17, 18 (except 18.12, 18.13), 19, 21-23, 24, 25.4-25.7, 26.4, 26.6, 27. |
| Alan Turner | Principal Mining Engineer | AMC Consultants | Yes | 28 January to 3 February 2018 | CEng | 1.6, 1.7, 3.2, 15, 16, 25.2, 26.2 |
| Jean-Sébastien Houle | Project Director, Geomechanical and Mining Environment | WSP Canada Inc. | Yes | N/A | P.Eng. | 1.11, 20, 25.5, 26.5 |
| Tudorel Ciuculescu | Senior Geologist | Roscoe Postle Associates (RPA) | Yes | 10-13 October 2014 | P.Geo. | 7-12, 14, and co-author of 1.3, 1.4, 14, 25.1, 26.1. |
| José Texidor Carlsson | Senior Geologist | Roscoe Postle Associates (RPA) | Yes | N/A | P.Geo. | Co-author of 1.3, 1.4, 1.5, 14, 25.1, 26.1. |
| Thomas Kerr | Senior Executive Project Engineer | Knight Piésold Consulting | Yes | 29 January to 2 February 2018 | P.Eng. | 18.12, 18.13, 25.3, 26.3 |

## 2.2     Site Visits

Neil Lincoln 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.

Jean-Sebastien Houle did not visit the site.

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, off 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 |
| 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 flourescence |

## 3.0 RELIANCE ON OTHER EXPERTS

### 3.1 Legal Standing of Tenements

Regarding the legal standing of the Orezone Bomboré tenements the authors of this report relied on a letter titled 'Avis juridique relative aux sociétés OREZONE INC SARL; OREZONE BOMBORÉ SA; et aux titres miniers détenus lesdites sociétés' (Legal notice relating to companies Orezone Inc SARL, Orezone Bomboré SA and mining titles held by these companies) written by Barthélemy Kere of KERE, Avocats – Numéro IFU 00005480P of 578, rue Gourma (secteur 28), 01 BP 2173, Ougadougou, 01, Burkina Faso and provided by Orezone.

The letter indicates that the tenements and ownership are in good standing and free of pending court action in Burkina Faso.

### 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.

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

### 3.3 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 by paved road, national highway N4 from Ouagadougou.

**Figure 4.1          Property Location**
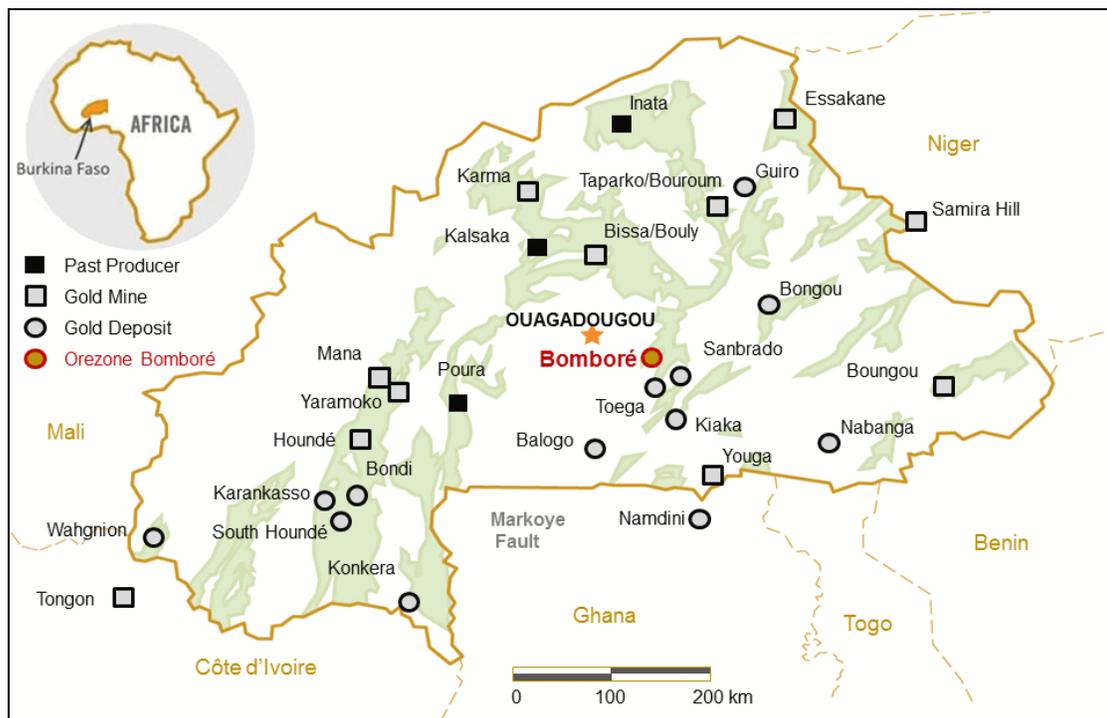


The UTM co-ordinates for the approximate centre of the Property are 1,348,800mN, 728,100mE (Zone 30, Clarke 1880 ellipsoid, Adindan datum). The geographic co-ordinates for the approximate centroid of the currently defined Bomboré gold deposit are 12°12'N Latitude and 0°12'W Longitude.

### 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 the 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**
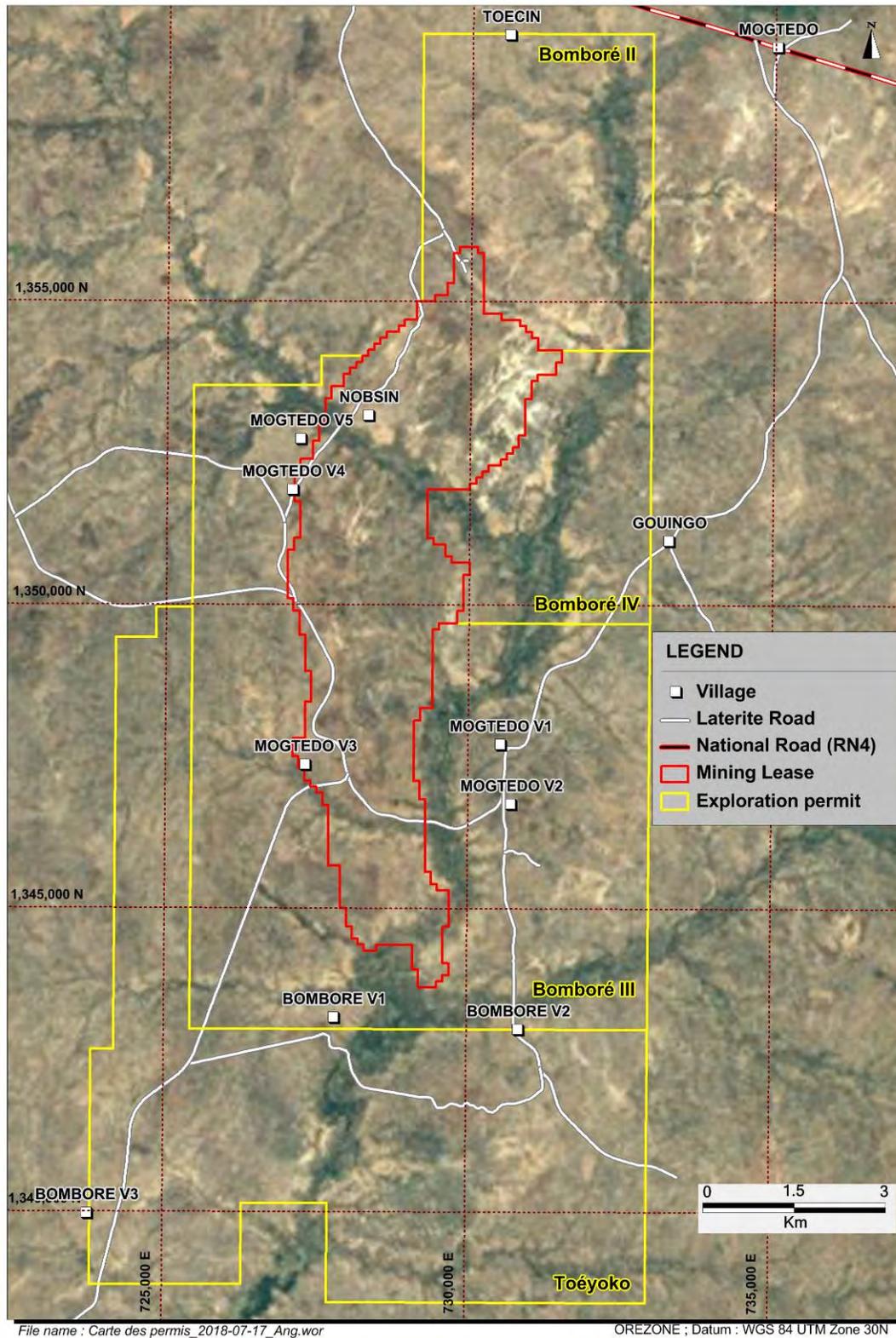
## 4.3    Underlying Agreements

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 Orezone Resources, 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 News 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 business agreement between ORINC and IAMGOLD, 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.

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## 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 by paved road, national highway N4 from Ouagadougou. 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.

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 by 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 a dry and wet season. 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 800mm 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 degrees Celsius in December and January to highs of about 43 degrees Celsius in March and April.   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 are able to mobilize, the resources necessary to construct and support mine development.

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The project site is within a thirty-minute drive from 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 majority of 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 Bomboré site is not serviced by the national power grid. Power will be provided by an on-site power station probably 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).

## 5.4     Physiography

The topography of the property is generally flat with low hills, in the order of 30 to 50 metres 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, like 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 monkeys have been reported.  The south-west corner of the property lies about 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**

## 6.0      HISTORY

Information in this section is updated from the Kappes, Cassiday & Associates (KCA) Technical Report dated April 28, 2015 which was in turn compiled from prior reports issued over the close to thirty-year duration of exploration activities at the site.

## 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 began its 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 Consulting (Canada) Inc. (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 Technical Report also included technical information and economic parameters used in the earlier PEA completed by G Ming Services Inc. (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 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 Section 14 of this Technical Report.

## 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**
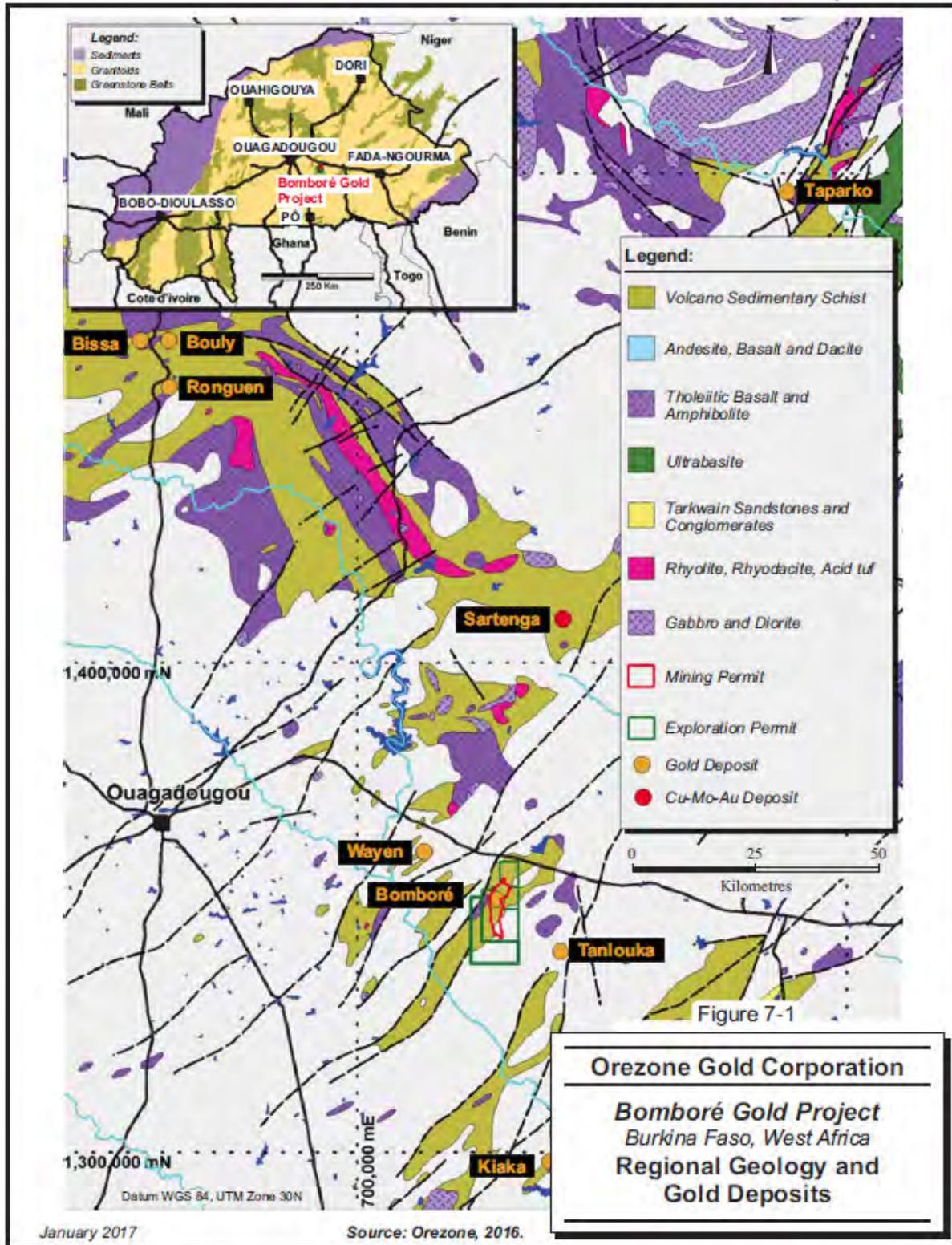
**Figure 7.2        Local Geology**
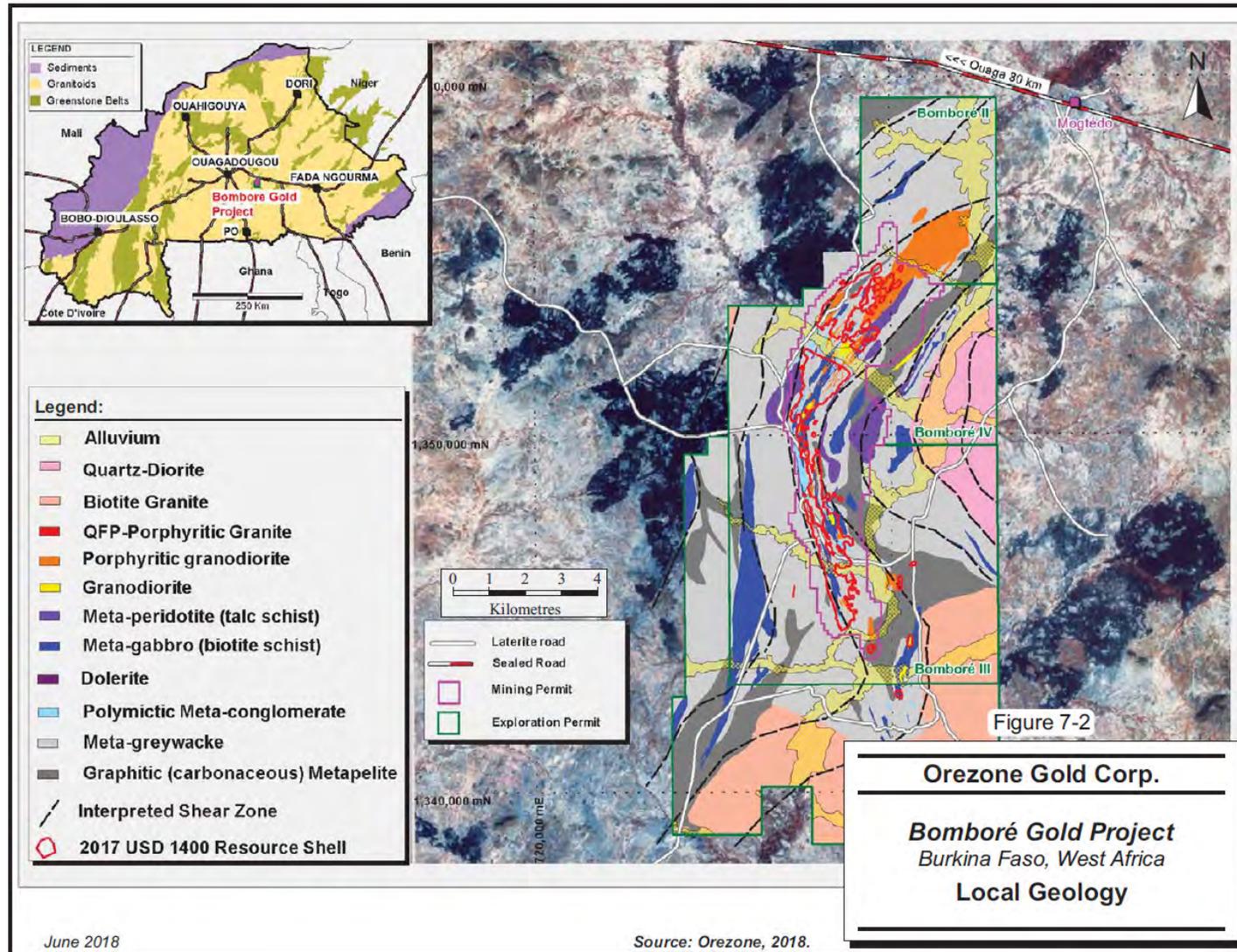
## 7.2     Property Geology

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 Au/t (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**
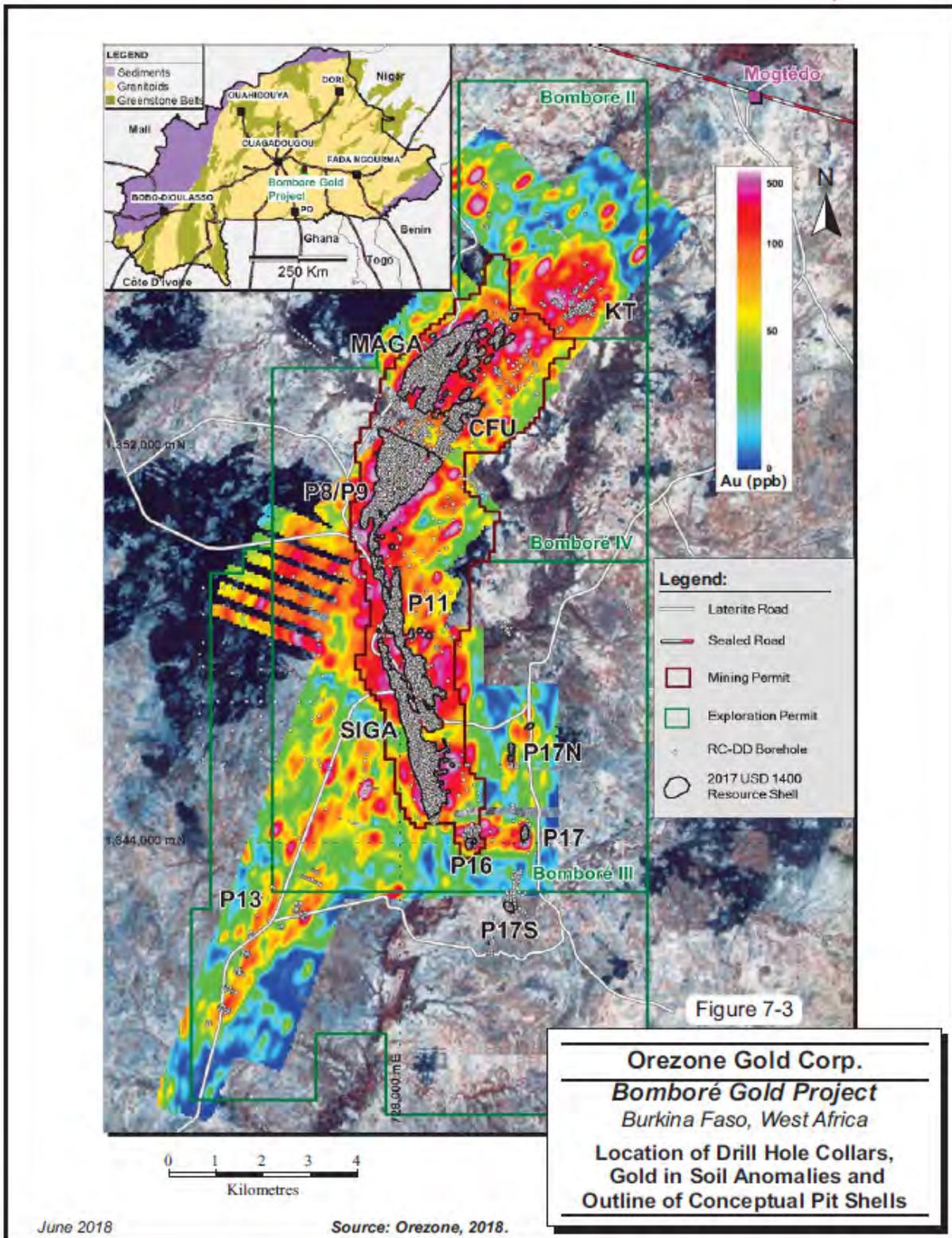
**Figure 7.4**        **Location of Auger and RAB Drill Hole Collars, Gold in Soils Results, and Outline of Conceptual Pit Shells**
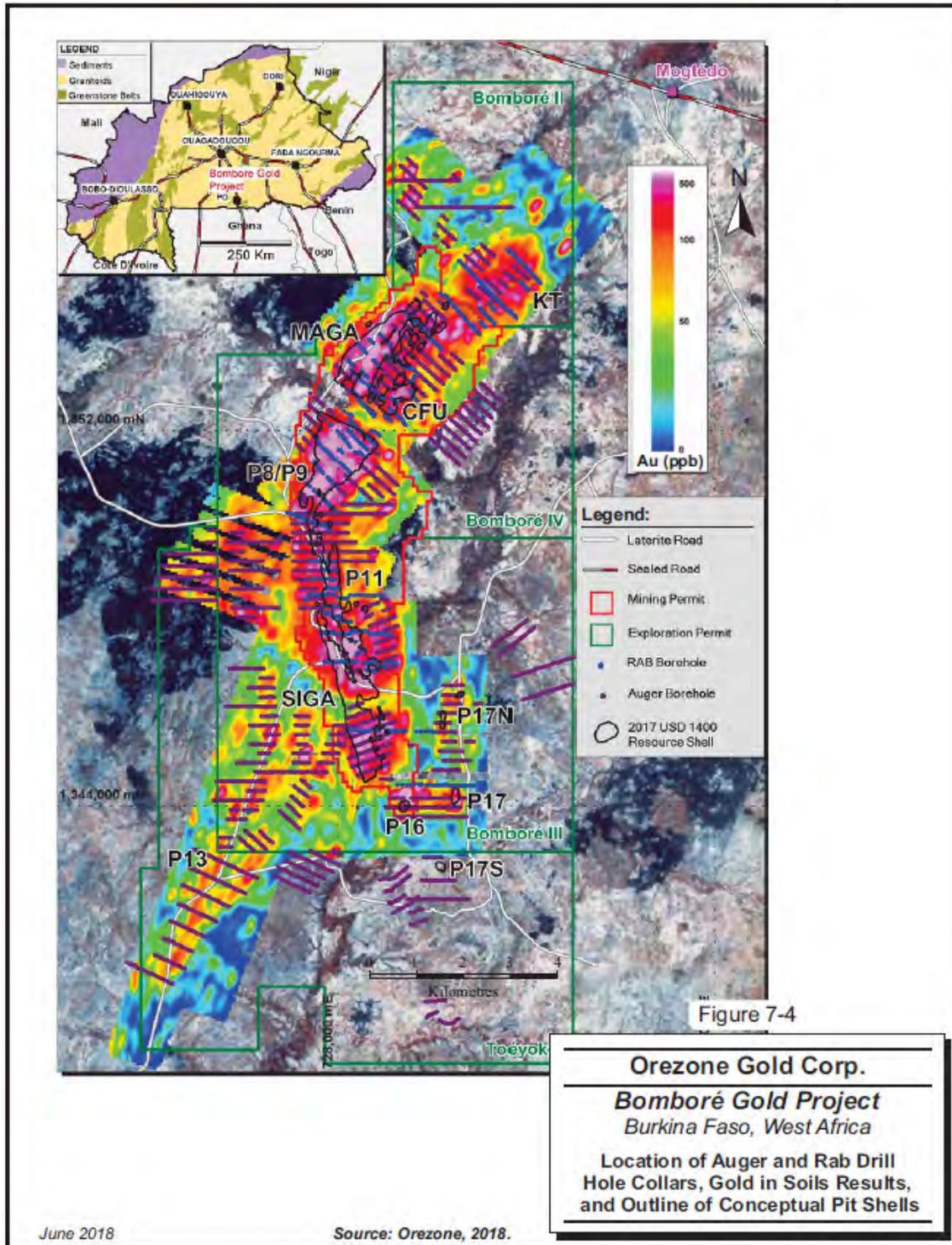
### 7.2.1 Lithologies

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 one kilometre 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**
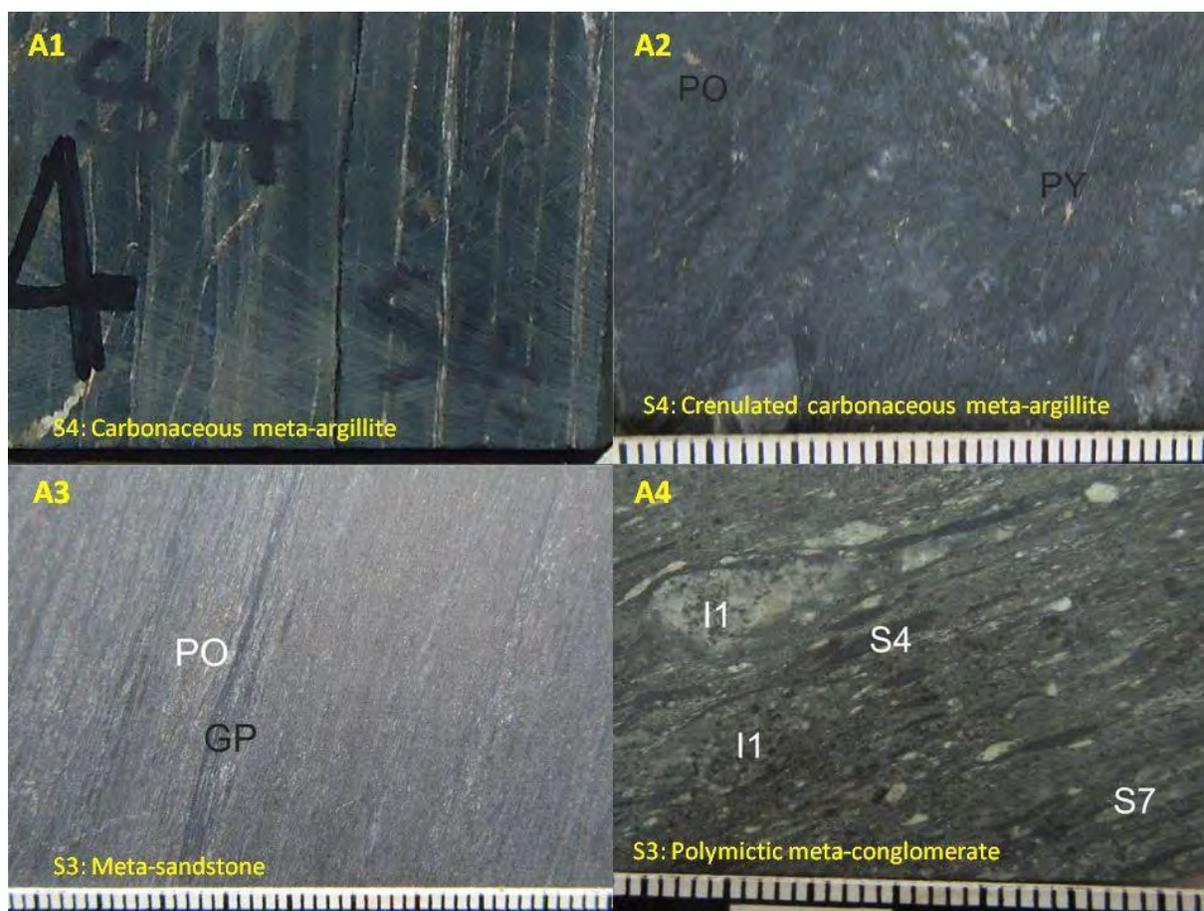


*A1: Carbonaceous meta-argillite.*

*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**
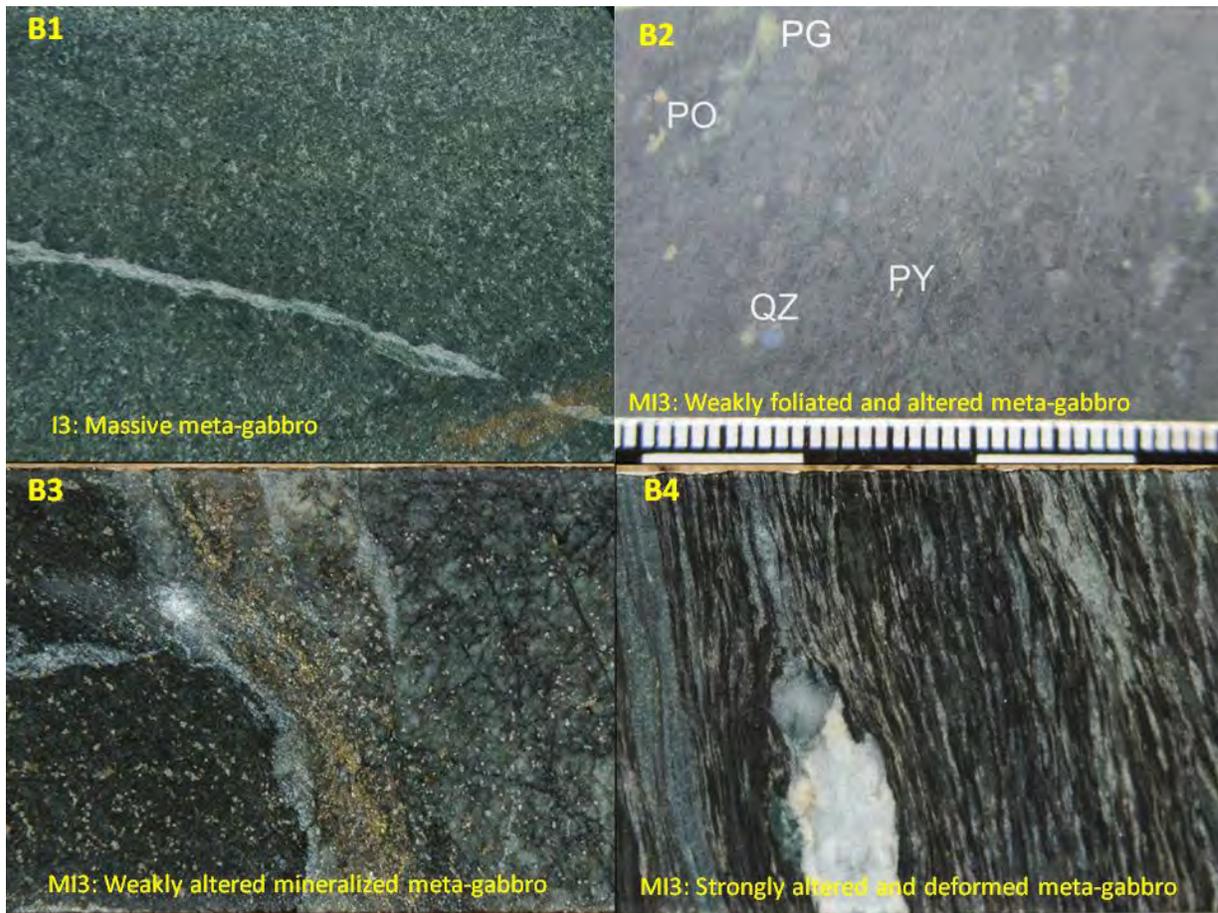


B1: Massive meta-gabbro. P11, BDD0029.

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**
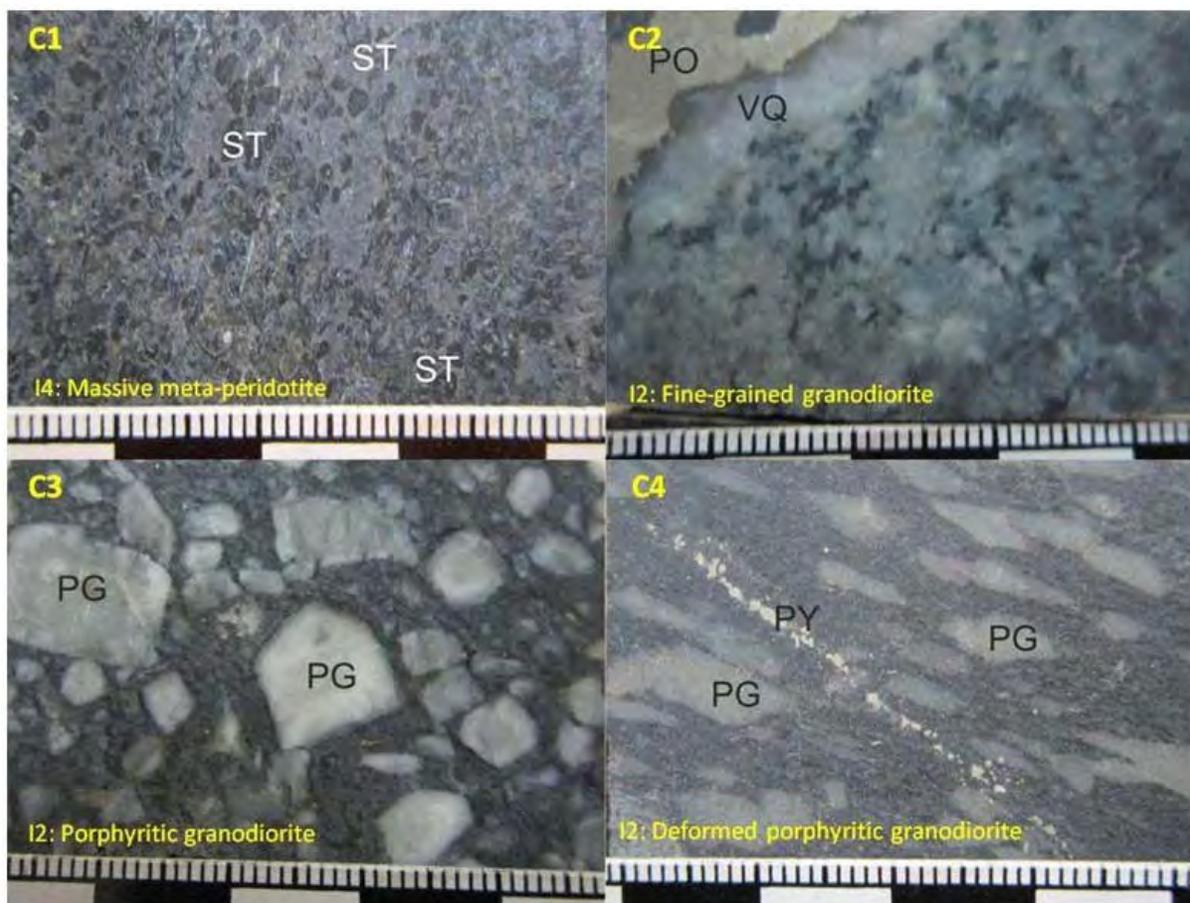


C1: Massive meta-peridotite with serpentine (ST) olivine pseudomorphs. Outcrop AE14, P11 east area.

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**
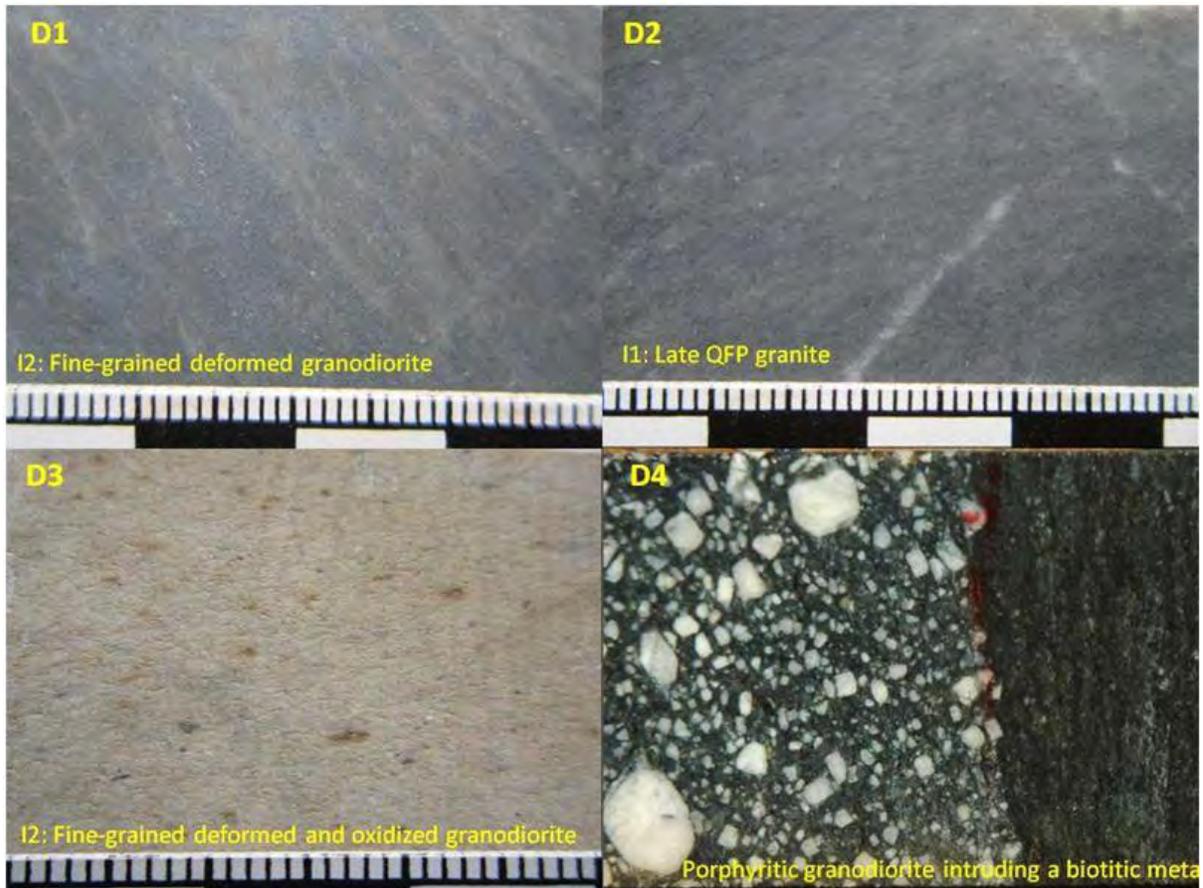


D1: Fine-grained deformed granodiorite. Plagioclase (PG) phenocrysts are altered and deformed. Pyrite (PY) veinlet. Maga, BBD0130, 83.0m.

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**
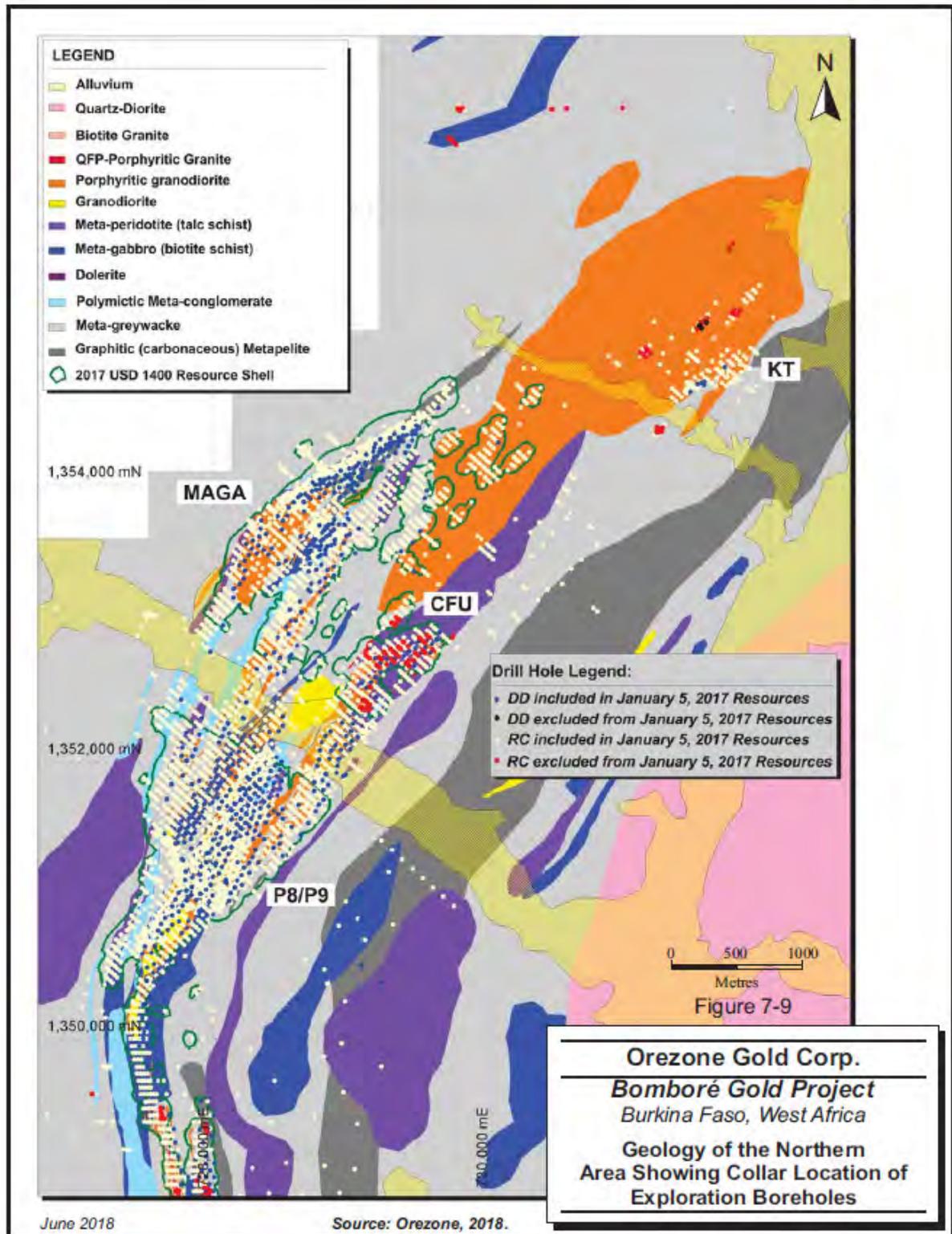
**Figure 7.10**          **Geology of the Southern Area Showing Collar Location of Exploration Boreholes**
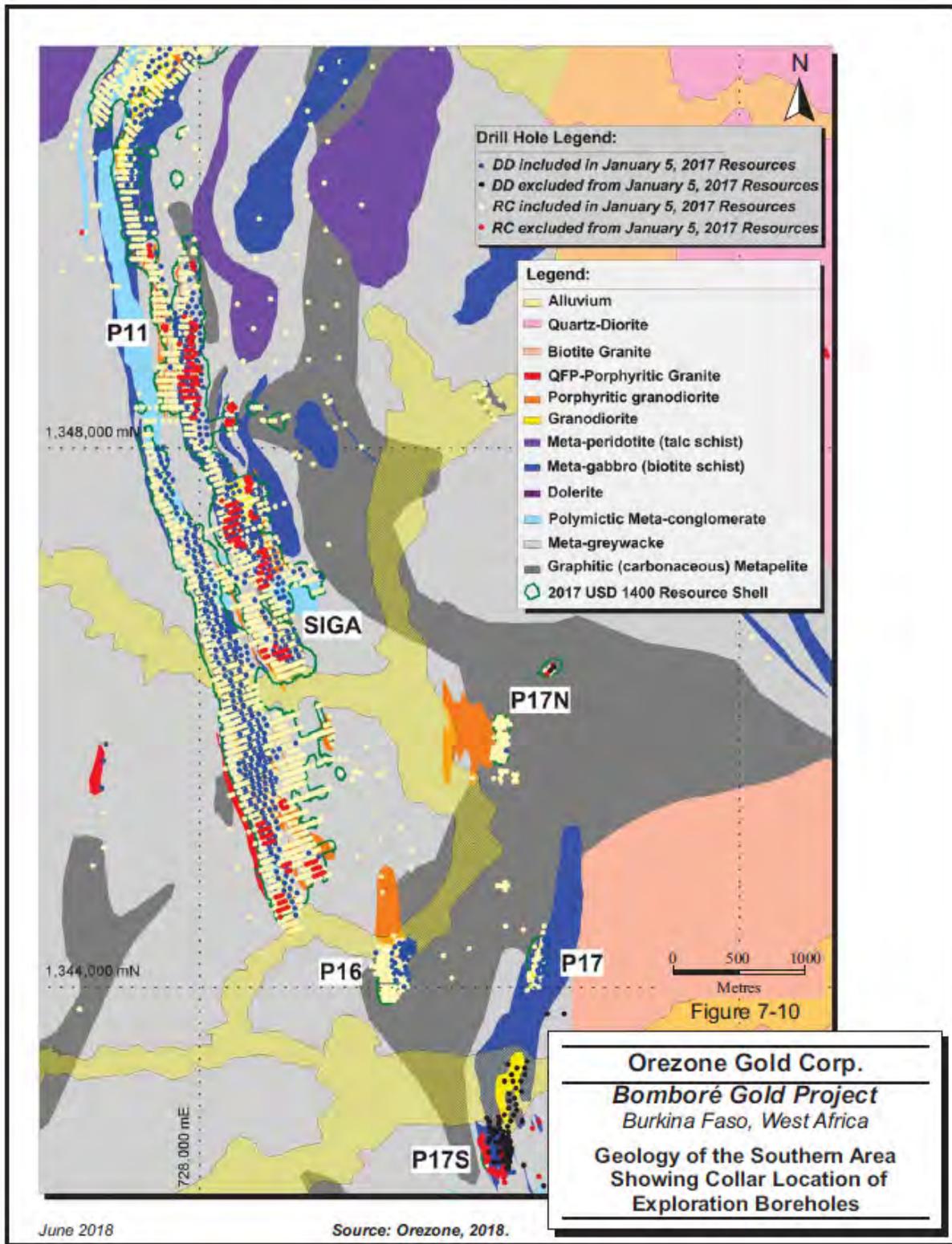
### 7.2.2      Structural Geology

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, 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 (sometimes 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 S0-S1 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 (S0 and S1).

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 south and southeast sectors of the project area.

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**
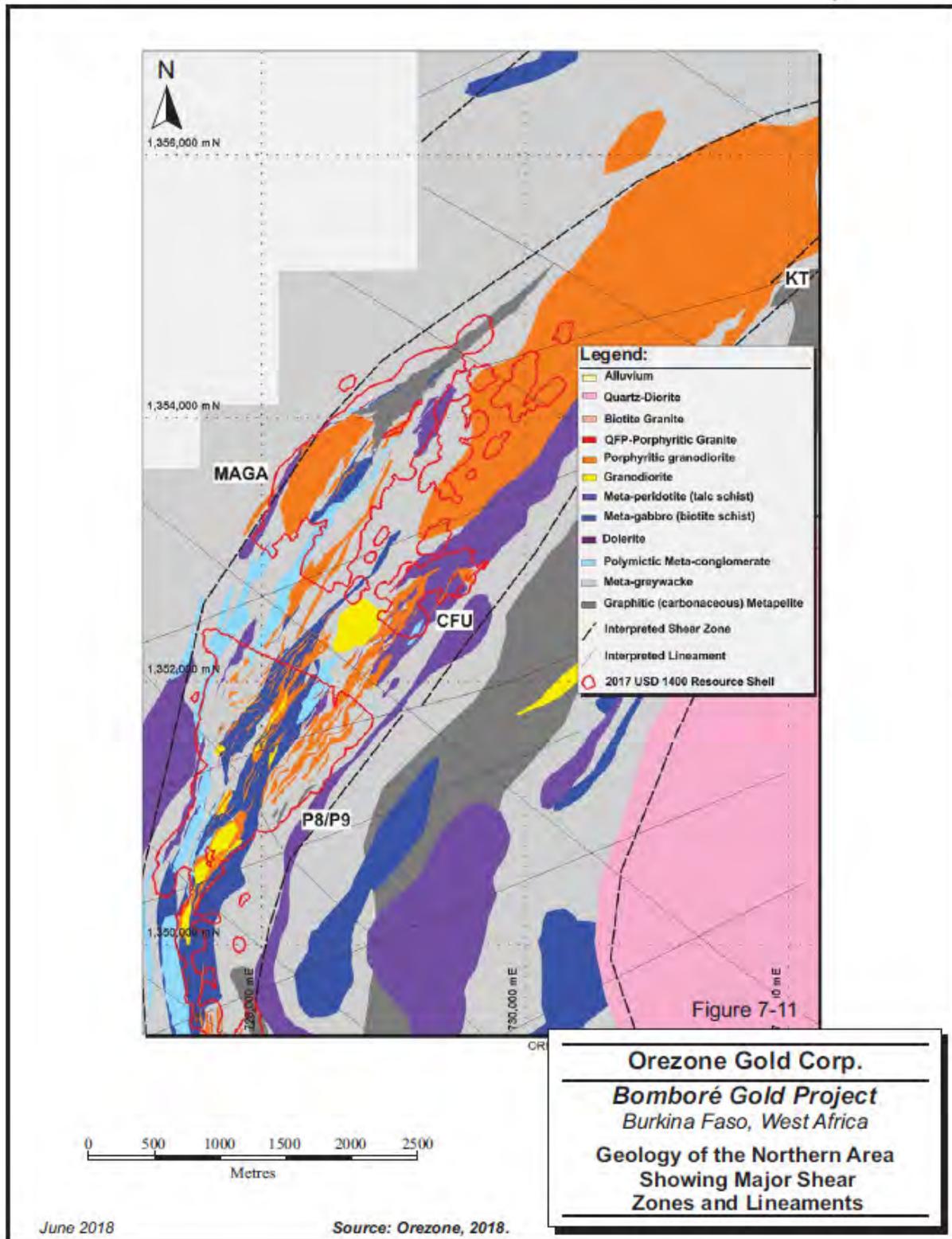
**Figure 7.12**         **Geology of the Southern Area Showing Major Shear Zones and Lineaments**
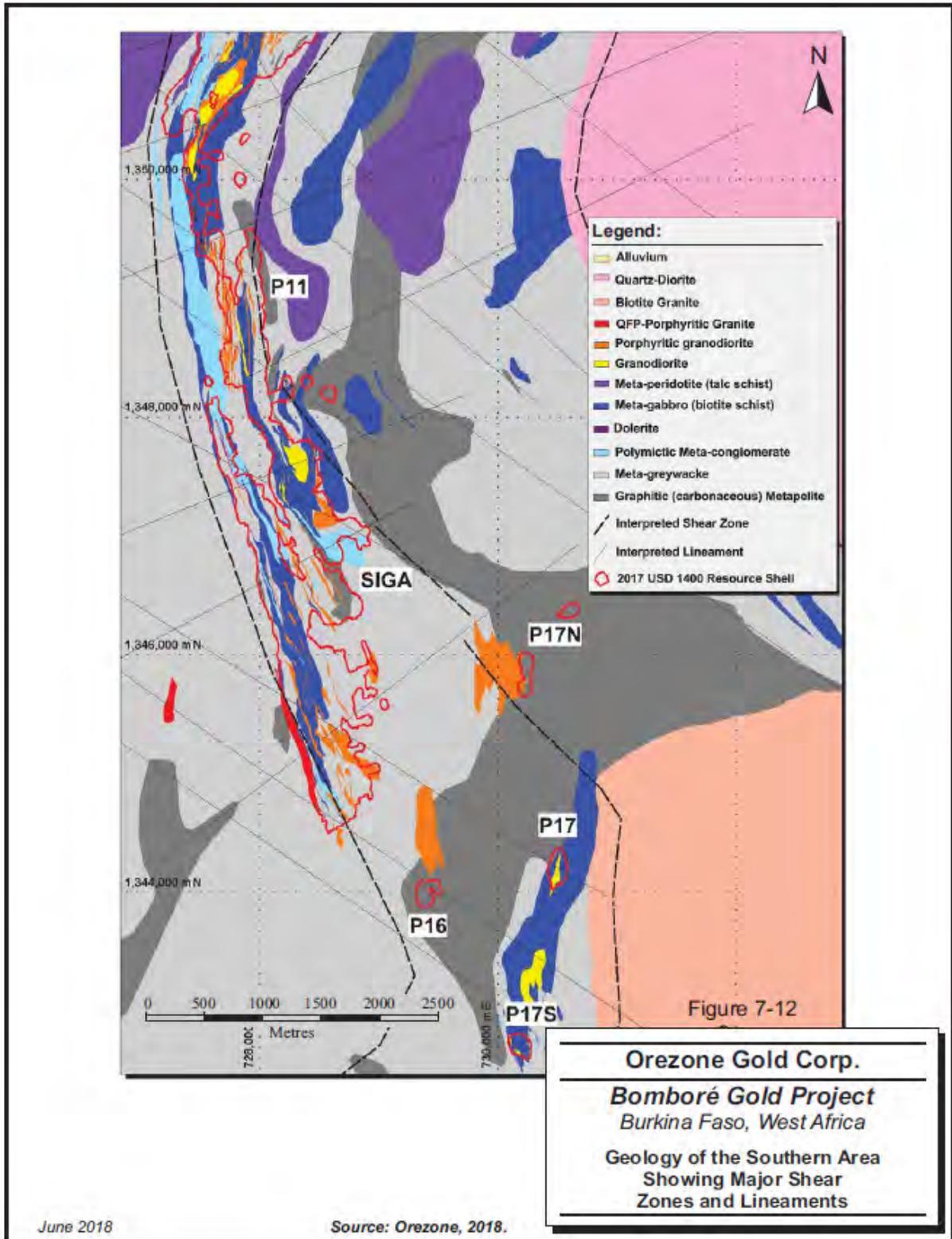
## 7.3    Mineralization

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 arrays of structurally controlled quartz veins and veinlets and attendant silica, sulphide, and carbonate alteration developed within 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 one metre, with an average of ten centimetres.  The near surface gold mineralization with grades of up to 0.2 g Au/t is pervasive regardless of quartz veining and is associated with fine disseminated sulphides, predominantly pyrite. Most of the quartz vein material is barren, as demonstrated by the scrubbing 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 in quartz veins and stringers 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 hosted by various rock types but most commonly in the biotite schist (meta-gabbro) and its host rocks (typically the meta-sandstones but also 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)**
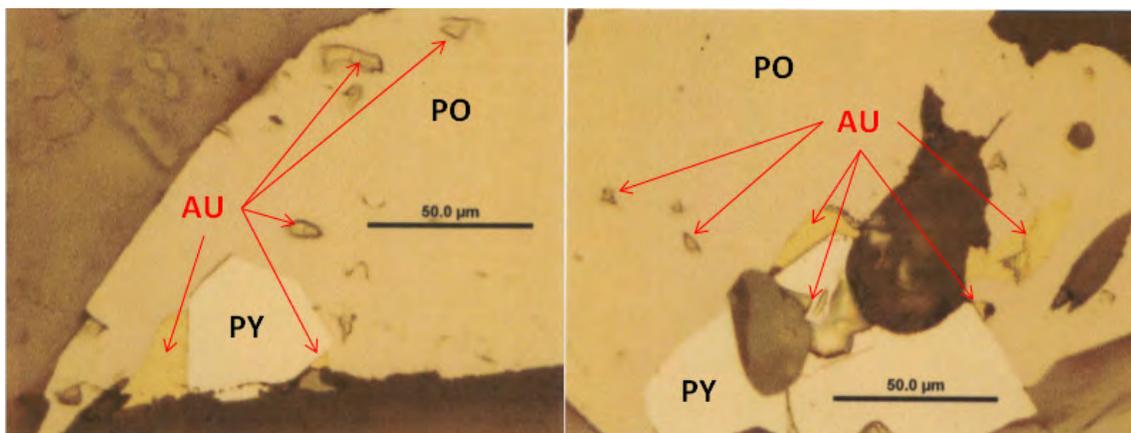
At a cut-off grade of approximately 0.2 g Au/t, the gold mineralization exhibits reasonable continuity over a strike length of approximately ten kilometres.  At this cut-off grade, the gold mineralization forms more restricted corridors (500 m to 1,000 m in length and 10 m to 100 m in width) defining 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 Au/t with the broader low grade domains based on a lower cut-off grade of 0.2 g Au/t.  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 PEA technical report 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 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 paleo-climate 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 program (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 Exploration Permit

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. 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 is included in the summary such as metallurgical testwork, geotechnical drilling, and environmental studies.

**Table 9.1          Summary of Exploration on the Bomboré I Exploration Permit**

| 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.<br>• Report on the 2003 RC drilling program. |
| 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.<br>• Survey of all RC and core boreholes that could be found.<br>• Photogrammetric airborne survey (112 km$^3$).<br>• 121 RC boreholes (8,770 m) at Maga, P8/P9, P11 and Siga.<br>• Ground gradient induced polarization (IP) survey (153.6 km; 100 m line spacing; 25 m stations) at Maga, P8/P9 and CFU.<br>• 1,450 check assays on the 2005 RC samples.<br>• 614 RC composite samples collected for cyanidation metallurgical test work.<br>• 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.<br>• Systematic mapping, prospecting, sampling, and gold assaying of outcrops and gold workings. |

| Period | Exploration Activities and Studies |
|---|---|
|  | • 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.<br>• 268 RC boreholes (19,963 m February to April 2008).<br>• Quality assurance/quality control (QA/QC) report, 2007-2008 RC-core programs.<br>• Cyanidation test work under the supervision of H.C. Osborne and Associates (Commerce City, CO, USA) completed in September 2008.<br>• Petrographic studies by Dr. Schandl (Toronto, ON, Canada) completed between April and September 2008.<br>• 662 multi-element ICP analyses from core samples.<br>• Re-logging of all RC and core boreholes to reconcile the surface mapping, petrography, and ICP data.<br>• SRK Resource Estimate based on RC and core data up to August 2008 – Initiated in June 2008, delivered in November 2008.<br>• Compilation of all historical RC borehole detailed journals to create a penetration rate model.<br>• 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.<br>• 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.<br>• High-resolution (50 m 10 m) resistivity surveys (237 km) over the Maga, P8/P9, P11, and Siga areas in March-April 2009.<br>• Core drilling program (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.<br>• Commissioning of bench top rotary sample dividers for all pulverized samples.<br>• Validation of all historical core specific gravity determinations, including new more closely spaced determinations.<br>• QA/QC reports on the April-June 2009 core drilling program and on various check assay programs.<br>• 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.<br>• Petrographic and structural study of the Bomboré gold deposits; Ph.D. progress report.<br>• Environmental Baseline Study (April to July 2009) by Bureau d'Études des Géosciences, des Énergies et de l'Environnement (BEGE) – report July 2009.<br>• Preliminary Environmental Impact Study (EIA) by BEGE (April to September 2009) – Report September 2009.<br>• Bench-top XRF program commenced in April 2009 – All available historical core pulp samples (> 10,000) were analyzed for a suite of 35 elements. |

| Period | Exploration Activities and Studies |
|---|---|
|  | • Core drilling program (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 programs 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 program. |
|  | • 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 program on SGS Tarkwa and Abilab Ouagadougou RC samples (n = 354) with a soluble gold grade > 1 g/t. |
|  | • 389-borehole auger drilling program 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 program (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 all historical collar positions. |
|  | • Several programs 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 program and various check assay programs. |
|  | • 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 program 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 program 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 program 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 program was completed in June 2012 and totalled 2,375 boreholes and 135,167 m. |
| 2011 | • 2,547 borehole auger drilling program 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. |
|  | • 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. |

| Period | Exploration Activities and Studies |
|---|---|
| | • 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 program 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 program 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 |
| | • Core drilling definition program 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 program was completed in February 2013 and totalled 121 boreholes for 23,109.5 m. |
| | • QA/QC report, RC, core, and auger programs completed from November 2010 to June 2012. |
| | • QA/QC report, RC, core, and check assay programs completed from June 2012 to September 2012. |

| Period | Exploration Activities and Studies |
|---|---|
| | • RC drilling program 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 program was completed in April 2013 and totalled 541 boreholes and 32,440 m.<br><br>• 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.<br><br>• 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.<br><br>• 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.<br><br>• Completion of the preliminary scrubbing test work completed by Orezone under the direction of GMSI, targeting the saprolite gold resources.<br><br>• QA/QC report, RC, core and check assay programs completed from October 2012 to December 2012.<br><br>• 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.<br><br>• 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 program for a scrubber test work program with Met-Solve Laboratories Inc. (Met-Solve) from Langley, BC, Canada. Report delivered in May 2013.<br><br>• Delivery by SOCREGE of an interim socio-economic study relevant to the 2011 CIL Definitive Feasibility Study (DFS) project.<br><br>• Final report of the study of the archaeological artifacts collected by BEGE.<br><br>• Final report from McClelland on the CIL DFS process metallurgical study.<br><br>• Initiation in March 2013 of a metallurgical study on pyrrhotitic samples with COREM from Quebec City, QC, Canada. Report delivered in May 2013.<br><br>• 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.<br><br>• 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.<br><br>• Interim report from BEGE on complementary botanical and archaeological studies completed for the CIL DFS.<br><br>• 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.<br><br>• QA/QC report, RC, core and check assay programs completed from November 2012 to June 2013.<br><br>• Preliminary PEA HL facility design delivered by Golder in November 2014.<br><br>• 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.<br><br>• Decision to proceed with an HL DFS in January 2014. |

| Period | Exploration Activities and Studies |
|---|---|
| | • 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 program. 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 program 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 program, 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 program (1,114 m) in May 2014 in the CFU and P17S areas. |
| | • RC drilling program (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 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 programs completed from July 2013 to August 2014. |

| Period | Exploration Activities and Studies |
|---|---|
|  | • 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 program 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 program, 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 programs 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 program 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 program. |
| 2017 | • A metallurgical test work program 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 programs totalling 462 boreholes and 1,648 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 program 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 programme 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 program totalling 144 boreholes and 528 m and a small RC drilling program 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. |
|  | • A small auger drilling program 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 program 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. |

| Period | Exploration Activities and Studies |
|---|---|
|  | • A RC drilling program 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. |
| 2018 | • A small drill program 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. <br><br> • A drill program completed from January to March 2018 on the Bomboré IV 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. <br><br> • A drill program completed from January to March 2018 on the Bomboré IV 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. |

## 10.0    DRILLING

## 10.1    Drilling Programs

The following is paraphrased from the French version of the 2015 FS prepared by Orezone and appended to the Orezone Bomboré Operating Mining permit application submitted to the MEMC in May 2015. Subsequent to that report, Orezone undertook several drill programs starting in late 2016 to the ongoing program in March 2018.

Orezone undertook core, RC, and auger drilling on the Property from 2003 to 2016 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 Orezone Drilling to March 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 | 55 | 2,904.00 |

| Company | Years | Drilling Type | Number of Holes Drilled | Length (m) |
|---|---|---|---|---|
| Orezone | 2018 | Core | 34 | 4,916.50 |
| Sub-total Channel (Included in January 5, 2017 resources) | | Auger | 0 | 0 |
| | | RAB | 1,000 | 34,249 |
| | | RC | 261 | 19,501 |
| | | Core | 10 | 1,080 |
| Sub-total Orezone (Included in January 5, 2017 resources) | | Auger | 4,221 | 20,058 |
| | | RAB | 0 | 0 |
| | | RC | 4,684 | 283,371 |
| | | Core | 1,026 | 156,069 |
| Sub-total Orezone (excluded from January 5, 2017 resources) | | Auger | 700 | 2,558 |
| | | RAB | 0 | 0 |
| | | RC | 412 | 21,819 |
| | | Core | 124 | 18,454 |
| Total as of March 31, 2018 | | Auger | 4,921 | 22,616 |
| | | RAB | 1,000 | 34,249 |
| | | RC | 5,357 | 324,691 |
| | | Core | 1,160 | 175,603 |

A total of 536 (412+124) RC and diamond core holes were not used to estimate Mineral Resources as their data were received after the effective date of the Mineral Resource. Most of these holes are located outside the resource area. 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**
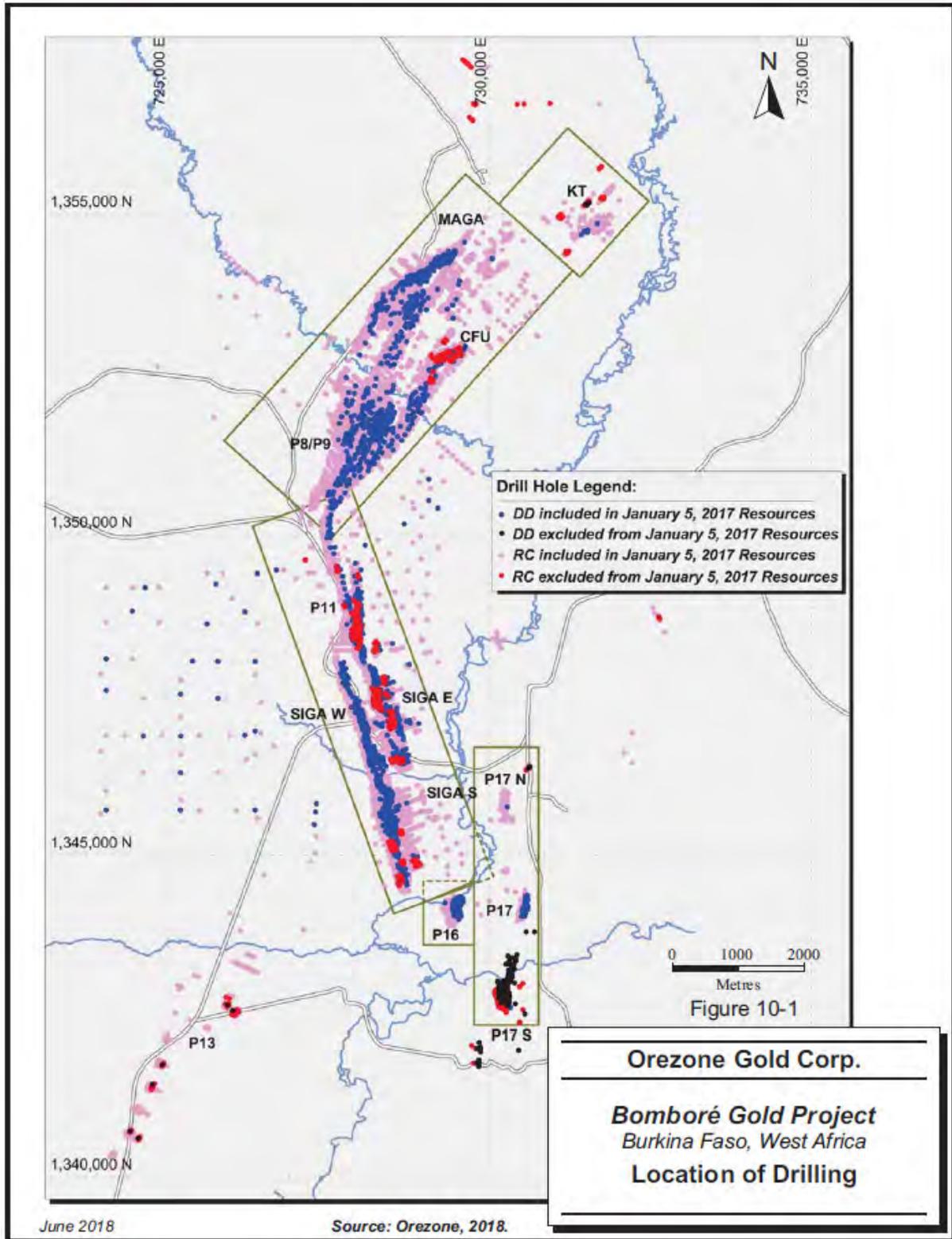
## 10.2    Drilling Procedures

### 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.  The Orezone-owned rig completed 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 22,006 from November 2016 to March 2018.

### 10.2.2    Water Table Elevation and Sample Recovery

The water table is encountered at a depth of approximately 20 m on average 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**
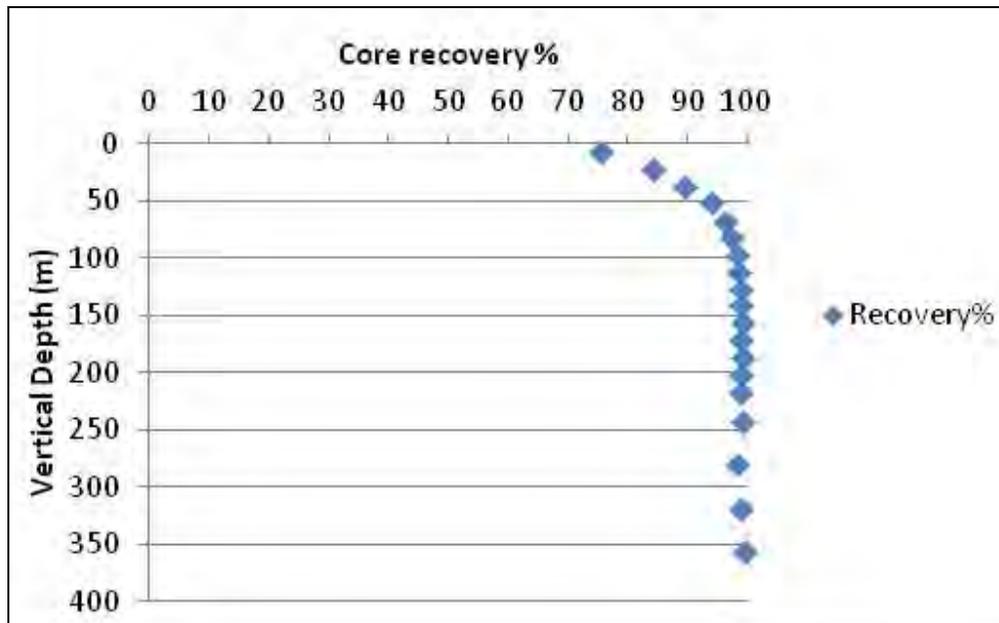


### 10.2.3    Borehole Orientation and Drilling Pattern

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 degrees, 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 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 surveyed using both DGPS and a Total Station instrument. Orezone used contractor La Boussole from Ouagadougou in 2006 for the Total Station surveys but then acquired a Leica Total Station instrument and surveyed the drill collars internally. Orezone acquired a new Trimble M3 Total Station instrument in 2011 and a Trimble R6 DGPS instrument in 2012, which is coupled with the Trimble M3 Total Station instrument and uses 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.

Orezone re-surveyed all drill holes back to 1994 using the Leica Total Station instrument or the DGPS instrument. A series of geodetic stations tied to the national grid system had been set up by La Boussole to build a network of control survey stations over the entire Project area. The planned and final collar coordinates are transferred into the database as "collar" files.

Upon completion, a three metre PVC pipe is inserted in RC holes and a six metre 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. A mock probe is first run down and up the hole to confirm that it is open and safe.

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 one metre intervals by collecting 100% of the material reporting to the cyclone. Small samples of screened and washed chips from each one metre run are saved in labelled plastic boxes (chip boxes). A quick 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 in 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 Au/t) zones are kept. Some sample bags damaged by UV light despite of the protective sheet are also discarded.

**Figure 10.3          Chip Box Photograph**
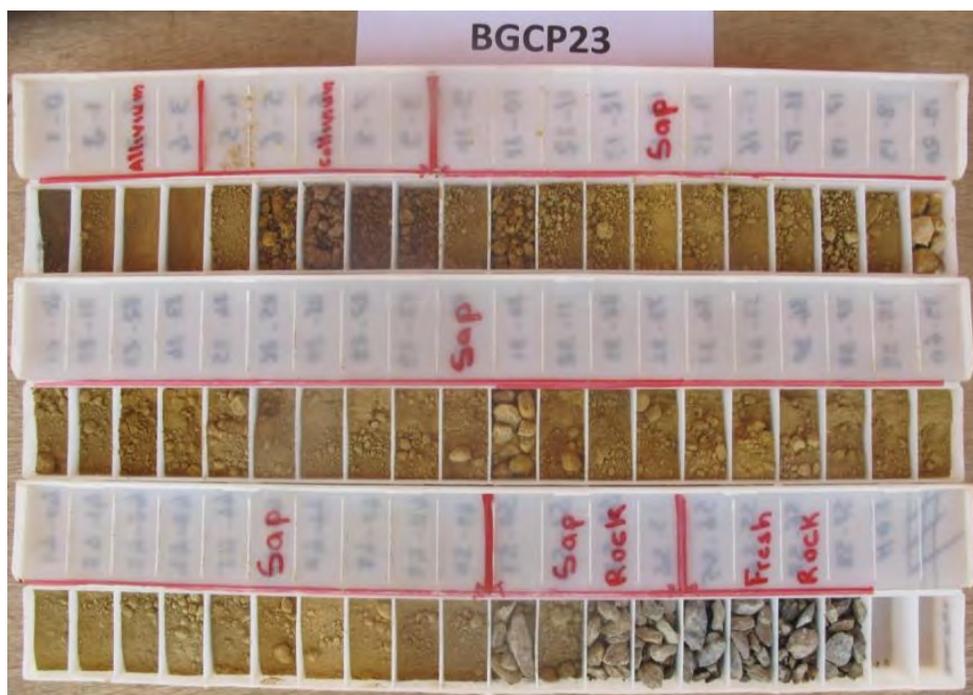
**Figure 10.4          RC Sample Storage at the Bomboré Main Camp**



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.  All drilling sampling was conducted by qualified personnel under the direct supervision of appropriately qualified geologists.

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 summarized from the French version of the 2015 FS prepared by Orezone and appended to the Orezone Bomboré Operating Mining permit application submitted to the MEMC in April 2016; and updated to include the 2017and 2018 drill programs.

## 11.1     Sampling and Analysis by Orezone 2003 to 2015

### 11.1.1    2003 RC Program

For the 2003 RC drilling program, samples were collected at the rig cyclone at one metre 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 done 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 one metre 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. |

| PROTOCOLS |
| --- |

**PROTOCOLS**

- The samples are then packaged in lots of 20 in 100 kg rice bags and sent in large lots to the analytical laboratory.

**QA/QC**

- A second riffle splitter fraction of five kilograms 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**

- 100% of the material reporting to the RC rig cyclone is collected at one metre 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 sample preparation, the weight recorded by the sample preparation laboratory, the weight of each fraction generated

| PROTOCOLS |
| --- |

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 one metre 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 returned to the Orezone Kossodo team.

| PROTOCOLS |
|---|
| • 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*

During the period 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 Au/t) | | | | | 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 Au/t) | | 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 Au/t) | | | | 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 Au/t) | | | | | 0 | 475 |
| RC (definition ≥ 5 g Au/t) | | | | | 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*

During the period 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 84,761 records generated by Orezone from measurements on core from 994 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 recognized that specific gravity increases with depth through the weathering profile and is then fairly homogeneous within the fresh zone. 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 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 form 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**

| | Program | | QA/QC Samples | | | | | Analyses |
|---|---|---|---|---|---|---|---|---|
| Period | (m) | Division | FD | LAPD | PD | BLK | STD | (Method/Weight) |
| 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 | | NC: | Non certified reference material |
| FD: | Field Duplicate, blind to the preparation laboratory | | IHC: | In-house referenced material |
| PD: | Pulp Duplicate, blind to the analytical laboratory | | FA: | Fire Assay, with AAS finish |
| LAPD: | Lab-Aware Pulp Duplicate, known to the analytical laboratory | | BLEG: | Bulk Leach Extractable Gold |
| BLK: | Blank, blind to the preparation laboratory | | 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* | *CM :* | *Certified Reference Material* |
| *CD:* | *Crush Duplicate, blind to the analytical laboratory* | *NC:* | *Non certified reference material* |
| *PD:* | *Pulp Duplicate, blind to the analytical laboratory* | *IHC:* | *In-house reference material* |
| *LAPD:* | *Lab-Aware Pulp Duplicate, known to the analytical laboratory* | *FA:* | *Fire Assay, with AAS finish* |
| *BLK:* | *Blank, blind to the preparation laboratory* | *BLEG:* | *Bulk Leach Extractable Gold* |
| *STD:* | *Standard; blind to the preparation laboratory* | *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 Au/t and 1.5 g Au/t as approximately 97% of the mineralized samples in the Project's assay database display a grade less than or equal to 1.5 g Au/t, with a lower cut-off grade of 0.2 g Au/t 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 (g Au/t) Average | Standard Deviation | Number of Samples Used |
|---|---|---|---|---|---|
| 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 (g Au/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, and 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 computerized IPAQ palm loggers 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 in Ouagadougou 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 are 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.  The site visit included a property tour guided by Pascal Marquis, Orezone SVP Exploration, with stops at drilling sites, artisanal mining sites, and sample handling, processing, and storage facilities.  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 | Hole ID | 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 via the VLOOKUP function.  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 FA (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 2018.  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.

**Table 13.1**                    **Summary of Testwork Programs**

| Program | Head Analysis | Variability | Cyanidation | Gravity | Flotation | Carbon-in-Leach (CIL) | 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 | | | ✓ | | ✓ | | | | ✓ | | | | | | |
| SGS Lakefield 2014 | | | | | | | | ✓ | | | | | | | |
| Kappes 2014 | | ✓ | ✓ | | | ✓ | ✓ | | ✓ | ✓ | | ✓ | ✓ | | |
| SGS Lakefield 2016 | | | ✓ | ✓ | ✓ | | | ✓ | | | ✓ | | | | |
| SGS Lakefield 2017/2018 | | ✓ | ✓ | | | | | ✓ | | | | | | ✓ | |
| Outotec 2018 | | | | | | | | | | | | ✓ | | | |

The following sections summarize selected testwork results that are pertinent to the project. For further details of each program, refer to the individual report.

Figure 13.1                    BGP Metallurgical Sampling Locations
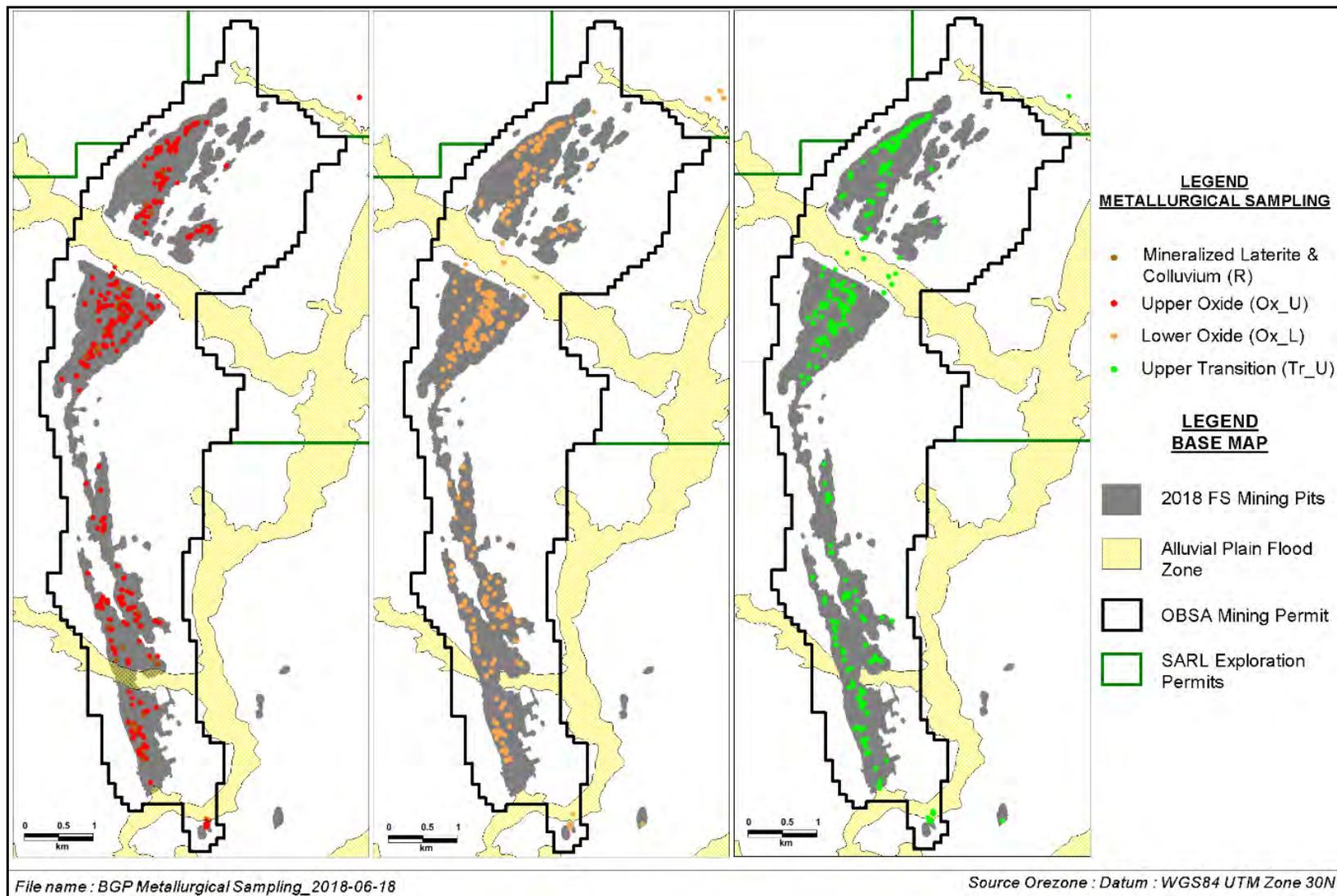
## 13.2    Historical Testwork Programs

### 13.2.1    SGS/ITS 1997

A total of 91 samples from the Siga, Maga and P8/P9 mineralized zones were collected for the 1997 testwork program.  Of these 67 were from RC drills and were used in bottle roll tests at SGS (Ghana). The remaining 24 were from RAB 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)*

- Out of the 67 RC samples, 57 were oxide and transition ore types, and 10 were sulphides.

- Gold extraction from 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 from cyanide leaching ranged from 63% to 95%, with an average of 85%.

*SGS Lakefield Research (Canada)*

- Out 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 0.5µm to 800µm, with ~85% less than 30µm in diameter.

### 13.2.2    Osborne 2008 Testwork Program

A total of 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 FA conducted at Alibab (Bamako).

Descriptions of each testwork stream are as follows:

- Stream 1 – Samples were pulverized to 75µm, subjected to 24 hours bottle roll BLEG, and FA on leach residue at 24 hours.

- Stream 2 – Samples were pulverized to 106µm, subjected to 24 hours 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 75µm, subjected to 72 hours 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 13.2.

**Figure 13.2**                    **Osborne Average Leach Curves**
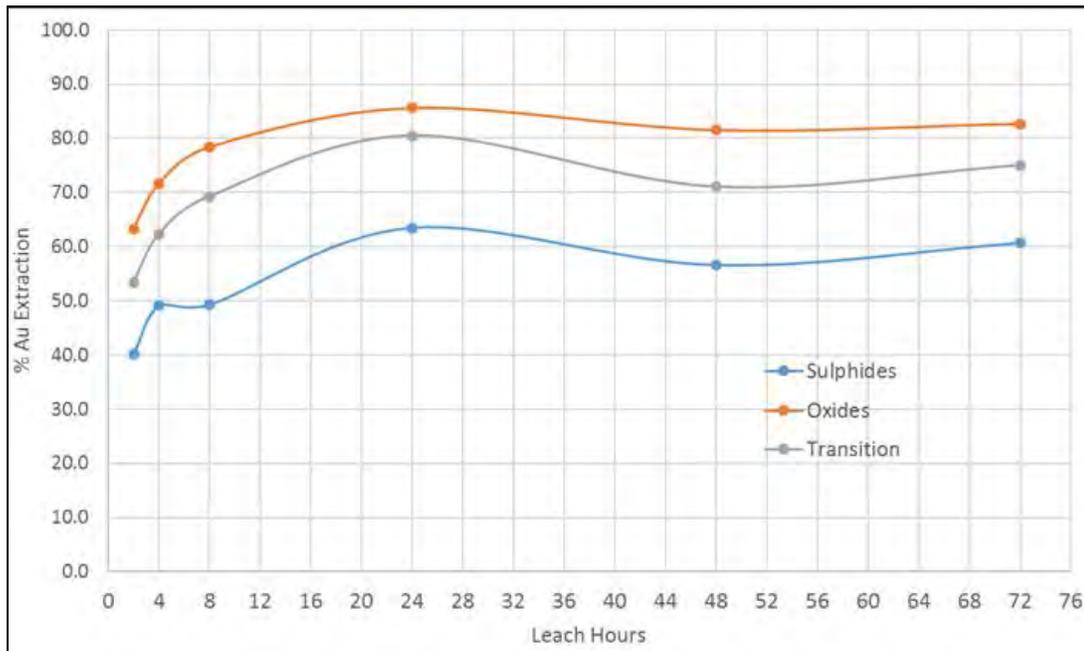
### 13.2.3 AMMTEC 2009 Testwork Program

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 Au/t | 0.90 | 0.90 | 0.62 | 0.59 | 0.61 | 0.65 |
| Head assays (g/t) | | | | | | |
| Au (ppm) | 1.09 | 0.56 | 0.61 | 0.59 | 0.63, 0.74 | 1.52, 0.80 |
| Ag (ppm) | < 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. As for the samples that contained oxide and transition metasediments, organic carbon is present in sufficient amount for Osborne to consider these as preg-robbing potentials 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 the ore is 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 9.26 16.78 3.71 6.04 | Shear | n/a |
| | | 69.47 41.19 74.12 60.96 | Shear | | | |

The results indicated that fresh rock 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 rock 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 rock, 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** | <25mm | <19mm | <12.5mm | <4mm | <25mm | <19mm | <12.5mm | <4mm |
| 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** | 150μm | 106μm | 75μm | 53μm | 150μm | 106μm | 75μm | 53μm |
| 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** | 150μm | 106μm | 75μm | 53μm | 150μm | 106μm | 75μm | 53μm |
| 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 @ $P_{80}$ 53μm** | | | | | | | | |
| 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** | <25mm | <19mm | <12.5mm | <4mm | <25mm | <19mm | <12.5mm | <4mm |
| 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** | 150μm | 106μm | 75μm | 53μm | 150μm | 106μm | 75μm | 53μm |
| 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 @ $P_{80}$ 53μm** | | | | | | | | |
| 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** | <25mm | <19mm | <12.5mm | <4mm | <25mm | <19mm | <12.5mm | <4mm |
| 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** | 150µm | 106µm | 75µm | 53µm | 150µm | 106µm | 75µm | 53µm |
| 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 @ $P_{80}$ 75µm** | | | | | | | | |
| Flotation Gold Extraction (%) | 65.01 | | | | 67.22 | | | |
| Mass Pull (%) | 11.64 | | | | 21.22 | | | |
| **Grav Conc/Grav Tails CIL, All @$P_{80}$ 53µm** | | | | | | | | |
| 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 minus 19 mm crush size would be sufficient for heap leaching transition ore and minus 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 10μm) 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 also conducted but 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 Au/t | 2.38 | 0.57 | 1.81 | 0.74 |
| Met Screen Assay | g Au/t | 2.62 | 0.61 | 1.70 | 0.85 |
| ICP Scans | g Ag/t | 1.0 | 2.0 | 1.0 | 1.0 |
| | Duplicate Ag g/t | 1.2 | 1.54 | 1.09 | 1.78 |
| | Al % | 8.05 | 7.87 | 6.06 | 6.39 |
| | g As/t | 715 | 251 | 1,810 | 1,065 |
| | g Cu/t | 120.5 | 174.5 | 84.5 | 75.0 |
| | Fe % | 5.55 | 6.49 | 6.23 | 6.17 |
| | g Hg/t | 0.02 | 0.03 | 0.02 | 0.03 |
| | g Ni/t | 50.7 | 54.9 | 27.2 | 55.8 |
| | g Te/t | 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 is negative 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 Au/t | | | | | % 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 <850μm 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 Au/t | | | | 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 to 0.71 mm.

Scoping level bulk sulphide flotation tests were conducted at $P_{80}$ of 75μm 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 Au/t | | | | 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 $P_{80}$'s – 212µm, 106µm and 53µm for the oxide ore, and five $P_{80}$'s - 150µm, 106µm, 75µm, 53µm, and 43µm 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 Au/t ore | | | Reagents kg/t ore | |
|---|---|---|---|---|---|---|---|
| | | | Extracted | Tails | Calc'd Head | NaCN | Lime Added |
| Oxide (avg. of HGO & MGO) | 100%-212µm | 93 | 1.29 | 0.09 | 1.37 | 0.12 | 3.1 |
| | 100%-106µm | 93.0 | 1.19 | 0.08 | 1.27 | 0.08 | 3.1 |
| | 100%-53µm | 91.3 | 1.13 | 0.08 | 1.21 | 0.28 | 3.0 |
| Sulphides (avg. of HGS & MGS) | 80%-150µm | 75.3 | 1.06 | 0.32 | 1.38 | 0.47 | 1.0 |
| | 80%-106µm | 78.2 | 1.05 | 0.29 | 1.34 | 0.39 | 1.0 |
| | 80%-75µm | 80.2 | 1.12 | 0.27 | 1.39 | 0.52 | 1.1 |
| | 80%-53µm | 84.1 | 1.22 | 0.23 | 1.44 | 0.76 | 1.3 |
| | 80%-45µm | 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 45µm case (HGS) for sulphides where the consumption was significantly higher. The $P_{80}$ of 45µm 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 13.3 to Figure 13.6.

**Figure 13.3**        **McClelland Gold Leach Profile for High Grade Oxide Composite (HGO)**
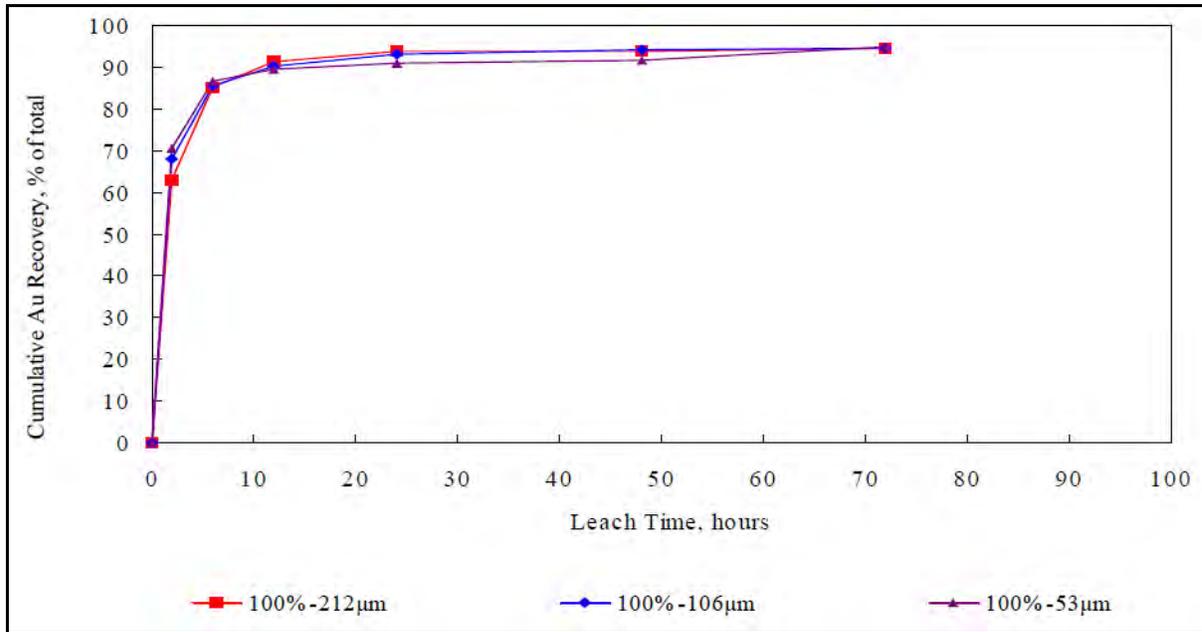


**Figure 13.4**        **McClelland Gold Leach Profile for Medium Grade Oxide Composite (MGO)**
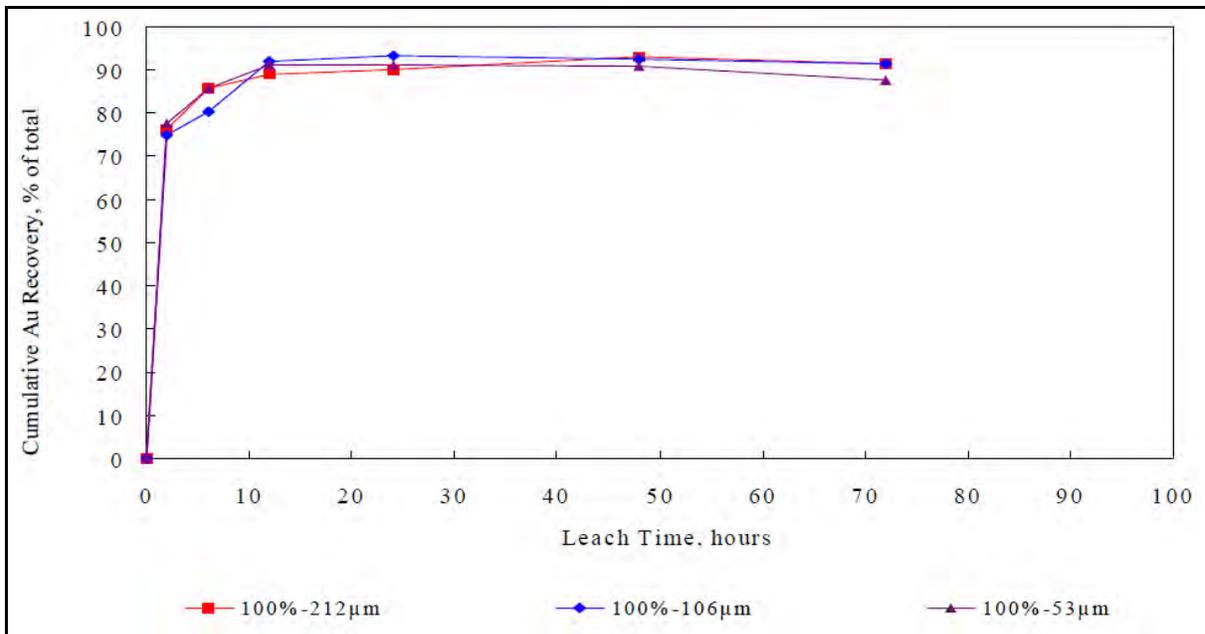
**Figure 13.5**     **McClelland Gold Leach Profiles for High Grade Sulphide Composite (HGS)**
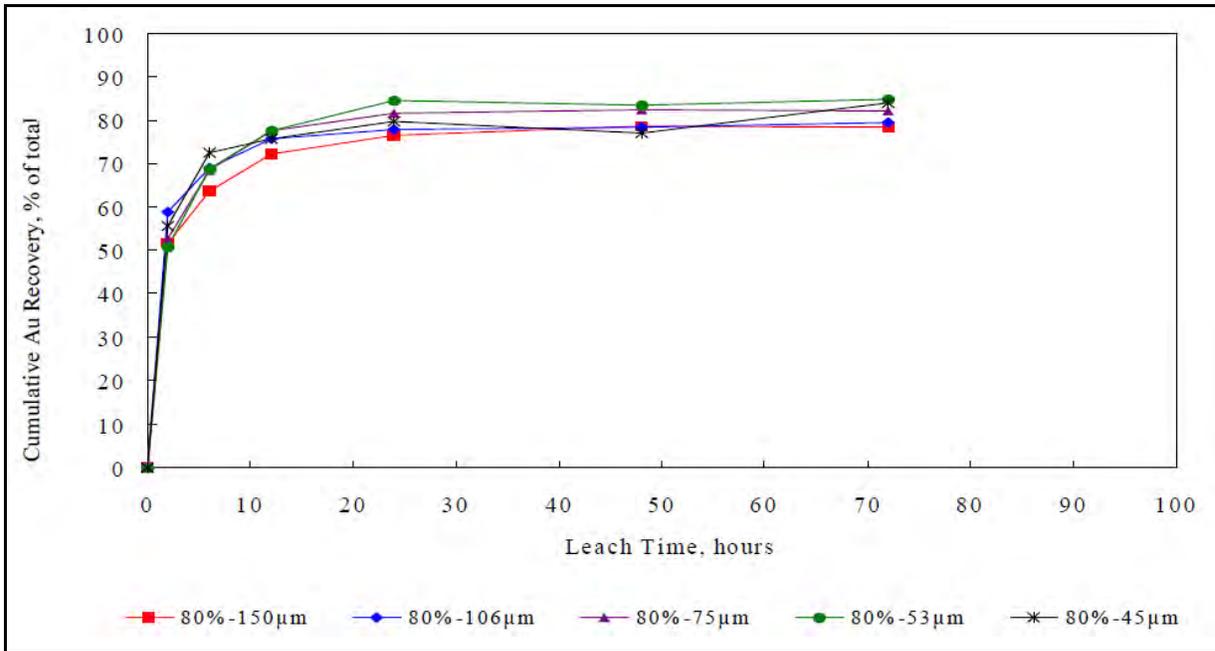


**Figure 13.6**     **McClelland Gold Leach Profiles for Medium Grade Sulphide Composite (MGS)**
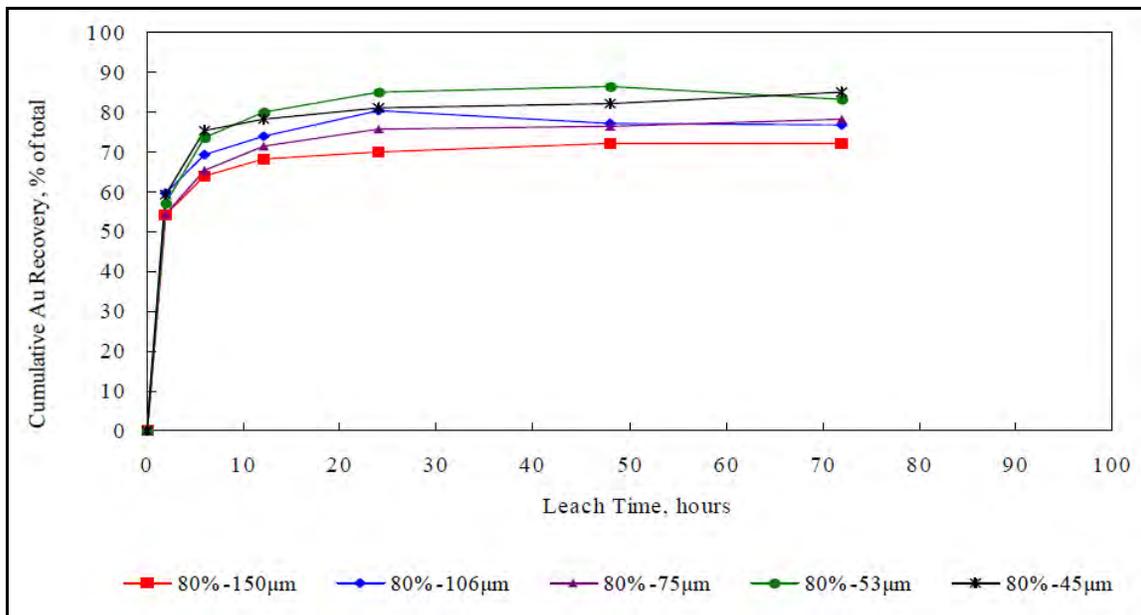


Additional bottle roll tests were conducted on all the composites to optimize the cyanide dosing rates. The oxide composites were tested at three grind sizes (-212, -106, and -53µm) and four NaCN concentrations (0.5, 1.0, 1.5, and 2.0 g/L). The sulphide composites were tested at two grind sizes (-75 and -53µm) 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 | NaCN g/L | No. of Tests | Gold Extraction % | g Au/t ore | | | Reagents, kg/t ore | |
|---|---|---|---|---|---|---|---|---|---|
| | | | | | Extracted | Tail | Calc'd Head | NaCN | Lime Added[2] |
| Oxide | 100%-212µm | 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 | 100%-150µm | 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 | 100%-106µm | 1.0 | 2 | 93.0 | 1.19 | 0.08 | 1.27 | 0.08 | 3.1 |
| Oxide | 100%-75µm | 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] | 100%-53µm | 1.0 | 2 | 94.1 | 1.13 | 0.07 | 1.20 | 0.09 | 4.0 |
| Sulphide[1] | 80%-150µm | 1.0 | 2 | 75.3 | 1.06 | 0.32 | 1.38 | 0.47 | 1.0 |
| Sulphide[1] | 80%-106µm | 1.0 | 2 | 78.2 | 1.05 | 0.29 | 1.34 | 0.39 | 1.0 |
| Sulphide | 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 | 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] | 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 minus 150µm feed size and 0.5 g NaCN/L.

For the sulphide composites, the results showed varying gold extraction percentages at the four cyanide concentrations, but no significant improvement beyond 1.0 g NaCN/L. 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 53µm feed size and 1.0 g NaCN/L.

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 Au/t ore | | | | | Reagents kg/t ore | |
|--------|-----------|-------------------|---|---------|----------|-------|------------|----------|------|---------------|--------------|------|----------------|
| | | Grav. Conc. | CN Leach | Combined | CN Tails | Total | Grav. Conc. | CN Leach | Tail | Calc'd Head | Assayed Head | NaCN | Lime Added |
| HGS | 80%-75µm | 30.9 | 54.2 | 85.1 | 14.9 | 100 | 0.61 | 1.07 | 0.30 | 1.98 | 1.76 | 0.30 | 1.1 |
| HGS | 80%-53µm | 22.3 | 63.5 | 85.8 | 14.2 | 100 | 0.41 | 1.16 | 0.26 | 1.83 | 1.76 | 0.52 | 1.3 |
| MGS | 80%-75µm | 14.0 | 69.5 | 83.5 | 16.5 | 100 | 0.10 | 0.52 | 0.12 | 0.74 | 0.80 | 0.34 | 1.1 |
| MGS | 80%-53µm | 18.2 | 68.4 | 86.6 | 13.4 | 100 | 0.14 | 0.53 | 0.10 | 0.77 | 0.80 | 0.17 | 1.1 |
| HGO | 100%-212µm | 6.2 | 89.7 | 95.9 | 4.1 | 100 | 0.10 | 1.5 | 0.07 | 1.67 | 2.50 | 0.04 | 2.9 |
| HGO | 100%-150µm | 12.0 | 84.9 | 96.9 | 3.1 | 100 | 0.22 | 1.54 | 0.06 | 1.82 | 2.50 | 0.04 | 2.9 |
| HGO | 100%-75µm | 12.8 | 82.8 | 95.6 | 4.4 | 100 | 0.21 | 1.38 | 0.07 | 1.66 | 2.50 | 0.07 | 3.1 |
| MGO | 100%-212µm | 7.7 | 84.1 | 91.8 | 8.2 | 100 | 0.05 | 0.54 | 0.05 | 0.64 | 0.59 | 0.18 | 3.2 |
| MGO | 100%-150µm | 10.2 | 82.6 | 92.8 | 7.2 | 100 | 0.07 | 0.55 | 0.05 | 0.67 | 0.59 | 0.07 | 4.4 |
| MGO | 100%-75µm | 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, -150, and -75µm. 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, $P_{80}$ 75µm and $P_{80}$ 53µm. 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 Au/t ore | | | NaCN Consumption kg/t ore | | | Lime added kg/tore | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Avg. | Min. | Max. | Avg. | Min. | Max. | Avg. | Min. | Max. | Avg. | Min. | Max. |
| Oxide Ore Type, 80%-150µm, 0.5 g NaCN/L | | | | | | | | | | | | |
| 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, 80%-150µm, 0.5 g NaCN/L | | | | | | | | | | | | |
| 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, 80%-53µm, 1 g NaCN/L | | | | | | | | | | | | |
| 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, 80%-53µm, 1 g NaCN/L | | | | | | | | | | | | |
| 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/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/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 to 0.43 kg/t) but varied from low to high for the sulphide samples (0.3 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 to 1 kg/t).

CIL and CIP carbon adsorption capacity tests were also conducted and the results are presented in Figure 13.7 to Figure 13.10.

**Figure 13.7        McClelland Carbon-In-Leach Adsorption Capacity for Oxides**
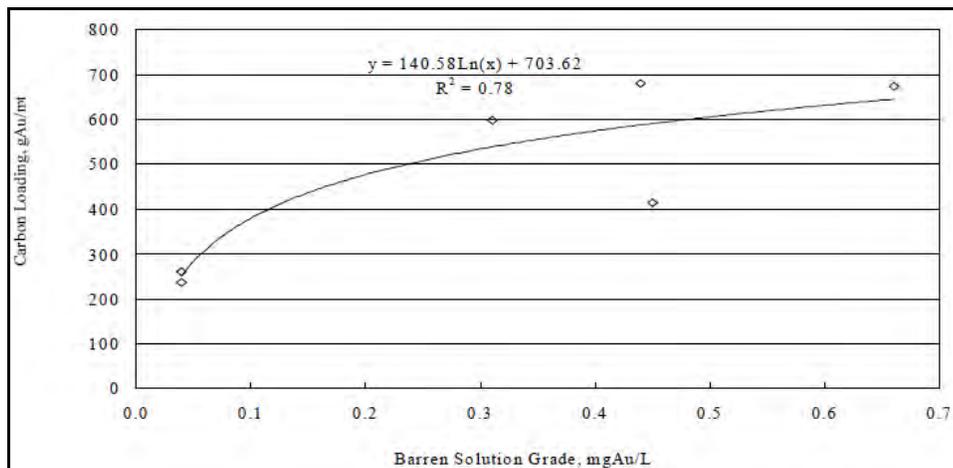
**Figure 13.8          McClelland Carbon-In-Leach Adsorption Capacity for Sulphides**
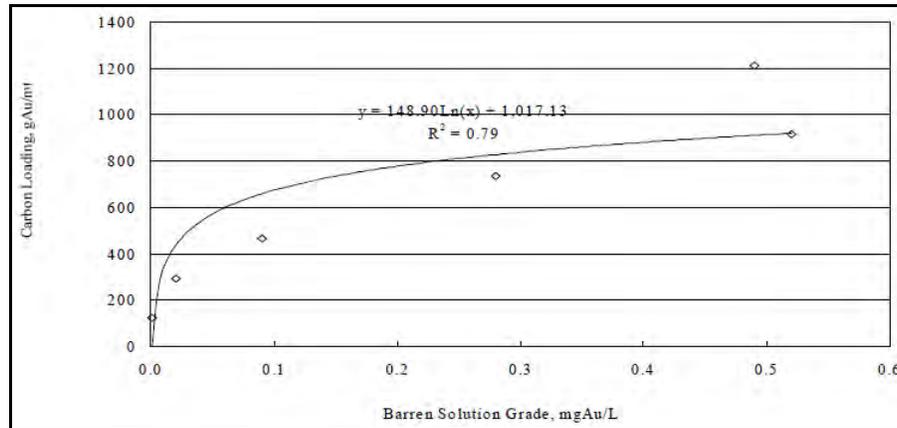


**Figure 13.9          McClelland Carbon-In-Pulp Adsorption Capacity for Oxides**
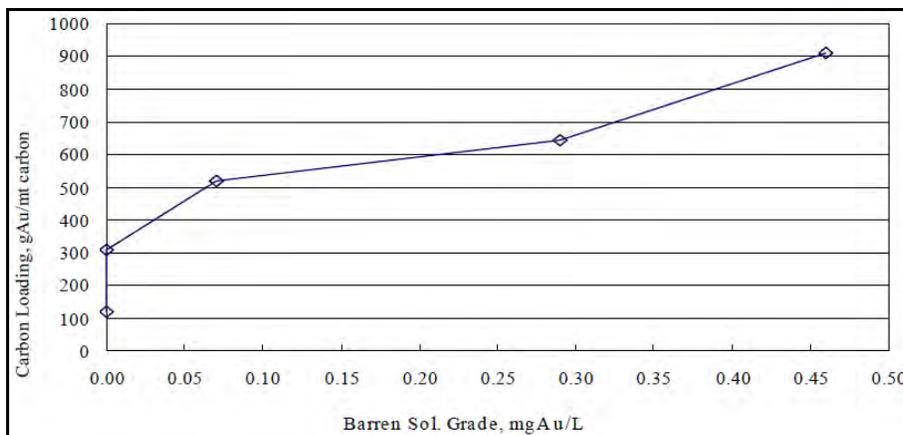


**Figure 13.10          McClelland Carbon-In-Pulp Adsorption Capacity for Sulphides**
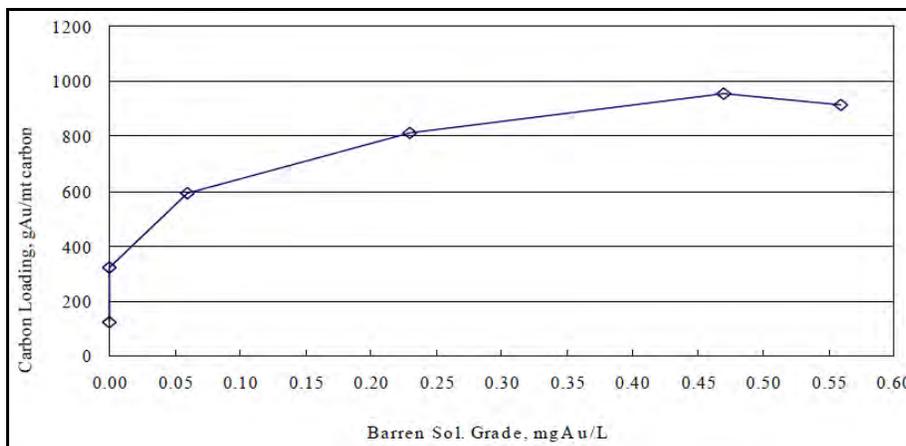
Standard cyanide destruction testing was conducted on both oxide and sulphide slurries using $SO_2$/Air treatment with SMBS used as the source of $SO_2$. A summary of the results is shown in Table 13.16.

**Table 13.16** **McClelland $SO_2$/Air Treatments Results for $P_{80}$ 150m Feeds**

| Sample | Stream | Retention Minutes | Run # | Slurry pH | $CN_{WAD}$ | Target | Actual | SMBS | $SO_2$ | $CuSO_4$ | $SO_2$/ g $CN_{WAD}$ Treated | SMBS | Lime | SMBS Utilization Efficiency % |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | **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 ore** | | |
| Oxide Master | Feed Effluent | 95.1 | 1st | 10.2 8.5 | 273* 0.024 | 3:1 | 2.3:1 | 6.86 | 5.57 | 0.58 | 5.57 | 3.51 | 8.93 | 44.2 |
| | Feed Effluent | 95.1 | 2nd | 8.5 | 238 0.036 | 3:1 | 2.7:1 | 8.12 | 6.59 | 0.69 | 6.60 | 3.51 | 5.95 | 37.3 |
| | Feed Effluent | 95.1 | 3rd | 8.6 | 178 0.034 | 3:1 | 4.3:1 | 13.08 | 10.63 | 1.11 | 10.63 | 3.51 | 5.36 | 23.2 |
| Sulphide Master | Feed Effluent | 95.1 | 1st | 10.6 8.5 | 200 0.28 | 3:1 | 4.9:1 | 14.87 | 12.08 | 0.83 | 12.09 | 5.31 | 8.12 | 20.4 |
| | Feed Effluent | 95.1 | 2nd | 8.6 | 158 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 | | Type | Dose (g/t) | Conc. (g/L) [2] | 10% Feed[1] | 15% Feed[1] | 20% Feed[1] | 25% Feed[1] | 30% Feed[1] | Max. U/F Solids Conc. (%) |
|---|---|---|---|---|---|---|---|---|---|---|
| | | **Flocculant** | | | **Min. Unit Area at Specified Feed Solids Conc. & U/F Density ($m^2$/t/d) [3]** | | | | | |
| 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 (m³/m³/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 cubic metres of feed slurry per hour per square metre of thickener area (m³/m² 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 0.45m 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 150μm closed screens. The oxide samples displayed much lower abrasion characteristics than the sulphides, as expected. The results are presented in Table 13.19.

**Table 13.19        McClelland Abrasion Index and Ball Mill Grindability Tests**

| Sample | Abrasion Index | BMWi (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 minus 212μm to 150μm.

- The optimum feed size for whole ore cyanidation of sulphide composites was $P_{80}$ 53μm.

- The optimum cyanide concentration for leaching of oxide composites was 0.5 g NaCN/L.

- The optimum cyanide concentration for leaching of sulphide composites was 0.5 to 1.0 g NaCN/L.

- 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 leaching at optimized conditions.

- Conventional $SO_2$/air tailings slurry treatment was effective in decreasing weak acid dissociable (WAD) CN concentration to acceptable levels.

### 13.2.5    Phillips 2012 Testwork Program

Further comminution testwork was performed by Phillips Enterprises on granodiorite, the most competent of all the lithologies which represents about 2% of the oxide reserves 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 to 32.5 m (saprolite), 32.5 to 37 m (bottom of the saprolite profile) and 37.0 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 of oxide ore from different ore zones and their gold deportment. On average, 62% of the oxides were at minus 100µm, and 83% of the oxides were at minus 200µm. 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 Au/t | Tails Avg. g Au/t | LeachWell + Tails Combined g Au/t | Gold Extraction % |
|---|---|---|---|---|---|---|---|---|
| Maga | +6.3mm | 18.72 | 15.1% | 100.0% | 829 | 0.058 | 0.887 | 93% |
| | +.200mm | 30.93 | 24.9% | 84.9% | 618 | 0.020 | 0.638 | 97% |
| | +.100mm | 5.53 | 4.4% | 60.0% | 667 | 0.047 | 0.714 | 93% |
| | -.100mm | 69.05 | 55.6% | 55.6% | 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% | 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% | 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% | 487 | 0.024 | 0.511 | 95% |
| Siga W-S | +6.3mm | 14.89 | 18.6% | 100.0% | 429 | 0.019 | 0.448 | 96% |
| | +.200mm | 15.26 | 19.1% | 81.4% | 868 | 0.057 | 0.925 | 94% |
| | +.100mm | 8.51 | 10.7% | 62.3% | 625 | 0.056 | 0.681 | 92% |
| | -.100mm | 41.22 | 51.6% | 51.6% | 569 | 0.051 | 0.620 | 92% |
| P16 | +6.3mm | 2.13 | 9.6% | 100.0% | 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% | 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% | 812 | 0.059 | 0.870 | 93% |
| | -.100mm | 271.15 | 55.4% | 55.4% | 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/m³ | 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.) | Sample #1263221 (49 min.) | Sample #1263226 (49 min.) |
|---|---|---|---|---|
| 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 13.11.

**Figure 13.11**      **COREM Leach Curves for Pyrrhotite-Rich Samples and Impact of Lead Nitrate**
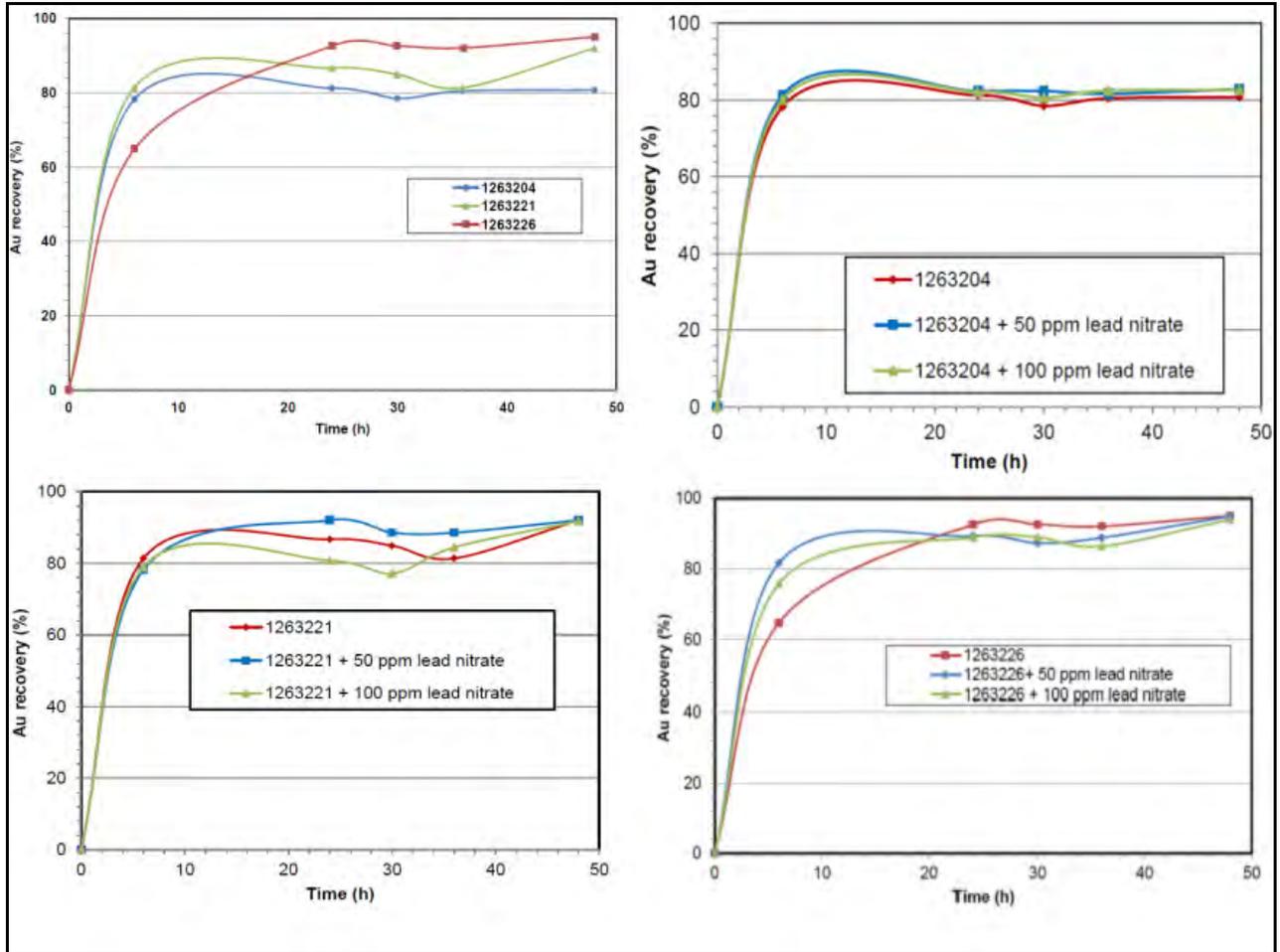


A comparative analysis of gold dissolution profiles for samples containing pyrrhotite showed that pyrrhotite content of the tested samples was not the only factor controlling the gold recovery. Sample #1263226 showed the highest gold recovery (92.6%) despite its high content of pyrrhotite (2.85%). The 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 13.12.

**Figure 13.12**          **2013 Met-Solve Scrubbing Test - Scrubbing Completeness % vs. Time**
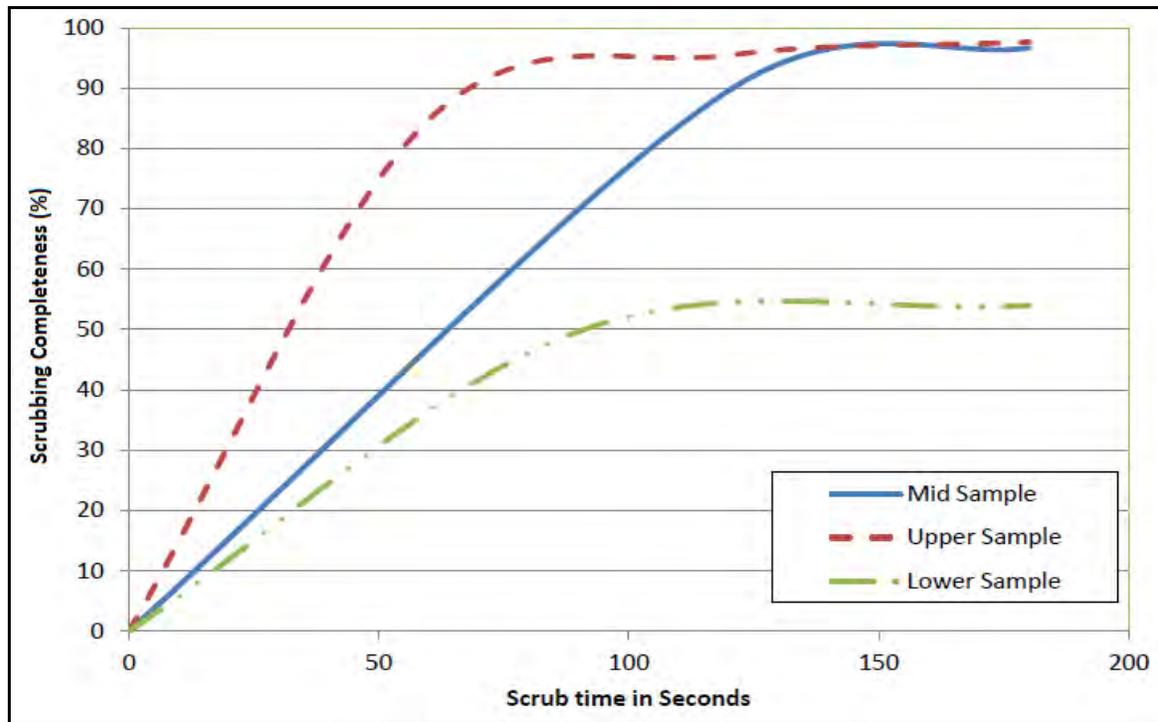


For scrubbing to be complete, a minimum of 95% completeness is required. As seen in Figure 13.12, the upper and middle oxide samples met this requirement, however, the lower oxide sample failed to scrub satisfactorily.

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 minus 150μm from the upper, middle and lower ore zones. The leach curves at different cyanide concentrations are shown in Figure 13.13. The cyanide and lime consumptions are shown in Table 13.24.

**Figure 13.13          Met-Solve Leach Curves at Different Cyanide Concentrations**
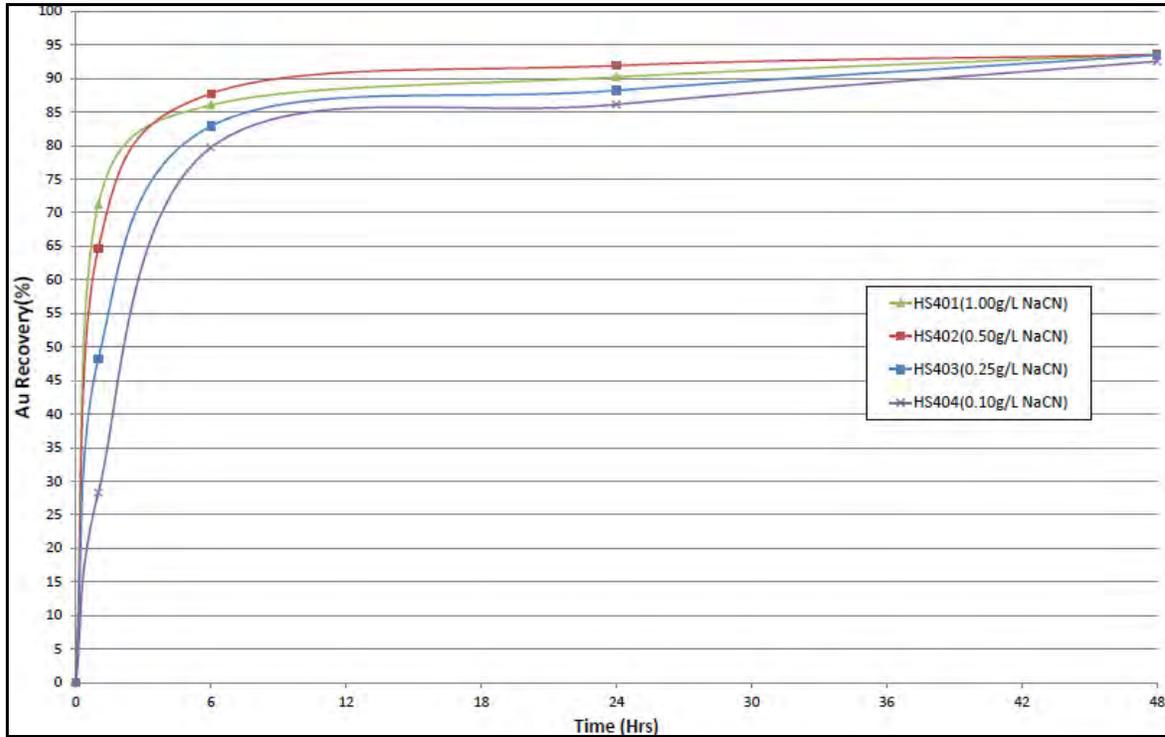


**Table 13.24          Met-Solve Cyanide Leach Results on Screened U/S (-150μm)**

| Test No. | Pulp Density % | NaCN Conc. (g/L) | Au Extraction % | | | | Residue Assay (g Au/t) | 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 | | 0.4 | 1.31 |
| HS402 | 33 | 0.5 | 65 | 88 | 92 | 94 | 0.06 | 0.85 | 0.76 | 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 NaCN/L, 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, where eight contained material to form four composites, and the other 18 contained material 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 Au/t | Average Assay, g Ag/t | Wt. Avg. Screen Head g Au/t | Wt. Avg. Screen Head g Ag/t |
|---|---|---|---|---|---|---|
| 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 | | | 0.685 | 0.31 | 0.689 | 0.39 |
| 70413 A | BHK-12 | Transition | -- | -- | 0.937 | 0.47 |
| 70413 B | | | 1.030 | 0.41 | 0.955 | 0.43 |
| 70414 A | BHK-13 | Transition | -- | -- | 0.790 | 0.55 |
| 70414 B | | | 0.855 | 0.51 | 0.670 | 0.55 |
| 70415 A | BHK-14 | Transition | -- | -- | 1.134 | 0.65 |
| 70415 B | | | 1.190 | 0.62 | 1.656 | 0.63 |
| 70416 A | BHK-15 | Transition | -- | -- | 1.114 | 0.61 |
| 70416 B | | | 1.426 | 0.62 | 1.070 | 0.58 |
| 70417 A | BHK-16 | Transition | -- | -- | 1.144 | 0.57 |
| 70417 B | | | 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 13.14 to Figure 13.15.

**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/m² | % Slump | Pass/Fail |
|---|---|---|---|---|---|---|---|---|---|
| 70401 | 70421 A | BHK-01 | Primary | 25 | 16 | 40 | 6 | 0 | Fail |
| | 70421 B | | | | 16 | 8 | 837 | 4 | Pass |
| | 70426 A | | | | 20 | 16 | 1,345 | 2 | Pass |
| | 70426 B | | | | 26 | 16 | 3,264 | 2 | Pass |
| 70402 | 70422 A | BHK-02 | Primary | 25 | 16 | 40 | 0 | 0 | Fail |
| | 70422 B | | | | 16 | 8 | 931 | 3 | Pass |
| | 70427 A | | | | 20 | 16 | 97 | 3 | Fail |
| | 70427 B | | | | 26 | 16 | 2,180 | 2 | Pass |
| 70403 | 70423 A | BHK-03 | Primary | 12.5 | 0 | 16 | 1,343 | 1 | Pass |
| | 70428 A | | | | 4 | 16 | 4,191 | 3 | Pass |
| 70404 | 70424 A | BHK-04 | Primary | 12.5 | 0 | 16 | 5,810 | 2 | Pass |
| | 70429 A | | | | 2 | 16 | 7,016 | 3 | Pass |
| 70405 | 70420 | Met-Solve | Primary | 25 | 0 | 40 | 88 | 2 | Fail |
| | 70425 | | | | 4 | 40 | 158 | 0 | Pass |
| | 70430 A | | | | 4 | 8 | 1,184 | 1 | Pass |

**Figure 13.14      KCA HL Program – Compacted Permeability Results for BHK-01 to BHK-04**
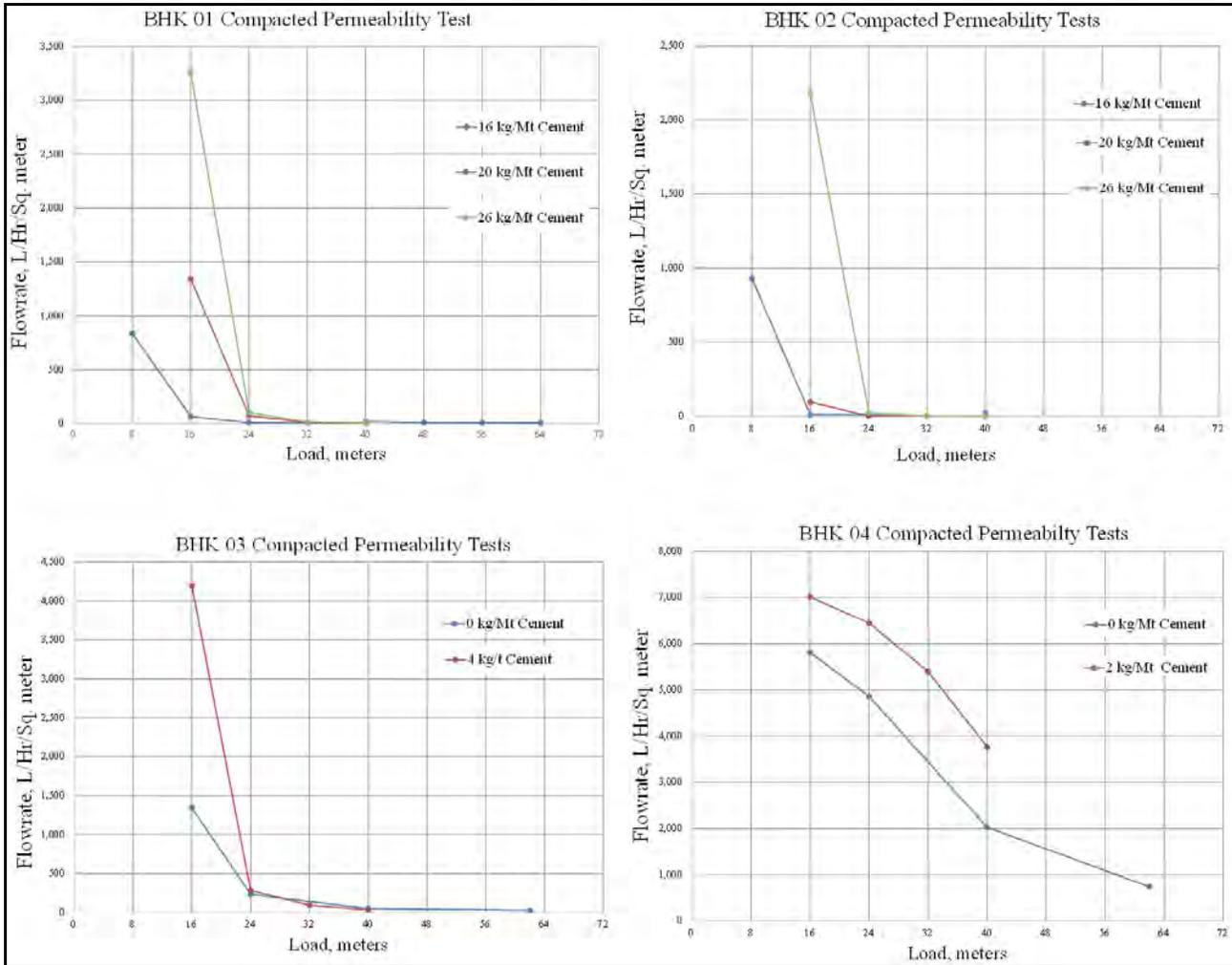


**Figure 13.15      KCA HL Program – Compacted Permeability Results for Met-Solve Composite**
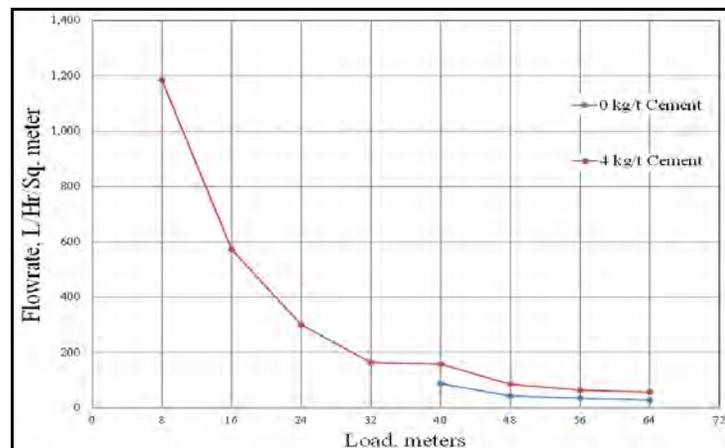
The results showed all except four tests passing the agglomeration and compacted permeability criteria as set out by KCA.  However, a large amount of cement is required to maintain the heap higher than 16 m for BHK-01 to BHK-04 tests. The Met-Solve material, which was previously scrubbed, required only 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 Au/t | 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 to 0.38 kg NaCN /t for BHK-05 to BHK-10, and 1.33 kg NaCN/t 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 to 0.96 kg NaCN/t.

- 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 Au/t | Weighted Avg. Head Assay, g Ag/t |
|---|---|---|---|---|
| 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 (1000g 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 Au/t | Au Extracted, % | Calculated Head, g Ag/t | Ag Extracted, % | Leach Time, hours | Consumption NaCN, kg/t | 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 Au/t | Calculated Head, g Au/t | Au Extracted, % | Leach Time, hours | Consumption NaCN, kg/t | 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 Ag/t | Calculated Head, g Ag/t | Ag Extracted, % | Leach Time, hours | Consumption NaCN, kg/t | 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 (m²/t/d) | Hydraulic Rate for High Rate Thick. Sizing (m³/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 from this testwork indicated no other exceedance other than arsenic at 0.029 ppm in decant solution and 0.012 ppm in filtered solution against a guideline of 0.010 ppm (WHO & EPA).

The oversized scrubbed material designated as KCA Sample No. 70489 A, at $P_{100}$ 25 mm and >0.212 mm, were used for column leach testwork. The results are presented in Table 13.35 and Figure 13.16.

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

| KCA Sample No. | Calculated Head, g Au/t | Extracted, Au % | Calculated Head, g Ag/t | Extracted, Ag % | Calculated Tail p80 Size, mm | Days of Leach | Consumption NaCN, kg/t | Addition Ca(OH)₂, kg/t |
|---|---|---|---|---|---|---|---|---|
| 70489 A | 1.083 | 81% | 0.88 | 40% | 13.9 | 82 | 0.99 | 3.94 |

**Figure 13.16        KCA Preliminary Hybrid Program – Oversized Scrubbed Material Column Leach Curve**
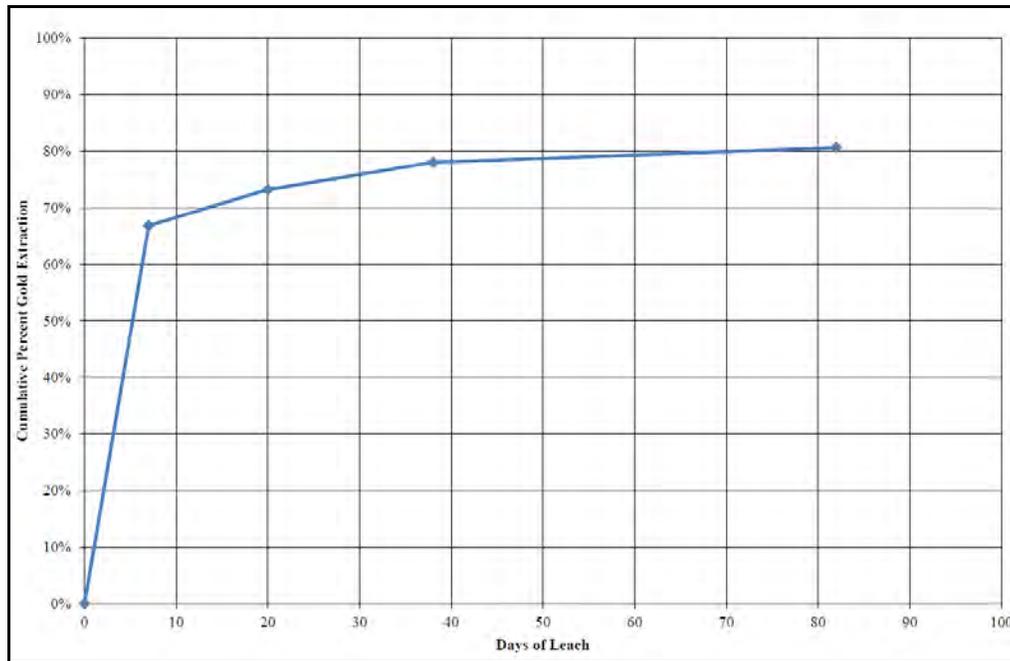


Copper and mercury content in column leach solutions were analyzed and reported low, indicating that excess reagent consumption from cyanide soluble species is unlikely.

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:

  o Wet head screen analysis and scrubbing program.

- Agglomeration and compacted permeability tests:

  o Eight +212µm samples, wet head screen analysis, physical characterization, agglomeration testwork, and compacted permeability testwork.

- Column leach tests:

  o Initial head assay characterization, wet head screen analysis with assays by size fraction, column leach testwork (minus 25 mm), and detox and environmental characterization.

- Direct bottle roll leach and CIL on undersized scrubber product:

  o Eight <212µm 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 Au/t | Weighted Avg. Head Assay, g Ag/t |
|---|---|---|---|---|---|
| 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 |

Particle size distributions of the samples are provided in Figure 13.17.

**Figure 13.17     KCA Hybrid FS – Particle Size Distribution of Samples**
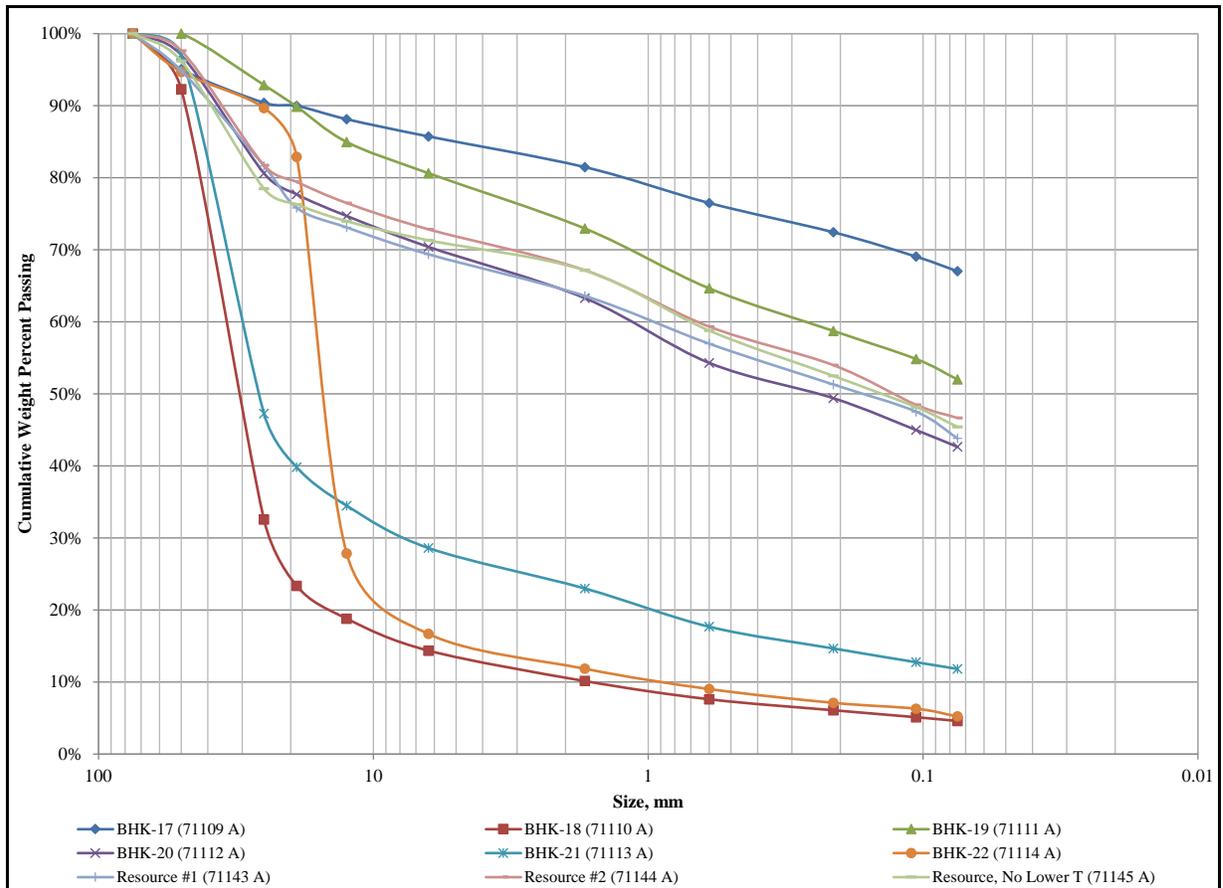
A summary of the compacted permeability test results is presented in Figure 13.18.

**Figure 13.18     KCA Hybrid FS – Compacted Permeability Tests**
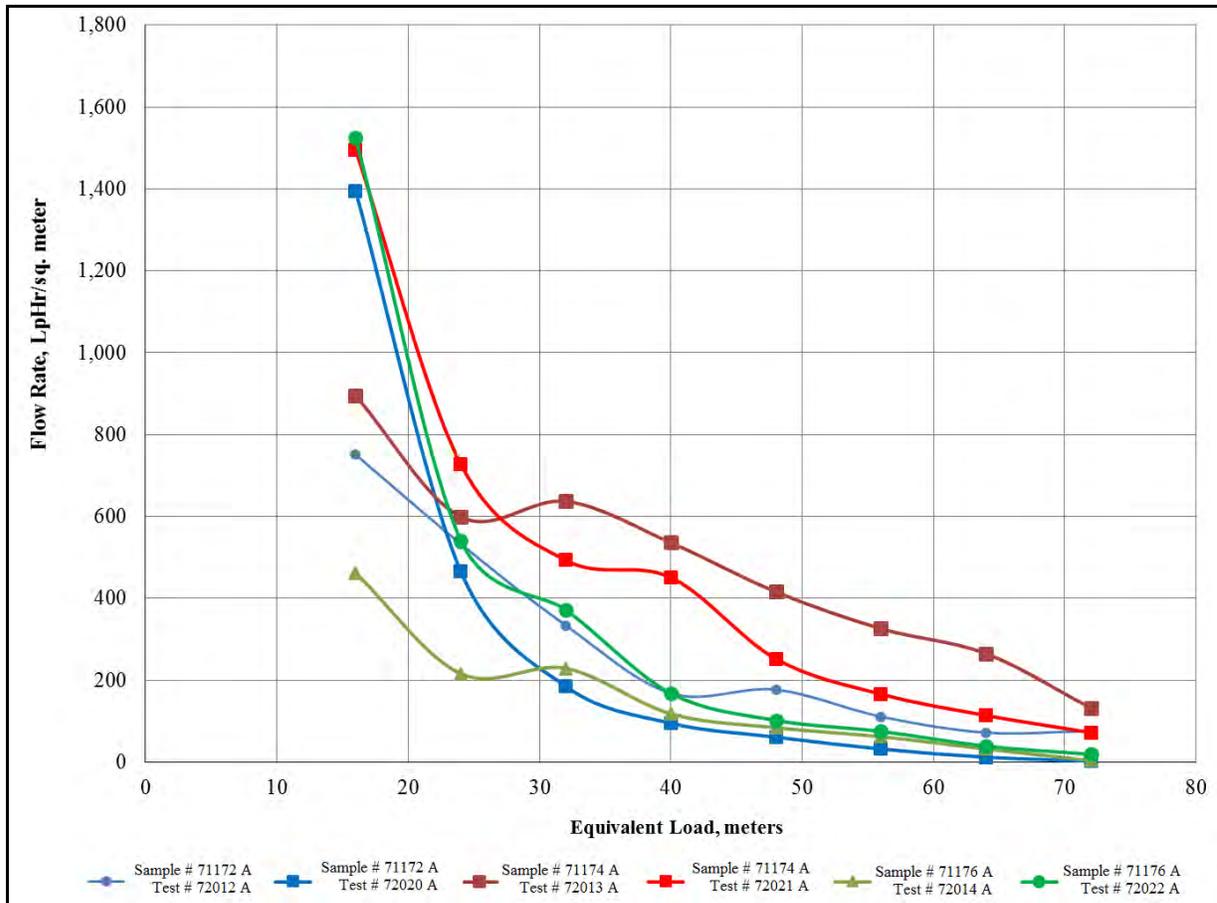
A summary of the column leach gold extraction results is presented in Figure 13.19 and Figure 13.20.

**Figure 13.19          KCA Hybrid FS – Column Test Leach Curves for BHK-17 to BHK-22**
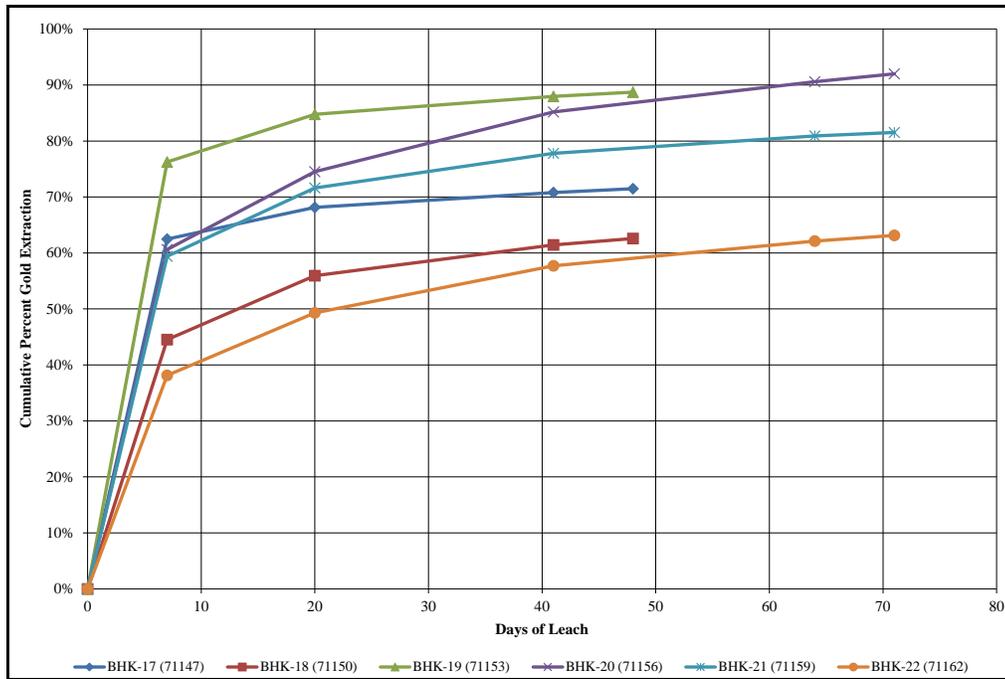


**Figure 13.20          KCA Hybrid FS – Column Test Leach Curves for Resource Composites**
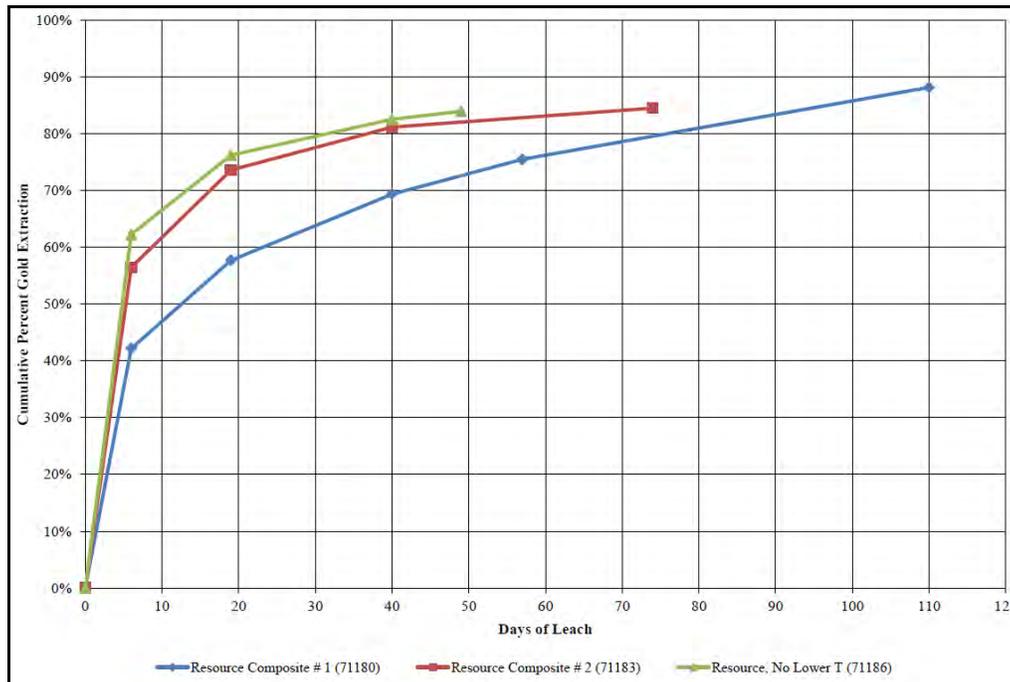
A summary of all the bottle roll and CIL gold leach extraction curves is presented in Figure 13.21.

Figure 13.21                    KCA Hybrid FS – Summary of Bottle Roll / CIL Leach Curves
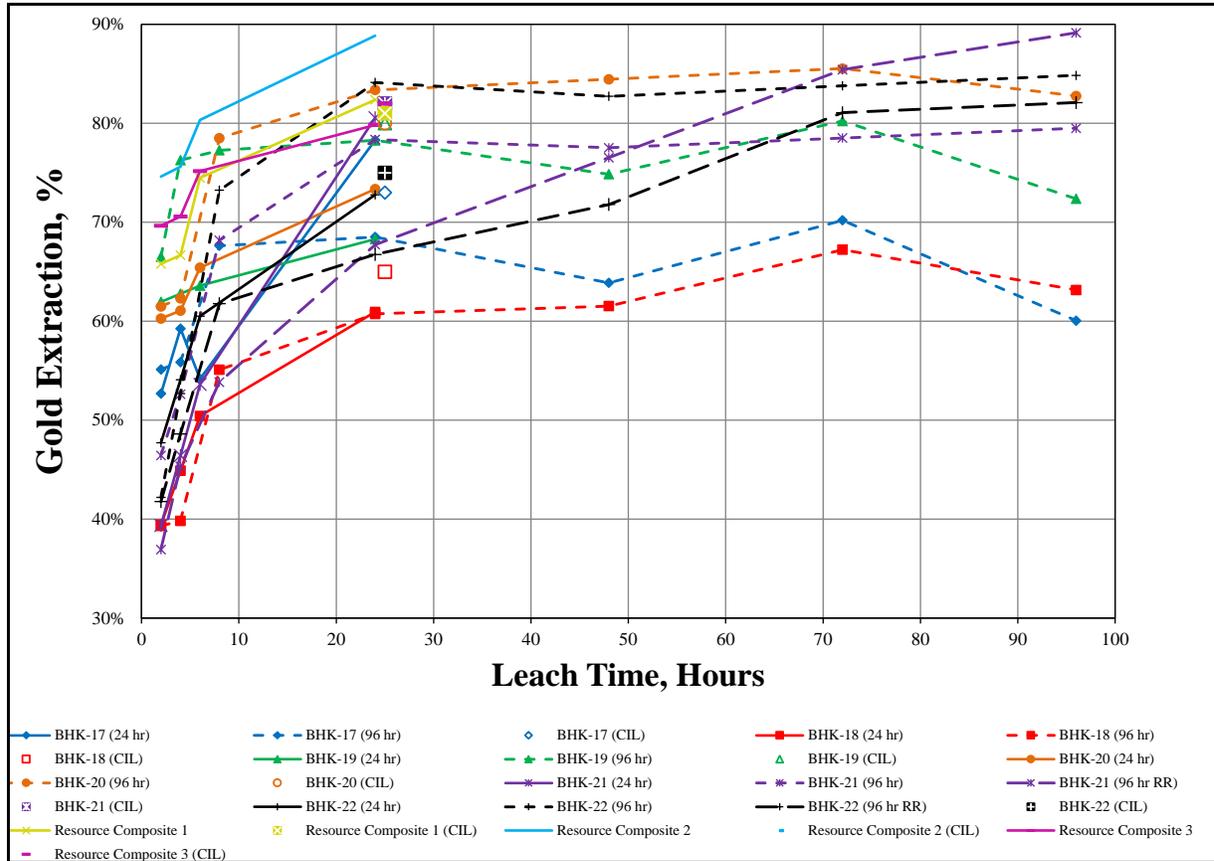


The overall extraction of the material was calculated by utilizing the weights of the scrubber products and the scrubber solutions. The oversized material was represented in the column leach tests, the undersized material was represented by the 24 hour CIL bottle roll leach tests, and the scrubber solutions 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 Au/t | Calc'd Head, g Au/t | Extraction +0.212mm, g Au/t | Extraction +0.212mm, % Au | Extraction -0.212mm, g Au/t | Extraction -0.212mm, % Au | Extraction Scrubber, g Au/t | 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% |

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 (ºC) | 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] (m²/t/d) | Hydraulic Rate for High Rate Thick. Sizing[5] (m³/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 underlined conventional type thickener design only and includes a 1.25 scale-up factor.*
*(5) Hydraulic loading rate for underlined 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 Au/t, 0.81% $S^=$, and 0.10% total carbonaceous matter. The main sulphide minerals were 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 for at grind size of $P_{80}$ 74µm. Any options with regrinding, the material was ground to One option included a regrind of the flotation concentrate to $P_{80}$ 26µm. 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 g Au/t | Head g Au/t |
|---|---|---|---|---|---|---|---|---|
| | Gravity | WO CN | Tails CN | Tails Flot | Conc CN | Combined | | |
| 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 Programs

In 2017, Orezone approached Lycopodium to conduct a feasibility study for a CIL process on the oxide and 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 submitted these to SGS Quebec for testing. The object 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 ore.

Dynamic thickening testwork was also conducted by Outotec Canada Ltd (Outotec) on an oxide composite sample.

The results of this program 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 Name | Au - Head (g/t) | LeachWell Au ppm | Calc Au Head (g/t) | As Received (%) Moisture | As Received (%) +3.35mm | BWI (kWh/t) Direct | BWI (kWh/t) O'All | Au CN Test P$_{80}$ (μm) | Au CN Test 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 13.22 to Figure 13.25 and Table 13.44.

**Figure 13.22**     **SGS 2018 Neutralization Test – pH vs. Lime Addition for Oxide Samples**
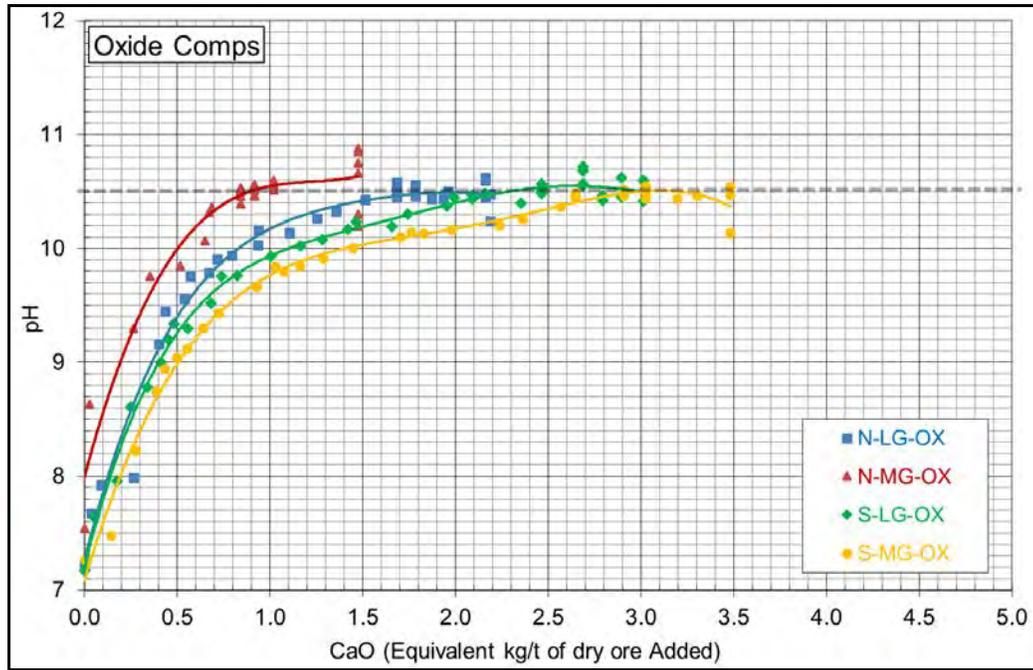


**Figure 13.23**     **SGS 2018 Neutralization Test – pH vs. Lime Addition for Transition Samples**
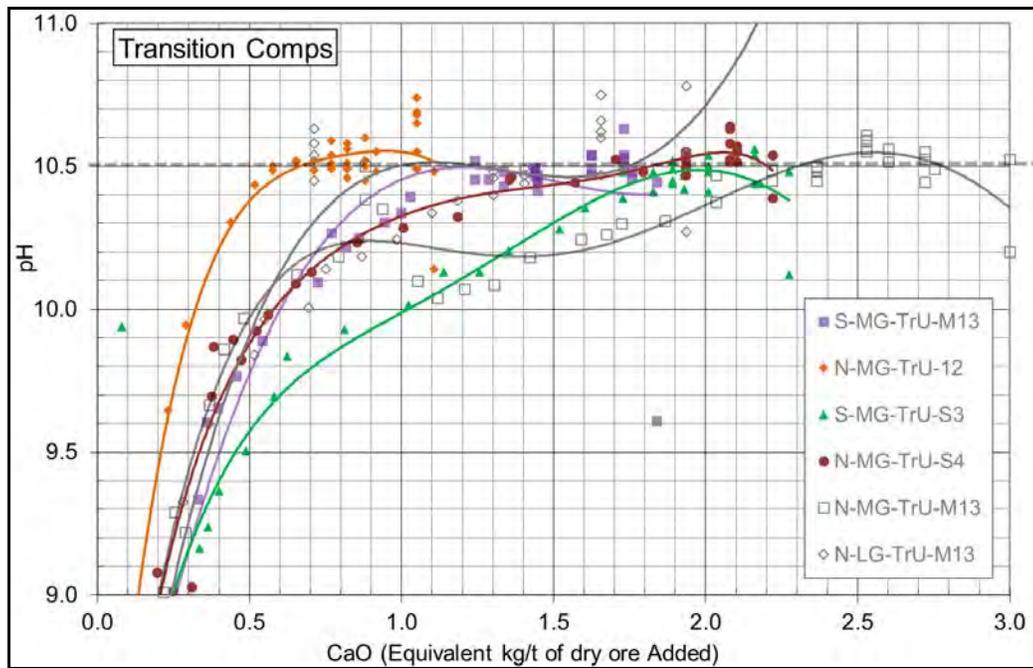
**Figure 13.24          SGS 2018 Neutralization Tests – Lime Addition vs. Time for Oxide Samples**
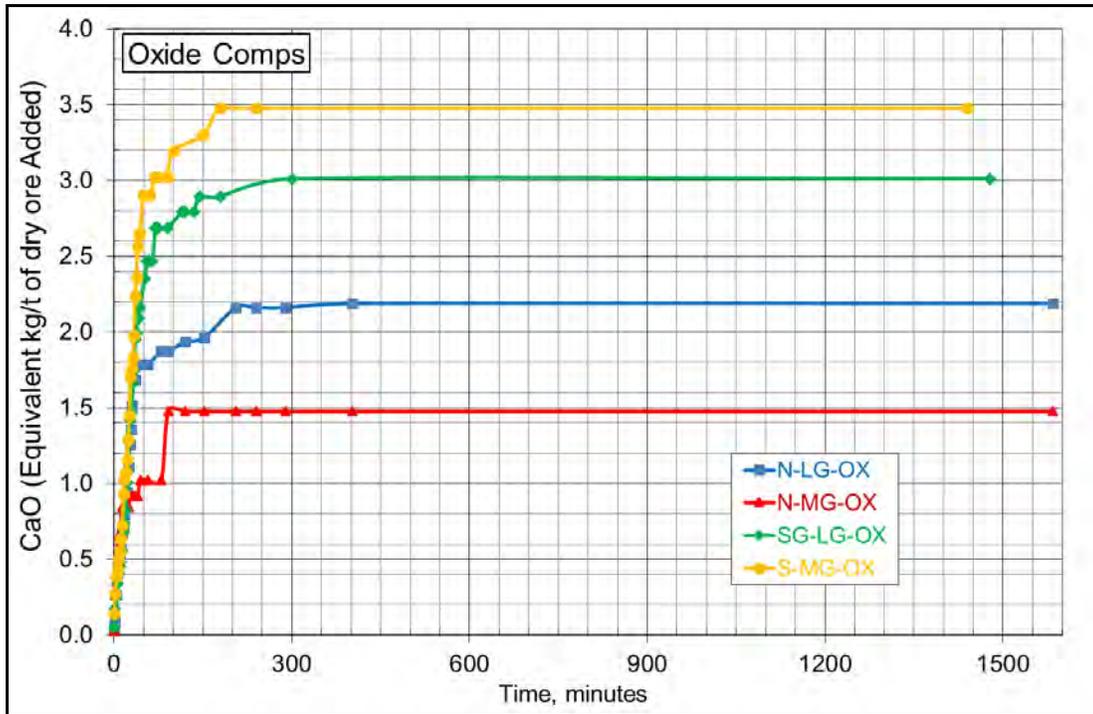


**Figure 13.25          SGS 2018 Neutralization Tests – Lime Addition vs. Time for Transition Samples**
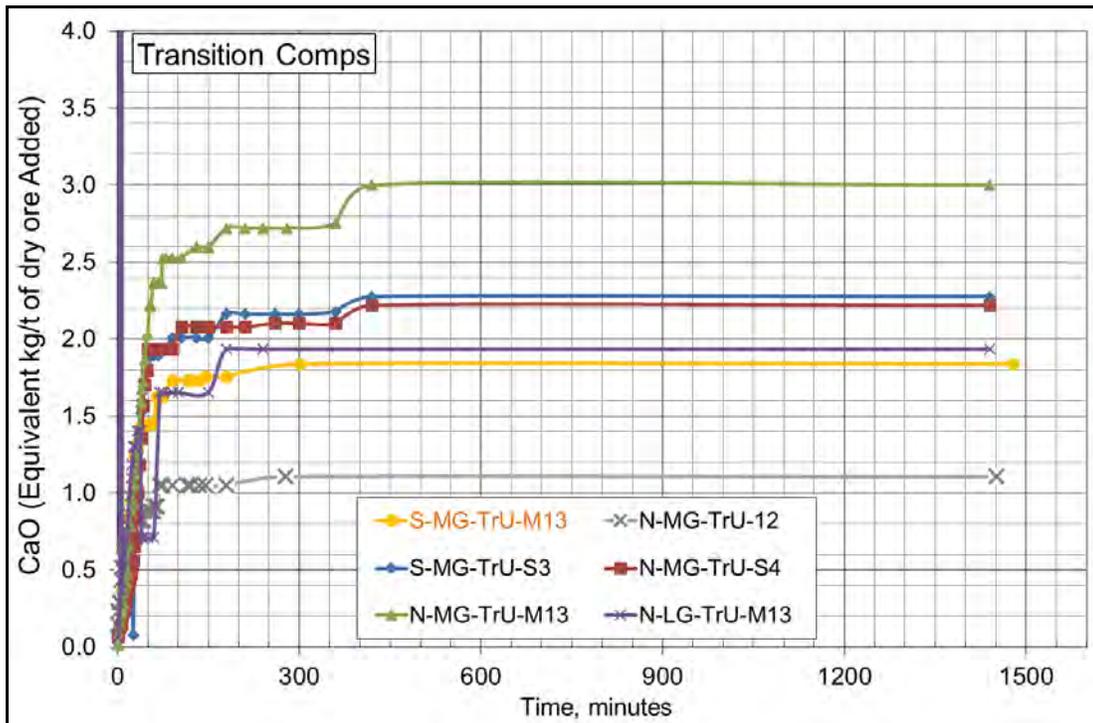
**Table 13.44          Summary Results for Neutralization (Lime Demand) Tests**

| Sample | CaO kg/tonne 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 125µm materials, was tested by Outotec to examine the flocculant screening and dynamic thickening. Results are shown in Figure 13.26 and Table 13.45. For other results provided by Outotec under this program, refer to Outotec Memorandum No. 11282017-TQ1-TM-001-R0.

**Figure 13.26          Outotec Flocculant Screening Results**
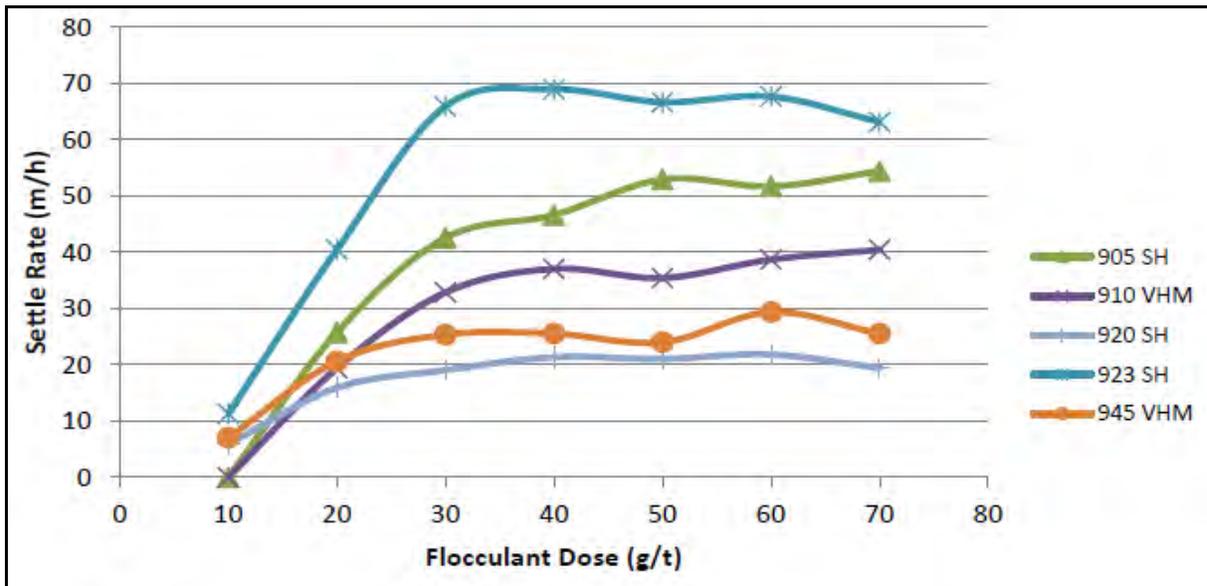
**Table 13.45          Outotec Dynamic Thickening Summary Results**

| Run No. | Feed | | | Flocculant | | Underflow | | Overflow |
|---|---|---|---|---|---|---|---|---|
| | Flux | Calc. Solids | Liquor RR | Type | Dose | Calc. Solids | YS | Solids |
| | (t/(m²·h)) | (% (w/w)) | (m/h) | | (g/t) | (% (w/w)) | (Pa) | (mg/L) |
| 1 | 0.80 | | 6.17 | | 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 | 12.0 | 1.54 | SNF 923 SH | 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/m²h. For thickener underflow density of 48.8 % solids, a flux of 0.60 t/m²h can be used for sizing the thickener.

## 13.4      Results Interpretation

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

### 13.4.1      Ore Characteristics

Orway Mineral Consultants (OMC) compiled and analysed all the historical comminution testwork and provided ore characteristic values for use in the comminution design as per Table 13.46. The comminution design parameters were also agreed to by Soutex.

**Table 13.46          Ore Characteristics for Comminution Design**

| Parameter | Units | Oxide | Transition Upper | Blend | Notes |
|---|---|---|---|---|---|
| **ROM Properties** | | | | | |
| Moisture Content | %H₂O | 5.70 | 2.80 | 4.83 | RPA Resource Update 2017 |
| Specific Gravity | - | | | 2.83 | Outotec 11282017-TQ1-TM-001-R0 |
| Al | g | 0.028 | 0.052 | 0.035 | Testwork Ave |
| CWI | kWh/t | 7.5 | 8.6 | 7.8 | Testwork Max |
| RWI | kWh/t | 5.4 | 7.8 | 6.1 | Testwork - 1 Sample |
| BWI | kWh/t | 4.3 | 7.8 | 5.3 | 85th Percentile |

### 13.4.2    Grind Size Selection

McClelland and AMMTEC both conducted extensive leach testwork at varying grind size, hence, results from the two programs were used to form the data in this grind size selection exercise. Figure 13.27 and Figure 13.28 are charts showing the grind sizes plotted against the residue grades for oxide and transition ores, respectively. Considering the life-of-mine plan 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 comminution circuit be designed for a grind size of $P_{80}$ 0.125mm (125µm).

**Figure 13.27        Grind Size vs. Residue Grade for Oxide Material <2 g/t**
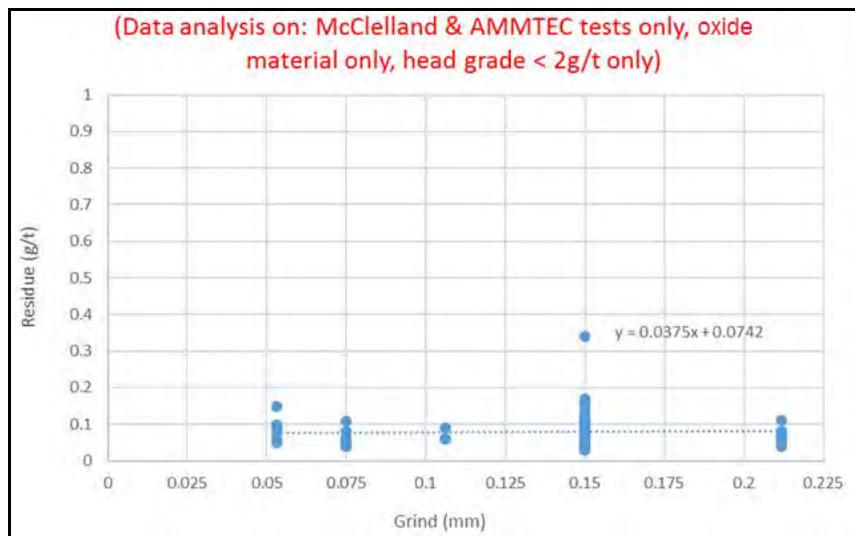


**Figure 13.28        Grind Size vs. Residue Grade Transition Material <2g/t**
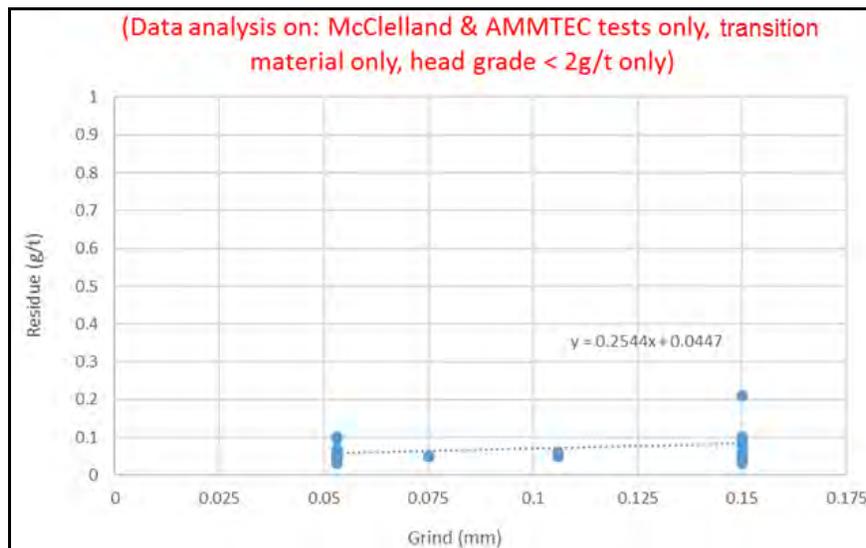
### 13.4.3    Gold Extraction Model

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

- Ore type: Upper Oxide, Lower Oxide and Upper Transition.

- Grind size: 74, 106 and 150 microns.

- 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 13.29 to Figure 13.31).

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 13.47.

**Table 13.47         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 |

The life-of-mine weighted average gold recovery equation shown in Table 13.48, including a correction for solution gold losses, was then used for the purposes of predicting annual gold production.

**Table 13.48         Gold Recovery Equation**

| Ore Type | Life-of-Mine Tonnage (Mt) | % of Total | Wt. Avg. Residue (g/t) | Solution Losses (g/t) |
|---|---|---|---|---|
| Upper Oxide | 27.55 | 49.2 | | |
| Lower Oxide | 20.90 | 37.3 | 0.07442 (~0.0745) | ~0.005 |
| Upper Transition | 7.54 | 13.5 | | |
| Total | 55.99 | 100 | | |
| Weighted Average Gold Recovery % = (Head Grade – 0.0695) / Head Grade x 100 | | | | |

**Figure 13.29        Gold Extraction Equation for Upper Oxide Ore**
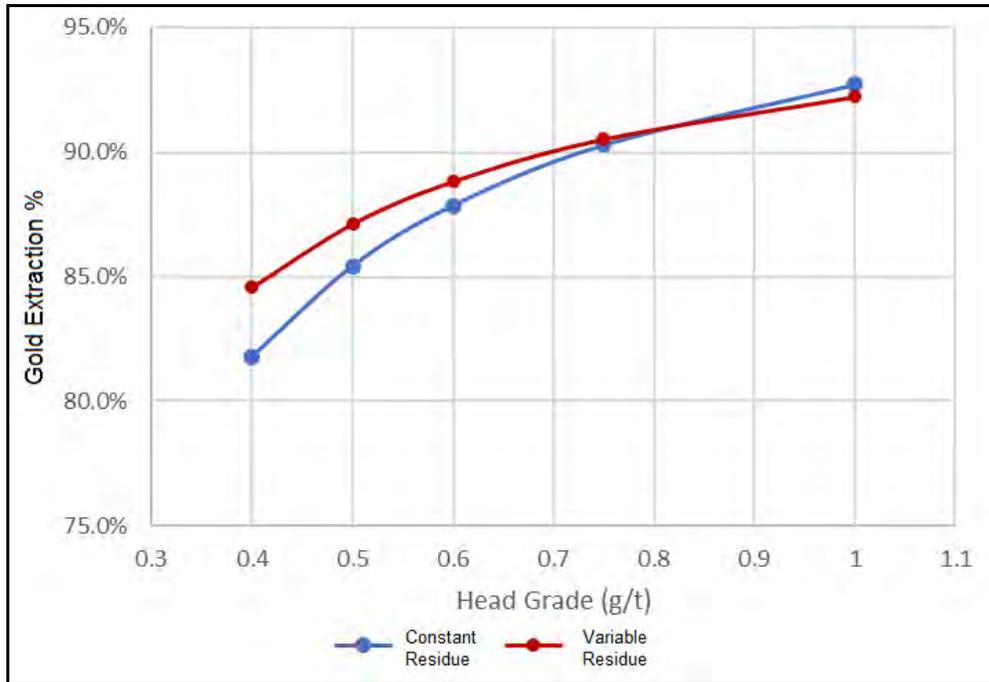


**Figure 13.30        Gold Extraction Equation for Lower Oxide Ore**
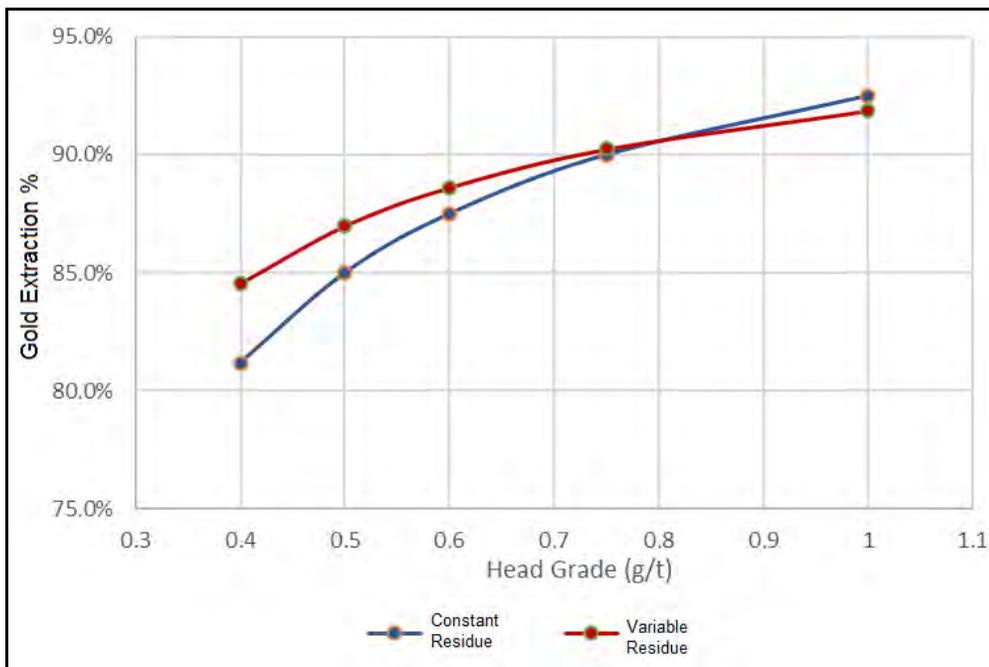
**Figure 13.31        Gold Extraction for Upper Transition Ore**
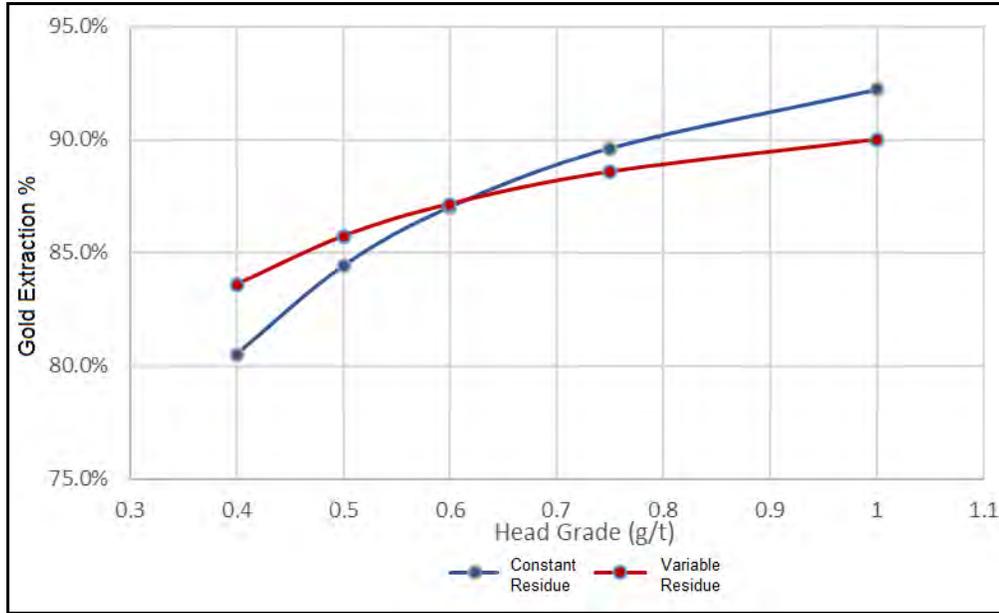


### 13.4.4    Major Reagents Consumption

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 13.49).

**Table 13.49        Estimated Lime Additions for 10.3 pH**

| Sample | CaO kg/tonne ore for 10.5 pH | Estimated CaO kg/tonne ore for 10.3 pH | Diff. (kg/tonne) 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 NaCN being maintained at 0.5 g/L 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 13.50 along with the average consumption value.

**Table 13.50          Estimation of Cyanide Consumption**

| 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** |

### 13.4.5    Summary of Metallurgical Design Criteria

A summary of the metallurgical inputs to the process design criteria is presented in Table 13.51.

**Table 13.51        Summary of Metallurgical Criteria**

| Criteria | Units | Design | Notes / Source |
|---|---|---|---|
| Plant Throughput | tpa | 4,500,000 | Agreed design throughput |
| Ore Type | - | Upper Oxide Lower Oxide Upper Trans | Mine plan |
| Ore Blend Per Mine Plan  - Oxide | % | 85 | Mine plan |
| - Transition | % | 15 | Mine plan |
| Ore Blend for Comminution Design | | | |
| - Oxide | % | 70 | Agreed design parameter |
| - Transition | % | 30 | Agreed design parameter |
| Head Grade        - Gold (Design) | g Au/t | 1.0 | Agreed design parameter |
| - Gold (LOM average) | g Au/t | 0.63 | Mine plan |
| - Silver | g Ag/t | 0.97 | Agreed design parameter |
| Gold Extraction Estimation at 1 g Au/t | | | |
| - Upper Oxide | % | 92.7 | Extraction equation |
| - Lower Oxide | % | 92.5 | Extraction equation |
| - Upper Transition | % | 92.2 | Extraction equation |
| - Per Mine Plan Ore Blend | % | 92.5 | Calculated |
| Silver Extraction Estimation | % | 50.3 | Avg. McClelland test result |
| Gold and Silver Losses Estimation at 1 g Au/t | | | |
| - Solution (solids loss equiv.) | g Au/t | 0.005 | Calc'd based on real operations |
| | g Ag/t | 0.005 | Assumption |
| - Tailings | g Au/t | 0.08 | Calculated |
| | g Ag/t | 0.48 | Calculated |
| Overall Recovery Estimation at 1 g Au/t | | | |
| - Gold | % | 92.2 | Calculated |
| - Silver | % | 49.8 | Calculated |
| Ore Specific Density | t/m³ | 2.8 | Testwork |
| Ore Bulk Density | t/m³ | 1.65 | Lycopodium/Orezone |
| Crushing Work Index (CWi, average) | kWh/t | 7.8 | Testwork |
| Bond Ball Mill Work Index (BWi, average) | kWh/t | 5.4 | Testwork |
| Bond Abrasion Index (Ai) | | 0.035 | Testwork |
| Grind Size $P_{80}$ | μm | 125 | Lycopodium |
| CIL Circuit Residence Time | hrs | 24 | Based on fast leach kinetics in testwork |
| CIL Slurry Density (for saprolitic ore) | % solids | 40.8% | Lycopodium |
| Thickener Solids Loading | t/m²·h | 0.60 | Based on Outotec testwork |
| Sodium Cyanide Consumption | kg NaCN/t | 0.28 | Calculated |
| - Consumption per Blended Ore | kg NaCN/t | 0.19 | Testwork |
| - NaCN Loss to Tails | kg NaCN/t | 0.17 | Calculated based on 60 ppm CN⁻ in Tails |
| - Recycled NaCN in Process Water | kg NaCN/t | 0.08 | Calculated |
| Lime Consumption (CaO at 90% purity) | kg/t | 2.07 | Testwork/Calculated |

## 13.5    Conclusions and Recommendations

### 13.5.1    Conclusions

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

- Oxide and transition ores at Bomboré are readily amenable to whole ore cyanidation.

- Gold recoveries are expected to be 90% plus for head grades over 0.75 g Au/t, high 80%'s for head grades of 0.5 to 0.75 g Au/t, and low 80%'s for head grades of 0.3 to 0.5 g Au/t.

- Optimum grind size was determined to be $P_{80}$ 125 µm based on grind size and recovery relationship.

- Leach extraction rates are essentially complete within 24 hours based on the observed leach kinetics.

- Cyanide consumption rates are expected to be low, averaging about 0.19 kg NaCN/t ore.

- Lime consumption rates are expected to be moderate, averaging at 2 kg/t ore for 90% CaO quicklime.

### 13.5.2    Recommendations

It is recommended that during plant operations:

- Natural cyanide attenuation (free and WAD) be monitored in the tailings storage facility.

- Site water quality (raw and process) be monitored during the initial wet and dry seasons to document the seasonal impact of water quality

- Gold adsorption rate and equilibrium loading on carbon be monitored as the plant head grade varies during the life of the operation to ensure that carbon movement and management is optimized.

## 14.0 MINERAL RESOURCE ESTIMATES

### 14.1 Summary

At Bomboré, 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 north-south, 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 four separate block model areas, referred to as the North, South, P16, and P17 areas. Together, the North and South block models contain the majority of the Mineral Resources. Low-grade mineralized wireframe models were created for the September 2016 estimate using a grade threshold of approximately 0.20 g Au/t, and a high-grade mineralized wireframe models that were created using a grade threshold of approximately 0.45 g Au/t.

Following completion of the September 2016 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 Au/t to capture material remaining outside the September 2016 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 newly constructed 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. 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 Au/t to 48.97 g Au/t for the low and high-grade mineralized wireframe domain assays for the September 2016 estimate and a universal value of 5.00 g Au/t 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 one metre length.

Gold grades within the September 2016 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 September 2016 and 2017 estimate wireframe models (low-grade, high-grade) for the P17 model area were estimated using the inverse distance squared (ID2) 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 ordinary kriging (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 ID2 interpolation algorithm. The first step used only composites outside wireframes and above 0.20 g Au/t to flag blocks with a grade above 0.00 g Au/t 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.

Data from 276 new holes totalling 15,387 m, located 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 resource model is still appropriate to be used as the basis for the 2018 FS 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 per ounce 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.

At a cut-off grade of approximately 0.2 g Au/t for oxide and transition material and 0.38 g Au/t for fresh material, the updated M&I Mineral Resources are estimated to total 218.1 Mt at an average grade of 0.68 g Au/t for 4.8 million ounces of contained gold (Table 14.1). Using the same cut-off grades, Inferred Mineral Resources are estimated to total an additional 48.2 Mt at an average grade of 0.64 g Au/t for 994,000 ounces of contained gold.

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

| Material Type | Measured Tonnes Mt | Measured Grade g Au/t | Measured Gold koz | Indicated Tonnes Mt | Indicated Grade g Au/t | Indicated Gold koz | M+I Tonnes Mt | M+I Grade g Au/t | M+I Gold koz | Inferred Tonnes Mt | Inferred Grade g Au/t | Inferred Gold koz |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Oxide+Tran HG | 16.9 | 0.94 | 513 | 36.5 | 0.83 | 974 | 53.4 | 0.87 | 1,487 | 4.8 | 0.77 | 117 |
| Oxide+Tran LG | 18.5 | 0.33 | 196 | 50.1 | 0.33 | 531 | 68.6 | 0.33 | 727 | 16.4 | 0.29 | 151 |
| **Total Oxide+Tran** | **35.4** | **0.62** | **709** | **86.7** | **0.54** | **1,505** | **122.0** | **0.56** | **2,214** | **21.2** | **0.39** | **268** |
| Fresh HG | 2.3 | 118 | 87 | 68.7 | 0.96 | 2,121 | 71.0 | 0.97 | 2,208 | 20.1 | 0.97 | 630 |
| Fresh LG | 0.8 | 0.43 | 11 | 24.2 | 0.43 | 337 | 25.0 | 0.43 | 348 | 6.9 | 0.43 | 96 |
| **Total Fresh** | **3.1** | **0.99** | **97** | **93.0** | **0.82** | **2,458** | **96.0** | **0.83** | **2,556** | **27.0** | **0.84** | **726** |
| **Total HG** | **19.2** | **0.97** | **600** | **105.3** | **0.91** | **3,095** | **124.5** | **0.92** | **3,695** | **24.9** | **0.93** | **747** |
| **Total LG** | **19.2** | **0.33** | **206** | **74.4** | **0.36** | **868** | **93.6** | **0.36** | **1,075** | **23.3** | **0.33** | **246** |
| **Total HG+LG** | **38.4** | **0.65** | **806** | **179.6** | **0.69** | **3,964** | **218.1** | **0.68** | **4,770** | **48.2** | **0.64** | **994** |

*Notes:*

1. *CIM (2014) definitions were followed for Mineral Resources.*

2. *HG indicates high-grade resources above the higher cut-offs (0.45 g Au/t for oxide and transition, and 0.5 g/t for fresh), LG indicates low-grade material between the breakeven cut-off grades (0.2 g Au/t for oxide and transition, and 0.38 g/t for fresh) and HG cut-offs.*

3. *Mineral Resources are estimated at variable cut-off grades depending on weathering layer and location; cut-off grades are approximately 0.2 g Au/t for oxide and transition material, and 0.38 g Au/t for fresh material.*

4. *Mineral Resources are estimated using a long-term gold price of US$1,400 per ounce.*

5. *A minimum mining width of approximately 3 m was used.*

6. *Bulk density varies by material type.*

7. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*

8. *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 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-south, to northwest, 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**



Figure 14-1

Orezone Gold Corporation

*Bomboré Gold Project*
*Burkina Faso, West Africa*
**Location of Gold Mineralized Zones and Block Models**
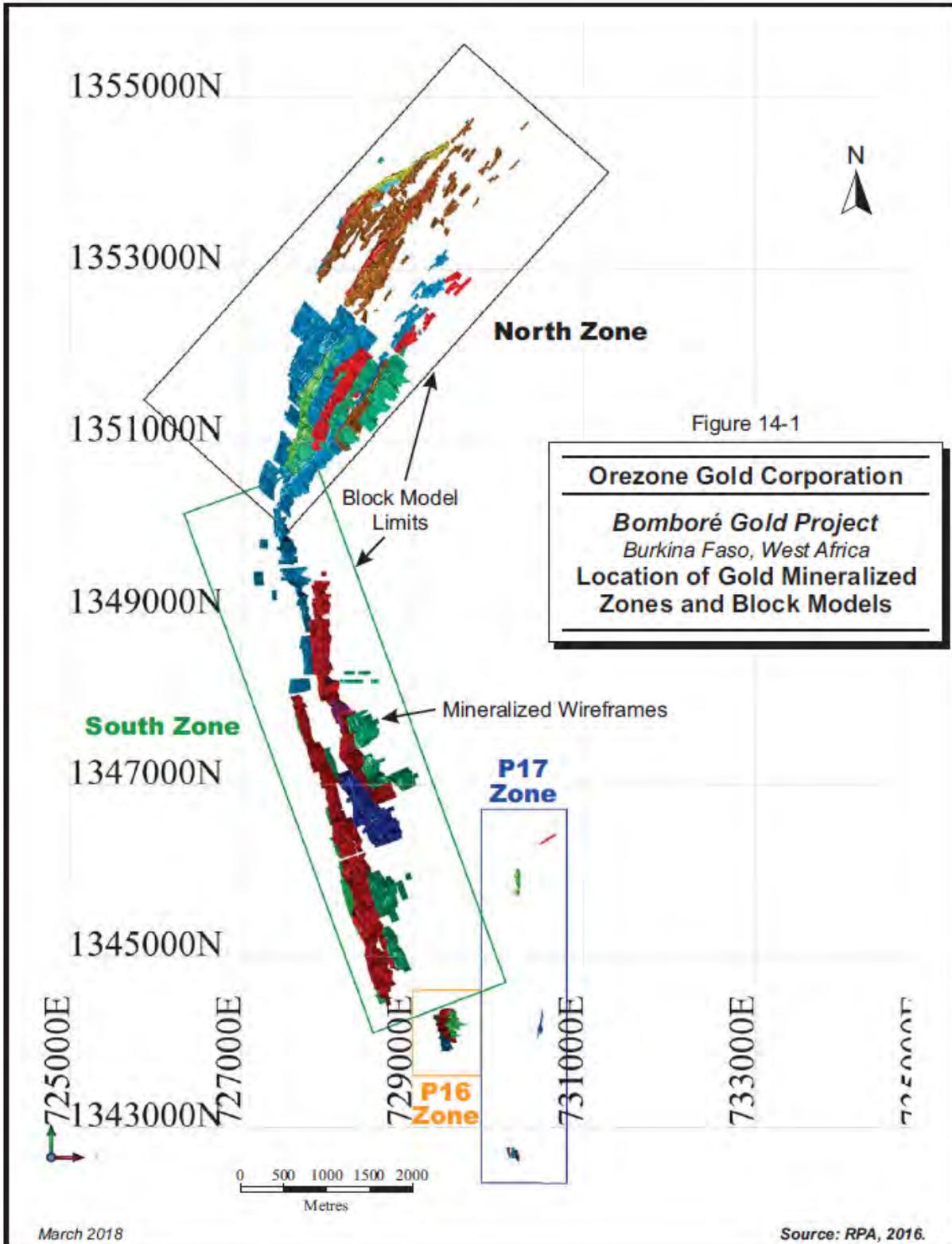
The gold mineralization along the 12 km strike length has been split into four separate block model areas in order to keep the size of the various block model files within functionally manageable limits. The block model areas are referred to as the North, South, P16, and P17 areas. Together, the North and South block models contain the majority of the 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. Both software packages used the MS Access database format for collecting and storing the drill hole database information.

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 January 5, 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 |

| Table Name | Data Type | Table Type | Records |
|---|---|---|---|
| Drillholes_structure_fract_jts | Point | time-independent | 0 |
| 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 |

| Table Name | Data Type | Table Type | Records |
|---|---|---|---|
| Drillholes_comp450ppb | interval | time-independent | 225 |
| 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**
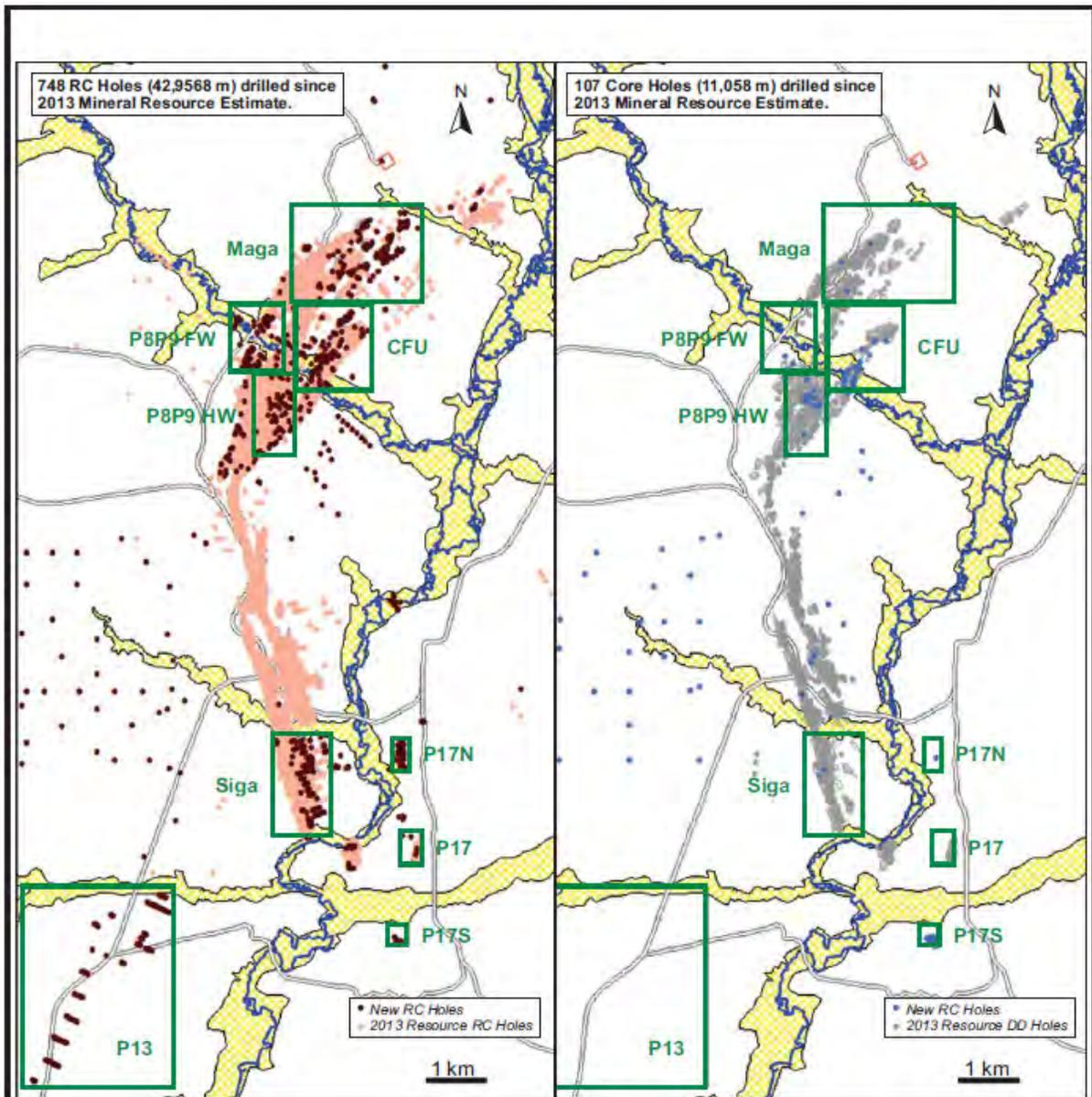
The spacing of the drill holes varies for each of the four model areas and is summarized in Table 14.3.

**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 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 profile has been divided into four major units, with sub-divisions as shown below.

1. Regolith
   a) Oxidized
   b) Upper
   c) Lower
2. Transition
   a) Upper
   b) Lower
3. Fresh rock
   a) Upper (<25 m)
   b) Lower (>25 m)

Digital surface models for each of these weathering zones have been created from the visual information collected during logging of the drill core or RC chips and from geochemical 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**
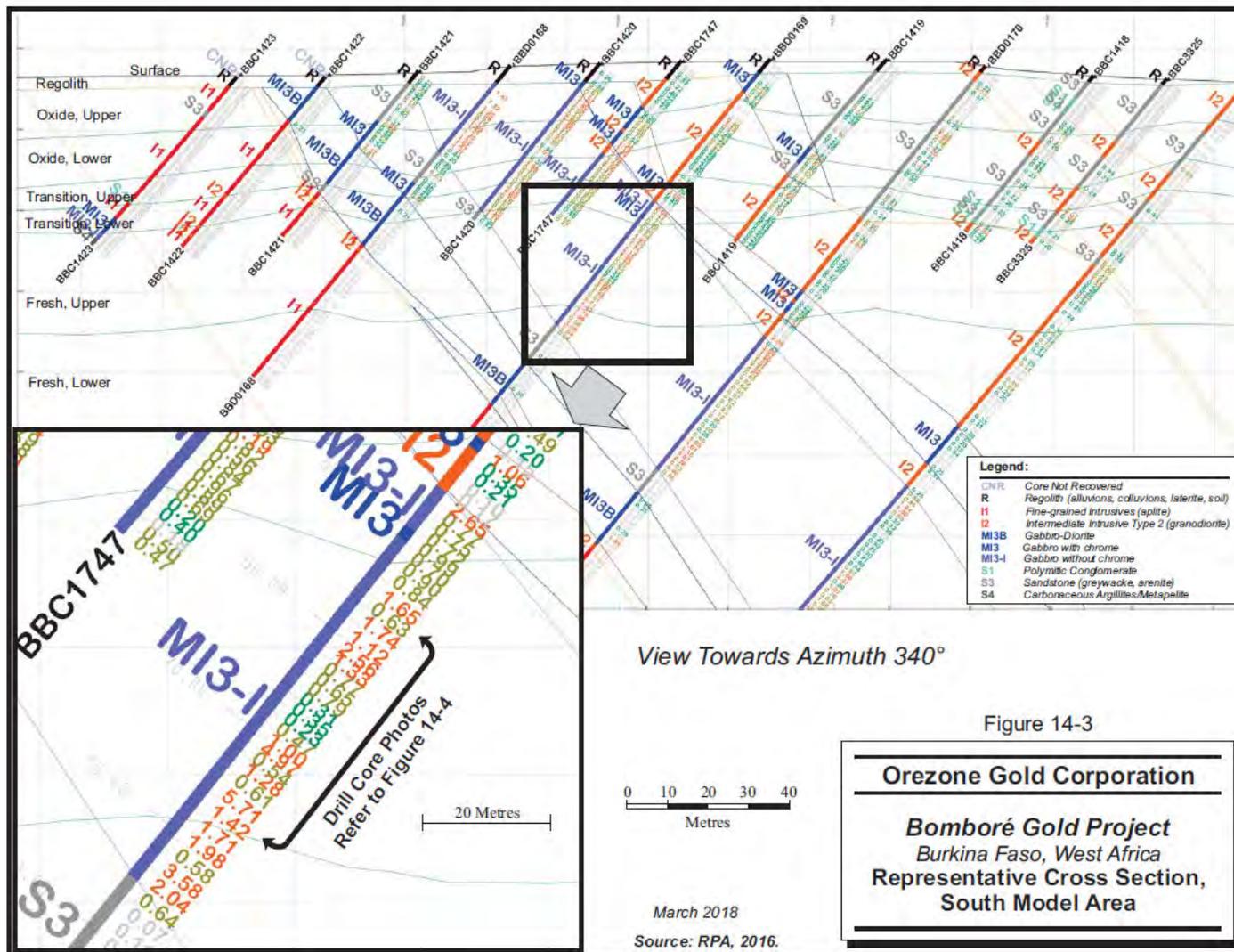
**Figure 14.4    Mineralized Gabbro, Drill Hole BBD0169, South Zone Area**

**Figure 14.5          Representative Cross Section, North Zone Area**

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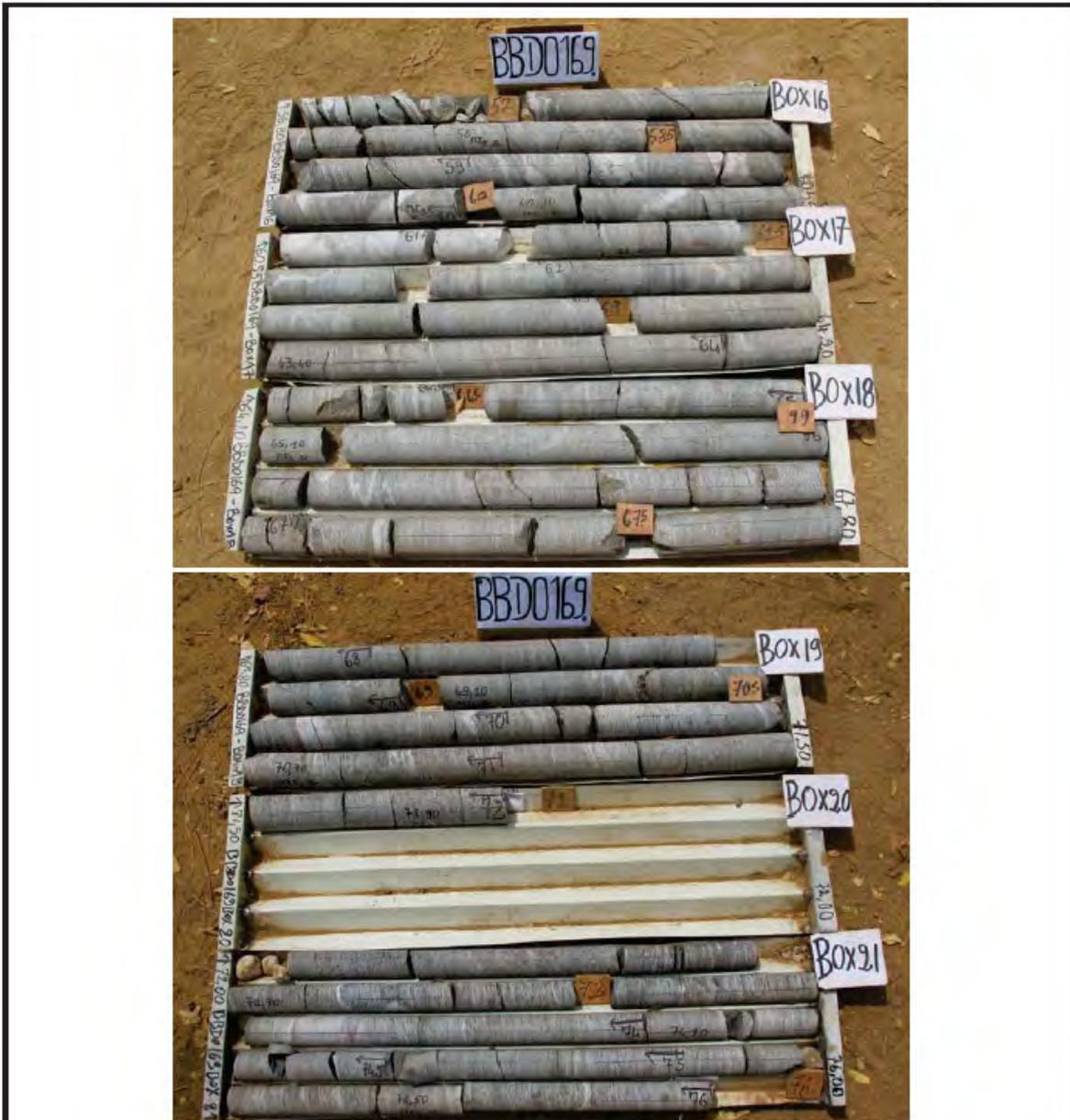


Figure 14-6

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 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 five metres was used for creation of the September 2016 estimate low-grade domains.  A minimum width of three metres was used for the creation of the new low grade domains for the January 2017 estimate.  Sample intervals with assay results less than the nominated cut-off grade (internal dilution) were included within the September 2016 estimate high and low-grade mineralized wireframes if the length was less than 1.5 m.  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 Au/t.

Both the September 2016 estimate and January 2017 estimate low-grade domain models were created using a lower grade limit of approximately 0.20 g Au/t, while the September 2016 estimate high-grade domain models were created using a grade limit of approximately 0.45 g Au/t.  RPA considers the selection of 0.20 g Au/t 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.

For the January 2017 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 Au/t can be observed to be present in adjacent drill sections and there is a minimum of two such intersects in at least one section.

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 so as 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**
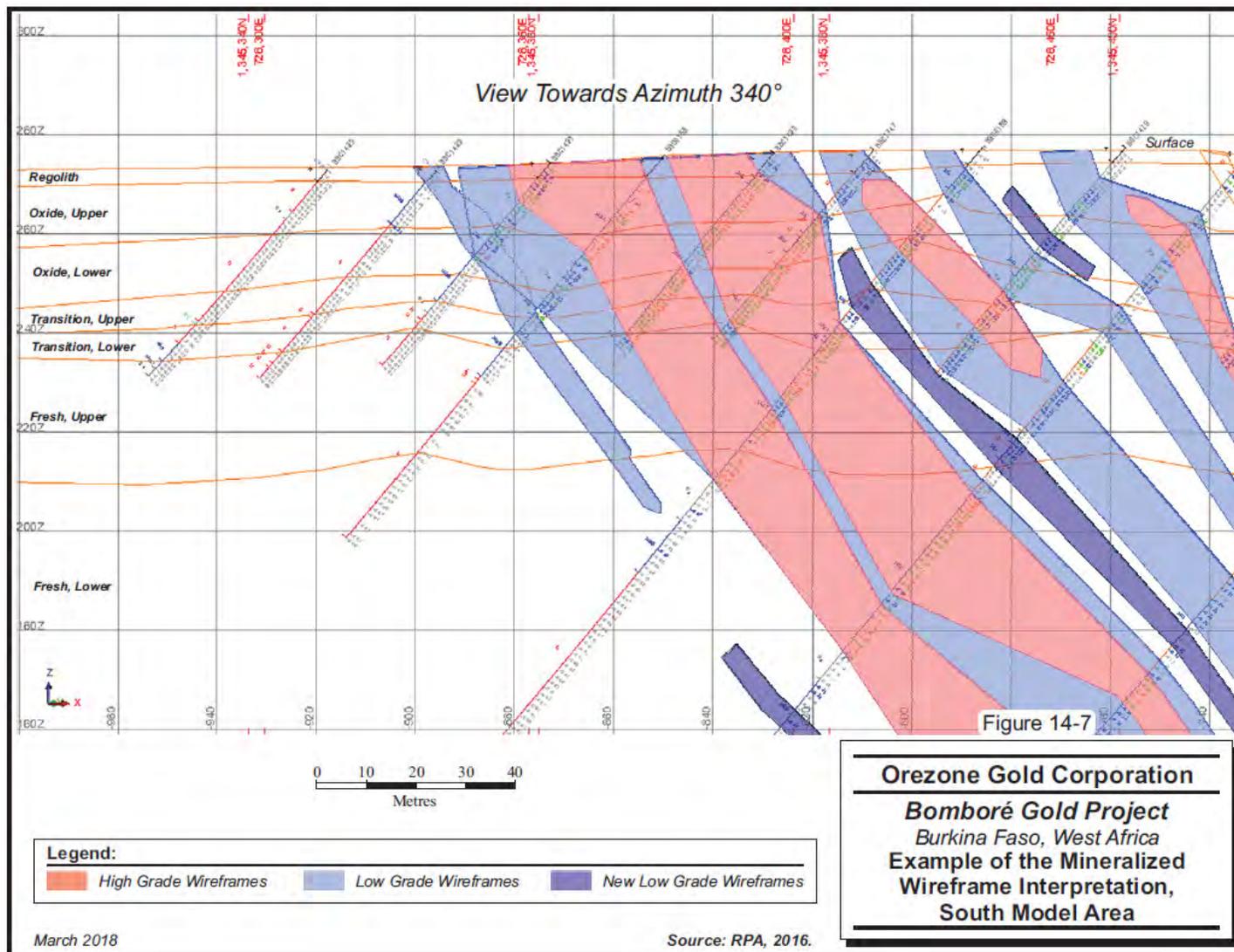
**Figure 14.8      Example of the Mineralized Wireframe Interpretation, North Model Area**



Figure 14-8

Orezone Gold Corporation

*Bomboré Gold Project*

*Burkina Faso, West Africa*

**Example of the Mineralized Wireframe Interpretation, North Model Area**

March 2018

Source: RPA, 2016.
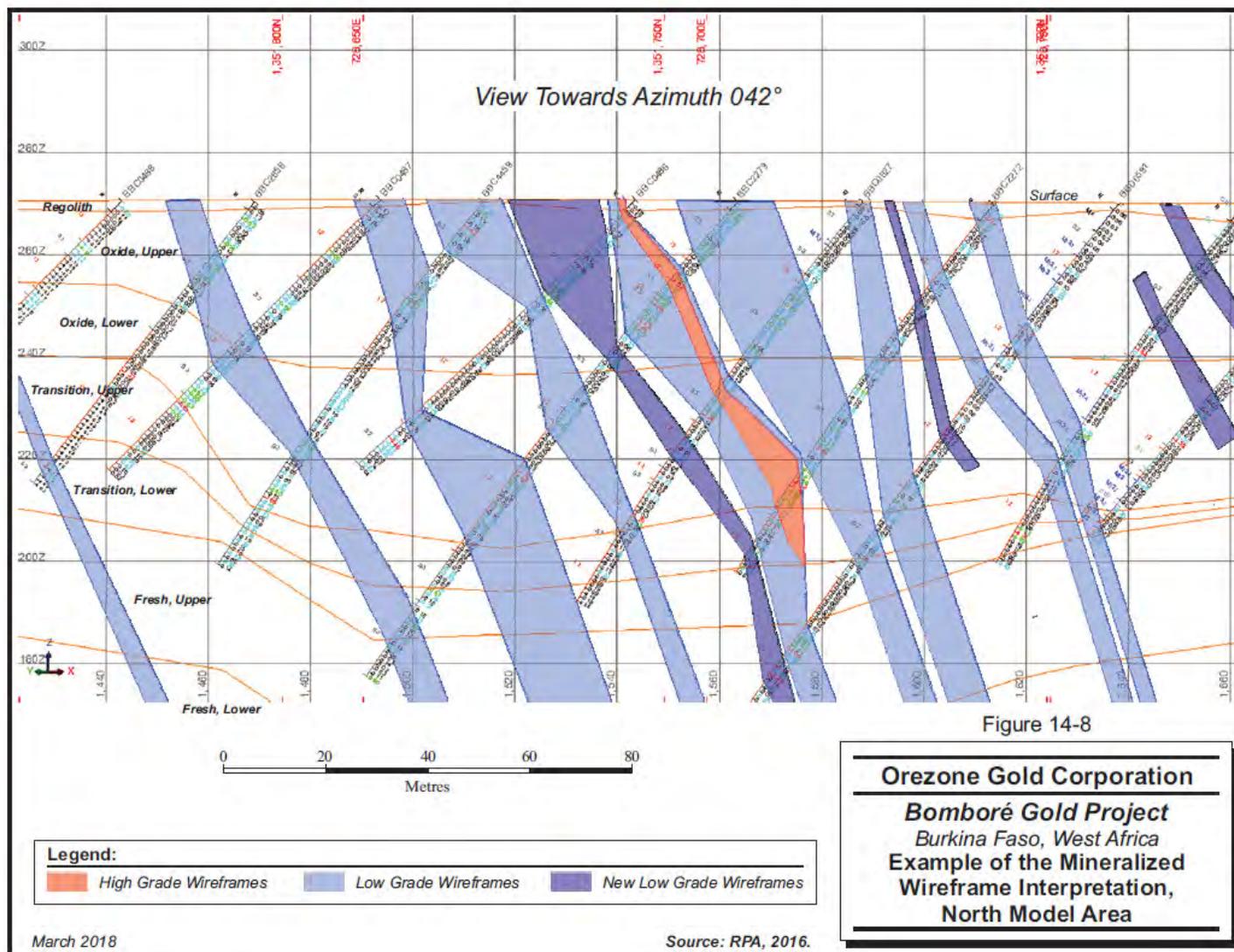
## 14.4     Mined Out Areas

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. Artisanal miners (orpailleurs) were essentially exploiting accessible quartz veins. 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 currently allowed only outside of the areas where the Project's 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 kilometers, and 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 ferruginous 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 often absent, 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 and P17 gold deposits are the only ones where a relatively large pit has been developed. |

**Figure 14.9**          **Location of Artisanal Mining Sites**
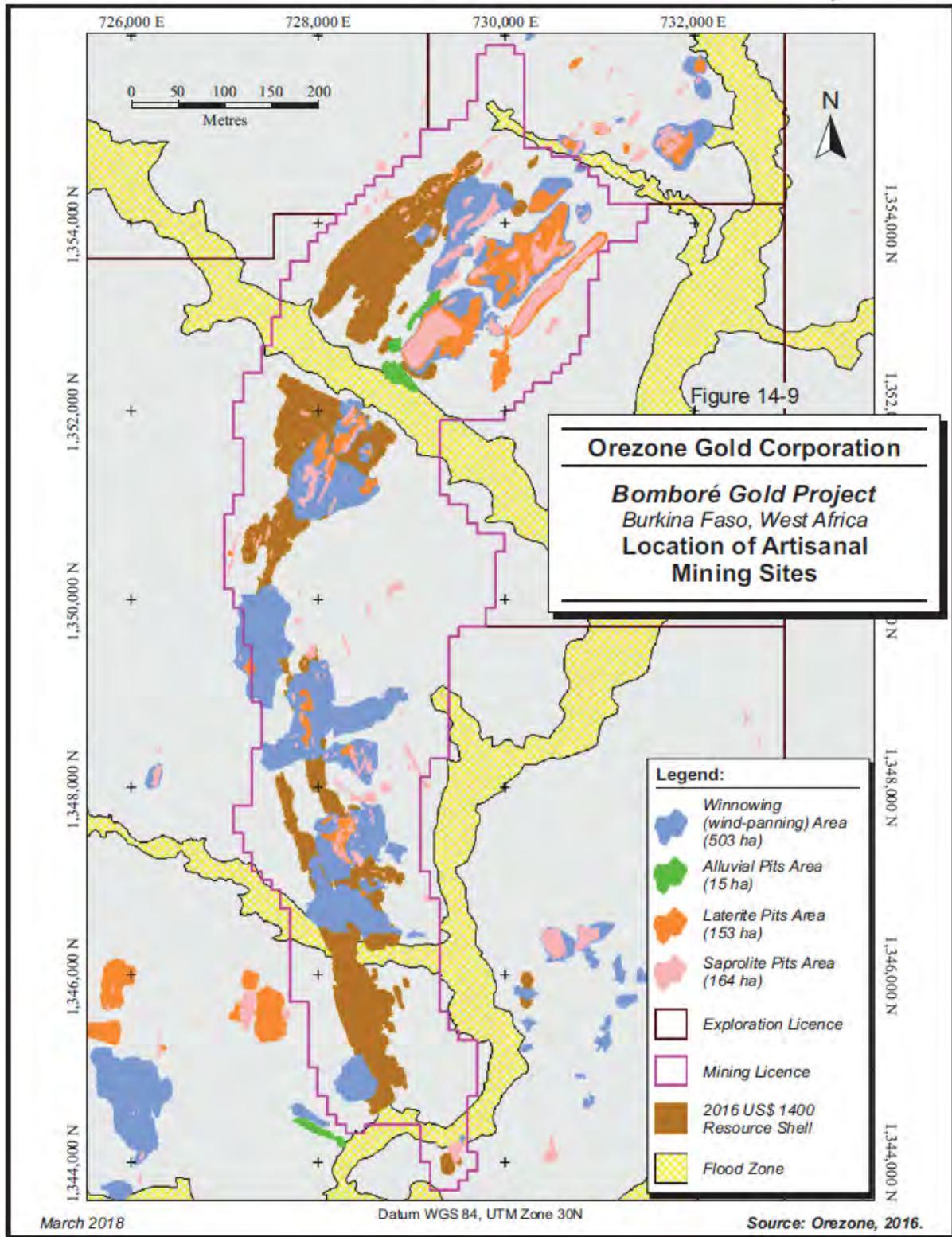
## 14.5　Topography Models

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 consisting mainly of drill hole collars surveyed with the Trimble TSC3 series instrument and earlier instruments with recalculated co-ordinates, including 20 control points within the PhotoSat topographic model area.　These control points were used to geo-reference the November 2012 re-processed topographic model.　In 2014, Geozone Geomatics validated the Project survey data against the PhotoSat topographic surface.

## 14.6　Sample Statistics and Grade Capping

The September 2016 estimate mineralization wireframe models and the new January 2017 mineralization wireframe models were used to code the drill hole database and identify those samples within all the mineralized wireframes.　All remaining samples 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 September 2016 estimate low and high-grade domain mineralized wireframes in all four block model areas.　A total of 15,003 samples were contained within the January 2017 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 Au/t) | 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 | 1,4707 |
| 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 (inc. 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 |
| Siga East_450fw | 0.83 | 1.44 | 1.74 | 0.01 | 28.00 | 526 |

| Wireframe | Mean (g Au/t) | StDev | CoV | Min | Max | Count |
|---|---|---|---|---|---|---|
| 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 (inc. 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 (inc. 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 (inc. 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 coefficient of variation (CoV) of less than approximately 2.0. For the September 2016 estimate the capping values were selected separately for the high-grade (0.45 g Au/t) and low-grade (0.20 g Au/t) domains for the North, South, and P16 block model areas, partially guided by the goal of achieving a target coefficient of variation (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 Au/t was applied 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 Au/t) | 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 (incl. 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 Au/t) | 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 September 2016 estimate a composite length of 1.5 m was used for the North, South, and P16 block model areas while a composite length of one metre was used for the P17 block model area.  The January 2017 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 September 2016 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.

As often happens, 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 Au/t) | 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 |
| P8/P9_450main_c | 1.05 | 1.15 | 1.09 | 0 | 15.88 | 3,359 |

| Wireframe | Mean (g Au/t) | StDev | CoV | Min | Max | Count |
|---|---|---|---|---|---|---|
| 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 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.

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**
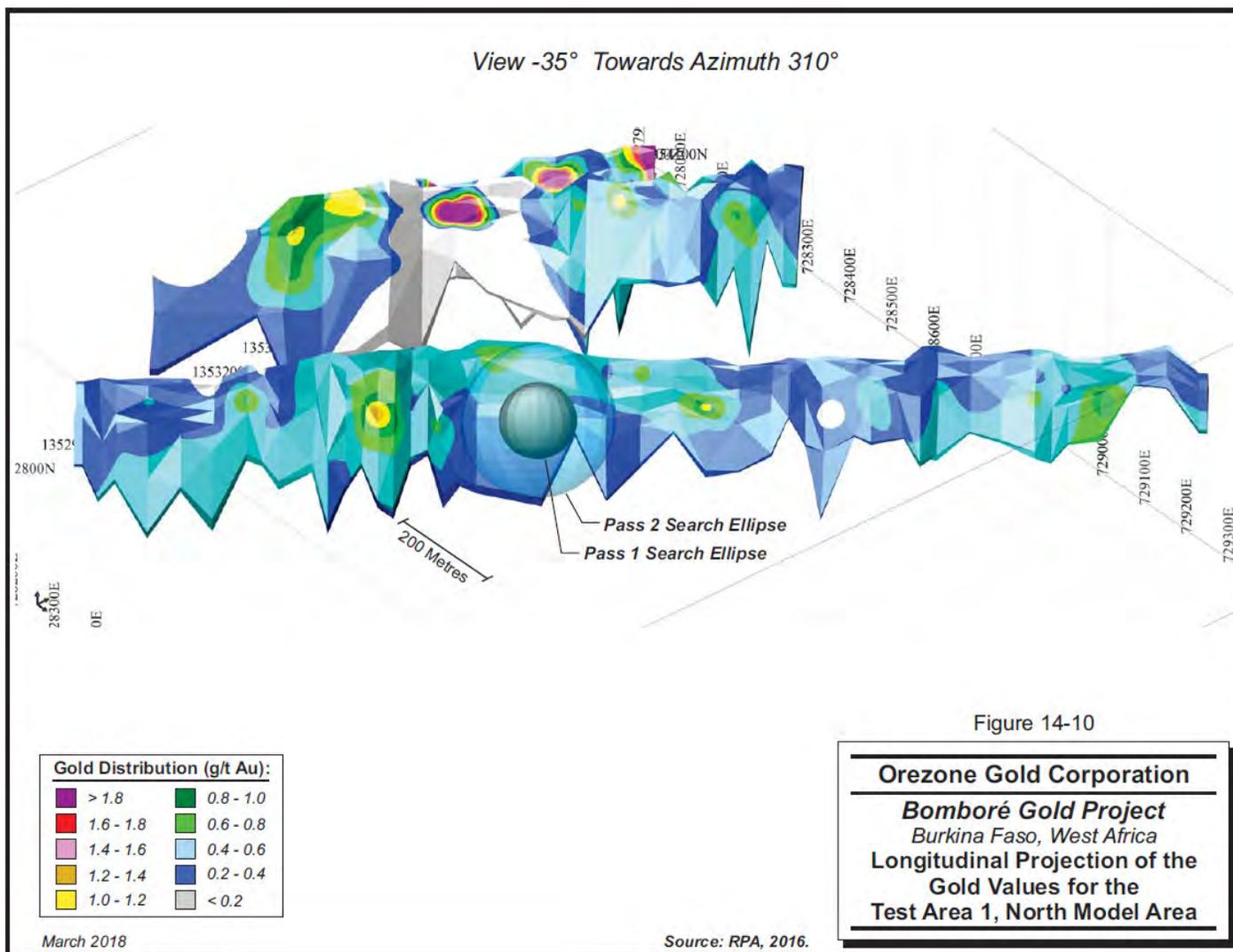
**Figure 14.11          Longitudinal Projection of the Gold Values for Test Area 2, North Model Area**
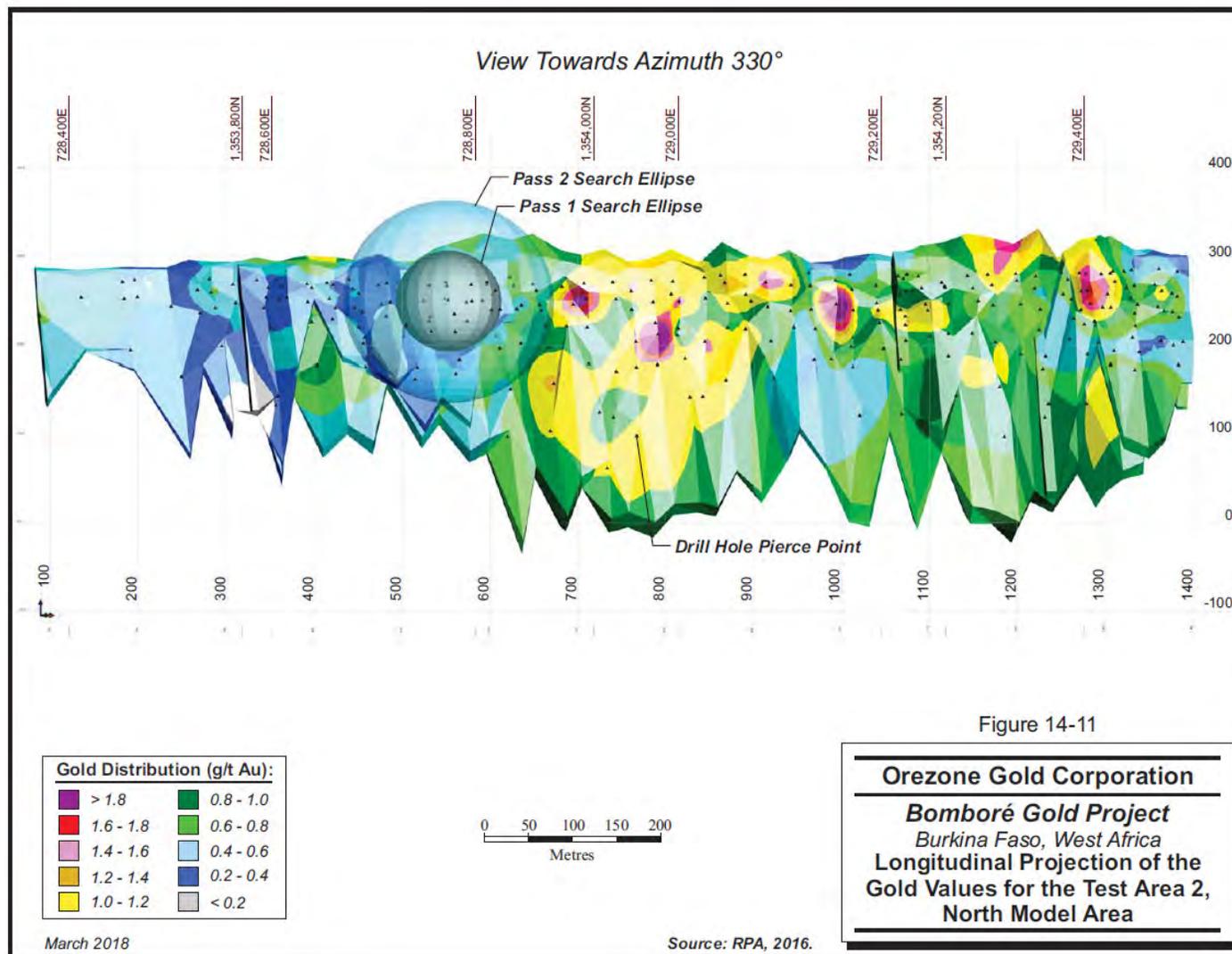
### 14.9.2    South Model Area

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**
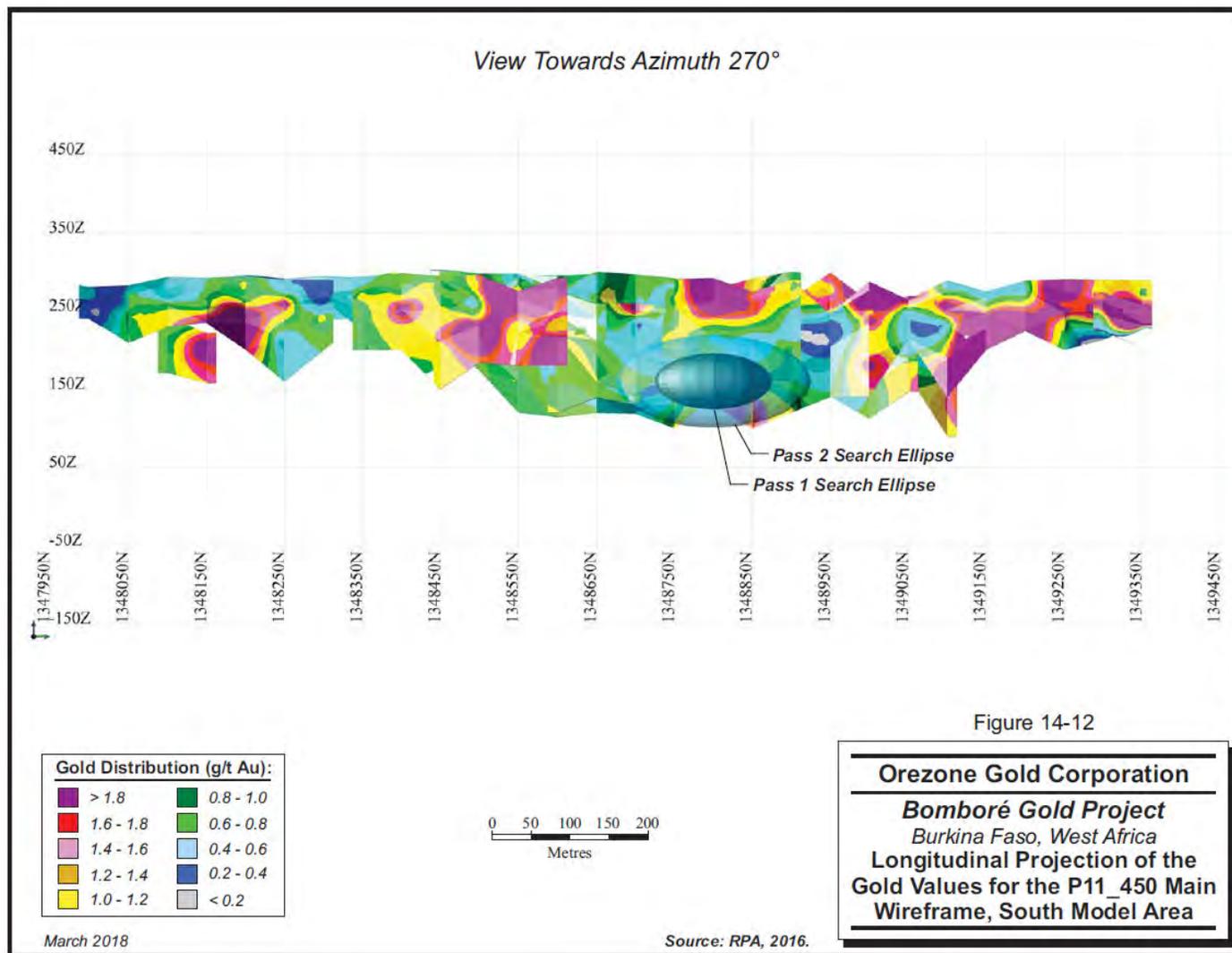
**Figure 14.13**        Longitudinal Projection of the Gold Values for the Siga SW_450 Main Wireframe, South Model Area
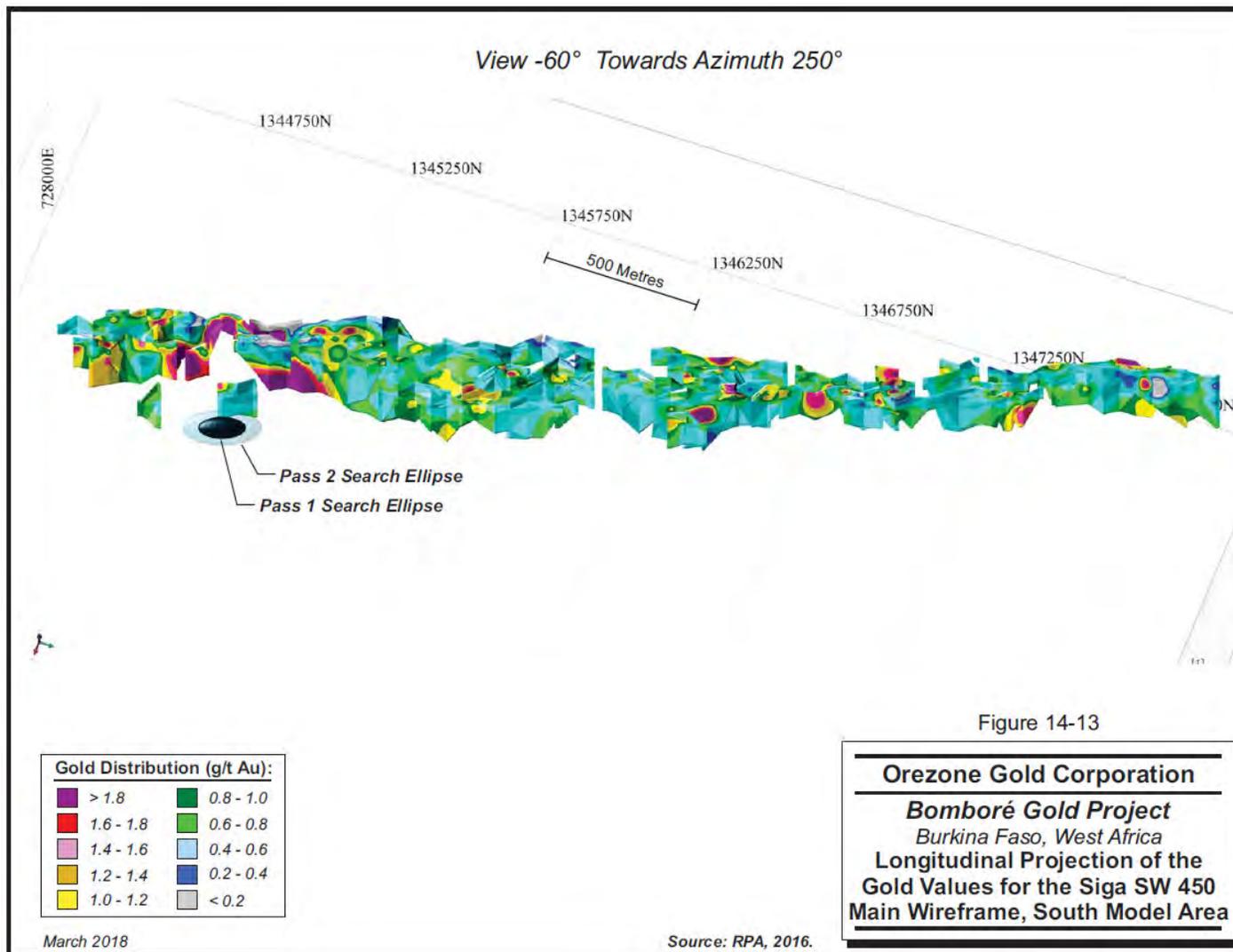
### 14.9.3    Other Areas

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 January 2017 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 September 2016 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 estimates. Gold grades were estimated using the OK interpolation algorithm for the January 2017 estimate within the new low-grade 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 ID3 interpolation algorithm which on the first step used only composites outside wireframes and above 0.2 g Au/t to flag blocks with a grade above 0.00 g Au/t 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, ID3 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<br>2: 0.27<br>3: 0.27 | 1: 0.441<br>2: 0.441<br>3: 0.441 | Same as South | N/A |
| Sill (Pass No., C2) | 1: 0.22<br>2: 0.22<br>3: 0.22 | 1: 0.19<br>2: 0.19<br>3: 0.19 | Same as South | N/A |
| Sill (Pass No., C3) | 1: 0.22<br>2: 0.22<br>3: 0.22 | 1: 0.129<br>2: 0.129<br>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<br>2: 4<br>3: Ellipsoid | 1: 2<br>2: 4<br>3: Ellipsoid | 1: 2<br>2: 4<br>3: Ellipsoid | 1: 2<br>2: Ellipsoid |
| Orientation | Varies by wireframe | Varies by wireframe | Varies by wireframe | Varies by wireframe |
| Length of Major Axis (Pass No., m) | 1: 60<br>2:120<br>3: 6,000 | 1: 75<br>2: 125<br>3: 300 | 1: 30<br>2: 60<br>3: 1000 | 1: 30<br>2: 60 |
| Anisotropy Ratio (Major/Semi-Major) | 1: 1<br>2: 1<br>3: 1 | 1: 2<br>2: 2<br>3: 2 | 1: 3<br>2: 3<br>3: 2 | 1: 1<br>2: 1 |
| Anisotropy Ratio (Major/Minor) | 1: 4<br>2: 4<br>3: 5 | 1: 4<br>2: 6<br>3: 6 | 1: 4.2<br>2: 6<br>3: 6 | 1: 8<br>2: 8 |
| Minimum Number of Samples | 1: 6<br>2: 4<br>3: 2 | 1: 4<br>2: 4<br>3: 2 | 1: 4<br>2: 2<br>3: 2 | 1: 5<br>2: 2 |
| Maximum Number of Samples | 1: 22<br>2: 16<br>3: 12 | 1: 22<br>2: 22<br>3: 16 | 1: 16<br>2: 12<br>3: 12 | 1: 10<br>2: 15 |
| Maximum Number of Samples per Hole | 1: 4<br>2: 4<br>3: 2 | 1: 4<br>2: 4<br>3: N/A | 1: 4<br>2: 4<br>3: N/A | 1: 3<br>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<br>2:35 | 1: 10<br>2: 35 | 1: 10<br>2: 35 | 1: 10<br>2: 35 |
| Anisotropy Ratio (Major/Semi-Major) | 1: 0.666<br>2: 1 | 1: 0.666<br>2: 1 | 1: 0.666<br>2: 1 | 1: 0.666<br>2: 1 |
| Anisotropy Ratio (Major/Minor) | 1: 4<br>2: 14 | 1: 4<br>2: 14 | 1: 4<br>2: 14 | 1: 4<br>2: 14 |
| Minimum Number of Samples (Block Tag) | 1: 2<br>2: 2 | 1: 2<br>2: 2 | 1: 2<br>2: 2 | 1: 2<br>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 per 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 Mineral Resources.

**Figure 14.14**     **Swath Plot by Northing for the P8/P9 High Grade Domains, North Model Area**
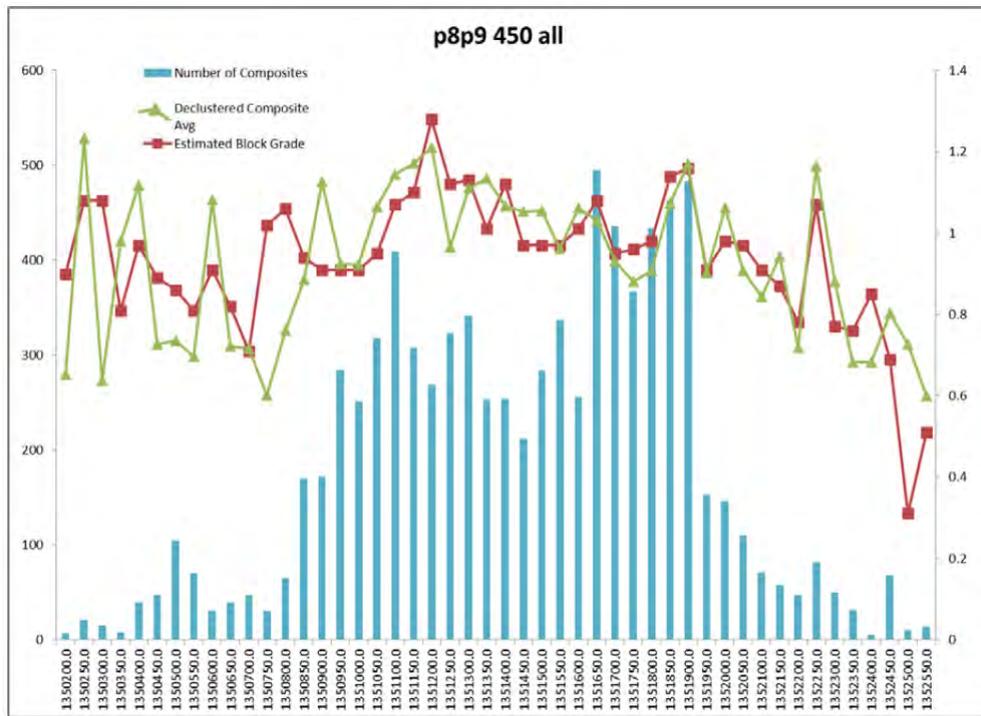


**Figure 14.15**     **Swath Plot by Northing for the Siga SW High Grade Domains, South Model Area**
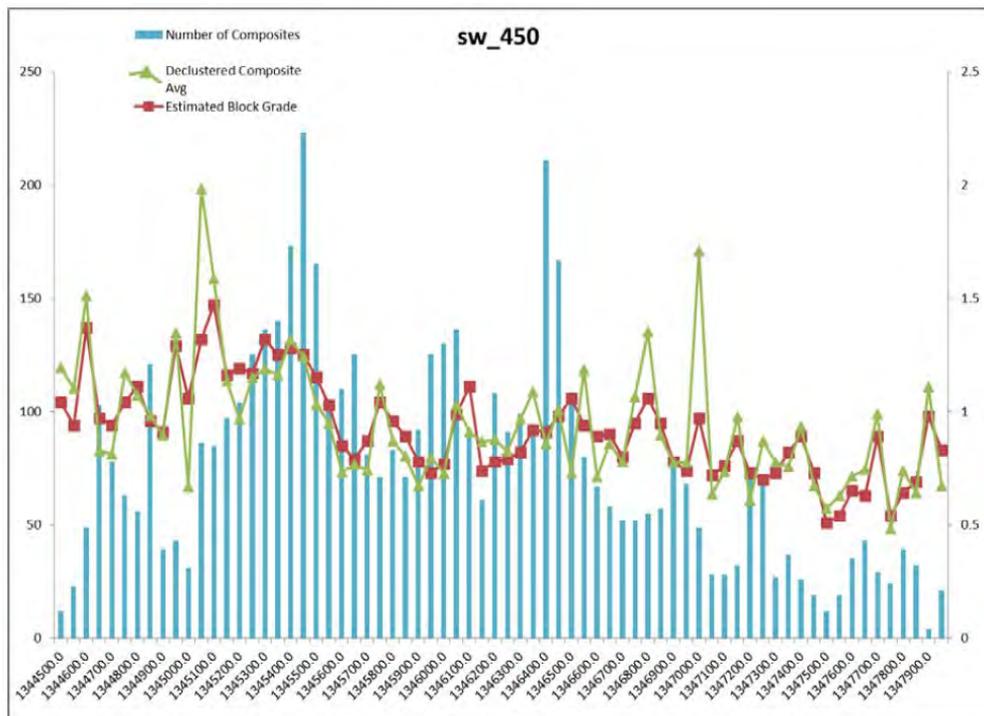
**Figure 14.16**     **Bomboré North Zone, New Low-Grade Domain Swath Plot (by Eastings), Rotated 40° Anti-clockwise**
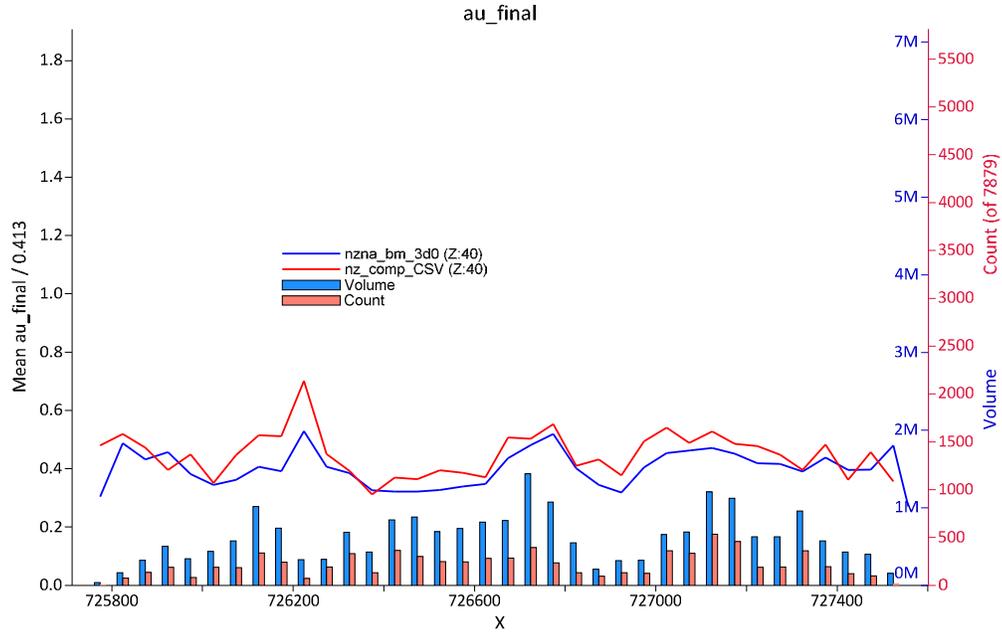


**Figure 14.17**     **Bomboré North Zone, New Low-Grade Domain Swath Plot (by Northings), Rotated 40° Anti-clockwise**
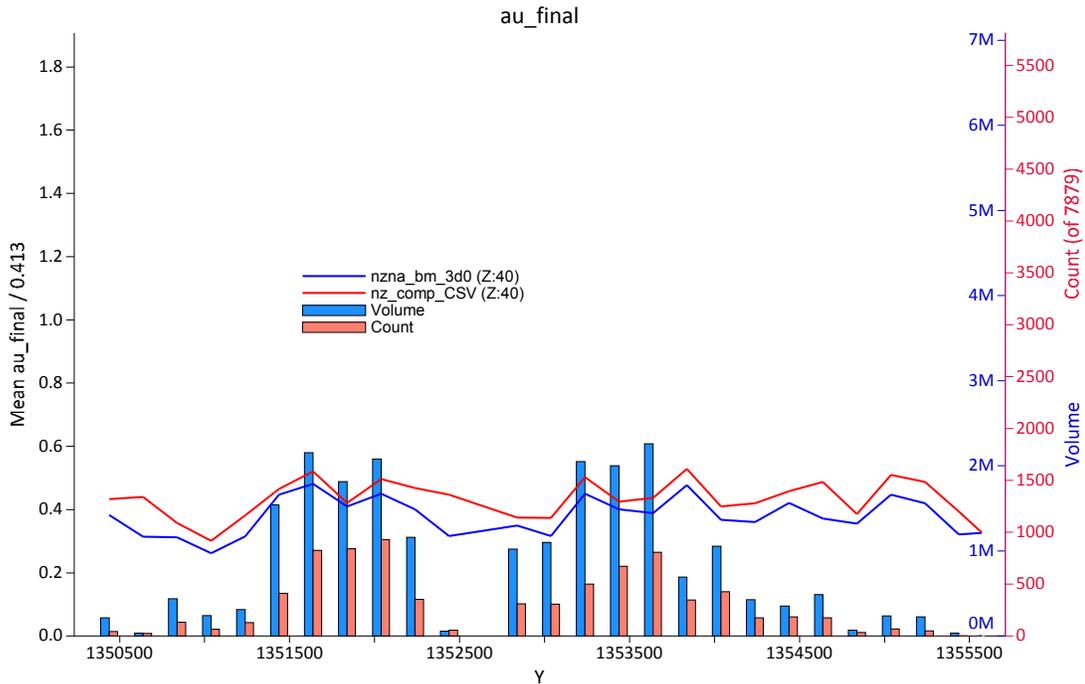
**Figure 14.18    Bomboré North Zone, New Low-Grade Domain Swath Plot (by Elevation)**
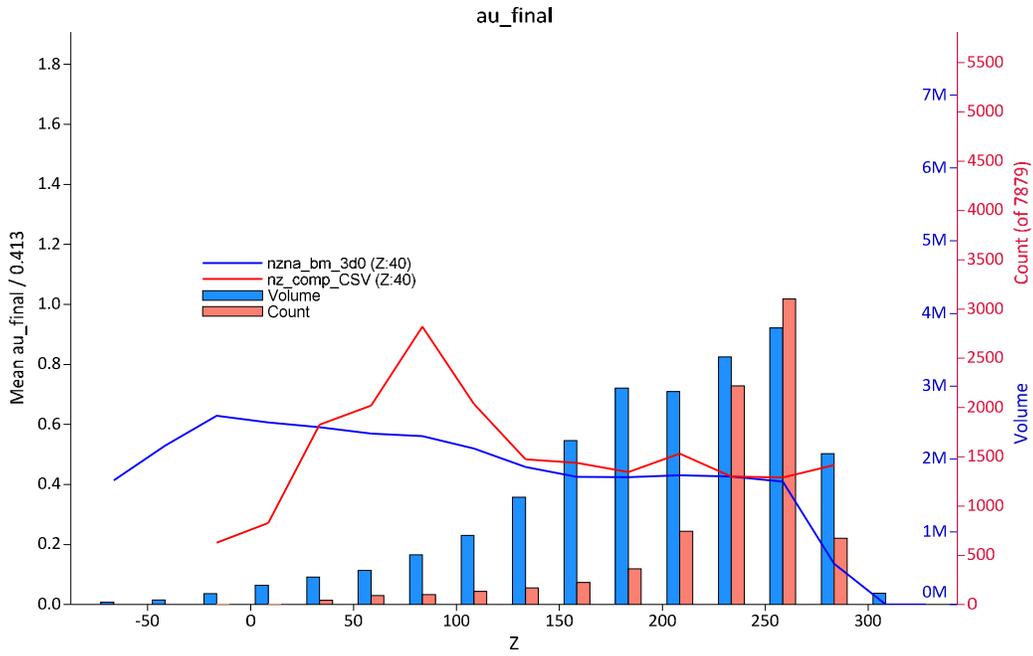


**Figure 14.19    Bomboré South Zone, New Low-Grade Domain Swath Plot (by Eastings), Rotated 13° Clockwise**
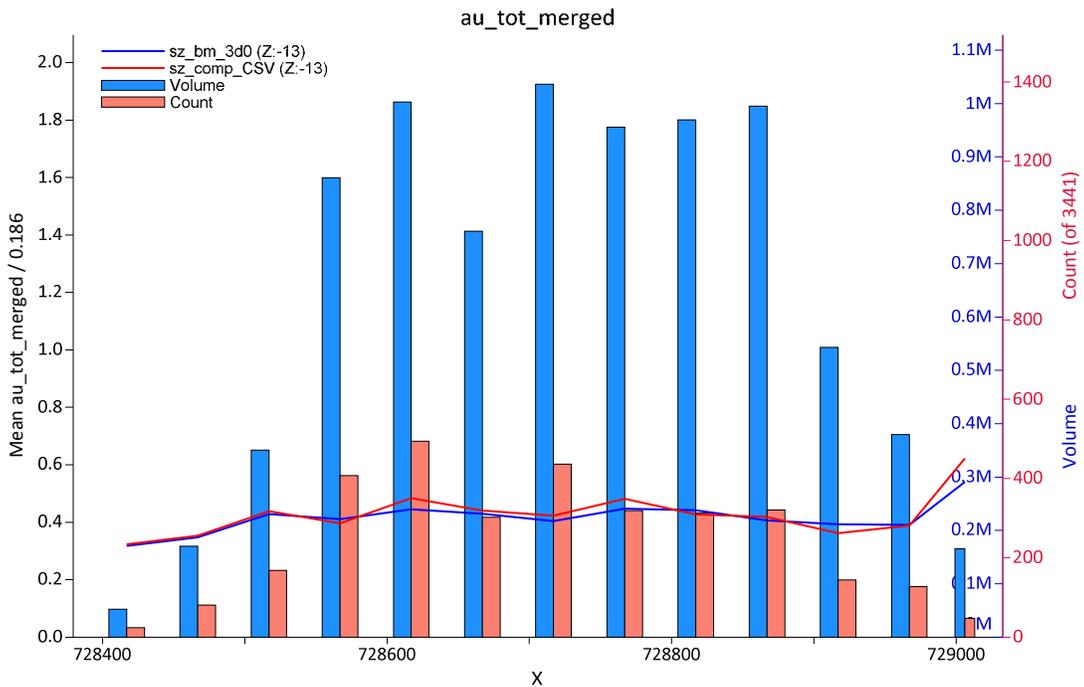
**Figure 14.20        Bomboré South Zone, New Low-Grade Domain Swath Plot (by Northings), Rotated 13° Clockwise**
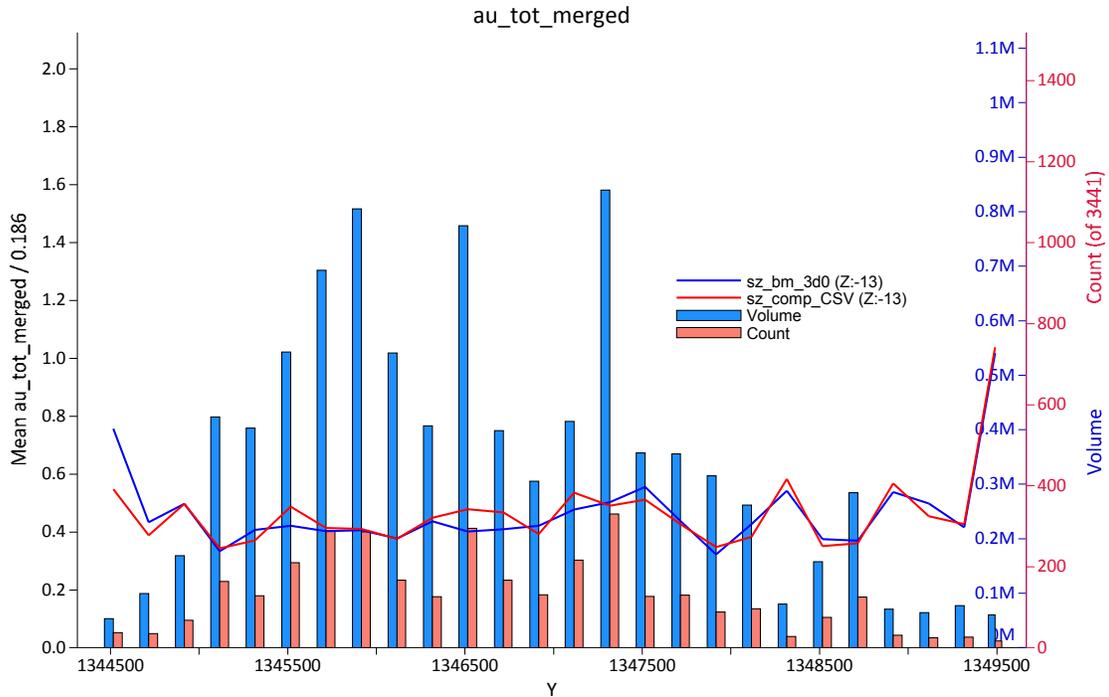


**Figure 14.21        Bomboré South Zone, New Low-Grade Domain Swath Plot (by Elevation)**
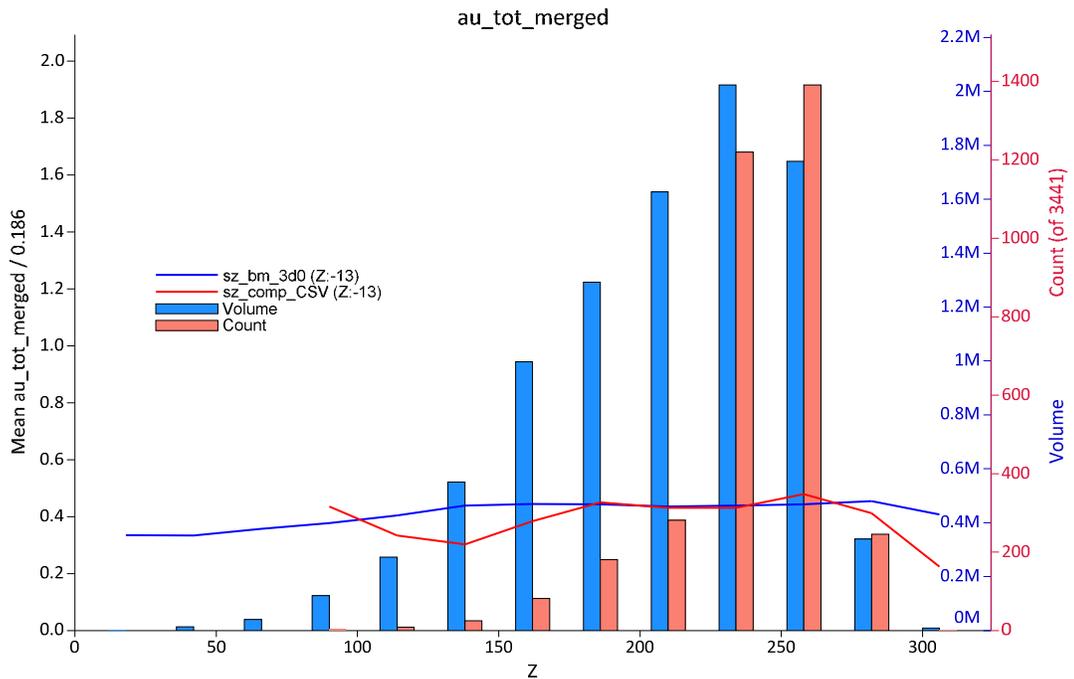
**Figure 14.22**    **Comparison of Block Grades vs. Contoured Gold Values, Test Area 1, North Model Area**
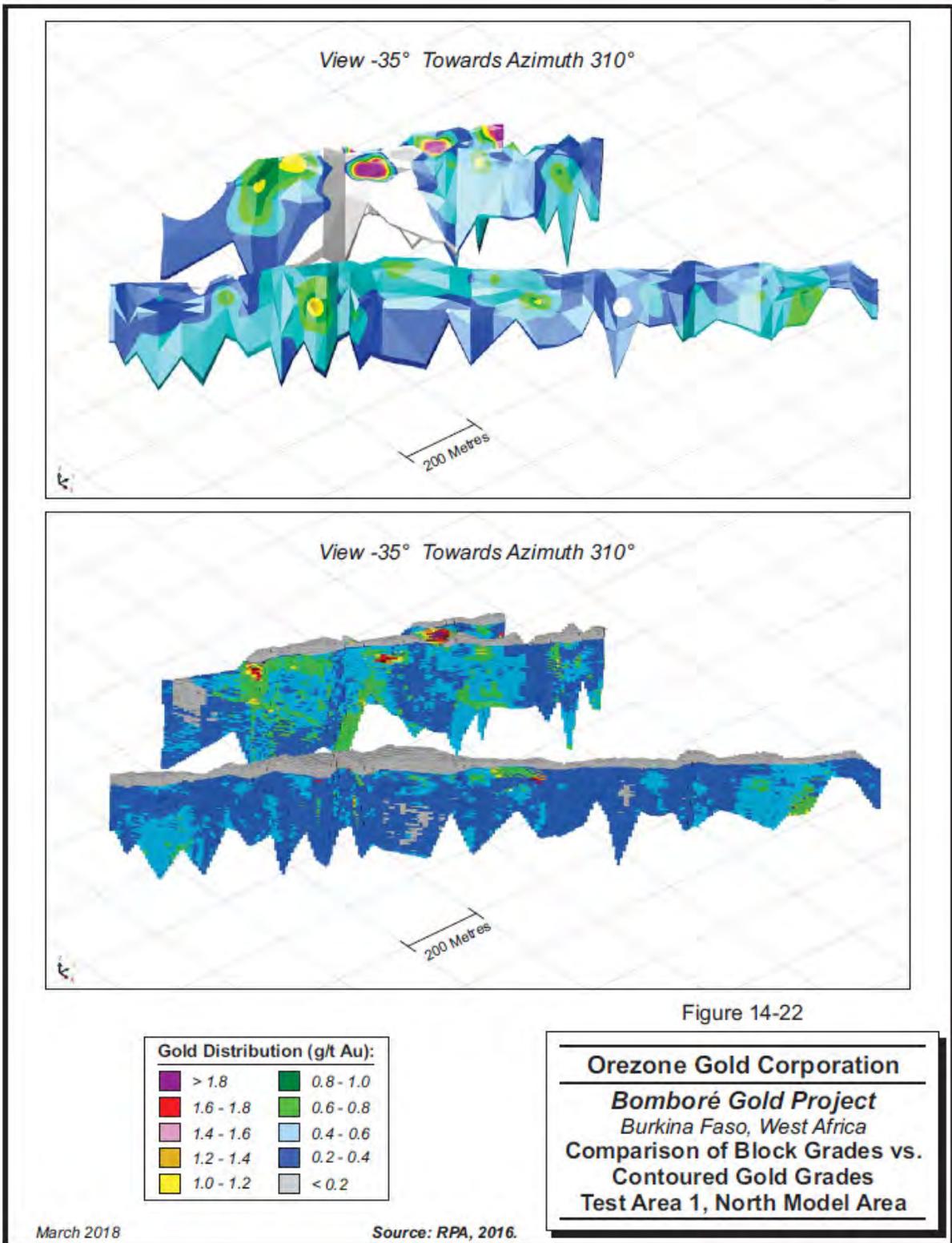
**Figure 14.23**     **Comparison of Block Grades vs. Contoured Gold Values, Test Area 2, North Model Area**
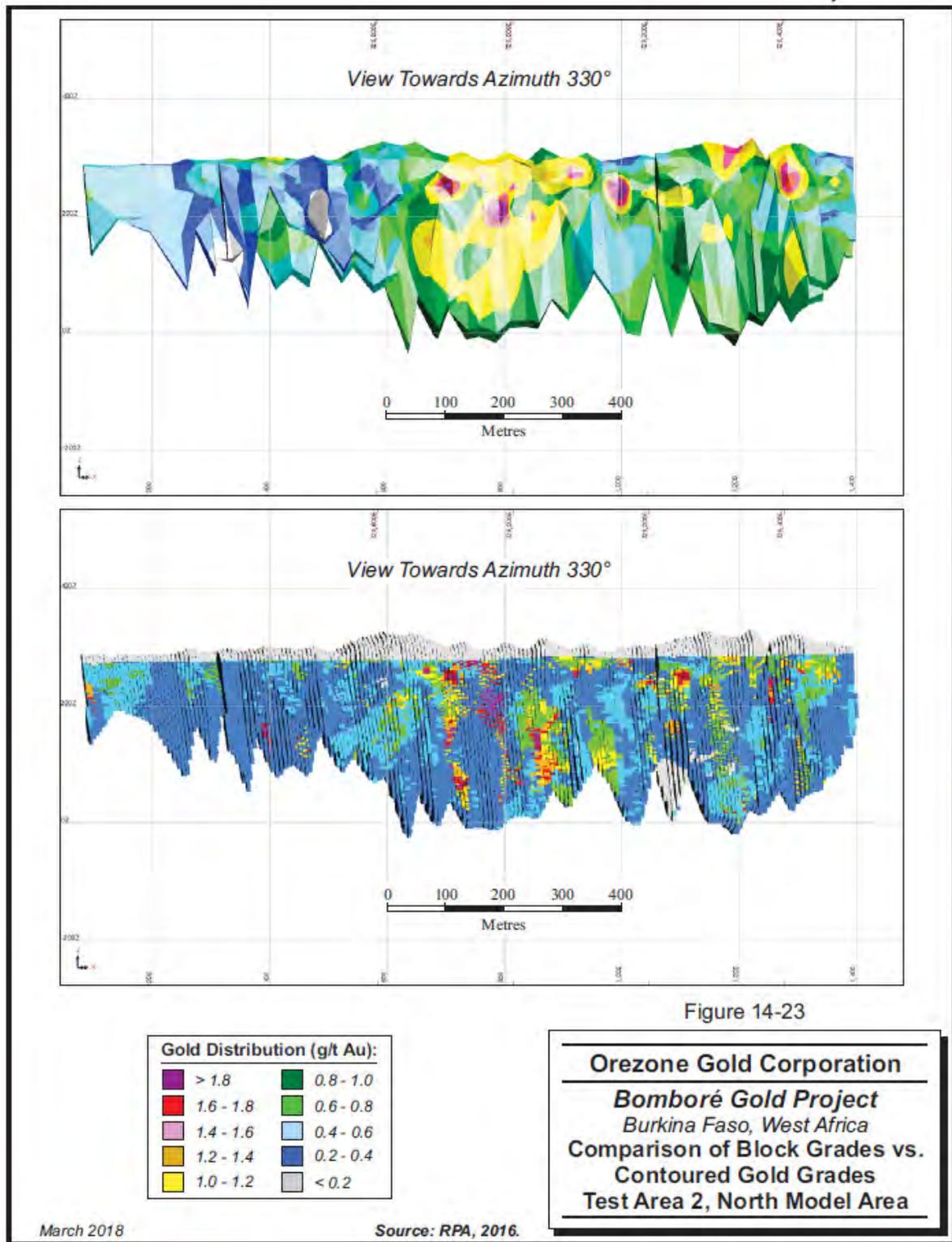
**Figure 14.24     Comparison of Block Grades vs. Contoured Gold Values, P11 450 Main Sub-Domain, South Model Area**
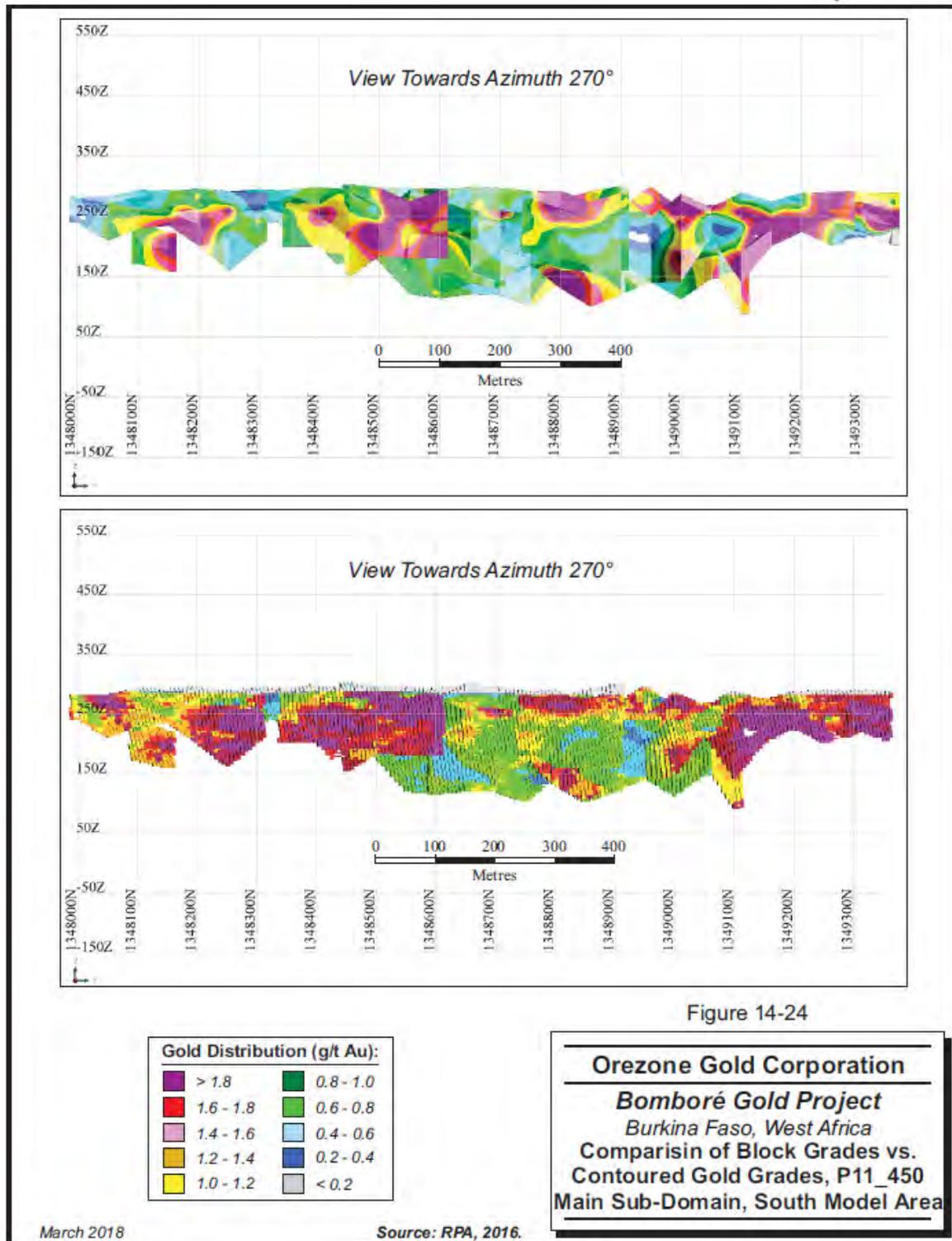
**Figure 14.25** **Comparison of Block Grades vs. Contoured Gold Values, Siga SW 450 Main Sub-domain, South Model Area**
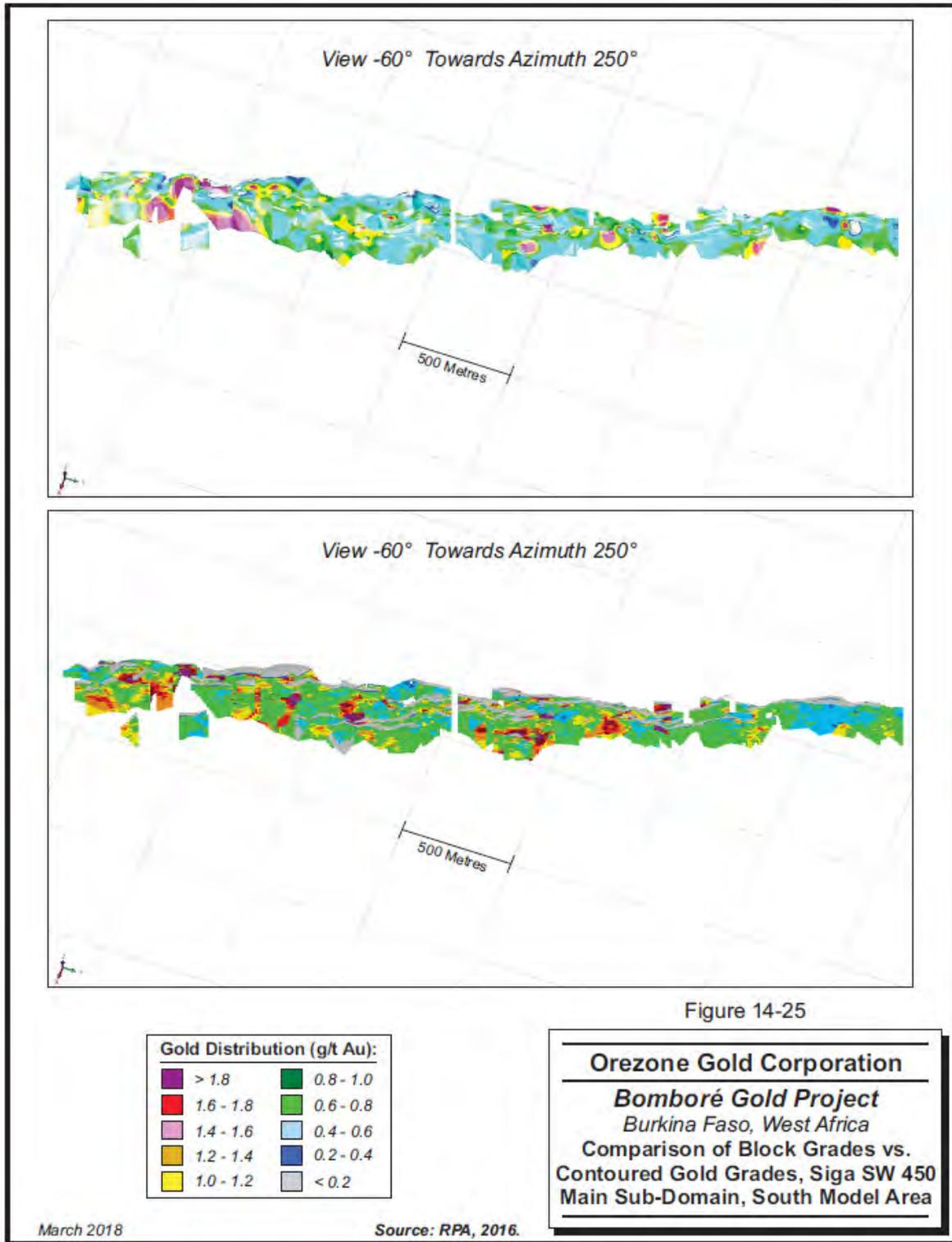
## 14.12    Mineral Resource Classification Criteria

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 and other workers 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. Of the new drilling, 23 holes totalling 1,148 m were drilled in P17S.   RPA expects these holes to upgrade at least a portion of the P17S resource from Inferred to the Indicated category the next resource update, however, this change would not affect the Mineral Reserves as most mineralization at P17S is sulphide which is excluded from reserves.

**Figure 14.26**        **Plan View of the Classified Resources, North Model Area**
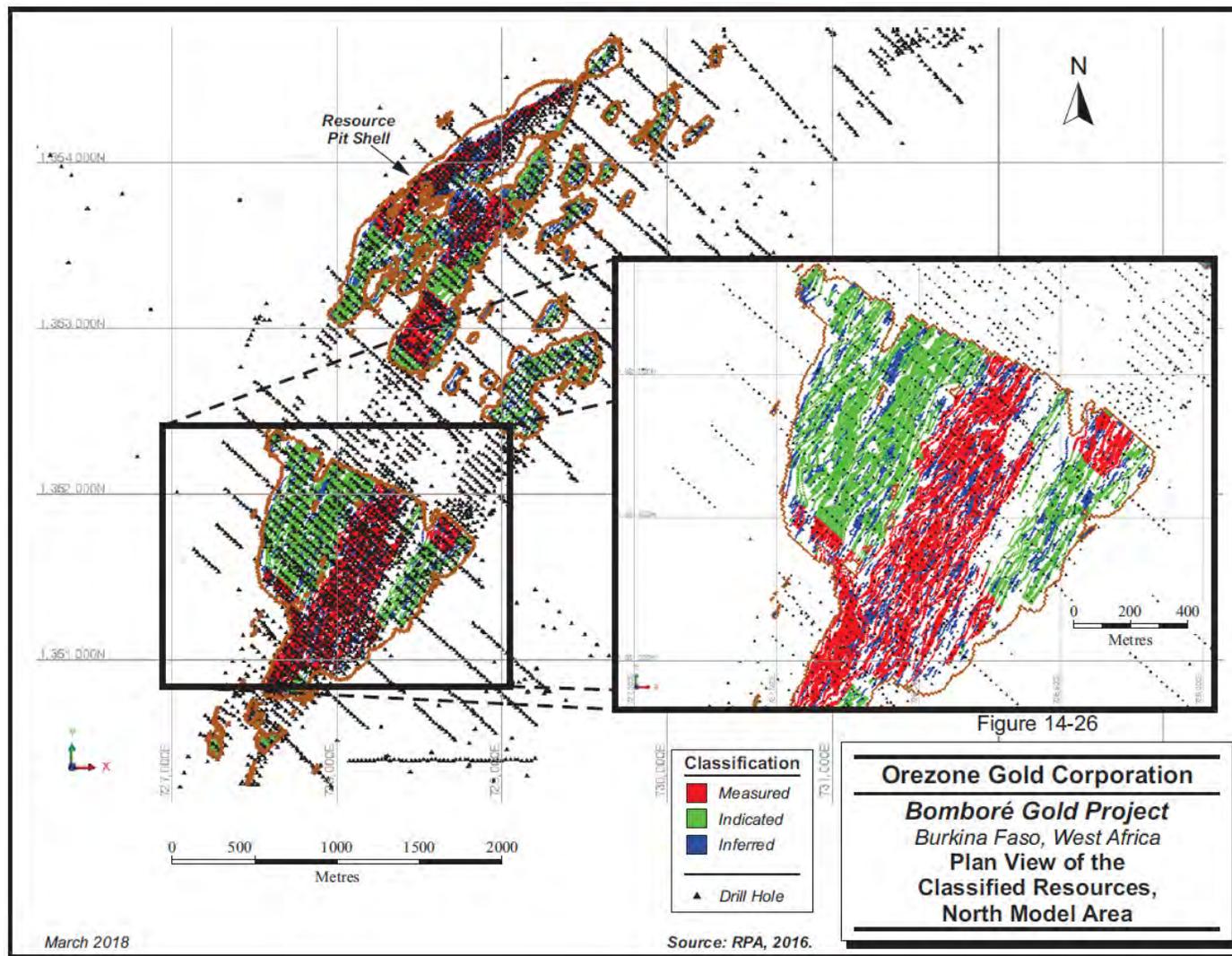


Figure 14-26

**Figure 14.27      Plan View of the Classified Resources, South Model Area**
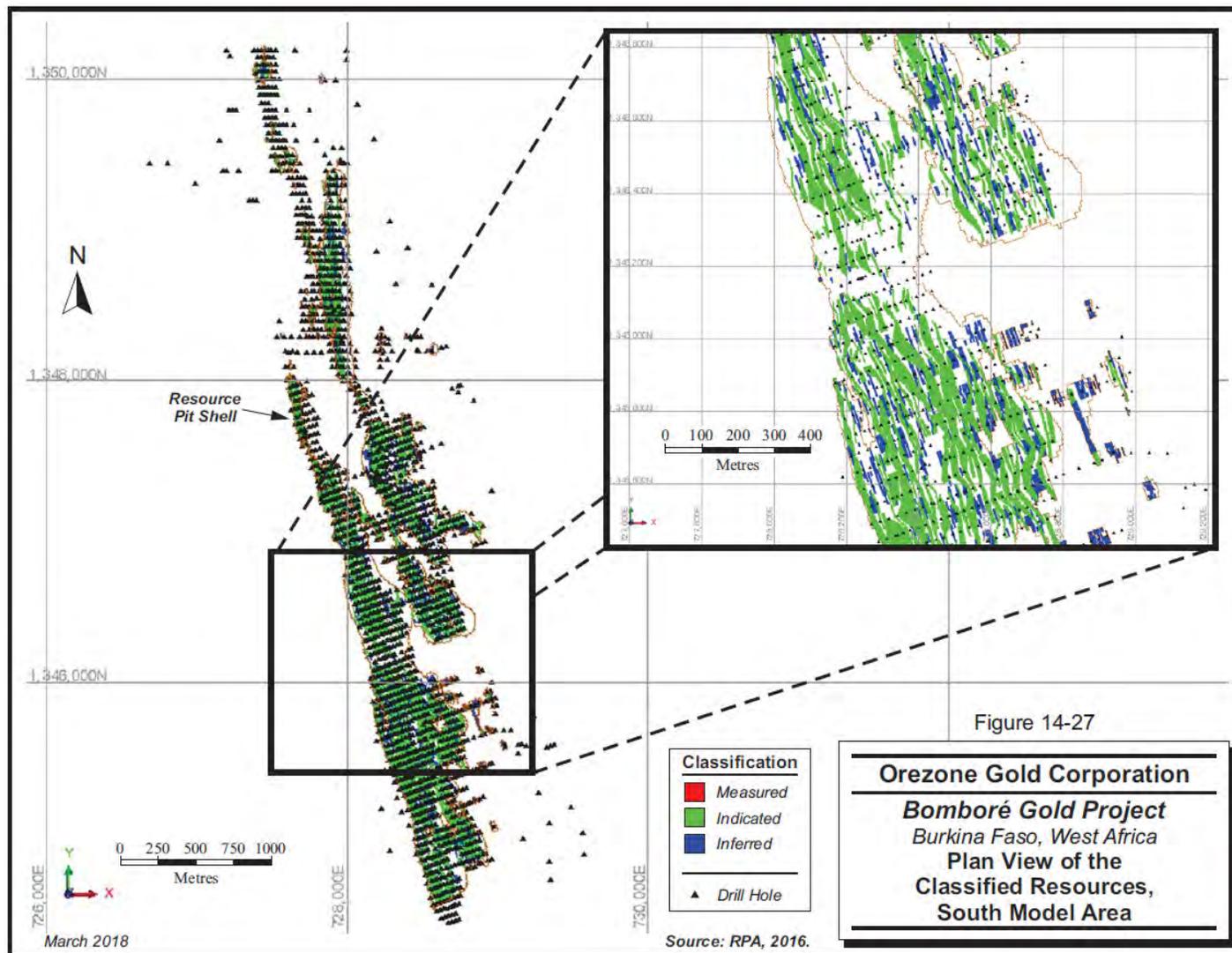
## 14.13    Responsibility for the Estimate

The estimate of the Mineral Resources for the Project presented in this 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 September 7, 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 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 and Resource Reporting

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 per ounce was used for all cut-off grades for reporting of the Mineral Resources.

Metal prices used for reserves are based on consensus, long term forecasts from banks, financial institutions, and other sources.  For mineral resources, metal prices used are slightly higher than those for reserves.

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 Au/t | 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 Au/t | 0.202 | 0.208 | 0.214 | 0.224 | 0.381 |
| P16 and P17 Estimated Cut-off Grade: | g Au/t | 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 $1.05/L, $0.62/L, and $0.14/kWh, respectively.

## 14.15    Mineral Resource Estimate

The Mineral Resources comprise those blocks that are classified into either the Measured, Indicated, or Inferred categories, contain estimated gold grades that are equal to or greater than those presented in Table 14.11, and are located within the respective resource pit shell.

At the estimated cut-off grades, M&I Mineral Resources are estimated to total 218.1 Mt at an average grade of 0.68 g Au/t for 4.77 million ounces of contained gold (Table 14.12). Using the same cut-off grades, Inferred Mineral Resources are estimated to total an additional 48.2 Mt at an average grade of 0.64 g Au/t for 994,000 ounces of contained gold.

**Table 14.12          Mineral Resource Statement as of January 5, 2017**

|  | Measured | | | Indicated | | | Measured + Indicated | | | Inferred | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
|  | Tonnes | Grade | Gold | Tonnes | Grade | Gold | Tonnes | Grade | Gold | Tonnes | Grade | Gold |
| Material Type | Mt | g Au/t | koz | Mt | g Au/t | koz | Mt | g Au/t | koz | Mt | g Au/t | koz |
| Oxide + Tran HG | 16.9 | 0.94 | 513 | 36.5 | 0.83 | 974 | 53.4 | 0.87 | 1,487 | 4.8 | 0.77 | 117 |
| Oxide + Tran LG | 18.5 | 0.33 | 196 | 50.1 | 0.33 | 531 | 68.6 | 0.33 | 727 | 16.4 | 0.29 | 151 |
| **Total Oxide + Tran** | **35.4** | **0.62** | **709** | **86.7** | **0.54** | **1,505** | **122.0** | **0.56** | **2,214** | **21.2** | **0.39** | **268** |
| Fresh HG | 2.3 | 118 | 87 | 68.7 | 0.96 | 2,121 | 71.0 | 0.97 | 2,208 | 20.1 | 0.97 | 630 |
| Fresh LG | 0.8 | 0.43 | 11 | 24.2 | 0.43 | 337 | 25.0 | 0.43 | 348 | 6.9 | 0.43 | 96 |
| **Total Fresh** | **3.1** | **0.99** | **97** | **93.0** | **0.82** | **2,458** | **96.0** | **0.83** | **2,556** | **27.0** | **0.84** | **726** |
| **Total HG** | **19.2** | **0.97** | **600** | **105.3** | **0.91** | **3,095** | **124.5** | **0.92** | **3,695** | **24.9** | **0.93** | **747** |
| **Total LG** | **19.2** | **0.33** | **206** | **74.4** | **0.36** | **868** | **93.6** | **0.36** | **1,075** | **23.3** | **0.33** | **246** |
| **Total HG+LG** | **38.4** | **0.65** | **806** | **179.6** | **0.69** | **3,964** | **218.1** | **0.68** | **4,770** | **48.2** | **0.64** | **994** |

*Notes:*

1. *CIM (2014) definitions were followed for Mineral Resources.*

2. *HG indicates high-grade resources above the higher cut-offs (0.45 g Au/t for oxide and transition, and 0.5 g Au/t for fresh), LG indicates low-grade material between the breakeven cut-off grades (0.2 g Au/t for oxide and transition, and 0.38 g Au/t for fresh) and HG cut-offs.*

3. *Mineral Resources are estimated at variable cut-off grades depending on weathering layer and location, and cut-off grades are approximately 0.2 g Au/t for oxide and transition material, and 0.38 g Au/t for fresh material.*

4. *Mineral Resources are estimated using a long-term gold price of US$1,400 per ounce.*

5. *A minimum mining width of approximately 3 m was used.*

6. *Bulk density varies by material type.*

7. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*

8. *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 Mineral Resource estimate.

## 14.16    Comparison to Previous Estimate

Using similar cut-off grades to the September 7, 2016 Mineral Resource estimate for comparison purposes (Table 14.13), the results indicate that the tonnage and gold ounces contained in the 2016 overall combined M&I resource have been increased by 27% and 19%, respectively.  The average gold grade has been reduced by 6% to 0.68 g Au/t.  Within this the oxide and transition portion of the M&I resource tonnage and gold ounces have increased by 20% and 14% and the average gold grade has reduced by 5% to 0.56 g Au/t.  The fresh rock (sulphide) M&I resource tonnage and gold ounces have been increased by 36% and 23% and the average gold grade has reduced by 10% to 0.83 g Au/t.

The increase in M&I resource is related to the new wireframes and the associated increase in pit shell depth allowing for recovery of material within previously defined wireframes.  Approximately 60% of the Inferred resource increase is due to the new wireframes and the new pit shell allowing for slightly deeper access to existing wireframes, with the remaining 40% belonging to the third domain.

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

**Table 14.13          Comparison to the Previous Mineral Resource Estimate**

| Sept 2016 to Jan 2017 | Cut-off Grade g/t Au | Measured Mineral Resource | | | Indicated Mineral Resource | | | Sub-Total Measured and Indicated | | | Inferred Mineral Resource | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Tonnage Mt | Grade g/t Au | Contained koz Au | Tonnage Mt | Grade g/t Au | Contained koz Au | Tonnage Mt | Grade g/t Au | Cont koz Au | Tonnage Mt | Grade g/t Au | Contained koz Au |
| 2016 Oxide+Trans | 0.20 | 32.1 | 0.66 | 683 | 69.4 | 0.56 | 1,252 | 101.6 | 0.59 | 1,935 | 2.5 | 0.50 | 40 |
| 2017 Oxide+Trans | 0.20 | 35.4 | 0.62 | 709 | 86.7 | 0.54 | 1,505 | 122.0 | 0.56 | 2,214 | 21.2 | 0.39 | 268 |
| Difference | | 3.2 | (0.04) | 26 | 17.2 | (0.02) | 254 | 20.5 | (0.03) | 279 | 18.7 | (0.11) | 227 |
| Percent Difference | | 10% | -6% | 4% | 25% | -4% | 20% | 20% | -5% | 14% | 748% | -21% | 567% |
| 2016 Fresh Total | 0.38 | 8.4 | 0.95 | 256 | 62.0 | 0.91 | 1,818 | 70.4 | 0.92 | 2,074 | 21.8 | 0.77 | 539 |
| 2017 Fresh Total | 0.38 | 3.1 | 0.99 | 97 | 93.0 | 0.82 | 2,458 | 96.0 | 0.83 | 2,556 | 27.0 | 0.84 | 726 |
| Difference | | (5.4) | 0.05 | (158) | 31.0 | (0.09) | 641 | 25.6 | (0.09) | 482 | 5.2 | 0.07 | 187 |
| Percent Difference | | -64% | 5% | -62% | 50% | -10% | 35% | 36% | -10% | 23% | 24% | 9% | 35% |
| 2016 All Layers Total | 0.20 and 0.38 | 40.6 | 0.72 | 939 | 131.4 | 0.73 | 3,069 | 172.0 | 0.73 | 4,008 | 24.3 | 0.74 | 579 |
| 2017 All Layers Total | 0.20 and 0.38 | 38.4 | 0.65 | 806 | 179.6 | 0.69 | 3,964 | 218.1 | 0.68 | 4,770 | 48.2 | 0.64 | 994 |
| Difference | | (2.1) | (0.07) | (133) | 48.2 | (0.04) | 894 | 46.1 | (0.04) | 762 | 23.9 | (0.10) | 415 |
| Percent Difference | | -5% | -9% | -14% | 37% | -6% | 29% | 27% | -6% | 19% | 99% | -14% | 72% |

## 15.0 MINERAL RESERVE ESTIMATES

The Reserve estimate is based on the Mineral Resource estimate prepared by RPA with an effective date of January 5, 2017. The Mineral Resource estimate (MRE) is described in Section 14 of this report. The Mineral Resource consists of three separate block models:

- North model including the KT, 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.

AMC developed Mineral Reserve models by applying modifying factors to the resource block models using Datamine's Studio OP software (Datamine). Diluted models were optimized using Gemcom's Whittle 4.X software (Whittle). The pit optimization was then used as basis for producing practical mine designs in Datamine. The reserve block models were evaluated against the mine designs to provide the Mineral Reserve Estimate.

This study considered only the weathered saprolite and upper transition horizons which reach a thickness of up to 90 m across the site; these material types can be excavated without the need for prior blasting (free dig material).

## 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 five mining blocks separated geographically by river flood plains 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.

Figure 15.1 shows a plan view of the mining block boundaries.

**Figure 15.1          Plan View of Mining Block Boundaries**
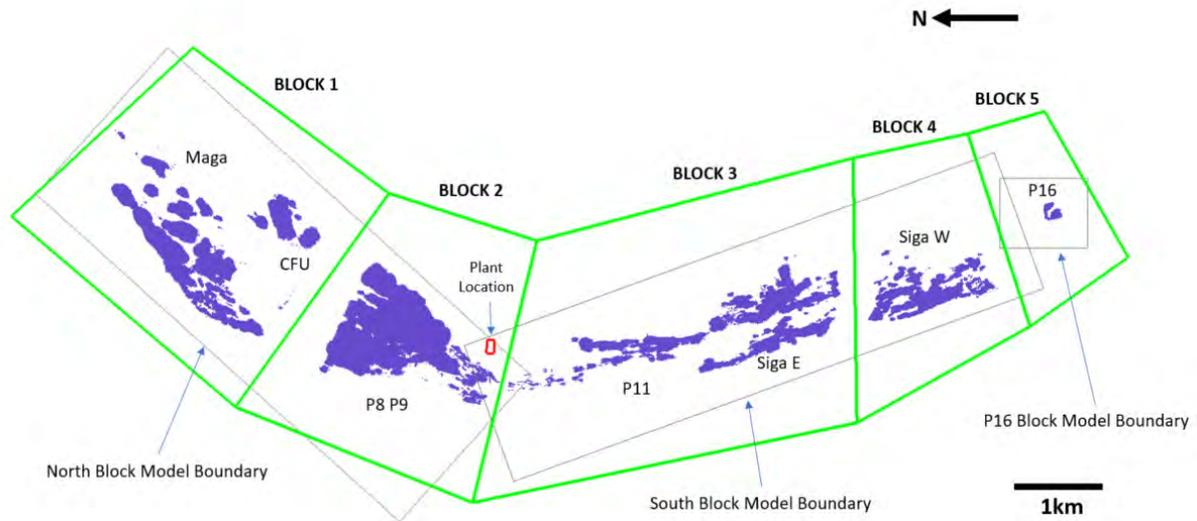


### 15.1.2    Optimization Inputs

AMC constrained the pit optimization to free dig material only. In addition, Inferred Mineral Resources were treated as waste, and only M&I Mineral Resources were considered as feed to the processing plant. The optimization inputs are shown in Table 15.1 and are not the same as the inputs used for the economic analysis of the project.

**Table 15.1 Optimization Inputs**

| Input | Units | Value |
|---|---|---|
| **Global Inputs** | | |
| • Gold Price | US$/oz Au | 1,250 |
| • Payable Gold | % | 99.9 |
| • Royalty | % | 4.0 |
| • Community Tax | % | 1.0 |
| • Offsite Charges | US$/oz Au | 3.50 |
| **Mining Inputs** | | |
| • Mining Recovery | % | 98 |
| • Mining Dilution | % | Variable |
| • Contractor Mining Base Rate | US$/t mined | 1.30 |
| • Owners Team Cost | US$/t ore | 0.75 |
| • Grade Control | US$/t ore | 0.30 |
| • Haulage Rate | US$/t km | 0.15 |
| **Process Inputs** | | |
| • Process Throughput | Mtpa | 4.5 |
| • Process Recovery | % | 90.0 |
| • Process Cost | US$/t ore | 5.00 |
| • Process Sustaining Cost (TSF development) | US$/t ore | 1.00 |
| **Other Costs** | | |
| • General & Administration Cost | US$/t ore | 2.22 |
| • Closure Cost | US$/t ore | 0.15 |
| • Re-settlement South Model & P16 | US$/t ore | 0.25 |

The gold price of $1,250 and associated offsite charges were provided by Orezone.

Gold royalties in Burkina Faso are calculated as follows:

• Spot price less than US$ 1,000: 3% of the NSR + 1% local community development tax.

• Spot price greater than US$ 1,000 and less than USD 1,300: 4% of the NSR + 1% local community development tax.

• Spot price greater than US$ 1,300: 5% of the NSR + 1% local community development tax.

Royalties are applied to 100% of the gold produced.

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 (WRD) and TSF. A mining contractor base rate of $1.30 per tonne was assumed for all material; a $0.15 per t-km adjustment was applied to ore to simulate the additional ore haulage distance to the crusher (Table 15.2).

**Table 15.2          Optimization – Mine Operating Cost**

| | | Ore Unit Cost | | | | |
|---|---|---|---|---|---|---|
| Block | Waste Unit Cost (US$/t) | Base Rate Contractor Cost (US$/t) | Additional Haulage (US$/t) | Owners Team (US$/t) | Grade Control (US$/t) | Ore cost (US$/t) |
| Block 1 | **1.30** | 1.30 | 0.44 | 0.75 | 0.30 | **2.79** |
| Block 2 | **1.30** | 1.30 | 0.00 | 0.75 | 0.30 | **2.35** |
| Block 3 | **1.30** | 1.30 | 0.24 | 0.75 | 0.30 | **2.59** |
| Block 4 | **1.30** | 1.30 | 0.68 | 0.75 | 0.30 | **3.03** |
| Block 5 | **1.30** | 1.30 | 0.95 | 0.75 | 0.30 | **3.30** |

For the pit optimization, AMC assumed that waste would be hauled a fixed distance of 1.5 km to a designated WRD in each mining block. This assumption changed during the study, as 82% of the waste is to be used for construction of the TSF. Subsequently, AMC conducted sensitivity runs to include the additional waste haulage distance; this did not cause any material change to the ultimate pit shells resulting in a 5% reduction in pit size.

***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 6m benches mined as two 3 m flitches. AMC evaluated that, based on the mineralization characteristics, equipment size and grade control procedures, the minimum selective mining unit (SMU) will be 3 m x 6.25 m x 3 m.

AMC generated diluted mining block models using the proprietary Drildil dilution algorithm in Datamine. The dilution algorithm modified the block models by:

- Adding 0.5 m wide dilution skins to the footwall and hangingwall contacts of the modelled mineralization. This is to account for exceeding dig lines and spalling of waste into ore excavations.

- Applying a minimum mining width of 3 m to incorporate blocks of waste between mineralized zones which would not be mined selectively and hence added dilution to the surrounding ore blocks. This internal dilution occurs where mineralized zones are less than 3 m apart.

Based on the grade tonnage curves and drill assay results outside the mineralized domains, a grade of 0.1 g Au/t was applied to the mining dilution.  AMC treated Inferred material as waste in the dilution process.

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**
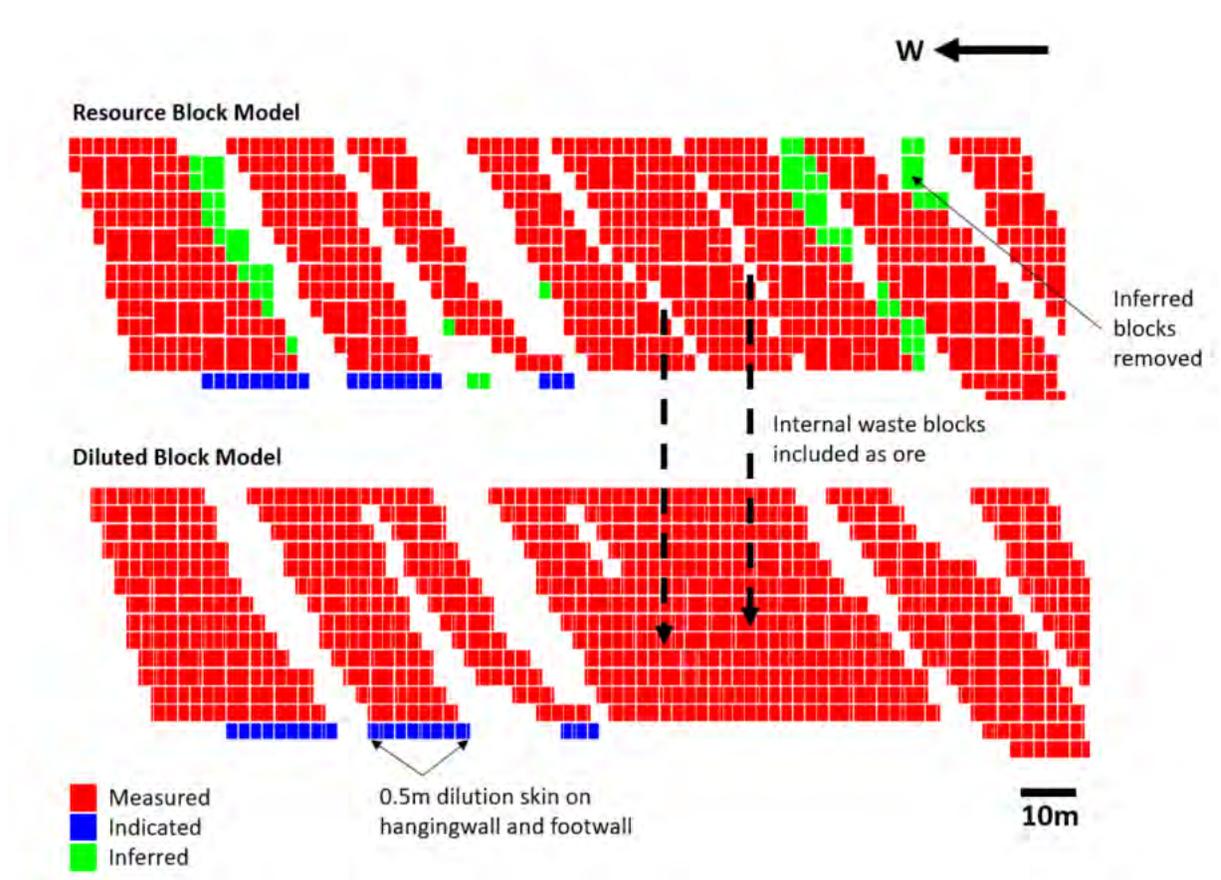


AMC applied a mine recovery factor of 98% to represent a 2% operational ore loss during the load, haul and dump cycle.

### 15.1.3    Sterilized Areas

AMC excluded the following areas from the pit optimization and the Mineral Reserves Estimate:

- OCR area – The OCR is located within an area of relatively low-grade mineralization. The footprint of the OCR was excluded to allow for the water storage facility. This is accounted for in the Mineral Reserve Estimate and production schedule.

- River flood plains – The Nobsin River in the North and the MV2 stream and Bomboré River in the South have been excluded from the Mineral Reserves. Mineralization continues through the river areas and Orezone are currently completing an ESIA to mine this area by adopting a staged mine and backfill approach during the dry season periods.

The sterilized areas are shown in Figure 15.3.

### Figure 15.3          Areas Sterilized in Optimization
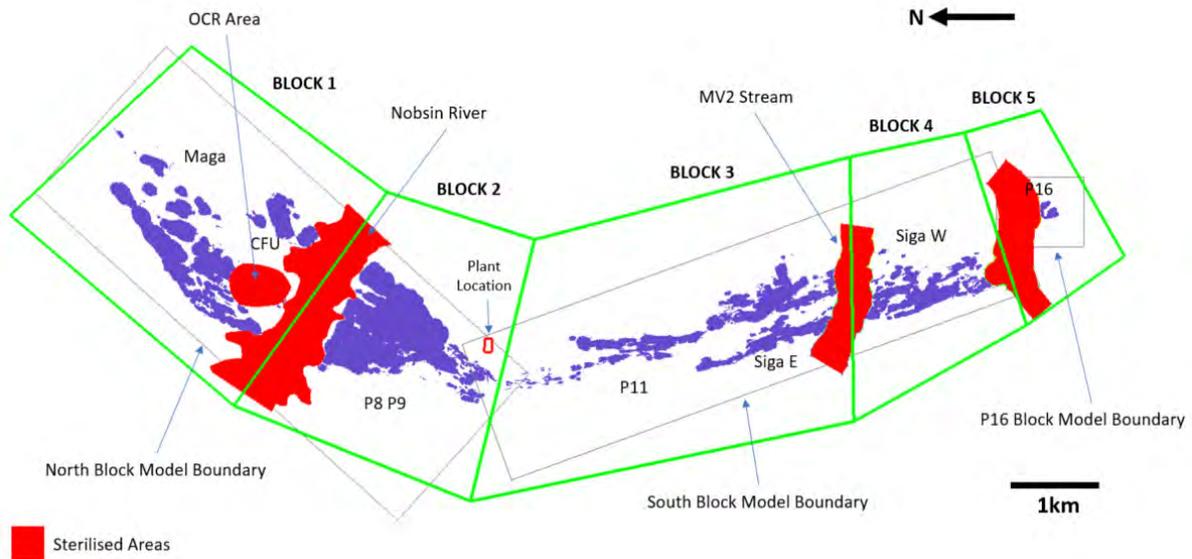


*Slope Regions*

Slope recommendations were provided by Golder Associates. Overall slope angle varies based on overall slope height and ground water saturation conditions.

*Saturation Zones*

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 these saturation zones is shown in Figure 15.4.

Figure 15.4          Groundwater Saturation Zones
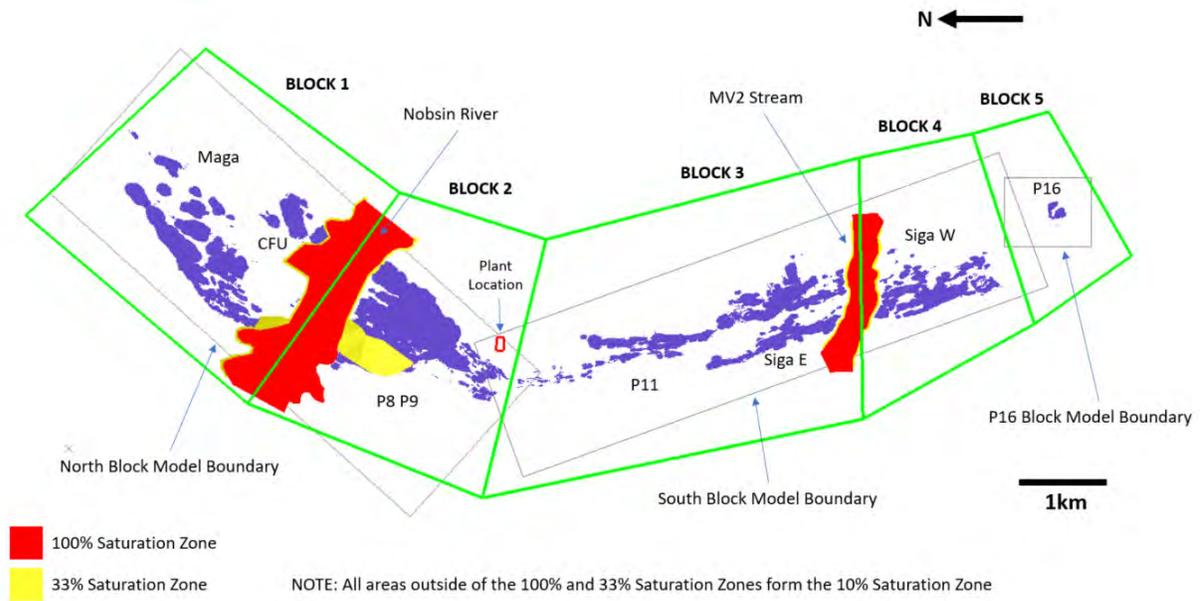


Most of the pits lie within the 10% saturation zone with pits adjacent to the river basins encroaching into the 33% saturation zone. The 100% saturation zone coincides with the river flood plains and has been excluded from the Mineral Reserve Estimate hence only the 10% and 33% saturation zones were used in the optimization.

*Slope Recommendations*

Golder's recommended inter-ramp slope angles for the 10% and 33% saturation zones are shown in Table 15.3.

**Table 15.3**　　　　　**Golder Slope Recommendations by Saturation Zone and Slope Height**

| Maximum Slope Height (metres) | Bench Height (metres) | 10% Saturation Zone | | | 33% Saturation Zone | | |
|---|---|---|---|---|---|---|---|
| | | Bench Face Angle (degrees) | Catch Bench Width (metres) | Inter-Ramp Slope Angle (degrees) | Bench Face Angle (degrees) | Catch Bench Width (metres) | Inter-Ramp Slope Angle (degrees) |
| 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 |

AMC completed an initial Whittle run with 40 degree slope angles to establish the likely final pit slope heights. The project area was then divided into 16 regions separated based on slope height. The slope regions and slope angles applied to each region are shown in Figure 15.5 and Table 15.4.

**Figure 15.5**　　　　　**Numbered Slope Regions by Pit Depth**
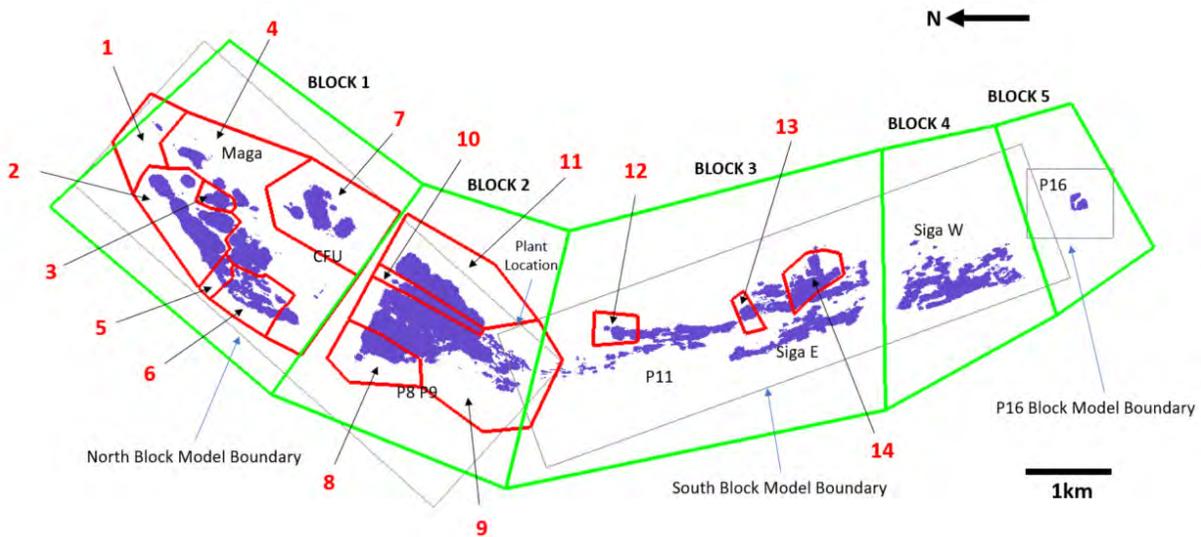
**Table 15.4**          **Slope Regions and Applied Inter-ramp and Overall Slope Angles**

| Slope Region | Max Slope Height (metres) | 10% Saturation zone | | 33% Saturation Zone | |
|---|---|---|---|---|---|
| | | Inter-Ramp Slope Angle (degrees) | Overall Slope Angle Including 14m wide ramp (degrees) | Inter-Ramp Slope Angle (degrees) | Overall Slope Angle Including 14m wide ramp (degrees) |
| 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 15.4 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 most of the waste to the Eastern side of the pits.

The deepest excavations with pit slopes up to 80m in height occur in the North block model where the saprolite weathering horizon is at its thickest. Except for of slope regions 12-14, the South block model consists of pits with a uniform 30 m depth.

### 15.1.4    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 while also ensuring sufficient tonnage for a minimum of 10-year mine life. Table 15.5 summarizes the selected pit shells which were used as guides for pit design.

**Table 15.5          Pit Shells Tonnes and Grades**

| Mining Block | Cut Off Grade (g Au/t) | Revenue Factor | Ore Tonnes (Mt) | Gold Grade (g Au/t) | Waste Tonnes (Mt) | Strip Ratio (W:O) |
|---|---|---|---|---|---|---|
| OCR | 0.29 | - | 3.1 | 0.45 | 6.4 | 2.09 |
| Block 1 | 0.29 | 1.0 | 13.0 | 0.71 | 28.2 | 2.17 |
| Block 2 | 0.28 | 1.0 | 25.0 | 0.61 | 36.4 | 1.46 |
| Block 3 | 0.29 | 1.0 | 10.0 | 0.66 | 15.0 | 1.50 |
| Block 4 | 0.30 | 1.0 | 5.6 | 0.67 | 5.4 | 0.96 |
| Block 5 | 0.28 | 1.0 | 0.2 | 0.85 | 0.4 | 2.07 |
| **Total** | | | **56.9** | **0.64** | **91.9** | **1.61** |

The cut-off grades shown in Table 15.5 represent the break-even cut-off grades.

AMC constrained the optimization to mine only free dig material and this imposes a physical limit on the pit depths. This is reflected by the relatively small gains in ore tonnage, large gains in waste and reduction in relative pit value as the revenue factor is increased above RF1.

In all mining blocks, AMC notes no obvious step changes or significant pushbacks.

Pit by pit graphs for the individual mining blocks are shown in Figures 15.6 to 15.10. Each graph shows discounted pit value for three cases:

- Best Case – The sequence that gives the maximum value by mining pit shells in the order they are generated by Whittle. This method gives the best value but does not consider a practical mining sequence or the spatial relationship between pushbacks.

- Worst Case – The simplest mining sequence whereby pits are mined in their entirety from top to bottom 'bench by bench'. This gives the most practical solution but the lowest relative value.

- Specified Case – Mining occurs in a sequence of specified practical steps generating a value in between the best case and worst case.

*Mining Block 1*

**Figure 15.6          Mining Block 1 Whittle Pit by Pit Graph**
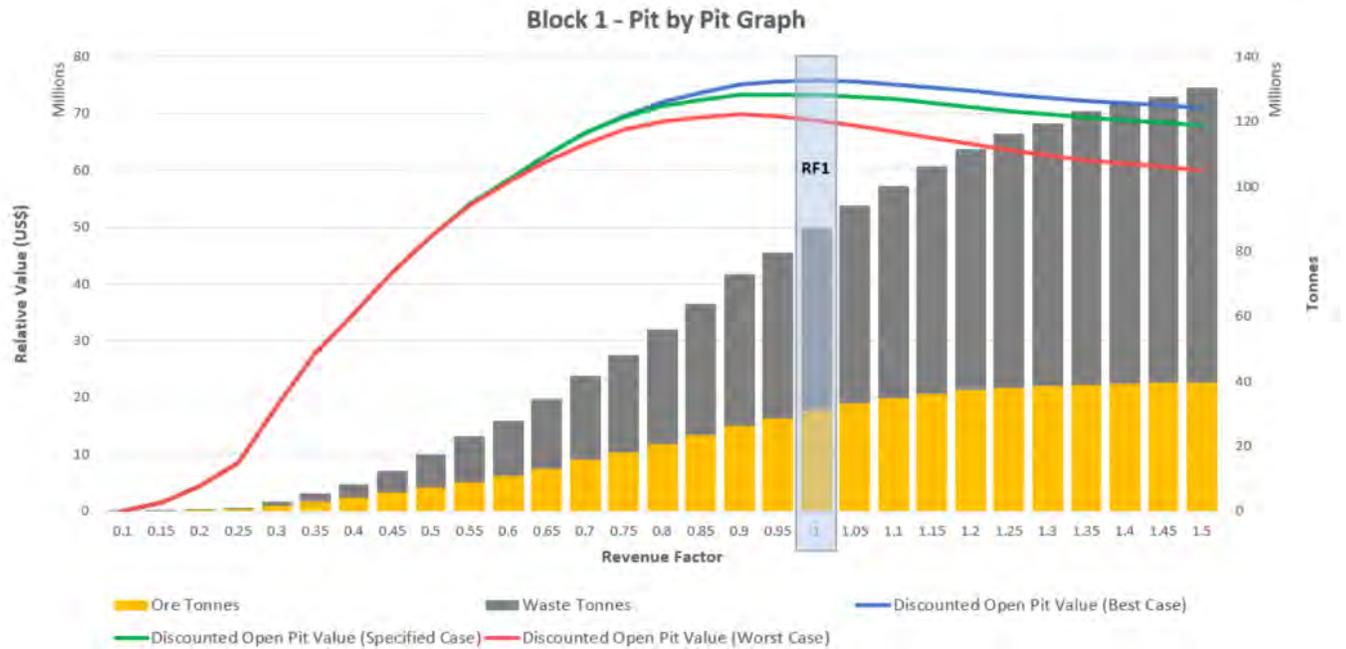


Figure 15.6 shows the Whittle pit by pit graph for Mining Block 1. The specified case utilizes Pit 9 (RF 0.5) as a starter pit. The RF1 pit shell gives the maximum relative pit value for the specified case and contains 13.0Mt of ore at 0.71 g Au/t and 28.2Mt of waste, accounting for 28% of the total ore tonnage. The break-even cut-off grade for the RF1 pit shell is 0.29 g Au/t.

The optimization excluded the OCR area to allow space for the separate design generated by Knight Piésold. The OCR contains 3.1Mt of ore at 0.45 g Au/t and 6.4Mt of waste.

***Mining Block 2***

**Figure 15.7          Mining Block 2 Whittle Pit by Pit Graph**
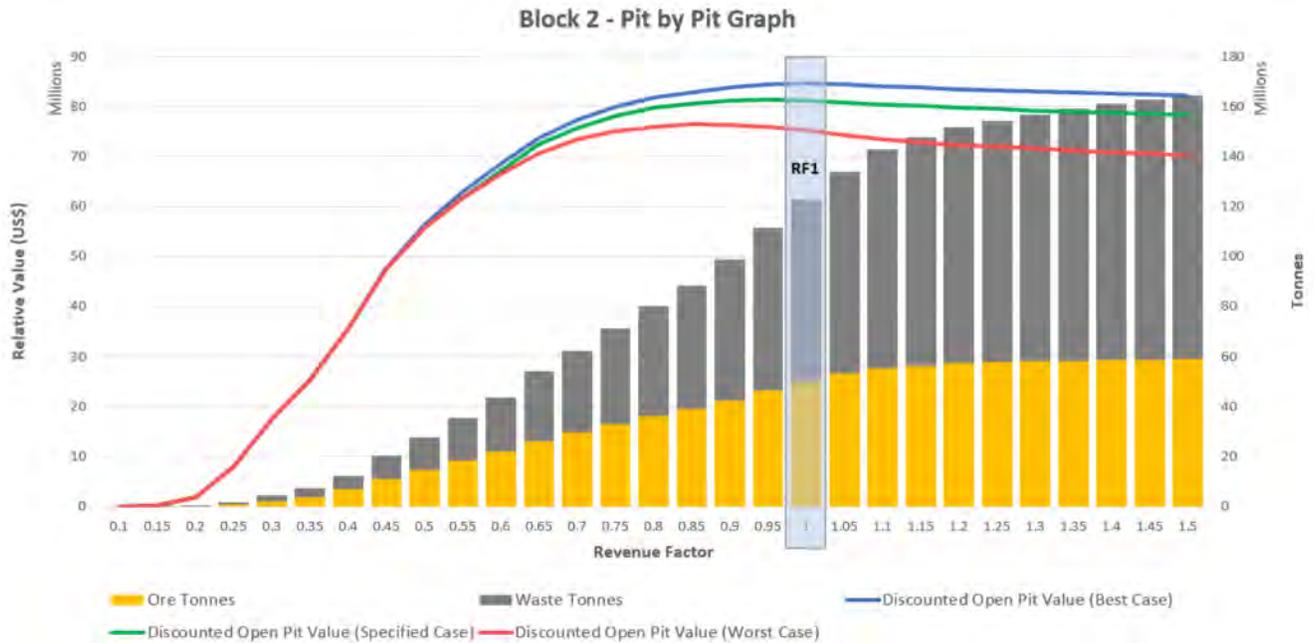


Figure 15.7 shows the Whittle pit by pit graph for Mining Block 2. The specified case utilizes Pit 9 (RF 0.5) as a starter pit. The RF0.95 pit shell gives the maximum relative pit value however AMC selected the RF1 pit shell to maximize available tonnes. The RF1 pit shell contains 25.0Mt at 0.61 g Au/t and 36.4Mt of waste, accounting for 44% of the total ore tonnage. The break-even cut-off grade for the RF1 pit shell is 0.28 g Au/t.

The northern margin and northwest area of the Block 2 pit are situated within the 33% groundwater saturation zone and hence have shallower slope angles applied in these areas.

*Mining Block 3*

**Figure 15.8          Mining Block 3 Whittle Pit by Pit Graph**
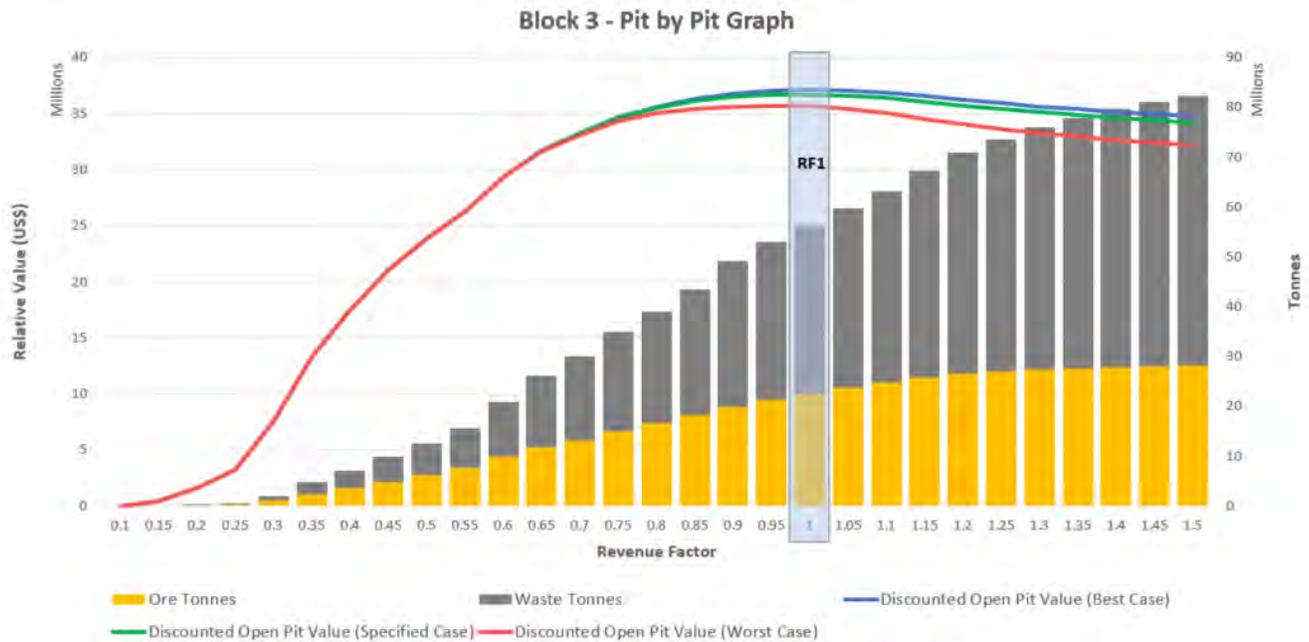


Figure 15.8 shows the Whittle pit by pit graph for Mining Block 3. The specified case utilizes Pit 9 (RF 0.55) as a starter pit. The RF1 pit shell gives the maximum relative pit value for the specified case and contains 10.0Mt of ore at 0.66 g Au/t and 15.0Mt of waste, accounting for 18% of the total ore tonnage. The break-even cut-off grade for the RF1 pit shell is 0.29 g Au/t.

*Mining Block 4*

**Figure 15.9          Mining Block 4 Whittle Pit by Pit Graph**
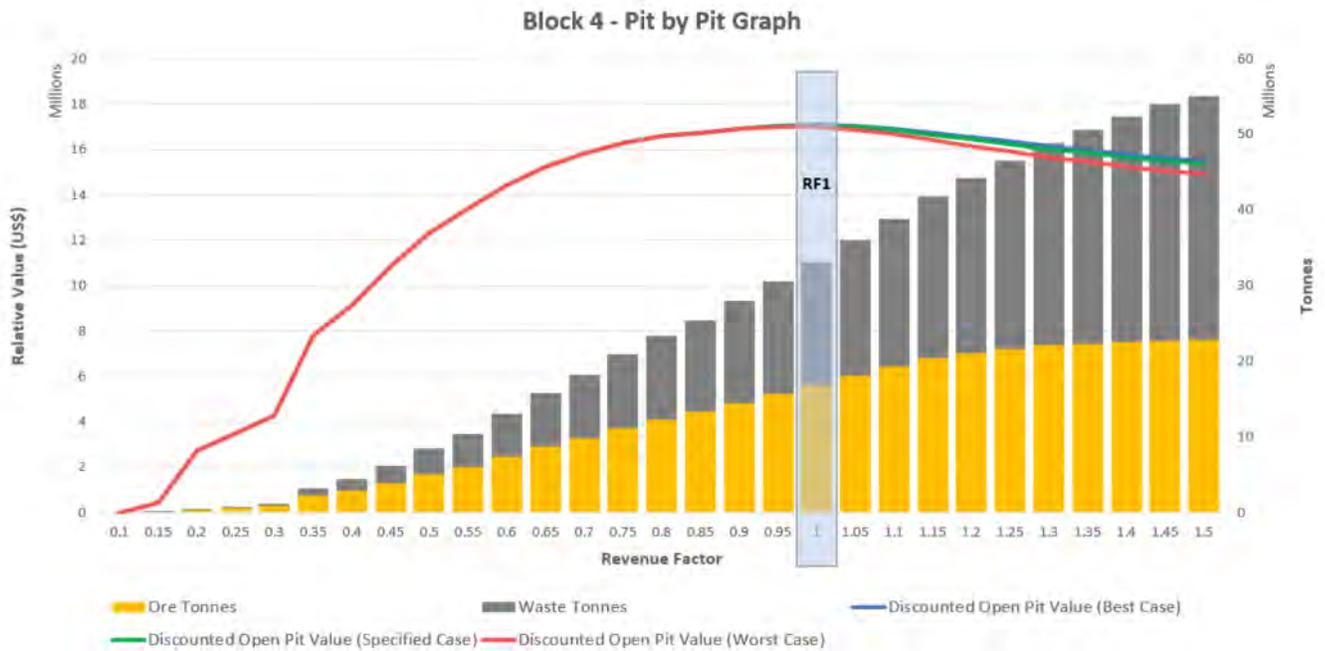


Figure 15.9 shows the Whittle pit by pit graph for Mining Block 4. The specified case utilizes Pit 9 (RF 0.55) as a starter pit. The RF1 pit shell gives the maximum relative pit value for the specified case and contains 5.6Mt of ore at 0.67 g Au/t and 5.4Mt of waste, accounting for 10% of the total ore tonnage. The break-even cut-off grade for the RF1 pit shell is 0.30 g Au/t.

*Mining Block 5*

**Figure 15.10**          **Mining Block 5 Whittle Pit by Pit Graph**
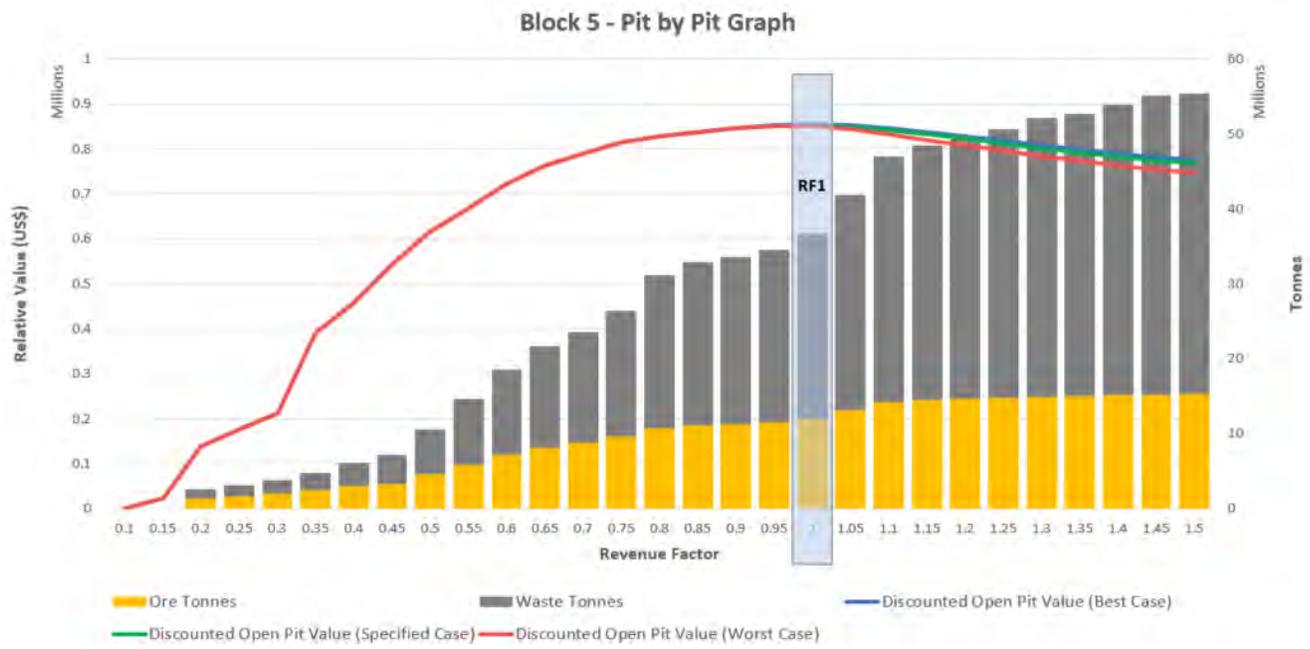


Figure 15.10 shows the Whittle pit by pit graph for Mining Block 5. The specified case uses the Whittle Milawa NPV algorithm. The RF1 pit shell gives the maximum relative pit value for the specified case and contains 0.2Mt of ore at 0.85 g Au/t and 0.4Mt of waste, accounting for <1% of the total ore tonnage. The break-even cut-off grade for the RF1 pit shell is 0.28 g Au/t.

## 15.2 Open-pit Design

AMC created an ultimate pit design in Datamine using the RF1 Whittle Pit Shells as a guide. The design incorporates over 60 separate pits varying in depth from 18 m to 90 m along 11 km of strike. A plan view of the design is shown in Figure 15.11.

**Figure 15.11          Plan View of Ultimate Pit Design**
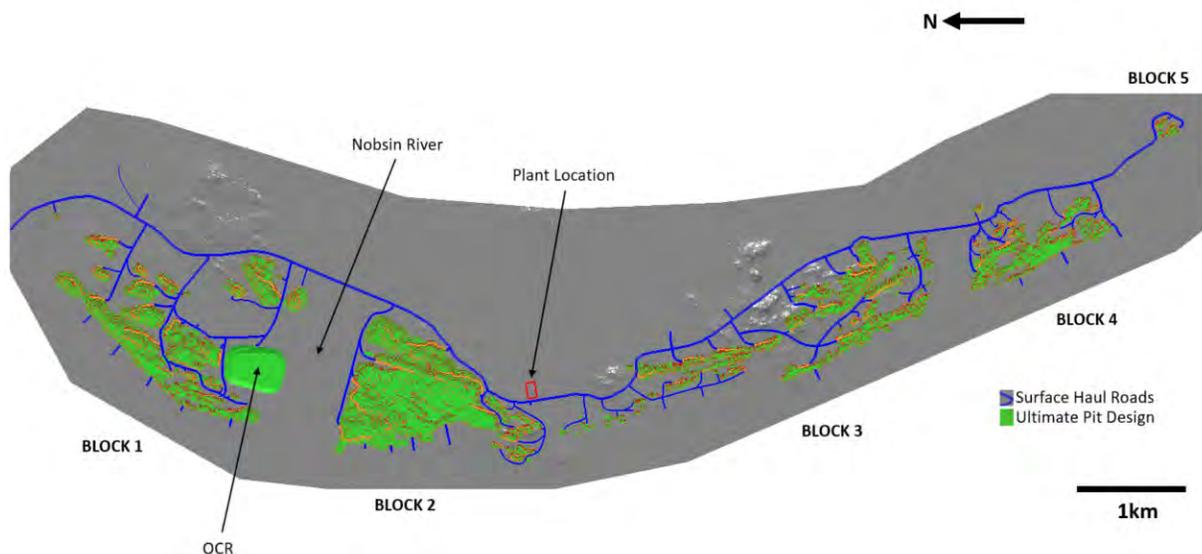


The design criteria used by AMC are described in detail in Section 16. In summary, the design is based on 6 m benches which will be mined in two 3m flitches with berm widths varied according to the parameters recommended by Golder in Table 15.3. The design incorporates ramps 7 m in width for single lane haulage and 14m wide for dual lane haulage which predominantly exit on the Eastern sides of the pits.

### 15.2.1     Reconciliation between Pit Optimization and Design

A comparison of the Whittle optimization to ultimate pit design reported at the same optimization breakeven cut-off grades is shown in Table 15.6.

**Table 15.6          Pit Shell Recommendation**

|  | Ore Tonnes (Mt) | Gold Grade (g Au/t) | Waste Tonnes (Mt) | Strip Ratio (W:O) |
|---|---|---|---|---|
| Whittle RF1 Shells (Including OCR Design) | 56.9 | 0.64 | 91.9 | 1.61 |
| Ultimate Pit Design (Including OCR Design) | 56.0 | 0.64 | 93.9 | 1.68 |
| **Difference** | **-0.9** | **0.0** | **2.0** | **0.1** |
| Difference % | -1.5% | -0.1% | 2.2% | 3.8% |

The ultimate pit design has 1.5% less ore tonnes and 2.2% more waste due primarily to small satellite pits not taken through into design and smoothing of pit floors in larger shallow areas. Some additional waste was incorporated with the inclusion of additional ramps both on Eastern and Western pit slopes.

## 15.3    Mineral Reserve Estimate

The Bomboré Mineral Reserve Estimate is summarised in Table 15.7 and includes only Measured and Indicated Resources with applicable mining dilution constrained to the ultimate pit design.

**Table 15.7          Mineral Reserve Estimate – July 9 2018**

| Classification | Units | Block 1 | Block 2 | Block 3 | Block 4 | Block 5 | Total |
|---|---|---|---|---|---|---|---|
| **Summary of Mineral Reserves** | | | | | | | |
| **Cut-off Grade** | g Au/t | 0.29 | 0.28 | 0.29 | 0.30 | 0.28 | |
| **Proven:** | | | | | | | |
| Ore | 000 t | 8,038 | 13,308 | 0 | 0 | 192 | **21,539** |
| Gold Grade | g Au/t | 0.71 | 0.66 | 0.00 | 0.00 | 0.85 | **0.68** |
| Contained Gold | 000 oz Au | 182 | 284 | 0 | 0 | 5 | **472** |
| **Probable:** | | | | | | | |
| Ore | 000 t | 7,684 | 11,859 | 9,398 | 5,526 | 2 | **34,468** |
| Gold Grade | g Au/t | 0.61 | 0.54 | 0.67 | 0.68 | 0.57 | **0.61** |
| Contained Gold | 000 oz Au | 151 | 205 | 201 | 120 | 0 | **678** |
| **Proven & Probable:** | | | | | | | |
| **Ore** | **000 t** | **15,722** | **25,167** | **9,398** | **5,526** | **194** | **56,007** |
| **Gold Grade** | **g Au/t** | **0.66** | **0.60** | **0.67** | **0.68** | **0.85** | **0.64** |
| **Contained Gold** | **000 oz Au** | **333** | **489** | **201** | **120** | **5** | **1,149** |
| **Mineral Reserves by Material Type** | | | | | | | |
| **Proven:** | | | | | | | |
| Regolith | | | | | | | |
| Ore | 000 t | 424 | 1,120 | 0 | 0 | 38 | **1,582** |
| Gold Grade | g Au/t | 0.58 | 0.47 | 0.00 | 0.00 | 0.43 | **0.50** |
| Contained Gold | 000 oz Au | 8 | 17 | 0 | 0 | 1 | **25** |
| | | | | | | | |
| Upper Saprolite | | | | | | | |
| Ore | 000 t | 3,650 | 4,997 | 0 | 0 | 84 | **8,731** |
| Gold Grade | g Au/t | 0.65 | 0.67 | 0.00 | 0.00 | 0.93 | **0.66** |
| Contained Gold | 000 oz Au | 76 | 107 | 0 | 0 | 3 | **186** |
| Lower Saprolite | | | | | | | |
| Ore | 000 t | 3,172 | 4,879 | 0 | 0 | 59 | **8,110** |
| Gold Grade | g Au/t | 0.72 | 0.69 | 0.00 | 0.00 | 1.02 | **0.71** |
| Contained Gold | 000 oz Au | 74 | 109 | 0 | 0 | 2 | **184** |
| Upper Transition | | | | | | | |
| Ore | 000 t | 792 | 2,312 | 0 | 0 | 11 | **3,115** |
| Gold Grade | g Au/t | 0.96 | 0.69 | 0.00 | 0.00 | 0.84 | **0.76** |
| Contained Gold | 000 oz Au | 25 | 51 | 0 | 0 | 0 | **76** |
| | | | | | | | |
| **Probable:** | | | | | | | |
| Regolith | | | | | | | |

| Classification | Units | Block 1 | Block 2 | Block 3 | Block 4 | Block 5 | Total |
|---|---|---|---|---|---|---|---|
| Ore | 000 t | 513 | 846 | 863 | 441 | 2 | **2,665** |
| Gold Grade | g Au/t | 0.51 | 0.46 | 0.50 | 0.49 | 0.57 | **0.49** |
| Contained Gold | 000 oz Au | 8 | 13 | 14 | 7 | 0 | **42** |
| Upper Saprolite | | | | | | | |
| Ore | 000 t | 3,776 | 5,275 | 3,692 | 1,843 | 0 | **14,587** |
| Gold Grade | g Au/t | 0.58 | 0.51 | 0.63 | 0.63 | 0.00 | **0.57** |
| Contained Gold | 000 oz Au | 71 | 86 | 75 | 38 | 0 | **269** |
| | | | | | | | |
| Lower Saprolite | | | | | | | |
| Ore | 000 t | 2,734 | 4,367 | 3,665 | 2,023 | 0 | **12,789** |
| Gold Grade | g Au/t | 0.65 | 0.56 | 0.71 | 0.71 | 0.00 | **0.65** |
| Contained Gold | 000 oz Au | 57 | 78 | 84 | 46 | 0 | **266** |
| Upper Transition | | | | | | | |
| Ore | 000 t | 660 | 1,371 | 1,177 | 1,219 | 0 | **4,428** |
| Gold Grade | g Au/t | 0.70 | 0.65 | 0.77 | 0.75 | 0.00 | **0.72** |
| Contained Gold | 000 oz Au | 15 | 29 | 29 | 29 | 0 | **102** |
| **Subtotals Proven & Probable:** | | | | | | | |
| **Regolith** | | | | | | | |
| **Ore** | **000 t** | **938** | **1,966** | **863** | **441** | **40** | **4,248** |
| **Gold Grade** | **g Au/t** | **0.54** | **0.47** | **0.50** | **0.49** | **0.44** | **0.49** |
| **Contained Gold** | **000 oz Au** | **16** | **29** | **14** | **7** | **1** | **67** |
| **Upper Saprolite** | | | | | | | |
| **Ore** | **000 t** | **7,426** | **10,272** | **3,692** | **1,843** | **84** | **23,318** |
| **Gold Grade** | **g Au/t** | **0.62** | **0.59** | **0.63** | **0.63** | **0.93** | **0.61** |
| **Contained Gold** | **000 oz Au** | **147** | **193** | **75** | **38** | **3** | **455** |
| **Lower Saprolite** | | | | | | | |
| **Ore** | **000 t** | **5,906** | **9,246** | **3,665** | **2,023** | **59** | **20,899** |
| **Gold Grade** | **g Au/t** | **0.69** | **0.63** | **0.71** | **0.71** | **1.02** | **0.67** |
| **Contained Gold** | **000 oz Au** | **131** | **187** | **84** | **46** | **2** | **450** |
| **Upper Transition** | | | | | | | |
| **Ore** | **000 t** | **1,452** | **3,683** | **1,177** | **1,219** | **11** | **7,543** |
| **Gold Grade** | **g Au/t** | **0.84** | **0.67** | **0.77** | **0.75** | **0.84** | **0.73** |
| **Contained Gold** | **000 oz Au** | **39** | **80** | **29** | **29** | **0** | **178** |

1.  *Mineral Reserves have been estimated in accordance with the CIM Definition Standards.*

2.  *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.*

3.  *Mineral Reserves are estimated at an average long-term gold price of US$ 1,250.*

4.  *Mineral Reserves are reported effective July 9, 2018*

## 16.0 MINING METHODS

The Bomboré mine will be a free dig open pit mining operation consisting of over 60 separate pits of variable size and depth across a mineralized zone 11km long and 2.5km wide. The production schedule is based on the Mineral Reserve Estimate described in Section 15. Mining is planned to span 12 years with run of mine (ROM) ore delivered to the plant followed by processing low grade stockpiles after closure of the open pits.

The key project highlights are:

- 149.8Mt total material mined:

  - 56.0Mt of ore

  - Diluted gold grade 0.64g Au/t

  - 1.1 million in situ ounces Au

  - 93.8Mt waste

  - 1.68 strip ratio

  - 13 year mine life

- Pre-production of 1 year for water storage.

- Process throughput of 4.5 Mtpa:

  - 41.1 Mt at 0.70g Au/t ROM ore

  - 14.9 Mt at 0.47g Au/t low grade ore rehandled from stockpiles

  - 1.024 Moz Au produced

The mine will be contractor operated with an owner's team on site responsible for site management, grade control and mine planning activities. Mining will be undertaken with 3-6 m$^3$ excavators and 30-50 t highway dump trucks.

ROM ore will be hauled to the process plant and low-grade ore hauled to one of three low grade stockpiles. Approximately 82% of waste produced will be used for TSF construction with the remainder being hauled to two WRD and four environmental barriers.

The project area generally consists of flat terrain crossed by wide, shallow river flood plains, which fill with water during the wet season. There are scattered low hills 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 a wet and dry season with rainfall generally occurring in the four months between June and September. On average, 0.8 m of rainfall occurs each year in daily short bursts of heavy rainfall during the wet season. Consideration for the impacts of the wet season on the mining operation has been taken into account in the mine schedule.

## 16.1    Mine Planning

Mining will consist of conventional free-dig open pit operations.

### 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 (not free dig, excluded from mine plan).

•        Fresh Rock (not free dig, excluded from mine plan).

AMC only included the free dig horizons down to the contact between Upper Transition and Lower Transition in the Mineral Reserve Estimate. Mining will occur through the free dig horizons until the point at which excavators can no longer proceed without prior blasting. 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**
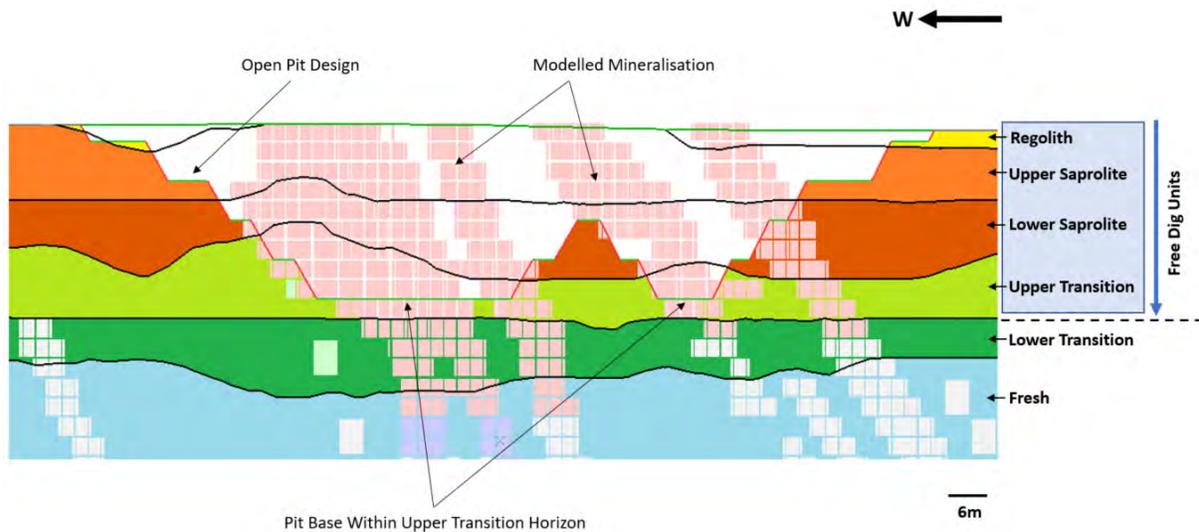


AMC has not allowed for any drill and blast operations in the mine plan. Material that is not free dig may be exposed at the base of pits but will not be mined.

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 80% of the mill feed, the Upper Transition accounts for 13% and the regolith accounting for the remaining 7%.

Weathering profiles are generally deeper in the Northern areas of the project gradually shallowing towards 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:

- Ore – Hauled to the ROM pad or the process plant.

- Low Grade – Hauled to low grade stockpiles for processing at the end of mine life.

- Waste – Hauled to the TSF for construction, environmental barriers and WRDs.

### 16.1.2    Open Pit Design

AMC created an ultimate pit design in Datamine using the RF1 Whittle Pit Shells as a guide. The design incorporates over 60 separate pits varying in depth from 18 m to 90 m along an 11 km strike. A plan view of the pit designs is shown in Figure 16.2.

**Figure 16.2**          **Plan View of Ultimate Pit Design**
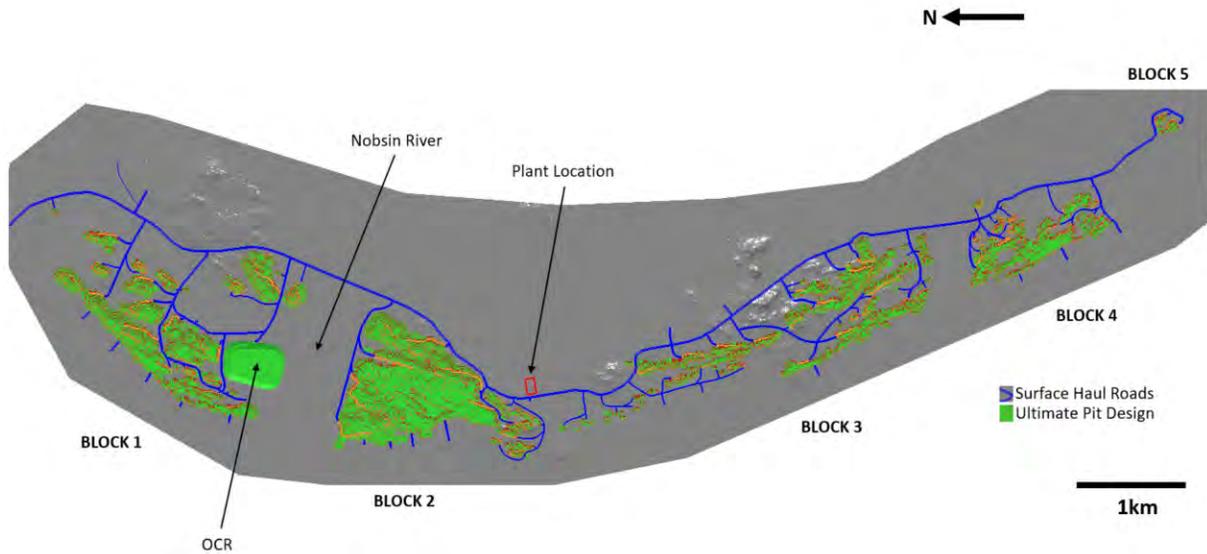


The ultimate pit design is based on 6 m benches which will be mined in two 3 m flitches. A typical layout of a mining area annotated with design parameters is shown in Figure 16.3.

**Figure 16.3**          **Cross Section of a Typical Bench Layout**
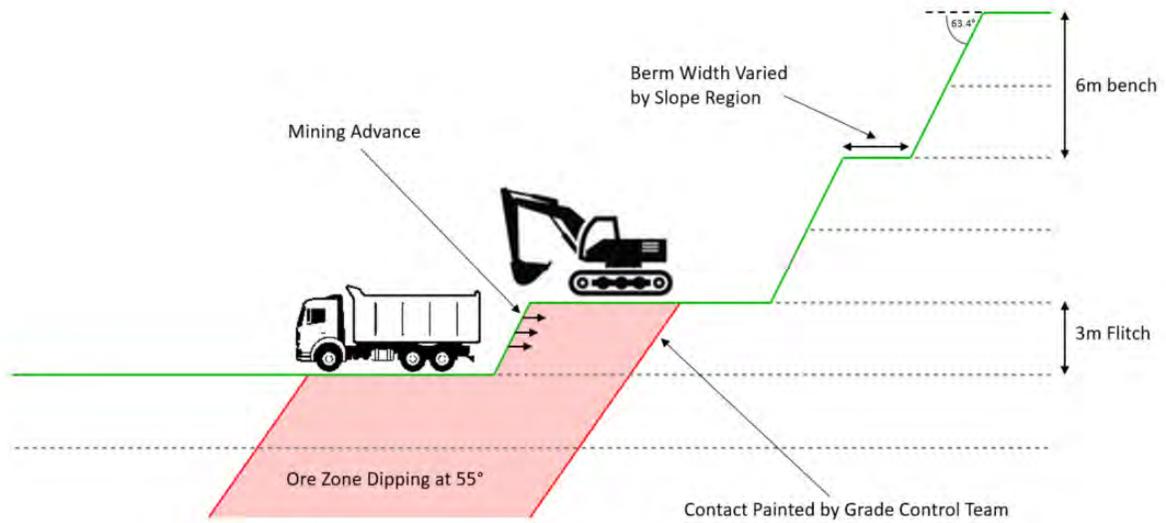


***Slope Parameters***

AMC was provided with slope recommendations by Golder (Golder, 2018). These slope parameters are presented in Table 15.3. AMC divided the project into slope regions for optimization (Figure 15.5 & Table 15.4)

and carried these slope regions through to design. To maintain the recommended inter-ramp slope angles, AMC varied the berm width by slope region as shown in Table 16.1.

**Table 16.1          Berm Widths Varied by Slope Region**

| Slope Region | Max Slope Height (m) | 10% Saturation zone | | 33% Saturation Zone | |
|---|---|---|---|---|---|
| | | Inter-Ramp Slope Angle (degrees) | Berm Width Applied (m) | Inter-Ramp Slope Angle (degrees) | 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 |

All benches were designed at 6m height with 63.4° bench angles as per Golder's recommendations.

*Mining Footprint*

The ultimate pit design generated by AMC has a surface footprint of approximately 400ha; 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 | 133 |
| North | Block 2 | 130 |
| South | Block 3 | 88 |
| South | Block 4 | 44 |
| P16 | Block 5 | 3 |
| **Total** | | **398** |

### 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 16.4.

**Figure 16.4      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

B = 0.5 x Tyre Diameter
C = 2.6 x height x 1.2
E = a x Operating Width

**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

B = 0.5 x Tyre Diameter
C = 2.6 x height x 1.2

Multiple specifications of truck were provided to Orezone as part of the mining contract tender which all fit within the 20-35 tonne highway dump truck range and have an operating width of between 2.5 m and 3.5 m. Dual lane 14 m haul roads will allow this size of truck to operate comfortably and provide some flexibility should larger trucks be considered in the future.

Due to the free dig nature of the project, the exact location of these ramps and pit exits may be optimized during detailed mine planning. Ramp routes may also be modified during operations when required to fit changing mining conditions. Many of the smaller satellite excavations may be mined using a temporary ramp along 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 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 and 7 m wide for single lane. Construction material for the surface roads will be mine waste and on-site borrow pits. Should more substantial aggregate sheeting be required, Orezone has identified a granitic quarry nearby. During detailed design, AMC recommends an assessment of all potential local borrow sources is conducted with the aim of minimising 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 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    Grade Control

The Owners Team will be responsible for grade control activities, mine planning, and supervise and manage the activities of the contractor on site.

Grade control on site will consist of three key components:

- Advance 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 3 m intervals and assayed using the LeachWELL method with 10% QA/QC samples sent for fire assay.

- Multi-element Niton XRF analyzes of drill samples and flitch cross sections. Using various indicator elements for example Sr 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 55º to the East. When a new mining flitch is opened, a cut perpendicular to the strike of the mineralisation will provide a face onto which the ore contact can be painted based on the drilling and XRF analyses. A spotter at the working face will ensure that dig lines are not exceeded to minimize dilution.

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 to test drilling, sampling, modelling and spotting so that refinements can be made prior to commencement of operations.

### 16.1.6    Pit De-Watering

On average, approximately 610,000 $m^3$ of water will be pumped out of the pits on an annual basis and in Year 3 (the maximum year for pit dewatering), 1.2 $Mm^3$ of water will be pumped out of the pits. These quantities are based on historical rainfall data and estimated groundwater inflow volumes. Nominally, the mine will have a maximum of 6 dewatering pumps in use at any one time and each pump is capable of pumping up to 150 litres/second of water; however, the pumps will operate at lower speeds to conserve diesel fuel and each pump will operate at 1,400 rpm with a pumping rate of approximately 100 litres/second.

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 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.

Water pumped from the pits will be collected and managed in mined-out pits, sediment ponds and the OCR. During detailed mine planning, mined-out pits should be assessed for their potential to store water, particularly in areas distant to the OCR.

### 16.1.7    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

Approximately 82% of 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, it could also be completed by a separate independent contractor. The remaining 18% of waste will be used to build environmental barriers to shield local communities from mining activities and hauled to two waste rock dumps (WRDs).

AMC generated designs for three low grade stockpiles, two WRD and four environmental barriers as shown in Figure 16.5.

**Figure 16.5** **Plan View of WRDs, Low Grade Stockpiles and Environmental Barriers**
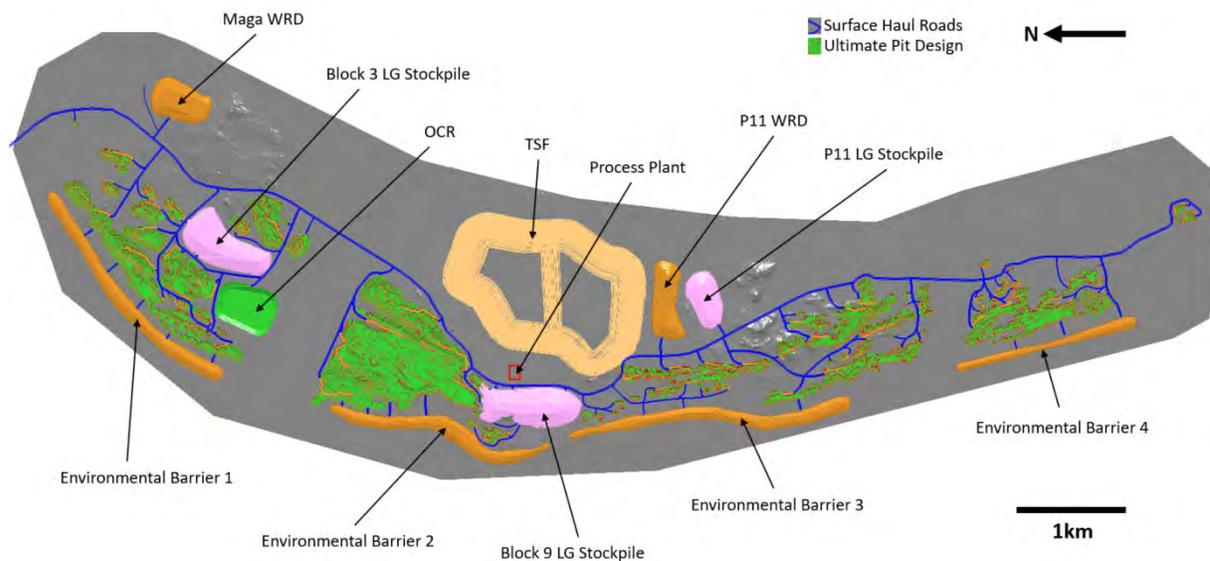


### 16.2.1 Design Parameters

As part of the 2015 FS (KCA, 2015), Golder made recommendations for WRD and stockpile design parameters. WRD and stockpiles designed by AMC are within the previous footprints and do not exceed the maximum crest elevations specified in the Golder report (Golder, 2015). The design criteria used are shown in Table 16.3.

**Table 16.3** **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 |
| Mine Swell Factor | % | 30 |

AMC has assumed a swell factor of 30% 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 material being mined and potential compaction from the mining fleet, a lower swell factor may be achieved during operation and should be quantified by measurements.

Topsoil stripped within the footprint of WRD 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

Three low-grade stockpiles will be built over first six years of production to a combined maximum size of 10.3Mt. The purpose of the stockpiles is to increase initial head grade in the early years of production. Low grade material will be re-handled and processed at the end of mine life. The maximum design capacity of the low-grade stockpiles in Million Loose Cubic Meters (MLCM) is summarized in Table 16.4.

**Table 16.4        Low Grade Stockpile Design Capacities**

| LG Stockpile | Footprint (ha) | Maximum Crest Elevation (m) | Overall Height (m) | Volume (MLCM) |
|---|---|---|---|---|
| Block 3 LG Stockpile | 29.6 | 300 | 25 | 4.97 |
| Block 9 LG Stockpile | 31.3 | 320 | 35 | 5.89 |
| P11 LG Stockpile | 14.5 | 300 | 30 | 2.12 |

The Block 3 LG Stockpile is located on a flat area between the Maga and CFU deposits where there are currently no planned open pit excavations and has a total design capacity of 4.97MLCM. This 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 Block 9 LG Stockpile situated west of the process plant has been designed with a maximum capacity of 5.89MLCM. This stockpile will provide additional environmental screening for the process plant during mining operations. The design backfills several small open pit excavations directly west of the process plant and these volumes are included in the stockpile capacity.

The P11 LG Stockpile South of the TSF has been designed with a maximum capacity of 2.12MLCM. This stockpile is intended for the storage of low grade material mined in the Southern area of the project and sits to the South of a natural ridge which separates it from the P11 waste dump.

### 16.2.3    Waste Management

The design capacity of the environmental barriers and WRDs is summarized in Table 16.5.

**Table 16.5**        **Environmental Barrier and WRD Design Capacities**

| WRD | Footprint (ha) | Maximum Crest Elevation (m) | Overall Height (m) | Volume (MLCM) |
|---|---|---|---|---|
| Maga WRD | 18.2 | 310 | 30 | 2.51 |
| P11 WRD | 18.9 | 300 | 30 | 2.48 |
| Environmental Barrier 1 | 33.3 | 295 | 15 | 2.62 |
| Environmental Barrier 2 | 29.8 | 295 | 15 | 2.26 |
| Environmental Barrier 3 | 34.5 | 295 | 15 | 2.40 |
| Environmental Barrier 4 | 15.8 | 275 | 10 | 0.81 |

Environmental barriers reach a maximum height of 15 m (three 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 must 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 WRDs.

The Maga WRD is situated in the farthest Northeast corner of the project and has a maximum capacity of 2.51MLCM. The location of the dump has been selected to avoid areas of proposed resource extension drilling.

The P11 WRD is situated south of the TSF and has been designed with a maximum capacity of 2.48MLCM. The WRD is located to the North of a natural ridge separating it from the P11 LG Stockpile.

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 recommends that the ROM is designed with a seven-day capacity to allow feed to the primary crusher during downtime caused by unfavourable weather on the mining operations, and conversely, dumping capacity during process downtime. Additional downtime capacity can be supplied by the nearby Block 9 LG stockpile when required.

## 16.3      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.

### 16.3.1    Schedule Inputs

*Mining Areas*

The production schedule is based on the reserve block models constrained to the ultimate pit design. AMC grouped the pits into 48 mining areas based on geography, tonnage and pit value. The mining areas and their associated ore and waste quantities are shown in Figure 16.6.

**Figure 16.6         Grouping of Mining Areas for Production Schedule**

| Block | Mining Area | Ore Tonnes Mt | Au grade g/t | Waste Tonnes Mt | Total Tonnes Mt |
|---|---|---|---|---|---|
| Mining Block 1 | 1 (OCR) | 3.1 | 0.45 | 6.4 | 9.5 |
| | 2 | 0.4 | 0.65 | 1.0 | 1.4 |
| | 3 | 1.4 | 0.70 | 3.8 | 5.1 |
| | 4 | 0.1 | 0.87 | 0.6 | 0.7 |
| | 5 | 0.2 | 0.65 | 0.9 | 1.0 |
| | 6 | 0.6 | 0.74 | 1.8 | 2.4 |
| | 7 | 0.3 | 0.66 | 1.2 | 1.5 |
| | 8 | 0.2 | 0.85 | 0.4 | 0.6 |
| | 9 | 0.3 | 1.09 | 1.8 | 2.1 |
| | 10 | 1.1 | 0.54 | 2.3 | 3.4 |
| | 11 | 2.5 | 0.71 | 4.3 | 6.8 |
| | 12 | 0.3 | 0.51 | 0.4 | 0.7 |
| | 13 | 0.7 | 0.51 | 0.9 | 1.7 |
| | 14 | 0.8 | 0.54 | 1.0 | 1.8 |
| | 15 | 3.8 | 0.82 | 7.9 | 11.7 |
| Mining Block 2 | 16 | 0.3 | 0.52 | 0.6 | 0.9 |
| | 17 | 0.3 | 0.69 | 0.9 | 1.2 |
| | 18 | 0.4 | 0.45 | 0.4 | 0.8 |
| | 19 | 2.3 | 0.57 | 1.7 | 4.0 |
| | 20 | 2.9 | 0.49 | 6.2 | 9.1 |
| | 21 | 3.3 | 0.48 | 6.4 | 9.8 |
| | 22 | 1.5 | 0.75 | 0.7 | 2.2 |
| | 23 | 4.0 | 0.50 | 4.4 | 8.4 |
| | 24 | 1.3 | 0.83 | 0.7 | 2.0 |
| | 25 | 1.8 | 0.53 | 2.3 | 4.1 |
| | 26 | 2.9 | 0.94 | 3.7 | 6.6 |
| | 27 | 4.2 | 0.59 | 10.8 | 14.9 |
| Mining Block 3 | 28 | 0.3 | 0.56 | 0.3 | 0.6 |
| | 29 | 0.2 | 0.58 | 0.3 | 0.5 |
| | 30 | 0.6 | 0.66 | 0.7 | 1.4 |
| | 31 | 0.4 | 0.89 | 1.9 | 2.3 |
| | 32 | 0.6 | 0.79 | 1.3 | 1.9 |
| | 33 | 0.8 | 0.46 | 0.7 | 1.5 |
| | 34 | 0.8 | 0.58 | 1.2 | 2.0 |
| | 35 | 1.0 | 0.64 | 1.2 | 2.2 |
| | 36 | 0.4 | 0.64 | 0.6 | 0.9 |
| | 37 | 0.2 | 0.80 | 0.7 | 0.9 |
| | 38 | 0.5 | 0.58 | 0.8 | 1.3 |
| | 39 | 0.2 | 0.63 | 0.5 | 0.7 |
| | 40 | 0.7 | 0.48 | 1.0 | 1.7 |
| | 41 | 2.6 | 0.78 | 3.3 | 5.9 |
| Mining Block 4 | 42 | 0.3 | 0.47 | 0.4 | 0.6 |
| | 43 | 0.4 | 0.81 | 0.4 | 0.8 |
| | 44 | 0.5 | 0.55 | 0.5 | 1.1 |
| | 45 | 0.3 | 0.52 | 0.3 | 0.6 |
| | 46 | 1.1 | 0.55 | 1.2 | 2.4 |
| | 47 | 3.0 | 0.76 | 2.5 | 5.4 |
| Mining Block 5 | 48 | 0.2 | 0.85 | 0.4 | 0.6 |
| | **TOTAL** | **56.0** | **0.64** | **93.8** | **149.8** |

*Schedule Constraints*

The following constraints were applied to the schedule:

- Maximum total material movement of 15.0Mt per year.

- Maximum of 10Mtpa per mining area based on equipment allocation constraints.

- Limit vertical advance rate of ten 6 m benches per year.

- Maximum number of 10 simultaneously active mining areas per year.

- Mining Blocks 4 and 5 to be mined during the dry season only; access to these blocks is only possible when the river bed can be traversed as the existing bridge will not be upgraded to accommodate haul trucks.

*Schedule Targets*

The key targets of the schedule are:

- 4.5 Mtpa steady state processing throughput.

- 1 g Au/t target head grade for the first two years of production.

- Waste mining production profile maintained in line with TSF and environmental barriers construction requirements.

### 16.3.2    Mine Schedule Summary

Mining operation extends over 12 years, including the pre-production period. The total annual ex-pit material movement peaks at 15 Mtpa from 2020 to 2022, before dropping to approximately 12 Mtpa for the next five years, and 10 Mtpa at the end of mine life. Year 2019 consists of 7 months.

The final two years of production, 2032 and 2033, are comprised of processing low grade material from stockpiles.

The production schedule is summarized in Table 16.6.

**Table 16.6**　　　　　　**Annual Mine Production Schedule**

| Material Mined | Unit | Total | 2019 | 2020 | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Ore | Mt | 56.0 | 2.2 | 4.6 | 6.5 | 5.5 | 5.7 | 4.2 | 4.3 | 4.1 | 4.6 | 4.5 | 3.3 | 3.5 | 3.1 | - | - |
| Gold Grade | g Au/t | 0.64 | 0.48 | 0.71 | 0.86 | 0.75 | 0.80 | 0.65 | 0.55 | 0.61 | 0.54 | 0.53 | 0.50 | 0.48 | 0.47 | - | - |
| Waste | Mt | 93.8 | 5.1 | 10.2 | 8.5 | 9.5 | 7.2 | 7.8 | 7.7 | 7.9 | 7.4 | 5.7 | 6.7 | 6.5 | 3.6 | - | - |
| Strip Ratio | W:O | 1.68 | 2.3 | 2.2 | 1.3 | 1.7 | 1.3 | 1.9 | 1.8 | 1.9 | 1.6 | 1.3 | 2.1 | 1.9 | 1.2 | - | - |
| | | | | | | | | | | | | | | | | | |
| Ore Mine to Process | Mt | 41.1 | - | 0.7 | 3.3 | 2.7 | 4.5 | 3.4 | 3.7 | 4.1 | 4.5 | 4.5 | 3.3 | 3.5 | 3.1 | - | - |
| Gold Grade | g Au/t | 0.70 | - | 1.20 | 1.28 | 1.11 | 0.92 | 0.72 | 0.59 | 0.61 | 0.54 | 0.53 | 0.50 | 0.48 | 0.47 | - | - |
| Ore Mine to Stockpile | Mt | 14.9 | 2.2 | 4.0 | 3.2 | 2.8 | 1.2 | 0.8 | 0.66 | - | 0.10 | - | - | - | - | - | - |
| Gold Grade | g Au/t | 0.47 | 0.48 | 0.62 | 0.44 | 0.40 | 0.34 | 0.32 | 0.31 | - | 0.35 | - | - | - | - | - | - |
| Ore Stockpile to Process | Mt | 14.9 | - | - | 1.2 | 1.8 | - | 1.1 | 0.8 | 0.4 | - | - | 1.2 | 1.0 | 1.4 | 4.5 | 1.3 |
| Gold Grade | g Au/t | 0.47 | - | 0.82 | 0.76 | 0.70 | - | 0.66 | 0.66 | 0.39 | - | 0.57 | 0.39 | 0.39 | 0.38 | 0.32 | 0.32 |
| | | | | | | | | | | | | | | | | | |
| **Material Processed** | | | | | | | | | | | | | | | | | |
| Process Feed | Mt | 56.0 | - | 0.7 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 1.3 |
| Gold Grade | g Au/t | 0.64 | - | 1.20 | 1.14 | 0.95 | 0.92 | 0.71 | 0.60 | 0.59 | 0.54 | 0.53 | 0.47 | 0.46 | 0.44 | 0.32 | 0.32 |

Figure 16.7 shows the annual production schedule graphically.

**Figure 16.7**　　　　　　**Annual Total Material Mined**
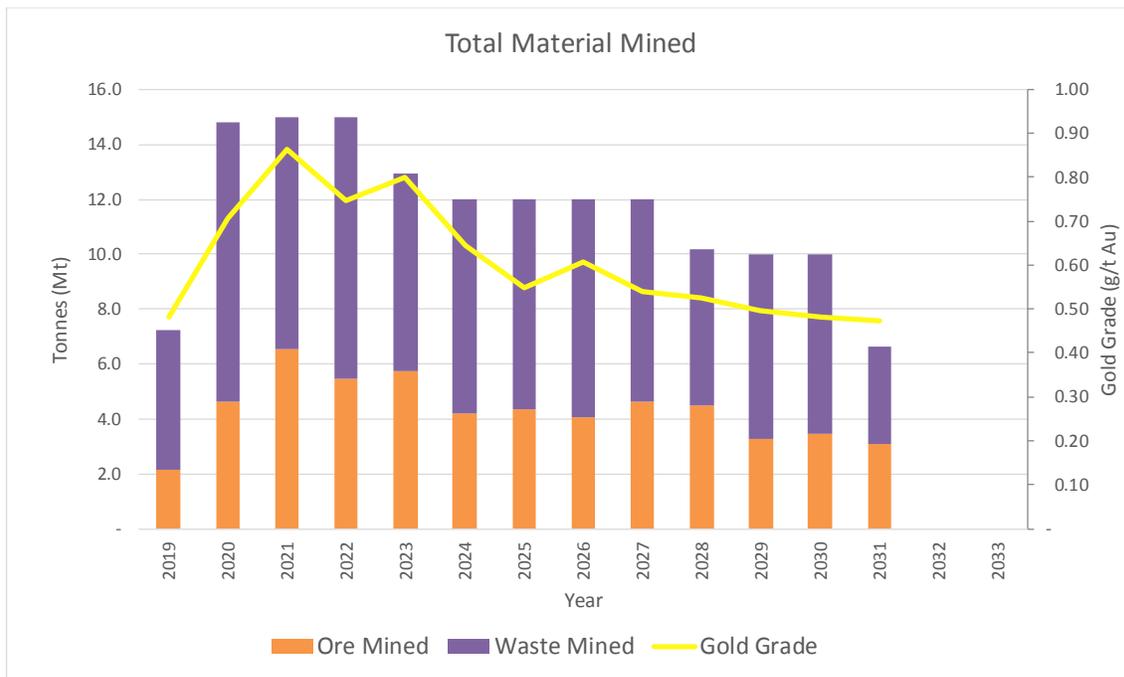
*Mining Sequence by Mining Block*

Mining commences with the pre-production development of the OCR in Mining Block 1. The OCR construction is scheduled to commence in June 2019 and takes 10 months to complete. Mining Blocks 1 and 2 commence in 2020 followed by the remaining mining areas.

Mining of Mining Blocks 3, 4 and 5 have been staggered in the schedule to allow for the development of the required surface haul road infrastructure.

Figure 16.8 and Figure 16.9 summarize the mining sequence.

**Figure 16.8          Mining Sequence by Mining Block**

| | 2019 | 2020 | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mining Block 1 | | | | | | | | | | | | | | | |
| Mining Block 2 | | | | | | | | | | | | | | | |
| Mining Block 3 | | | | | | | | | | | | | | | |
| Mining Block 4 | | | | | | | | | | | | | | | |
| Mining Block 5 | | | | 1.0 | | | | | | | | | | | |

**Figure 16.9          Annual Total Material Mined by Mining Block**
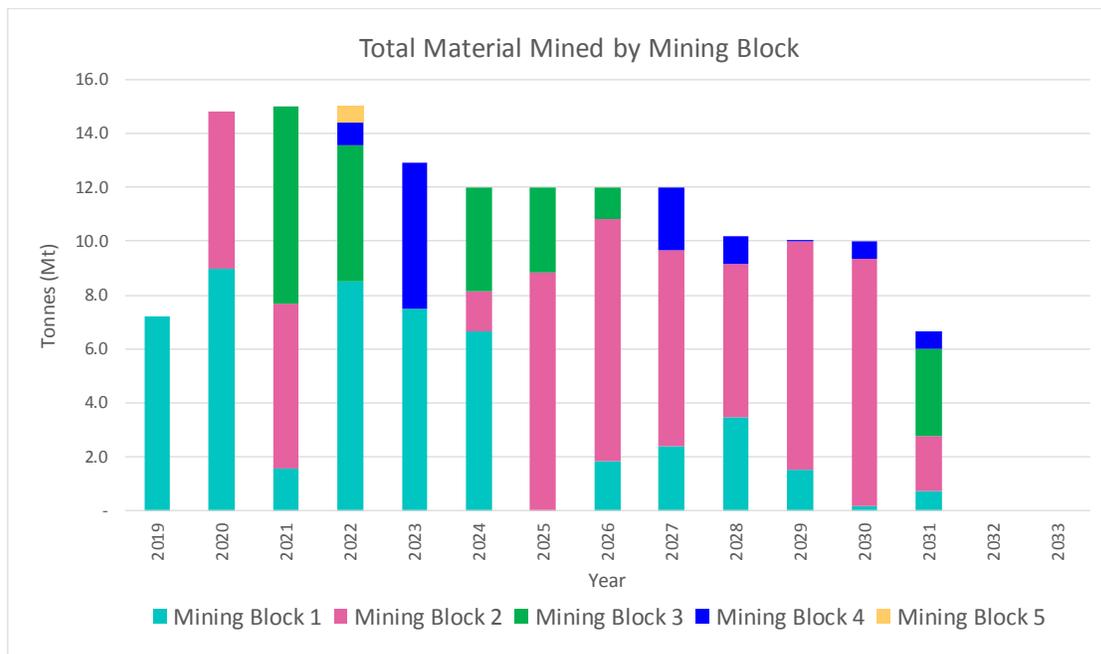


*Mining Sequence by Mining Area*

The mining sequence of the 48 mining areas is summarized in Figure 16.10.

**Figure 16.10          Mining Sequence by Mining Area**

| BLOCK | Mining Area | 2019 | 2020 | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mining Block 1 | 1 | ■ | ■ | | | | | | | | | | | |
| | 2 | | | | | | | | | ■ | | | | |
| | 3 | | | | | | ■ | | | | | | | |
| | 4 | ■ | | | | | | | | | | | | |
| | 5 | | | | | | | | | ■ | | | | |
| | 6 | | ■ | | | | | | | | | | | |
| | 7 | | | | | | ■ | | | | | | | |
| | 8 | | ■ | | | | | | | | | | | |
| | 9 | | ■ | | | | | | | | | | | |
| | 10 | | | | | | | | | | ■ | | | |
| | 11 | | | ■ | ■ | | | | | | | | | |
| | 12 | | | | | | | | | | | | | ■ |
| | 13 | | | | | | | | | | | ■ | ■ | |
| | 14 | | | | | | | | ■ | | | | | |
| | 15 | | ■ | ■ | ■ | ■ | | | | | | | | |
| Mining Block 2 | 16 | | | | | | | | | ■ | | | | |
| | 17 | | ■ | | | | | | | | | | | |
| | 18 | | | | | | | | | | | | | ■ |
| | 19 | | | | | | ■ | ■ | | | | | | |
| | 20 | | | | | | | | | | | ■ | ■ | |
| | 21 | | | | | | | | | | | ■ | ■ | ■ |
| | 22 | | | ■ | | | | | | | | | | |
| | 23 | | | | | | | | | ■ | ■ | | | |
| | 24 | | ■ | | | | | | | | | | | |
| | 25 | | | | | | | | ■ | ■ | | | | |
| | 26 | | ■ | ■ | | | | | | | | | | |
| | 27 | | | | | | | ■ | ■ | | | | | |
| Mining Block 3 | 28 | | | | | | | ■ | | | | | | |
| | 29 | | | | | | | ■ | | | | | | |
| | 30 | | | | ■ | | | | | | | | | |
| | 31 | | | ■ | | | | | | | | | | |
| | 32 | | | | ■ | | | | | | | | | |
| | 33 | | | | | | | | | | | | | ■ |
| | 34 | | | | | | | ■ | | | | | | |
| | 35 | | | | | | ■ | | | | | | | |
| | 36 | | | | | | ■ | | | | | | | |
| | 37 | | | | ■ | | | | | | | | | |
| | 38 | | | | | | | ■ | ■ | | | | | |
| | 39 | | | | | | ■ | | | | | | | |
| | 40 | | | | | | | | | | | | | ■ |
| | 41 | | | ■ | ■ | | | | | | | | | |
| Mining Block 4 | 42 | | | | | | | | | | | | | ■ |
| | 43 | | | | ■ | | | | | | | | | |
| | 44 | | | | | | | | | | ■ | ■ | | |
| | 45 | | | | | | | | | | | | ■ | |
| | 46 | | | | | | | | | ■ | | | | |
| | 47 | | | | | ■ | | | | | | | | |
| Mining Block 5 | 48 | | | | ■ | | | | | | | | | |

## 16.3.3    Process Feed Schedule

The schedule includes the following process plant production ramp-up:

- 40% capacity in October 2020.

- 65% capacity in November 2020.

- 75% capacity in December 2020.

As a result, the total process plant throughput in 2020 is 0.68Mt, followed by a yearly throughput of 4.5Mtpa from year 2021.

As presented in Figure 16.11, targeted process feed is achieved on a yearly basis from 2021 to 2032. In 2032 and 2033, process feed consists of low grade ore re-handled from the stockpiles.

**Figure 16.11          Process Feed Schedule**
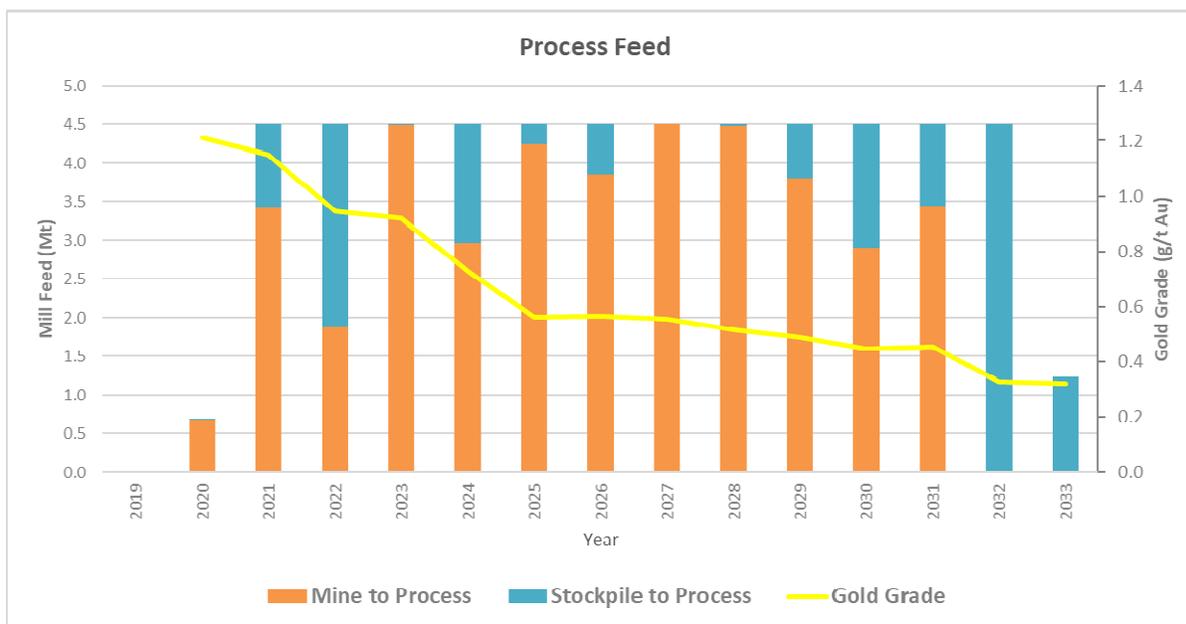


### 16.3.4    Stockpiling Strategy

To achieve the target head grade of 1 g Au/t in the first two years, AMC split the ore into three categories as follows:

- Low grade: Gold grade > mining block cut-off grade and < 0.35 g Au/t.

- Medium grade: Gold grade >=0.35 g Au/t and <0.45 g Au/t.

- High grade: Gold grade >=0.45 g Au/t.

As illustrated by Figure 16.12, stockpiling reaches a peak of 10.3Mt by 2023. The stockpile tonnages presented are total quantities by grade ranges (low, medium, and high) and not by stockpile location.

**Figure 16.12          Total Stockpile Size over Life of Mine by Grade Category**
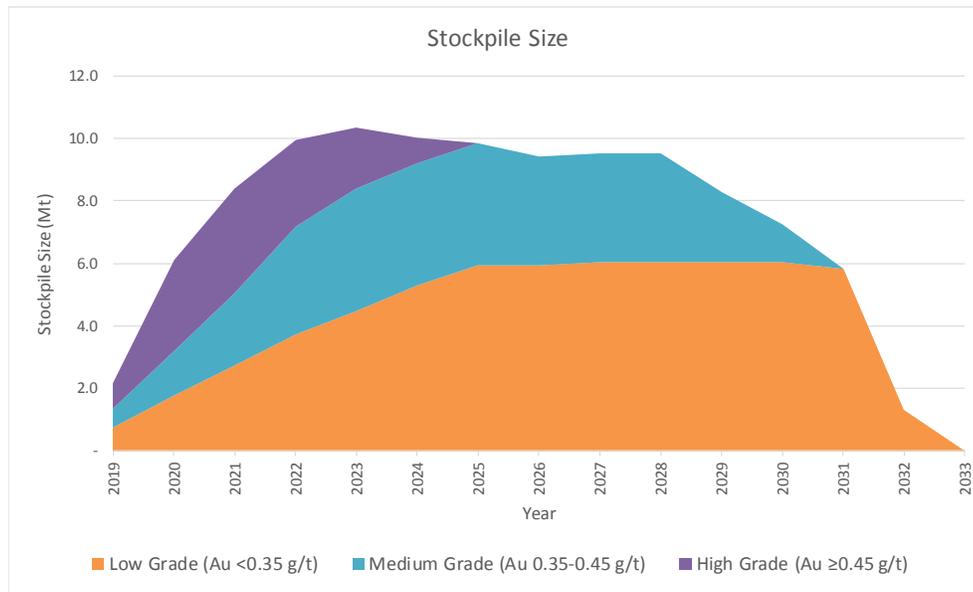


Low and medium grade material will be sent to the closest available stockpile. High grade material stockpiled in initial years will be stored at the Block 9 LG Stockpile adjacent to the process facilities.

The high-grade stockpile is built up during the initial pit start-up; no additional high-grade material is stockpiled after 2022 (refer to Figure 16.13). The medium and low-grade materials are sent to stockpiles until 2024 after which these stockpiles are depleted until the end of the mine life.

**Figure 16.13          High Grade Stockpile Movement**
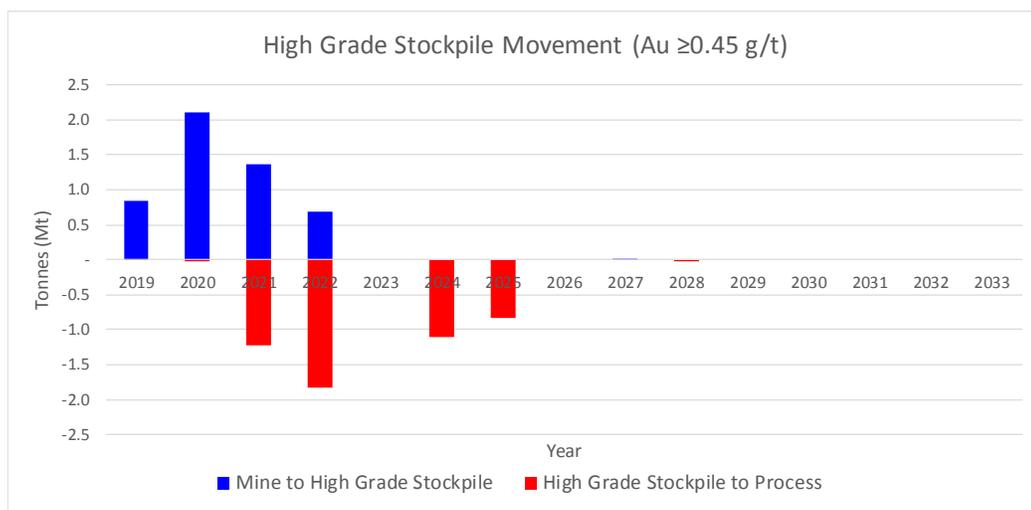
### 16.3.5 Waste Movement

In total, 77.1Mt of waste which accounts for 82% will be sent to the TSF for construction. The remainder of the waste is prioritized for environmental barrier construction followed by development of the two WRD. Life of mine waste movements are summarized in Table 16.7.

**Table 16.7          Life of Mine Waste Movement Schedule**

| Waste Movement | Unit | Total | 2019 | 2020 | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| % of Waste sent to TSF | % | 82% | 59% | 93% | 94% | 94% | 79% | 37% | 90% | 90% | 61% | 99% | 100% | 96% | 55% |
| | | | | | | | | | | | | | | | |
| Mine to TSF | Mt | 77.1 | 3.0 | 9.4 | 8.0 | 8.9 | 5.6 | 2.9 | 6.9 | 7.2 | 4.5 | 5.6 | 6.7 | 6.3 | 2.0 |
| Mine to Environmental Barrier 1 | Mt | 3.5 | 2.1 | 0.0 | - | 0.0 | 1.5 | - | - | - | - | - | - | - | - |
| Mine to Environmental Barrier 2 | Mt | 3.2 | - | 0.8 | - | - | - | 0.4 | 0.4 | - | 1.3 | 0.0 | - | 0.2 | 0.1 |
| Mine to Environmental Barrier 3 | Mt | 3.2 | - | - | 0.5 | 0.2 | - | 2.2 | 0.3 | - | - | - | - | - | - |
| Mine to Environmental Barrier 4 | Mt | 1.1 | - | - | - | 0.4 | 0.1 | - | - | - | 0.6 | - | 0.0 | 0.0 | 0.0 |
| Mine to Maga WRD | Mt | 3.5 | - | - | - | - | 0.0 | 2.4 | - | - | 1.0 | 0.0 | - | 0.0 | 0.0 |
| Mine to P11 WRD | Mt | 2.2 | - | - | - | - | - | - | - | 0.8 | - | - | - | - | 1.4 |
| **Total Waste** | **Mt** | **93.8** | **5.1** | **10.2** | **8.5** | **9.5** | **7.2** | **7.8** | **7.7** | **7.9** | **7.4** | **5.7** | **6.7** | **6.5** | **3.6** |

### 16.3.6 Production Schedule Details

The detailed production schedule was generated in monthly periods for the first two years, followed by quarterly periods for two years, and in yearly increments thereafter. The detailed production schedule is shown in Table 16.8.

**Table 16.8    Detailed Production Schedule**

| Category | Item | Unit | LOM Total | Jun 2019 | Jul 2019 | Aug 2019 | Sep 2019 | Oct 2019 | Nov 2019 | Dec 2019 | Jan 2020 | Feb 2020 | Mar 2020 | Apr 2020 | May 2020 | Jun 2020 | Jul 2020 | Aug 2020 | Sep 2020 | Oct 2020 | Nov 2020 | Dec 2020 | Q1 2021 | Q2 2021 | Q3 2021 | Q4 2021 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Process Feed** | Process Feed | Mt | 56.0 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 0.2 | 0.2 | 0.3 | 1.1 | 1.1 | 1.1 | 1.1 |
| | Gold Grade | g Au/t | 0.64 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 1.19 | 1.20 | 1.21 | 1.17 | 1.41 | 0.95 | 1.05 |
| **Movement** | Ore Mined | Mt | 56.0 | 0.2 | 0.3 | 0.3 | 0.3 | 0.3 | 0.4 | 0.4 | 0.4 | 0.4 | 0.4 | 0.3 | 0.2 | 0.2 | 0.4 | 0.4 | 0.4 | 0.5 | 0.6 | 0.5 | 1.5 | 2.2 | 1.3 | 1.5 |
| | Gold grade | g Au/t | 0.64 | 0.56 | 0.45 | 0.45 | 0.45 | 0.46 | 0.47 | 0.53 | 0.44 | 0.45 | 0.46 | 0.74 | 0.81 | 1.11 | 0.73 | 0.68 | 0.77 | 0.73 | 0.78 | 0.84 | 0.93 | 0.92 | 0.75 | 0.82 |
| | Waste Mined | Mt | 93.81 | 0.2 | 0.7 | 0.7 | 0.8 | 0.8 | 0.9 | 0.9 | 0.8 | 0.9 | 0.9 | 1.0 | 1.1 | 0.9 | 0.8 | 0.8 | 0.8 | 0.7 | 0.7 | 0.7 | 2.3 | 1.5 | 2.4 | 2.3 |
| | Total Mined | Mt | 149.80 | 0.5 | 1.0 | 1.0 | 1.1 | 1.1 | 1.3 | 1.3 | 1.2 | 1.3 | 1.3 | 1.3 | 1.3 | 1.2 | 1.2 | 1.3 | 1.3 | 1.3 | 1.3 | 1.3 | 3.8 | 3.8 | 3.8 | 3.8 |
| | Strip ratio | W:O | 1.68 | 1.0 | 2.5 | 2.5 | 2.7 | 2.6 | 2.5 | 2.3 | 2.3 | 2.4 | 2.6 | 3.4 | 5.6 | 3.9 | 2.1 | 2.0 | 2.0 | 1.4 | 1.2 | 1.4 | 1.6 | 0.7 | 1.8 | 1.6 |
| **BCM** | Ore BCM | M BCM | 31.1 | 0.1 | 0.2 | 0.2 | 0.2 | 0.2 | 0.2 | 0.2 | 0.2 | 0.2 | 0.2 | 0.2 | 0.1 | 0.1 | 0.2 | 0.2 | 0.2 | 0.3 | 0.3 | 0.3 | 0.8 | 1.3 | 0.8 | 0.8 |
| | Waste BCM | M BCM | 53.8 | 0.1 | 0.4 | 0.4 | 0.5 | 0.5 | 0.5 | 0.5 | 0.4 | 0.5 | 0.5 | 0.5 | 0.6 | 0.5 | 0.5 | 0.5 | 0.4 | 0.4 | 0.4 | 0.4 | 1.3 | 0.9 | 1.4 | 1.3 |
| | Total BCM | M BCM | 84.9 | 0.3 | 0.6 | 0.6 | 0.6 | 0.7 | 0.7 | 0.7 | 0.6 | 0.7 | 0.7 | 0.7 | 0.7 | 0.6 | 0.7 | 0.7 | 0.6 | 0.7 | 0.7 | 0.7 | 2.2 | 2.1 | 2.2 | 2.1 |
| **Ore movement** | Mine to Process | Mt | 41.1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 0.2 | 0.2 | 0.3 | 0.9 | 1.1 | 0.6 | 0.7 |
| | Gold grade | g Au/t | 0.70 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 1.19 | 1.20 | 1.21 | 1.26 | 1.41 | 1.11 | 1.28 |
| | Mine to Stockpile | Mt | 14.9 | 0.2 | 0.3 | 0.3 | 0.3 | 0.3 | 0.4 | 0.4 | 0.4 | 0.4 | 0.4 | 0.3 | 0.2 | 0.2 | 0.4 | 0.4 | 0.4 | 0.4 | 0.3 | 0.2 | 0.6 | 1.1 | 0.7 | 0.8 |
| | Gold grade | g Au/t | 0.47 | 0.56 | 0.45 | 0.45 | 0.45 | 0.46 | 0.47 | 0.53 | 0.44 | 0.45 | 0.46 | 0.74 | 0.81 | 1.11 | 0.73 | 0.68 | 0.77 | 0.54 | 0.46 | 0.43 | 0.42 | 0.43 | 0.45 | 0.46 |
| | Stockpile to Process | Mt | 14.9 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 0.0 | 0.2 | - | 0.5 | 0.5 |
| | Gold grade | g Au/t | 0.47 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 0.82 | 0.81 | - | 0.76 | 0.74 |
| | Total to Process | Mt | 56.0 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 0.2 | 0.2 | 0.3 | 1.1 | 1.1 | 1.1 | 1.1 |
| | Gold grade | g Au/t | 0.64 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 1.19 | 1.20 | 1.21 | 1.17 | 1.41 | 0.95 | 1.05 |
| | % of Waste sent to TSF | % | 82% | 0% | 0% | 0% | 51% | 100% | 100% | 100% | 100% | 100% | 100% | 100% | 100% | 100% | 77% | 65% | 96% | 100% | 100% | 66% | 100% | 100% | 80% | 100% |
| **Waste movement** | Mine to TSF | Mt | 77.1 | - | - | - | 0.4 | 0.8 | 0.9 | 0.9 | 0.8 | 0.9 | 0.9 | 1.0 | 1.1 | 0.9 | 0.6 | 0.5 | 0.8 | 0.7 | 0.7 | 0.5 | 2.3 | 1.5 | 1.9 | 2.3 |
| | Mine to Environmental Barrier 1 | Mt | 3.5 | 0.2 | 0.7 | 0.7 | 0.4 | - | - | - | 0.0 | 0.0 | 0.0 | - | - | - | - | - | - | - | - | - | - | - | - | - |
| | Mine to Environmental Barrier 2 | Mt | 3.2 | - | - | - | - | - | - | - | - | - | - | - | - | - | 0.2 | 0.3 | 0.0 | - | - | 0.3 | - | - | - | - |
| | Mine to Environmental Barrier 3 | Mt | 3.2 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 0.0 | - | 0.5 | - |
| | Mine to Environmental Barrier 4 | Mt | 1.1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| | Mine to Maga WRD | Mt | 3.5 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| | Mine to P11 WRD | Mt | 2.2 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| | Total Waste | Mt | 93.8 | 0.2 | 0.7 | 0.7 | 0.8 | 0.8 | 0.9 | 0.9 | 0.8 | 0.9 | 0.9 | 1.0 | 1.1 | 0.9 | 0.8 | 0.8 | 0.8 | 0.7 | 0.7 | 0.7 | 2.3 | 1.5 | 2.4 | 2.3 |

| Category | Item | Unit | LOM Total | Q1 2022 | Q2 2022 | Q3 2022 | Q4 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Process Feed** | Process Feed | Mt | 56.0 | 1.1 | 1.1 | 1.1 | 1.1 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 1.3 |
| | Gold Grade | g Au/t | 0.64 | 0.92 | 0.95 | 0.94 | 0.97 | 0.92 | 0.71 | 0.60 | 0.59 | 0.54 | 0.53 | 0.47 | 0.46 | 0.44 | 0.32 | 0.32 |

| | | Unit | Total | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Movement** | Ore Mined | Mt | 56.0 | 1.2 | 1.3 | 1.4 | 1.6 | 5.7 | 4.2 | 4.3 | 4.1 | 4.6 | 4.5 | 3.3 | 3.5 | 3.1 | - | - |
| | Gold grade | g Au/t | 0.64 | 0.76 | 0.75 | 0.74 | 0.74 | 0.80 | 0.65 | 0.55 | 0.61 | 0.54 | 0.53 | 0.50 | 0.48 | 0.47 | - | - |
| | Waste Mined | Mt | 93.81 | 2.5 | 2.5 | 2.3 | 2.2 | 7.2 | 7.8 | 7.7 | 7.9 | 7.4 | 5.7 | 6.7 | 6.5 | 3.6 | - | - |
| | Total Mined | Mt | 149.80 | 3.8 | 3.8 | 3.8 | 3.8 | 12.9 | 12.0 | 12.0 | 12.0 | 12.0 | 10.2 | 10.0 | 10.0 | 6.7 | - | - |
| | Strip ratio | W:O | 1.68 | 2.0 | 2.0 | 1.7 | 1.4 | 1.3 | 1.9 | 1.8 | 1.9 | 1.6 | 1.3 | 2.1 | 1.9 | 1.2 | - | - |
| **BCM** | Ore BCM | M BCM | 31.1 | 0.7 | 0.7 | 0.8 | 0.9 | 3.2 | 2.4 | 2.4 | 2.2 | 2.5 | 2.3 | 2.0 | 1.9 | 1.6 | - | - |
| | Waste BCM | M BCM | 53.8 | 1.5 | 1.5 | 1.4 | 1.3 | 4.1 | 4.6 | 4.4 | 4.5 | 4.1 | 3.1 | 4.1 | 3.7 | 2.0 | - | - |
| | Total BCM | M BCM | 84.9 | 2.1 | 2.2 | 2.2 | 2.1 | 7.3 | 6.9 | 6.8 | 6.7 | 6.6 | 5.5 | 6.1 | 5.6 | 3.6 | - | - |
| **Ore movement** | Mine to Process | Mt | 41.1 | 0.7 | 0.6 | 0.7 | 0.7 | 4.5 | 3.4 | 3.7 | 4.1 | 4.5 | 4.5 | 3.3 | 3.5 | 3.1 | - | - |
| | Gold grade | g Au/t | 0.70 | 1.03 | 1.14 | 1.11 | 1.18 | 0.92 | 0.72 | 0.59 | 0.61 | 0.54 | 0.53 | 0.50 | 0.48 | 0.47 | - | - |
| | Mine to Stockpile | Mt | 14.9 | 0.5 | 0.7 | 0.7 | 0.9 | 1.2 | 0.8 | 0.7 | - | 0.1 | - | - | - | - | - | - |
| | Gold grade | g Au/t | 0.47 | 0.39 | 0.38 | 0.40 | 0.41 | 0.34 | 0.32 | 0.31 | - | 0.35 | - | - | - | - | - | - |
| | Stockpile to Process | Mt | 14.9 | 0.4 | 0.5 | 0.4 | 0.5 | - | 1.1 | 0.8 | 0.4 | - | 0.0 | 1.2 | 1.0 | 1.4 | 4.5 | 1.3 |
| | Gold grade | g Au/t | 0.47 | 0.72 | 0.71 | 0.70 | 0.68 | - | 0.66 | 0.66 | 0.39 | - | 0.57 | 0.39 | 0.39 | 0.38 | 0.32 | 0.32 |
| | Total to Process | Mt | 56.0 | 1.1 | 1.1 | 1.1 | 1.1 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 4.5 | 1.3 |
| | Gold grade | g Au/t | 0.64 | 0.92 | 0.94 | 0.94 | 0.97 | 0.92 | 0.71 | 0.60 | 0.59 | 0.54 | 0.53 | 0.47 | 0.46 | 0.44 | 0.32 | 0.32 |
| | % of Waste sent to TSF | % | 82% | 100% | 92% | 100% | 82% | 79% | 37% | 90% | 90% | 61% | 99% | 100% | 96% | 55% | 0% | 0% |
| **Waste movement** | Mine to TSF | Mt | 77.1 | 2.5 | 2.3 | 2.3 | 1.8 | 5.6 | 2.9 | 6.9 | 7.2 | 4.5 | 5.6 | 6.7 | 6.3 | 2.0 | - | - |
| | Mine to Environmental Barrier 1 | Mt | 3.5 | - | - | - | 0.0 | 1.5 | - | - | - | - | - | - | - | - | - | - |
| | Mine to Environmental Barrier 2 | Mt | 3.2 | - | - | - | - | - | 0.4 | 0.4 | - | 1.3 | 0.0 | - | 0.2 | 0.1 | - | - |
| | Mine to Environmental Barrier 3 | Mt | 3.2 | - | 0.2 | - | 0.0 | - | 2.2 | 0.3 | - | - | - | - | - | - | - | - |
| | Mine to Environmental Barrier 4 | Mt | 1.1 | - | - | - | 0.4 | 0.1 | - | - | - | 0.6 | - | 0.0 | 0.0 | 0.0 | - | - |
| | Mine to Maga WRD | Mt | 3.5 | - | - | - | - | 0.0 | 2.4 | - | - | 1.0 | 0.0 | - | 0.0 | 0.0 | - | - |
| | Mine to P11 WRD | Mt | 2.2 | - | - | - | - | - | - | - | 0.8 | - | - | - | - | 1.4 | - | - |
| | Total Waste | Mt | 93.8 | 2.5 | 2.5 | 2.3 | 2.2 | 7.2 | 7.8 | 7.7 | 7.9 | 7.4 | 5.7 | 6.7 | 6.5 | 3.6 | - | - |

### 16.3.7    Equipment Requirements

AMC estimated equipment requirements based on the production schedule. Table 16.9 summarizes the truck and excavator productivity inputs used in the calculations.

**Table 16.9          Truck and Excavator Productivity Inputs**

| Input | Units | Value |
|---|---|---|
| **Shift** | | |
| Days per year | days | 365 |
| Hours Per Day | hr | 24 |
| Shift time | hr | 12 |
| Shifts per day | # | 2 |
| Effective shift time | mins/hr | 55 |
| Effective hours per day | hr | 22 |
| Downtime | days | 9 |
| Wet Season Downtime Days | days | 6 |
| Total Downtime | days | 15 |
| Working days per year | days | 350 |
| Calendar time | hr | 7700 |
| **Truck Parameters** | | |
| Truck payload | t | 27.7 |
| Truck speed | | |
|    In-pit loaded | km/hr | 12 |
|    Ex-pit loaded | km/hr | 40 |
|    Ex-pit empty | km/hr | 50 |
|    In-pit empty | km/hr | 35 |
|    On-bench/on-lift | km/hr | 25 |
| Truck availability | % | 85 |
| Truck utilization | % | 85 |
| Operator efficiency | % | 90 |
| Productive truck hours | hr | 5,007 |
| **Excavator Parameters** | | |
| Bucket size | m$^3$ | 4.5 |
| Fill factor | % | 95 |
| Bucket load | t | 5.7 |
| Passes per truck | # | 5 |
| Load time per truck | sec | 174 |
| Propel factor | % | 95 |
| Presentation factor | % | 95 |
| Excavator availability | % | 85 |
| Excavator utilization | % | 85 |
| Productivity | t/hr | 437 |
| Productivity | Mt/yr | 2.8 |

The average haul distances used in the schedule are summarized in Table 16.10.

**Table 16.10        Average Haul Distances by Material Movement and Period**

| | | | 2019 | 2020 | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mine to Process | km | 3.5 | - | 1.9 | 2.8 | 5.0 | 6.0 | 4.1 | 2.6 | 2.7 | 3.4 | 3.2 | 2.5 | 2.3 | 3.7 | - | - |
| Mine to Stockpile | km | 1.8 | 1.6 | 1.5 | 1.6 | 2.1 | 3.4 | 1.7 | 1.4 | - | 1.4 | - | - | - | - | - | - |
| Stockpile to Process | km | 3.4 | - | 3.6 | 3.2 | 3.3 | - | 3.4 | 3.4 | 3.6 | - | 1.4 | 3.6 | 3.6 | 3.5 | 3.3 | 3.3 |
| Mine to TSF | km | 4.3 | 4.7 | 4.1 | 4.1 | 4.9 | 6.5 | 6.6 | 3.2 | 3.5 | 4.2 | 4.9 | 3.5 | 3.2 | 4.6 | - | - |
| Mine to Environmental Barrier 1 | km | 1.8 | 1.7 | 2.0 | - | 2.0 | 2.0 | - | - | - | - | - | - | - | - | - | - |
| Mine to Environmental Barrier 2 | km | 1.7 | - | 2.0 | - | - | - | 1.5 | 1.5 | - | 1.6 | 2.0 | - | 2.0 | 2.0 | - | - |
| Mine to Environmental Barrier 3 | km | 2.3 | - | - | 2.4 | 2.5 | - | 2.2 | 2.8 | - | - | - | - | - | - | - | - |
| Mine to Environmental Barrier 4 | km | 1.8 | - | - | - | 2.3 | 1.6 | - | - | - | 1.6 | - | 1.6 | 1.6 | 1.6 | - | - |
| Mine to Maga WRD | km | 2.9 | - | - | - | - | 2.9 | 2.9 | - | - | 2.9 | 2.9 | - | 2.9 | 2.9 | - | - |
| Mine to P11 WRD | km | 2.7 | - | - | - | - | - | - | - | 2.7 | - | - | - | - | 2.7 | - | - |

The resulting truck hours and number of trucks required are shown in Figure 16.14. The minimum number of trucks was estimated based on 5,000 operating truck hrs/year and does not include spares. AMC completed a detailed haulage analysis of Mining Block 1.

**Figure 16.14        Truck Hours and Minimum Truck Requirements**
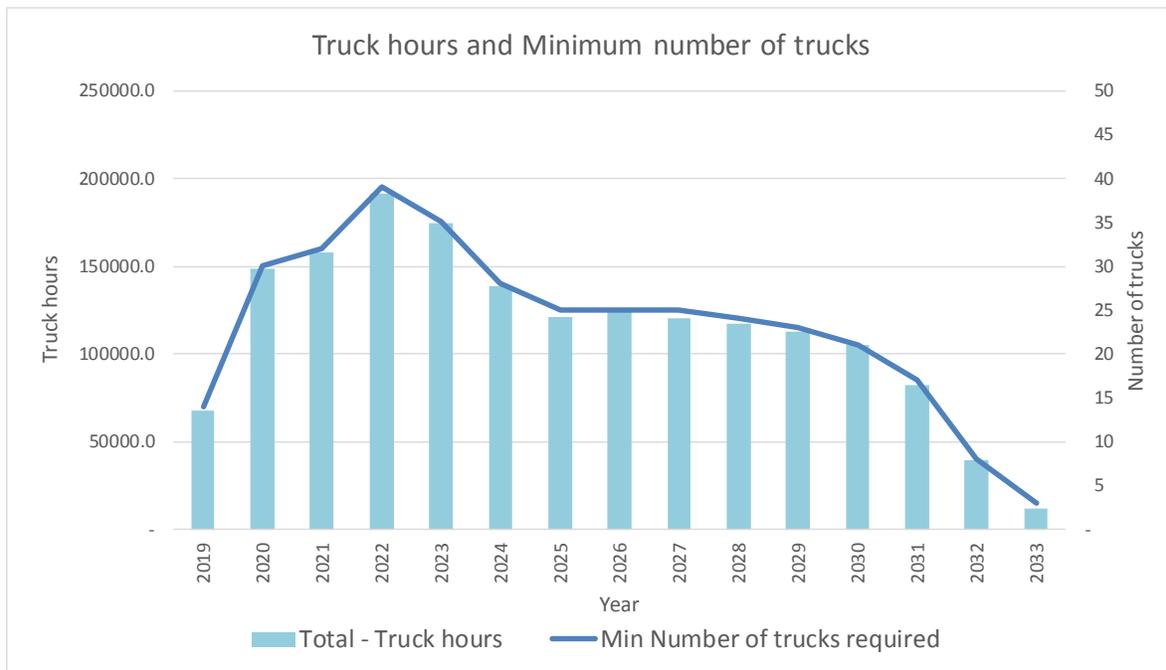


The mining fleet recommended by AMC is listed in Table 16.11.

**Table 16.11          Recommended Mining Fleet**

| Equipment | Number Required |
|---|---|
| Excavator (4.5m$^3$ bucket) e.g. Komatsu PC850 | 6 |
| Haul Truck (30t Rigid highway Truck) e.g. 30t Sinotruck Howo | 40 |
| Bulldozer e.g. CAT D8R | 2 |
| Front End Loader e.g. Komatsu WA600 | 1 |
| Water Truck (25,000L) | 2 |
| Grader e.g. CAT 140M | 2 |
| Compactor e.g. BOMAG BW 214 | 1 |
| Lowbed Transport Truck | 1 |
| Crew Bus (50 person) | 2 |
| Fuel Truck (15,000L) | 1 |
| Crane (40t) | 1 |
| Tyre Vehicle | 1 |
| Service Vehicle | 1 |
| Potable Light Towers | 10 |
| Light Vehicles e.g. Toyota Hilux | 6 |
| Mine Rescue Truck | 1 |

In May 2018, four quotations were received from local contractors with varying mining fleets specified all with truck numbers in line with AMC's assessment. The final specification of equipment will take place during contractor negotiations in detailed engineering.

### 16.3.8     Fuel Consumption

Fuel requirements for the mining equipment have been estimated based on the mine production schedule and fuel consumption assumptions listed in Table 16.12.

**Table 16.12          Parameters for Fuel Consumption**

| Equipment | Units | Value |
|---|---|---|
| Excavator (4.5m$^3$ bucket) e.g. Komatsu PC850 | L/hr | 57 |
| Haul Truck (30t Rigid highway Truck) e.g. 30t Sinotruck Howo | L/hr | 28 |

The annual fuel requirements for mobile equipment in million litres are summarized in Table 16.13. Fuel for support equipment has been estimated as 15% of total fuel requirements for excavators and trucks.

**Table 16.13          Fuel Requirements for Mining Fleet**

| Equipment | Unit | Total | 2019 | 2020 | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Trucks | ML | 47.9 | 1.9 | 4.2 | 4.4 | 5.4 | 4.9 | 3.9 | 3.4 | 3.5 | 3.4 | 3.3 | 3.1 | 2.9 | 2.3 | 1.1 | 0.3 |
| Excavators | ML | 19.5 | 0.9 | 1.9 | 2.0 | 2.0 | 1.7 | 1.6 | 1.6 | 1.6 | 1.6 | 1.3 | 1.3 | 1.3 | 0.9 | - | - |
| Support Equipment | ML | 10.1 | 0.4 | 0.9 | 1.0 | 1.1 | 1.0 | 0.8 | 0.7 | 0.8 | 0.7 | 0.7 | 0.7 | 0.6 | 0.5 | 0.2 | 0.0 |
| Total | ML | 77.6 | 3.3 | 7.0 | 7.3 | 8.4 | 7.6 | 6.3 | 5.7 | 5.8 | 5.7 | 5.3 | 5.1 | 4.9 | 3.7 | 1.3 | 0.4 |

### 16.3.9 Surface Haul Road Development

The surface haul roads will be developed based on the production schedule. Table 16.14 summarizes the haul road requirements.

**Table 16.14        Surface Haul Road Schedule**

|  | Unit | Total | 2019 | 2020 | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mining Block 1 | km | 8.6 | 3.1 | 3.5 | 0.1 |  | 0.2 | 0.3 |  | 0.7 | 0.2 | 0.2 |  |  | 0.2 |
| Mining Block 2 | km | 4.5 |  | 2.9 | 0.2 |  |  |  |  |  | 0.7 |  | 0.5 |  | 0.1 |
| Mining Block 3 | km | 8.5 |  |  | 2.5 | 2.6 |  | 1.3 | 2.0 | 0.2 |  |  |  |  |  |
| Mining Block 4 | km | 3.5 |  |  |  | 0.7 | 1.0 |  |  |  | 0.8 |  |  | 0.3 | 0.5 |
| Mining Block 5 | km | 0.1 |  |  |  | 0.1 |  |  |  |  |  |  |  |  |  |
| **Total** | **km** | **25.1** | **3.1** | **6.5** | **2.8** | **3.4** | **1.3** | **1.6** | **2.0** | **0.9** | **1.7** | **0.2** | **0.5** | **0.3** | **0.8** |

### 16.3.10 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 required 53 personnel are listed in Table 16.15.

**Table 16.15        Owner's Team Personnel**

| Position | Expat/National | Count |
|---|---|---|
| **Mine Management & Technical Services** | | |
| Technical Services Manager | Expat | 1 |
| Senior Engineer | National | 2 |
| Intermediate Engineer | National | 3 |
| Junior Engineer | National | 4 |
| Chief Surveyor | National | 2 |
| Surveyor / Rodman | National | 4 |
| Senior Geologist | National | 1 |
| Intermediate Geologist | National | 2 |
| Junior Geologist | National | 6 |
| Grade Control Technician | National | 8 |
| Secretary | National | 1 |
| **Mine Production** | | |
| Mine Superintendent | Expat | 1 |
| Mine Shift Foreman | National | 8 |
| Pump Foreman | National | 1 |
| Secretary | National | 1 |
| Pump/General Service Crew | National | 4 |
| Labourer | National | 4 |
| **Subtotal Mine** | | **53** |

The Mine Management & Technical Services team will be responsible for ensuring grade control procedures are competed 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. Contractor quotations received in May 2018 varied in number of personnel with approximately 6-10 management staff with mine production crews of approximately 50 workers per shift predominantly composed of truck operators.

## 16.4    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 for additional mining of materials and areas excluded in the Reserve Estimate.

- 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:

### 16.4.1    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 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.

- Non free dig material unexpectedly encountered

- **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.

- 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 per 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 move mining areas and allow time to pump the flooded workings should the event occur. Mine planning must be cognisant of wet season and dry season to minimize disruption on 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:** The mine optimization has excluded the flood plain areas. During operations, this boundary must be carefully revised and monitored and the mine plan altered accordingly.

- 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 mineralisation. There are numerous similar projects in West Africa. In addition, this project is 85 km from the capital Ouagadougou which will be attractive to technical personnel**.**

### 16.4.2 Opportunities

- Potential mining in sterilized areas

    - For the Mineral Reserve Estimate, AMC sterilized the river flood plain areas. Mineralization continues through these areas and has been drilled. Currently Orezone is in the process of negotiations with local authorities to seek permission for staged mining and backfilling during the dry season when the rivers are dry. AMC has recently commenced a separate study to assess the feasibility of operating within the flood plains.

- 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 minimise road construction. For some smaller satellite excavations, temporary haul road solutions may be adopted.

- Optimization of in-pit ramps

    - Due to mining in free dig material, in-pit ramps can be relatively easily re-located in detailed design to shorten haul distances. When the contractor fleet is finalized, ramps may be narrowed to save on construction and maintenance costs.

- Potential future reserves

    - This project has considered only the free dig oxide material. Oxide mineralization continues through the transition horizon and there is a potential for extracting sulphide mineralization below. AMC recommends a study to consider the future potential of mining the additional material excluded from this study.

## 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 operating costs. The flowsheet is based on unit operations including milling, Carbon in Leach (CIL) leaching, Zadra elution, gold electrowinning and carbon regeneration that are well proven in the industry.

Process plant feed comprises two main ore types, oxide (also called saprolite) and transition (specifically upper transition) material.  Both ore types are mined by 'free digging' methods with no drill and blast required in the pits. The ore has a fine particle size distribution and crushing is not required.

The plant design is considered appropriate for a project with a 13-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 project design criteria for the plant are:

- Nominal throughput of 4.5 Mtpa of a 70% oxide/30% transition blend.

- Process plant availability of 91.3%, supported by the incorporation of surge capacity and standby equipment where required.

- Sufficient automated plant control to minimize the need for continuous operator interface and allow manual override and control if and when required.

Study design documents have been prepared incorporating engineering design criteria and key metallurgical design criteria derived from the results of the metallurgical testwork carried out on Bomboré ores. The sampling protocol and metallurgical testwork are described in detail in Section 13 of this report.

### 17.1.1     Selected Process Flowsheet

The treatment plant design incorporates the following unit process operations:

- ROM ore fed through a static grizzly to a surge bin providing 15 minutes of surge capacity.

- 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 125µm.

- A hydrocyclone overflow slurry density of 40% solids for direct feed to the leach tanks.

- A leaching circuit with one (1) leach and six (6) CIL tanks to achieve the required 24 hours of residence time for optimum leach recovery. A pre-CIL leach tank is used to reduce capital cost and raise the gold tenor of the leach solution prior to loading on the carbon in the first CIL tank.

- A tailings thickener for cyanide and water recovery and to provide an optimum slurry density for disposal to the tailings storage facility to minimise water losses by evaporation on the dam.

- 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.

An overall process block flow diagram depicting the unit operations incorporated in the selected process flowsheet is presented in Figure 17.1. Plan and isometric views of the process plant are provided in Figures 17.2 and 17.3.

**Figure 17.1      Overall Process Block Flow Diagram**
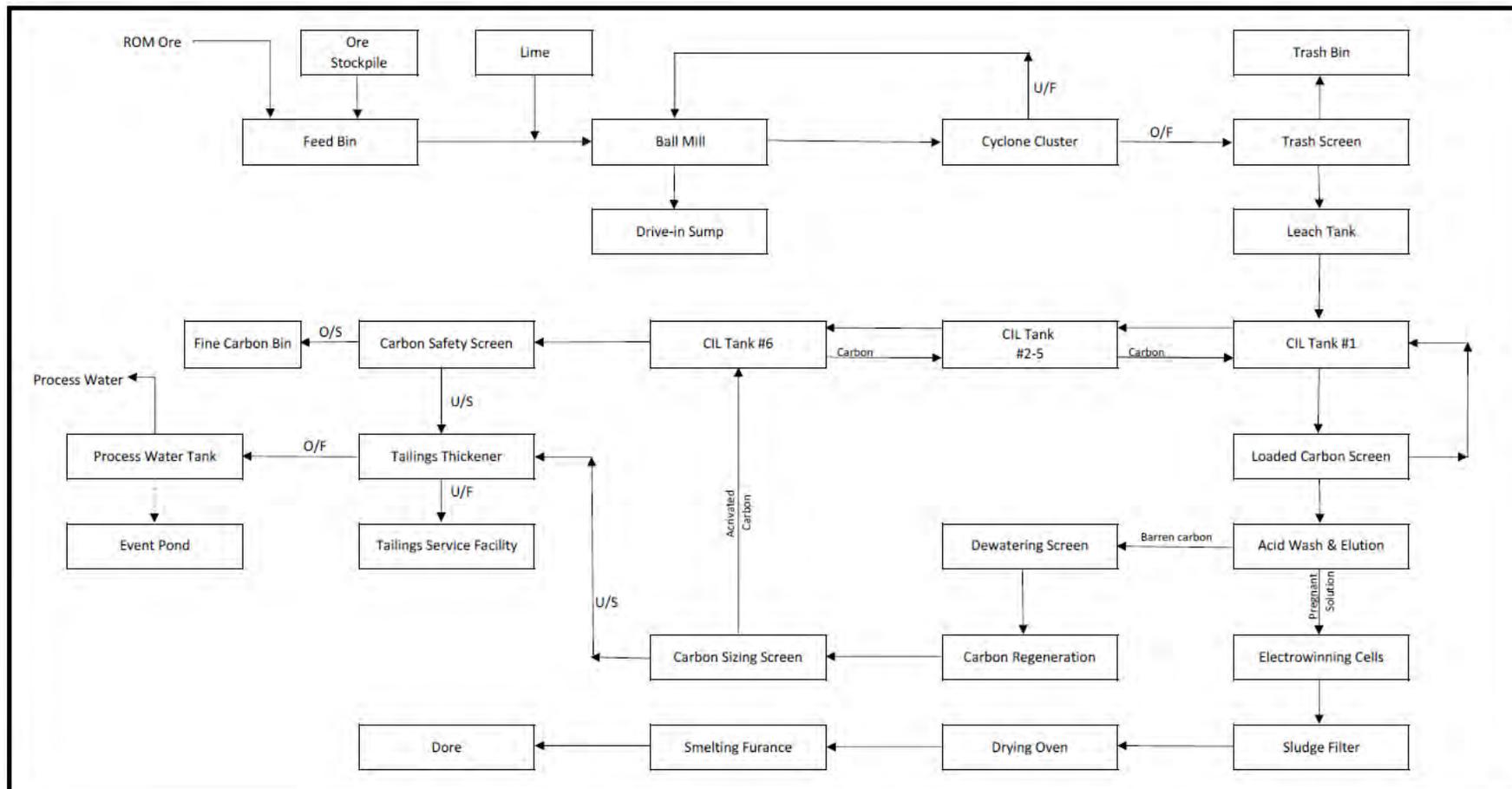
**Figure 17.2        Process Plant Plan**
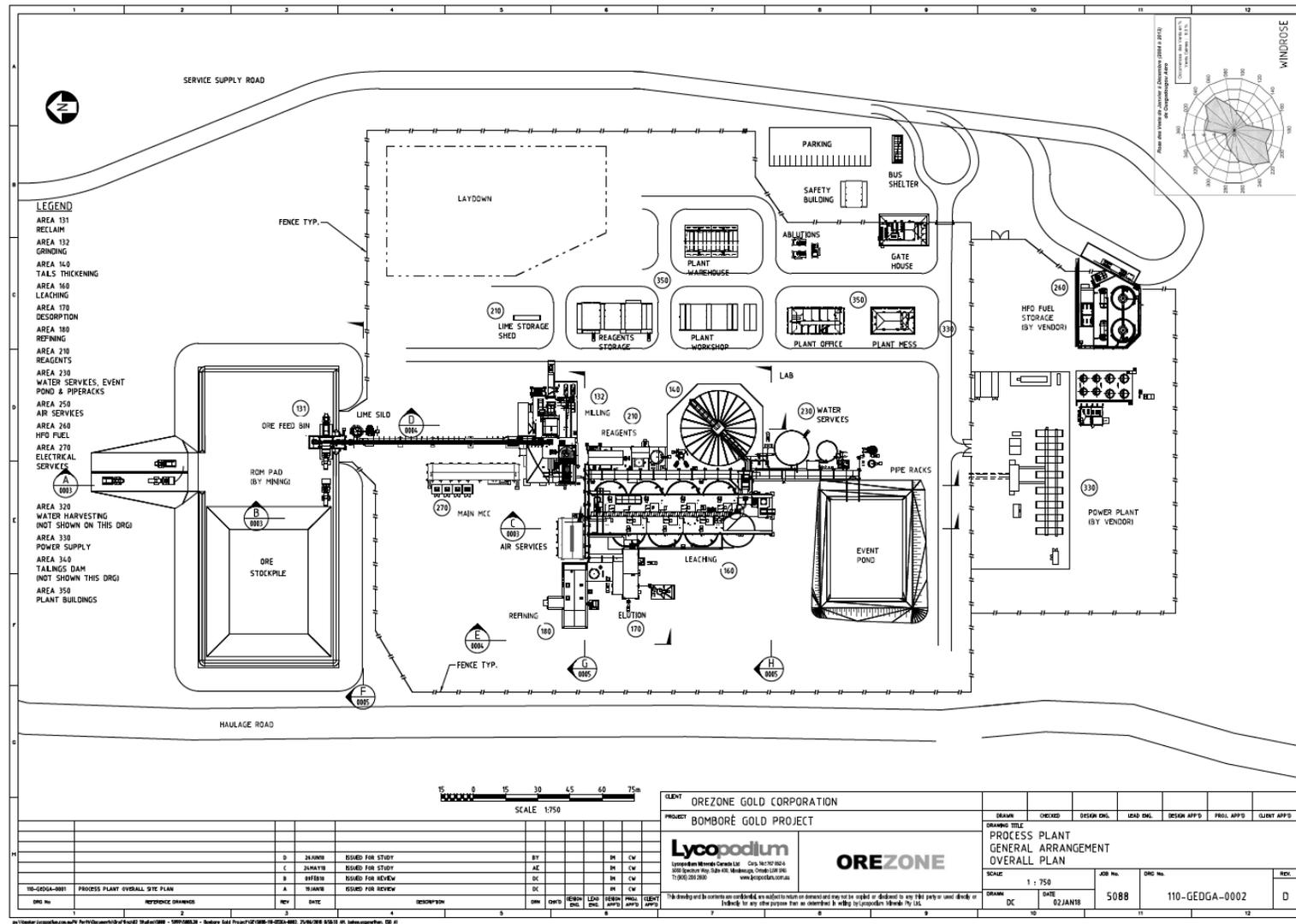
**Figure 17.3          Process Plant Isometric View**
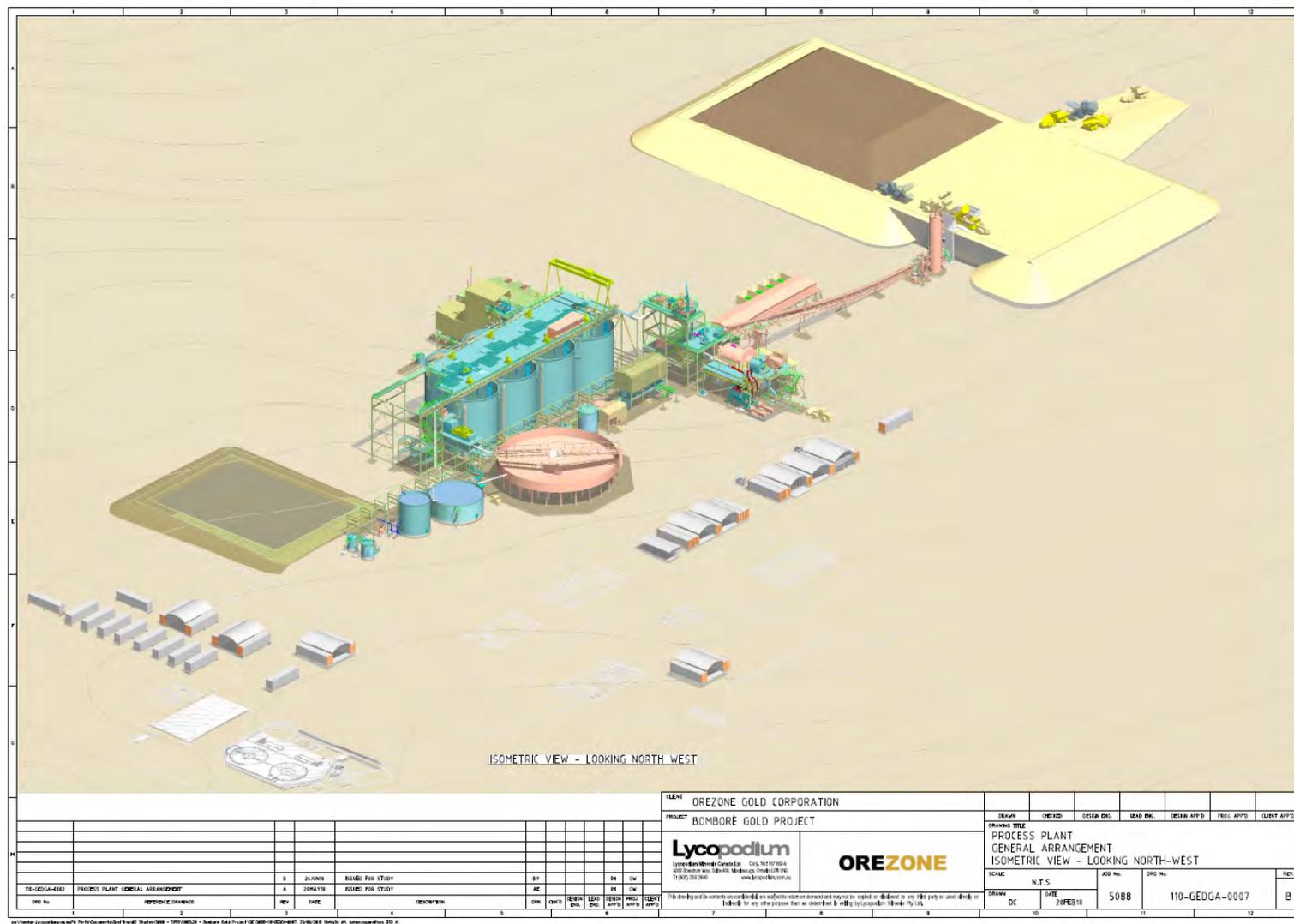
### 17.1.2 Key Process Design Criteria

Key process design criteria are summarized in Table 17.1 below.

**Table 17.1          Summary of Key Process Design Criteria**

|  | Units | Design | Source |
|---|---|---|---|
| Plant Throughput | tpa | 4,500,000 | Orezone |
| Ore Type |  |  |  |
| Avg. Blend per Mine Plan - Oxide | % | 85 | Mine plan |
| - Transition | % | 15 | Mine plan |
| Comminution Design - Oxide | % | 70 | Lycopodium |
| - Transition | % | 30 | Lycopodium |
| Head Grade - Design |  |  |  |
| Gold | g Au/t | 1.0 | Mine Schedule |
| Silver | g Ag/t | 0.97 | Mine Schedule |
| Overall Gold Recovery[1] | % | 92.2 | Lycopodium |
| Overall Silver Recovery[2] | % | 49.8 | Lycopodium |
| Ore Specific Density | t/m$^3$ | 2.8 | Testwork |
| Ore Bulk Density | t/m$^3$ | 1.65 | Testwork |
| Angle of Repose | degrees | 37.0 | Lycopodium |
| Plant Availability | % | 91.3 | Lycopodium |
| Crushing Work Index (CWi, average) | kWh/t | 7.8 | Testwork |
| Bond Ball Mill Work Index (BWi, average) | kWh/t | 5.4 | Testwork |
| Bond Abrasion Index (Ai, average) |  | 0.035 | Testwork |
| Grind Size (P$_{80}$) | μm | 125 | Lycopodium |
| CIL Circuit Residence Time | hrs | 24 | Testwork |
| CIL Slurry Density | % solids (w/w) | 40.8 | Lycopodium |
| Number of Leach Tanks |  | 1 | Lycopodium |
| Number of CIL Tanks (Stages) |  | 6 | Lycopodium |
| Air Addition Rate | Nm$^3$/h/m$^3$ | 0.15 | Lycopodium |
| Thickener Solids Loading | t/m$^2$.h | 0.60 | Testwork |
| Sodium Cyanide Consumption | kg/t ore | 0.28 | Lycopodium |
| Lime Consumption[3] | kg/t ore | 2.07 | Lycopodium |
| Sodium Hydroxide Addition | kg/t ore | 0.016 | Lycopodium |
| Elution Circuit Type |  | Pressure Zadra | Lycopodium |
| Elution Circuit Size | t | 5.0 | Lycopodium |
| Frequency of Elution | strips / week | 7.0 | Lycopodium |
| Kiln Capacity | kg/h | 250 | Lycopodium |

1. At design head grade of 1.00 g Au/t including 0.08 g Au/t (residue plus soluble) loss

2. At design head grade of 0.97 g Ag/t including 0.49 g Ag/t (residue plus soluble) loss

3. Lime addition based on 90% CaO content of supplied quicklime.

## 17.2       Process and Plant Description

### 17.2.1      Introduction

The oxide (saprolite) ore and a portion of the transition ore is free digging and drill and blast will not be required in the open pits. Once the in-situ ore at depth becomes so competent such that rip and excavate mining methods are no longer effective it will be considered that the economic base of the pit has been reached. This is estimated to occur at an average depth of 50% through the transition layer.  Extensive mapping and XRF analysis have been completed throughout the exploration program and all data are incorporated within the Bomboré database and this will also be used during mine planning purposes to help predict the free digging interface during operations.

The life of mine (LOM) mining plan averages 85% upper and lower oxide (saprolite), and 15% upper transition ore. The average LOM gold grade is 0.64 g Au/t. Initially low-grade ore will be stockpiled to provide above average grade to the mill in the early years of the operation.  A life of mine milling and gold production schedule is provided in Section 24.

### 17.2.2      Run of Mine Ore Receipt and Mill Feed

Run-of-mine (ROM) ore from the open pit will be transported to the plant by 30 t and 50 t capacity rear dump tip trucks. The trucks used are anticipated to be heavy-duty triple axle road 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 15 minutes plant feed to the ball mill).  A static grizzly (150 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.

Quick lime 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 80 t, equivalent to 2.5 days consumption. To mitigate the risk of lime shortages a small inventory of 70, 1 t capacity lime bags 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.

Any spillage generated, within the conveying area, will be manually recovered and transported to the ROM pad for feeding back into the plant.

### 17.2.3    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. It is anticipated that pebble rejection from the mill will be minimal and that the pebbles will have a low gold assay. Should it be found at a future date that the pebbles contain enough gold to make treatment a viable option then they can be batch processed through a contract crushing plant and fed back to the mill. If this is not the case the pebbles will be used for stabilizing slopes and landscaping. Pebbles will be removed from a sump at the mill discharge by FEL.

The mill has been designed to be capable of achieving the required grind size with a mill feed blend of 70% saprolite and 30% transition ore, which is harder than the life of mine average blend. The installed variable speed will ensure that the mill can be controlled to draw the required power to achieve the required leach product size when treating softer blends.  The mill will operate with a nominal 24% 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 control the mill pulp density.  The slurry will overflow 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 ROM bin (by FEL) and ore feed system.

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 one of two cyclone feed pumps (one standby). Process water will be added to the pumpbox to control the hydrocyclone performance and overflow density to the leach circuit. The hydrocyclone overflow will gravitate, via a trash screen and automatic sampler, to the leach feed distribution box.  The trash removed (wood chips etc) will be discharged to a trash bin for disposal.

Cyclone underflow will flow by gravity back to the ball mill feed box for further grinding.

Fine spillage within the grinding circuit sump area will be managed through area sump pumps. Coarse spillage will be cleaned using a Bobcat or similar equipment.

### 17.2.4    CIL Circuit

Trash screen underflow will gravitate directly to the leach feed distribution box and to the leach tank.  If the 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 leach and six CIL, mechanically agitated, tanks operating in series.  The tanks each have a live volume of approximately 3,500 m³ providing a 24-hour leach residence time at the design feed rate of 1016 m³/h.  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.

Cyanide as a 20% w/w solution will be added to the circuit from a ring main fed by the cyanide dosing pumps. The primary cyanide dosing point will be the first leach and first CIL tank, with further addition points located down the CIL train.  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 blowers and distributed down the agitator shafts.

Slurry will flow sequentially through the leach/CIL circuit tanks driven by the hydraulic gradient across the circuit provided by the inter-tank weirs. Activated carbon will be retained in each of the CIL tanks by a self-cleaning inter-tank 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 CIL tank #3.

To replace the recovered loaded carbon, regenerated carbon (or fresh carbon) will be pumped to CIL tank #6, 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 leach or a CIL tank be off-line for any reason, it will be possible to bypass the off-line tank using pneumatically actuated gate valves located within the CIL inter-stage launders, diverting slurry to the following CIL tank.

The leach/CIL area will be serviced by two sump pumps.  The sump pump closer to the leach tank will return spillage to the leach feed distribution launder. The sump pump closer to the other end of the sump will return spillage to the #4 CIL tank. The leach/CIL bund area will overflow in 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.

### 17.2.5    Desorption and Carbon Regeneration

A pressure Zadra elution circuit was selected for gold recovery from the activated carbon as it requires a lesser quantity and lower quality water than an AARL circuit. A more rapid strip is useful to provide flexibility and extra capacity when wide swings in feed gold grade can be expected but this is not the case at Bomboré.

The desorption circuit will consist of separate acid wash and elution columns.  The circuit will be designed to complete one strip per 22 hour period.  At the design carbon gold loading of 2,840 g/t and silver loading of 1,553 g/t the required daily carbon movement will be 5 tonnes.

*Acid Wash*

The cold acid wash sequence is required to remove any accumulated calcium scale (from the lime) from 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 CIL tank #1.  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. The stored (32% w/w) hydrochloric acid 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.  The first 2 BV will include a caustic injection, to neutralize the acid waste, while the last 2 BV are comprised of a fresh water rinse only.  Displaced solution from both the acid rinse and wash steps will pass through the Acid Wash Discharge Strainer before discharging to the tailings thickener standpipe.

The acid wash sequence will conclude with carbon being transferred by pump to the Elution Column.

*Elution*

A 1% w/w caustic soda (NaOH) and 0.2% w/w sodium cyanide (NaCN) solution (barren eluate) from the strip solution surge tank will be pumped and pre-heated to 95°C using a diesel fired solution heater.  Once pre-heated, the solution will be diverted through the heat recovery exchanger to be further heated, under pressure, to 125°C and then fed to the elution column to strip gold and silver from the loaded carbon.

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 30 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 carbon and transfer water will be pumped 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 250 kg/h. The carbon will be heated within the diesel-fired, horizontal rotary kiln to 700°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 #6 with a bypass option to CIL tank #5. Sizing screen undersize will be discarded to the tailings thickener underflow standpipe. Fresh carbon will be added to the CIL circuit from the carbon quench tank.

### 17.2.6    Electrowinning and Gold Room

Soluble gold and silver recovery, from the pregnant eluate, will be achieved by electrowinning onto stainless steel cathodes. The electrowinning circuit will consist of two 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 and borax), and then charged into the diesel fired gold furnace to produce slag and doré ingots, which 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 feedbox.

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.

### 17.2.7 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-tank screen.  The safety screen will also serve to prevent carbon loss to tailings in the event of a holed inter-tank 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 used for circuit monitoring and for metal 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.

Residual cyanide in the plant tailings will degrade naturally through hydrolysis and UV irradiation in the TSF.

### 17.2.8 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 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.

### 17.2.9 Reagents Mixing and Storage

The major reagents utilized within the process plant are:

- Quicklime (90% CaO content) for pH control.

- Sodium cyanide (NaCN) for gold dissolution and desorption.

- Caustic soda (NaOH) for neutralization and desorption.

- Hydrochloric acid (HCl) for carbon acid washing.

- Flocculant for thickening.

In addition, fluxes (silica, nitre and borax) are required for smelting charge preparation and antiscalant is used as required to reduce scaling in the process water distribution, carbon wash and stripping circuits. Sulphamic acid will be used to descale 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 80 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 70-1t bulk bags of lime will be retained on site for use in 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 tonnes) as briquettes in standard 1,000 kg double bagged containment in 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.

The cyanide bulk bag will be lifted, by the cyanide 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% cyanide concentration.

During operations cyanide will be drawn from the mixing tank and circulated around the plant cyanide ring main. The ring main returns to a small cyanide storage tank that overflows back into the mixing tank. When mixing a fresh batch of cyanide solution, the cyanide circulation 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.

### Caustic Soda (NaOH)

Sodium hydroxide (caustic soda or caustic) will be delivered to site as pearls in 1,000 kg bulk bags. The bulk bag will be lifted, by the caustic soda hoist, to the bulk bag breaker mounted above the caustic storage hopper.

Caustic will be fed slowly, from the storage hopper, by the caustic rotary feeder to the caustic mixing tank which will 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%). 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%) 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% 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 delivered to the tailings thickener discharge stand pipe by the 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%).

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% prior to dosing to the tailings thickener feed.

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 ($SiO_2$).

### 17.2.10    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 applications where quality water with low dissolved solids is required and as make-up the process water circuit.

*Raw Water*

Raw water will be harvested from the Nobsin River during the wet season and stored in the OCR until needed. Three (3) OCR pumps will be available to deliver water to the plant site process water tank (as make-up if required) and the raw water tank. Three 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 600 $m^3$ of which 288 $m^3$ is fire water reserve. This provides a 3 hour supply of raw water for all uses other than 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:

- Dust suppression.

- Eluate make-up.

- Carbon regeneration and quenching.

- Flocculant mixing and dilution.

- Reagent mixing.

- Potable water treatment plant.

- Grinding mill cooling water.

### 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).

### Cooling Water

Cooling water will be stored within a dedicated cooling water tank for pumping through the milling lubrication skid cooling circuit. The cooling water will be recirculated to the storage tank and the circuit fitted with a temperature sensor to dump water and replace with raw water if the temperature rises excessively.

### 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 ball mill. 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 1,000 m³ process water tank, which provides 1.3 hours of surge capacity.

From the process water tank, process water will be circulated around the plant by the process water pumps, with off-takes supplied for the main user points, namely:

- Milling area (mill feed chute, cyclone feed dilution and trommel spray water).

- CIL circuit.

- Cyanide mixing.

### 17.2.11    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 via a ring-main and injected into each tank down the agitator shafts.

## 17.3     Water, Power and Reagent Consumption

### 17.3.1    Water Consumption

A water balance for the process plant has been completed by Lycopodium and incorporated into the site wide water balance undertaken by KP.

Irrespective of season the process plant requires an average of 120 m³/h of raw water for applications for which process water is unsuitable.

When available, up to 460 m³/h of decant return water will be recycled from the TSF to the process plant. This quantity will likely only be available during the wet season when rainfall on the tailings storage facility increases the size of the decant pond. The shortfall will be made up by pumping additional 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.

### 17.3.2    Energy Consumption

Electrical power for the site and camp, will be provided by on-site power generation using HFO and diesel fuel. The average power demand is summarized in Table 17.2.

**Table 17.2          Average Power Demand Summary**

| Plant Areas | Installed Power (kW) | Average Continuous Draw (kW) | Total Annual Power Consumption (kWh / year) |
|---|---|---|---|
| Area 130 - Reclaim & Milling | 5,210 | 2,992 | 26,205,540 |
| Area 140 - Screening & Tailings | 1,300 | 340 | 2,949,755 |
| Areas 160 - Leaching & CIL | 1,004 | 637 | 5,580,295 |
| Area 170 - Elution | 62 | 9 | 75,949 |
| Area 180 - Gold Room | 132 | 43 | 374,578 |
| Area 210 - Reagents | 129 | 24 | 206,123 |
| Area 230 - Plant Water Services | 1,034 | 387 | 3,391,522 |
| Area 250 - Air Services | 595 | 289 | 2,529,012 |
| Area 260 - Fuel Dispensing | 3 | 1 | 10,162 |
| Area 270 - Electrical Services - Lighting | 508 | 134 | 1,169,723 |
| Area 340 – TSF | 209 | 42 | 371,512 |
| Area 350 – Plant Buildings | 270 | 81 | 709,560 |
| Area 310 – Environmental | 39 | 1 | 11,034 |
| Area 380 - Camp and Camp Services | 335 | 168 | 1,467,300 |
| Area 440 - Mine Facilities | 150 | 75 | 657,000 |
| **Total** | **10,980** | **5,218** | **45,709,063** |

### 17.3.3    Reagent and Consumable Consumption

Table 17.3 provides a summary of major reagent and consumable usage for the process plant at the design throughput of 4.5 Mtpa.

**Table 17.3          Annual Reagent and Major Consumable Consumption**

| Reagent / Consumable | Annual Consumption |
|---|---|
| Ball Mill Liners | 0.63 sets |
| Ball Mill Grinding Media | 520 t |
| Quicklime | 9,350 t |
| Sodium Cyanide | 1,250 t |
| Activated Carbon | 45 t |
| Sodium Hydroxide (Caustic) | 80 t |
| Hydrochloric Acid | 230 t |
| Flocculant | 45 t |
| Distillate Diesel Oil DDO (plant usage only) | 690 m$^3$ |

## 17.4 Plant Control System

### 17.4.1 General Overview

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.

## 17.5 Sampling and Assaying

Multiple commercial laboratories provide gold fire assay facilities in Ouagadougou so assay facilities on site will be limited to simple AAS assay of gold in solution. This can be supplemented by LeachWell type rapid cyanide soluble gold assays if required.

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 the process plant. 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.

## 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 includes major facilities including mine open pit, process plant, TSF, mine services, 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 defer access by unauthorized people and prevent grazing animal access.

**Figure 18.1          Project Site – Plant Commissioning**
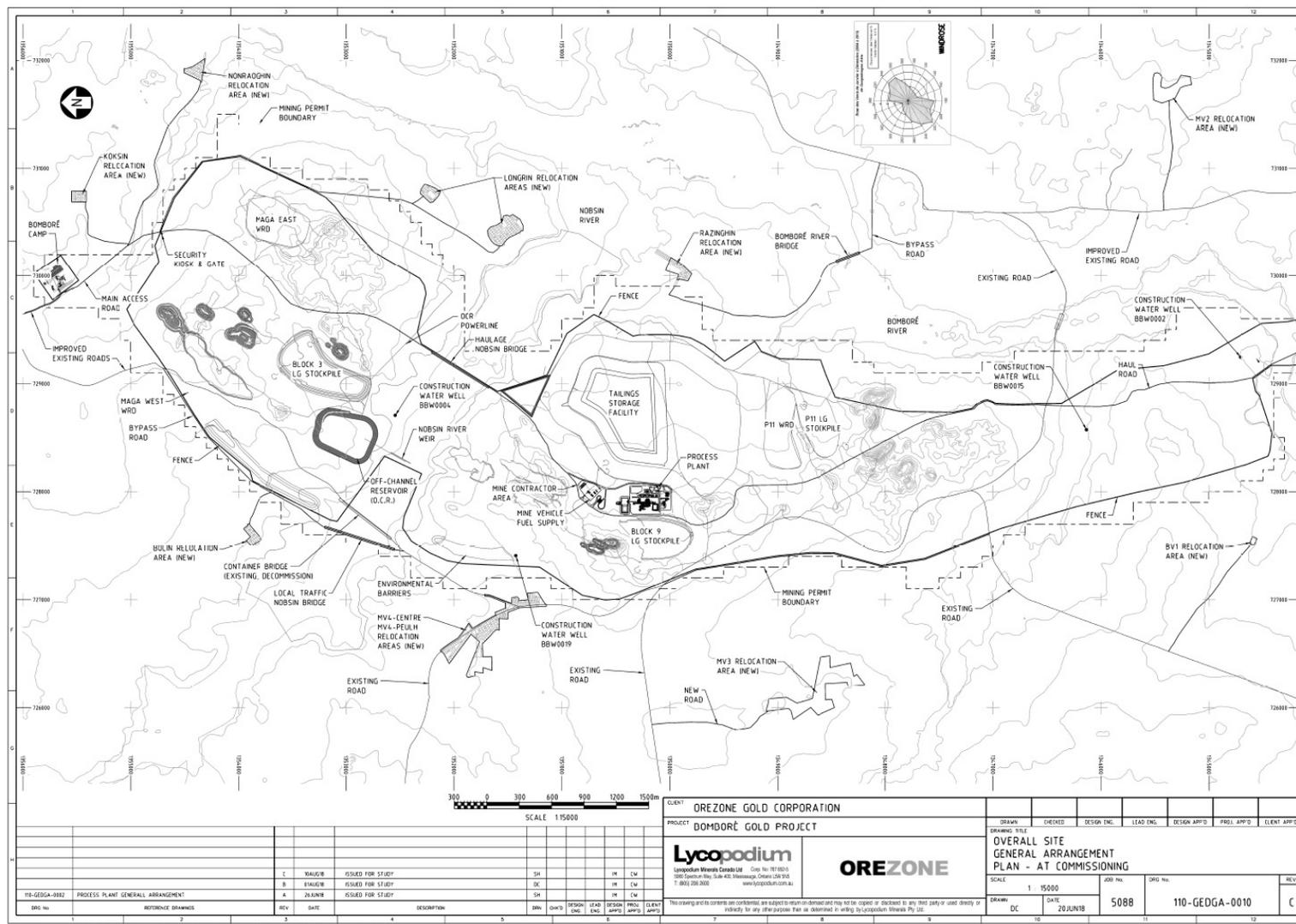
**Figure 18.2** **Project Site – Prior to Decommissioning**
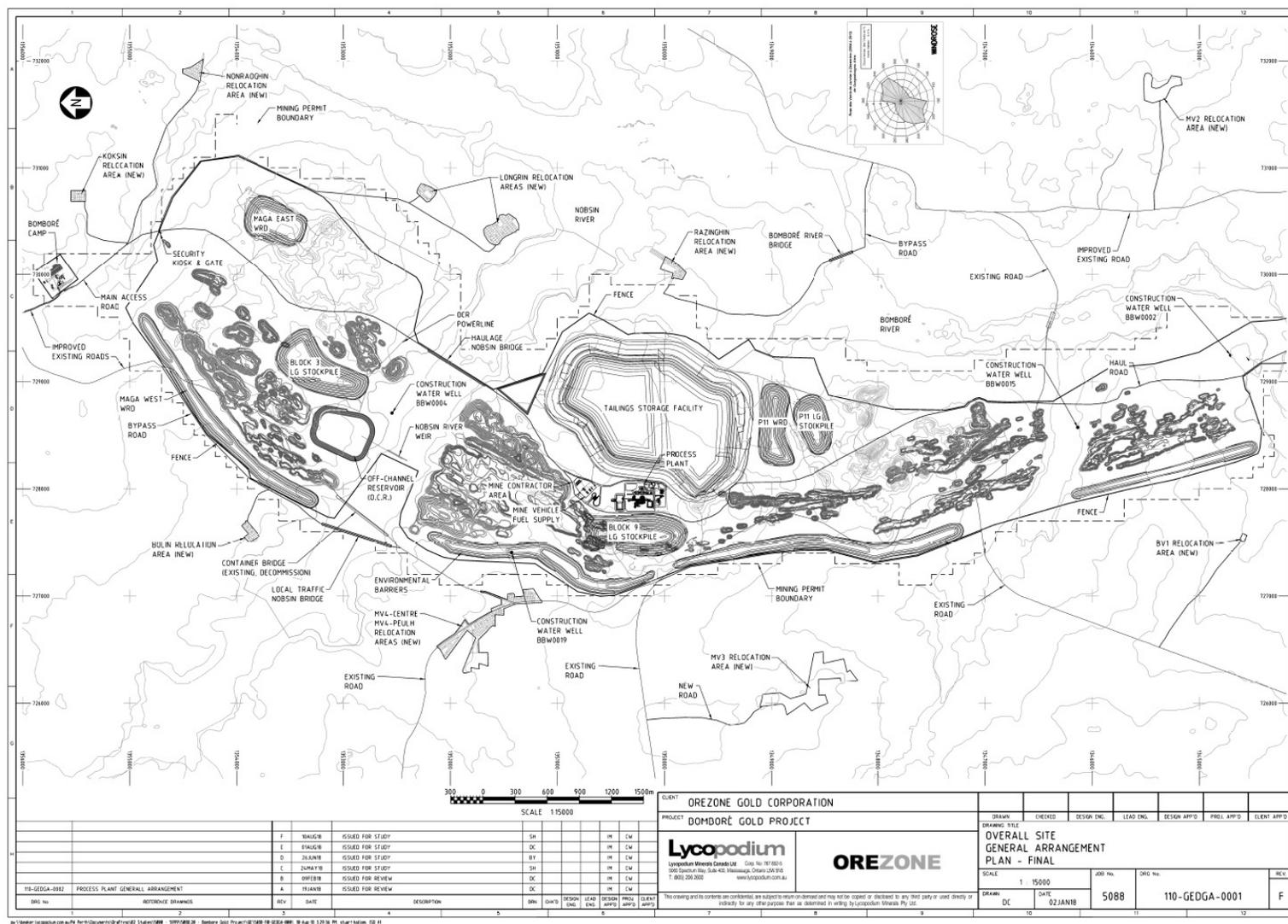
## 18.1    Site Access

Site access will be from the north using the upgraded 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 main camp buildings are constructed from locally produced red bricks with terrazzo floors.  The office is a 480 $m^2$ building with reception, meeting rooms and open plan facilities. The camp can currently accommodate 76 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. Bottled water is provided for drinking. A kitchen/dining room operated by a contract service provider serves meals for camp residents and guests.

Planned additions to the camp, included under Owner's costs in the project capital estimate, consists of an additional block of 18 bedrooms with private bathrooms. Rooms will also be freed up for project personnel when the local police detachment moves into a new facility across the road from the camp.  There is room within the existing camp boundary fence to install several additional accommodation blocks or other facilities should they be required in the future.

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.

- Kitchen/dining room upgrade.

- A recreation room.

- A clinic.

Site accommodation, at the expanded Bomboré Camp, will be provided for the owner's team, EPCM contractor's team, 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 be required to make their own accommodation arrangements with local businesses. Contractors will also be required to make arrangements for bussing their employees to and from site and for providing a midday meal.

Contractor quotations used for capital cost estimation 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 ore and waste are free digging and the pits small and shallow, it is planned that mine operations will be undertaken with a fleet of small excavators, 30 to 50 t capacity (probably Chinese built or equivalent) road-legal dump trucks and a modest support fleet.

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:

- 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.

## 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' and provided adequate operating flexibility with 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 average electrical load on site is estimated to be 5.45 MW with a peak demand of 7.45 MW.  Annual energy consumption is estimated to be 46.0 GW h/yr.

The electrical system is based on 11 kV distribution and 415 V working voltage.

A single switchroom in the process plant area will house the 415 V motor control centres (MCCs).  Outdoor control panels and distribution boards have been allowed for plant lighting and small power distribution and un-interrupted power supply (UPS) power distribution.

The 3.5 MW mill motor and the 600 kW cyclone feed pump motors will operate at the 11 kV supply 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.

## 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).

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.

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.

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 and will be provided with a manned entry gate to prevent unauthorized access. Security personnel contracted to Orezone will be 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 and water.

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 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 of the TSF to build a consolidated and stable deposit. Bleed water will be recovered from a small supernatant pond on the surface of the tailings and continuously recycled to the process plant for re-use as process water.

The site is relatively flat, and thus the facility will be developed within embankments wrapping entirely around an internal basin. The slopes range from less than 1% at the north-eastern and south-eastern limits of the tailings facility footprint up to approximately 3% along its western side. In the southwest region, the slopes become locally steeper in an area where a rock outcrop, that will contain the closure spillway, forms a local high point.

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 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 about 60 Mt of tailings to be produced at a rate of 4.5 Mt/year over a period of approximately 13.3 years.

Most of the construction materials will be obtained from the mining operations as mine waste. It will be placed and compacted in controlled, thin, horizontal lifts to form dense and stable wrap around the dams. Since mining will be conducted only in oxide and upper transition zones of saprolite and saprock, 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 with approximately 80% of the waste being taken into the dams on a year to year basis. An 80/20 ratio has been estimated as an appropriate split of suitable versus unsuitable material for the dams for this design owing to the relative consistency and make-up of the geologic deposit, although a greater percentage of mine waste may prove to be suitable. The remaining 20% of the mine waste will be used to construct perimeter berms along the western side of the project site and placed in waste dumps. In certain years the 80% material allowance will allow for a pre-build of a portion of the next stage of the tailings facility dams, which has been accounted for in the TSF design. The overall objective is to construct the dams with as much mine waste as possible, and to schedule this waste into the dam construction sequence directly from the mine as and when it is available, to improve efficiency and reduce cost, and to reduce the environmental footprint of the project.

The TSF has been designed to the highest industry standards 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.

Containment of tailings and water in the tailings storage facility will be by embankments wrapping entirely around an internal basin in a paddock type arrangement. The embankments will be stage raised using the downstream method of construction and thus will not rely on the tailings for any support. The tailings storage basin will be developed in two adjacent halves as shown in Figure 18.3. A northern half will be developed first and will receive tailings for the first 3 years. This will be followed by a southern half that will receive tailings for the following 3 years. After this, the dam separating the two halves will be overtopped, and the tailings will be combined such that the basin will be operated as one facility for the remaining 7+ yrs.

**Figure 18.3**        **Tailings Storage Facility Layout**
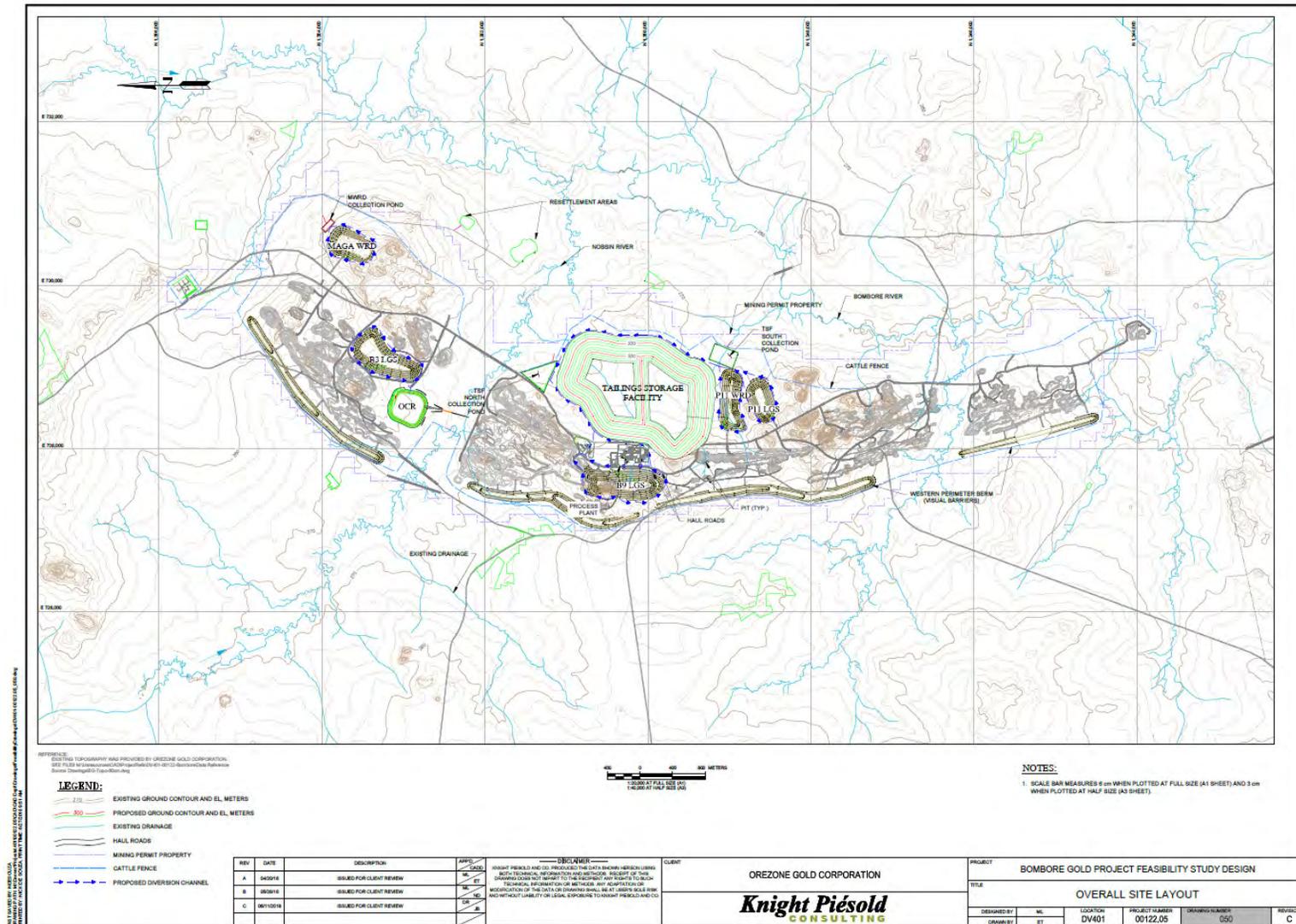
The basin areas will be fully lined with a 300 mm thick low permeability soil bedding layer and a 1.5 mm (60 mil) HDPE geomembrane placed immediately over top, similar to the inside facing on the dams described above. The low permeability soil layer and geomembrane will be continuous with the soil layer 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) [metres] | | | Maximum Tailings Elevation (September) [metres] | | | Tailings Capacity (Million Tonnes)* | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | North Area | South Area | Combined Area | North Area | South Area | Combined Area | North Areas | South Area | Combined Area* |
| 1 | 2020 | 288 | - | - | - | - | - | - | - | - |
| 2 | 2021 | 296 | - | - | 287 | - | - | 4.5 | - | - |
| 3 | 2022 | 304 | - | - | 295 | - | - | 9.0 | - | - |
| 4 | 2023 | - | 288 | - | 302.5 | - | - | 13.5 | - | - |
| 5 | 2024 | - | 304 | - | - | 287 | - | - | 4.5 | - |
| 6 | 2025 | - | 304 | - | - | 295 | - | - | 9.0 | - |
| 7 | 2026 | - | - | 313 | - | 302.2 | - | - | 13.5 | - |
| | 2027 | - | - | | - | - | 305.3 | - | - | 31.5 |
| | 2028 | - | - | | - | - | 308.5 | - | - | 36.0 |
| 8 | 2029 | - | - | 321 | - | - | 311.7 | - | - | 40.5 |
| | 2030 | - | - | | - | - | 315.7 | - | - | 45.0 |
| 9 | 2031 | - | - | 325 | - | - | 319.3 | - | - | 49.5 |
| | 2032 | - | - | | | - | 321.6 | - | - | 54.0 |
| | 2033 | - | - | | | - | 324.0 | - | - | 60.0 |

*\* The storage capacity may exceed 60 Mt if the actual in situ tailings density is greater than that assumed for this study*

A summary of the years of tailings filling of 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 and 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 and Dams**

| Stage # | Years of Construction | Year of Tailings Filling | Planned Tailings Dry Density (t/m³) | Cumulative Tailings Production at End of Stage (M t) | Min. Tailings Elev. (m) | Max. Tailings Elev. (m) | Max. Water Level Elev. of 72-Hour PMP (m) | Embankment Crest Elev. (m) | Comments |
|---|---|---|---|---|---|---|---|---|---|
| 1 | 1 | 1 | 1.1 | 4.5 | 282 | 287 | 285.7 | 288 | North half of TSF |
| 2 | 1 | 2 | 1.2 | 9 | 291 | 295 | 294.0 | 296 | North half of TSF |
| 3 | 1 | 3 | 1.25 | 13.5 | 299 | 303 | 301.1 | 304 | North half of TSF |
| 4 | 1 | 4 | 1.1 | 18 | 284 | 287 | 287.1 | 288 | South half of TSF |
| 5 | 1 | 5 | 1.2 | 22.5 | 292 | 295 | 294.3 | 304 | South half of TSF |
| 6 | 1 | 6 | 1.25 | 27 | 300 | 303 | 301.4 | 304 | South half of TSF |
| 7 | 1 | 7, 8, 9 | 1.3 | 40.5 | 307 | 312 | 310.4 | 313 | Both halves of TSF |
| 8 | 3 | 10, 11 | 1.3 | 49.5 | 313 | 320 | 316.2 | 321 | Both halves of TSF |
| 9 | 2.3 | 12, 13, 14 | 1.3 | 60 | 318 | 324 | 319.6 | 325 | Both halves of TSF |

The supernatant pond will be located at the low points on the beach areas. 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 probable maximum flood (PMF) volume (stage to stage volumes described above) plus freeboard. Tailings deposit and pond modelling show that the pond limits for the PMF 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 tailings storage facility. 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 probable maximum precipitation (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**
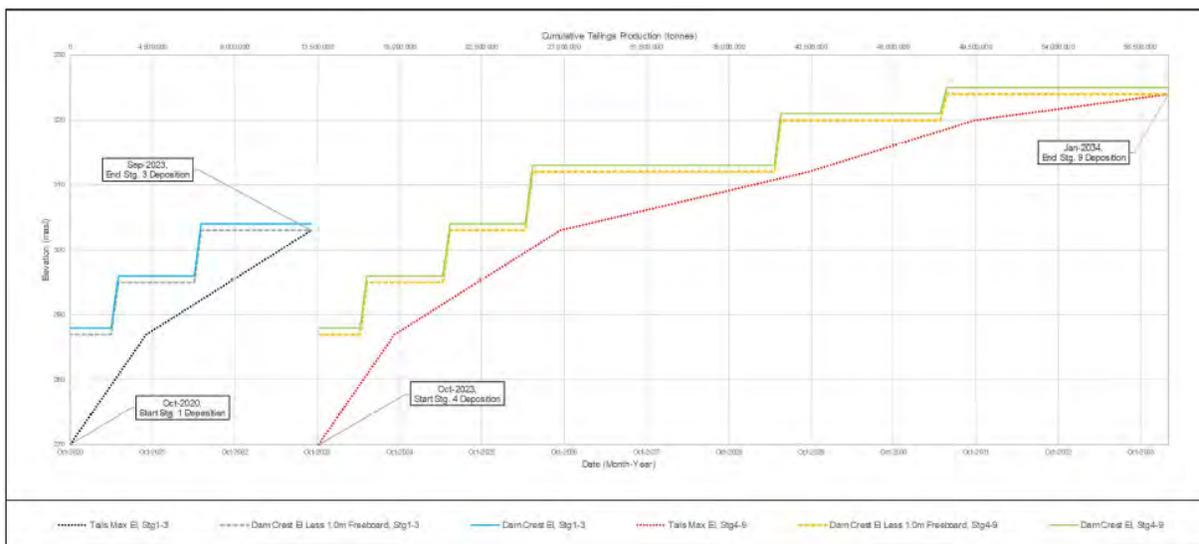


Supernatant water will be removed from each pond via a high inflow capacity, slotted, vertical decant tower pipe that will be surrounded by clean rockfill. This arrangement is designed to enable the pond to be well drawn down on the beach at all times with only a small head pool present for much of the time. This will maximize air drying and densification of the tailings and maximize water recovery by minimizing evaporation losses.

The tailings slurry will be pumped from the tailings pump station, located at the plant, to the tailings storage facility 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. One pipe will be in operation while the other is being lifted and raised.

The deposition pipeline will deliver the tailings to multiple thin wall HDPE drop-bars around the facility. Two drop-bars should always operate at a time to avoid solids settling in the distribution pipeline between the active deposition points. The drop bars 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 tailings storage facility.

## 18.13    Site Water Management

The overall operational water management strategy is to utilize all water captured within the mine limits to the maximum practical extent in an efficient manner. This 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 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 seasonal 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 for the process plant, dust control, and TSF construction water (i.e., moisture conditioning of the embankment fill) demands.

- The water volumes that will accumulate in the TSF supernatant pond will be reclaimed via a 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 from the OCR.

- Water will be pumped from the OCR to the TSF for temporary storage and subsequent use in the process plant on an as needed and seasonal basis.  However, consideration will be made in the future to replacing this with a direct connection from the OCR to the process plant raw water tank to reduce evaporative losses.  Transferring water to the TSF is advantageous at times because the TSF has a large storage capacity that can be utilized in extreme dry conditions and it would allow greater volumes to be captured in the OCR if needed.

- Water yield from groundwater wells will also be utilized as necessary for process plant, dust control, and TSF dam construction water make-up.

- Rainfall-run-off that comes into contact with the stockpiles, waste dumps, and general surface run-off will be collected and retained for use via diversion channels, collection ponds as well as several mine pits.

The climate in the region of the Project is classified as semi-arid. The average annual rainfall depth is approximately 780 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 50-year Mogtédo Station record are approximately 560 mm and 1,240 mm, respectively.

On average, rainfall occurs 33% of the time in the wet season and 4% of the time 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 probabilistic water balance model to predict water quantities in the mine circuit from May 2020 through early 2033, which includes the production timeframe starting in October 2020 was developed. As presented in the operational water balance schematic on Figure 18.5, the mine water circuit considered in the analyses consisted of the OCR, process plant, TSF, and groundwater wells.

**Figure 18.5          Operations Water Balance Schematic**
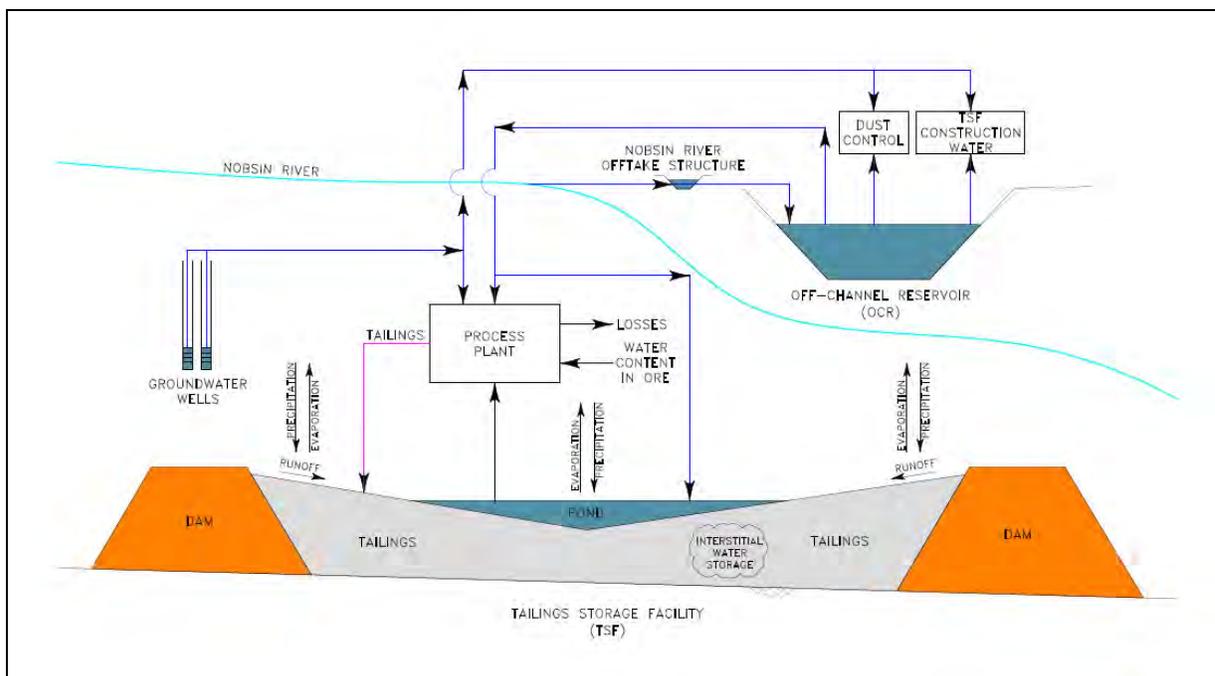


The results of the water balance are sensitive to OCR seepage losses. Variations or modifications to the losses estimated in the analyses could affect the required size of the OCR and the water transfer philosophy and amounts to the TSF and process plant.  Further testing and hydrogeologic characterization of the subsurface materials are recommended to more accurately quantify the potential seepage losses from the OCR.

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 1.0 m minimum freeboard) from May 2020 through February 2033. This is illustrated on Figures 18.6 and 18.7 for TSF Stages 1 through 3 and Stages 4 through 9, respectively, which show the simulated pond elevations from the water balance.

- The TSF pond volume will be most affected by water transfers from the OCR to the TSF pond. These transfers were performed in years 2020 through 2025 when the evaporation rates from the TSF pond would not negate the volume of water that was transferred. Other, less significant, 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) and tailings unit weights, which vary over time.

- 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 water storage volume was selected by Orezone to reduce the risk of water shortfalls based on Knight Piésold's water balance analyses.

- The OCR water storage volume will be full from September through December of each year for average and drier conditions based on an assumed river flow capture efficiency to the OCR of 50%. This could extend from June through February in wet conditions. There is a chance that the total impounded water volume in the OCR could be exhausted by the end of the dry season in extreme dry conditions. 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 water volume will be most affected by the losses to seepage, which were estimated based on information from previous Golder (2015a, 2015b, 2014) documentation. The results of the water balance are significantly sensitive to the OCR seepage losses. Variations or modifications to the flows estimated in the analyses 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.

- On average, the process plant water demand will generally be met via water from the OCR and TSF reclaim, which will each provide near equal proportions of water. Relatively small deficits may occur at the end of each dry season for average conditions. Figure 18.5 presents the simulated inflows to the process plant from the OCR, TSF, and groundwater wells combined.

- On average, a total volume of approximately 160,000 m$^3$ may be required from outside water sources from May 2020 through February 2023. This estimate increases to approximately 570,000 m$^3$ for a 1.0% chance of exceedance. Water surpluses were not predicted.

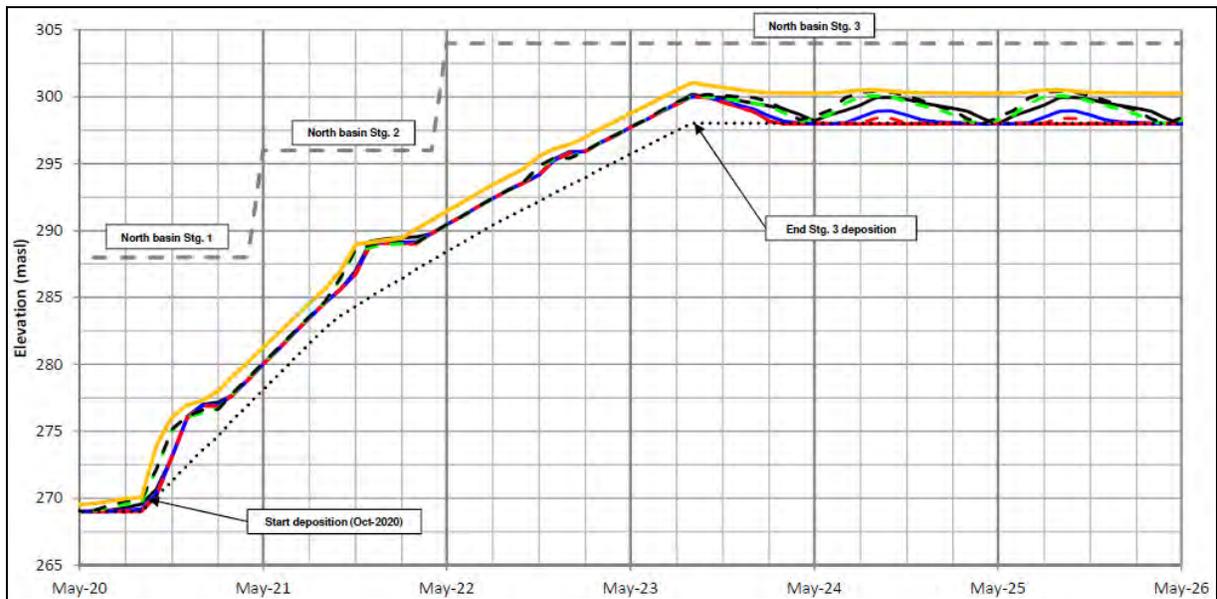**Figure 18.6         TSF Pond Elevations, Stages 1 through 3**



**Figure 18.7         TSF Pond Elevations, Stages 4 through 9**
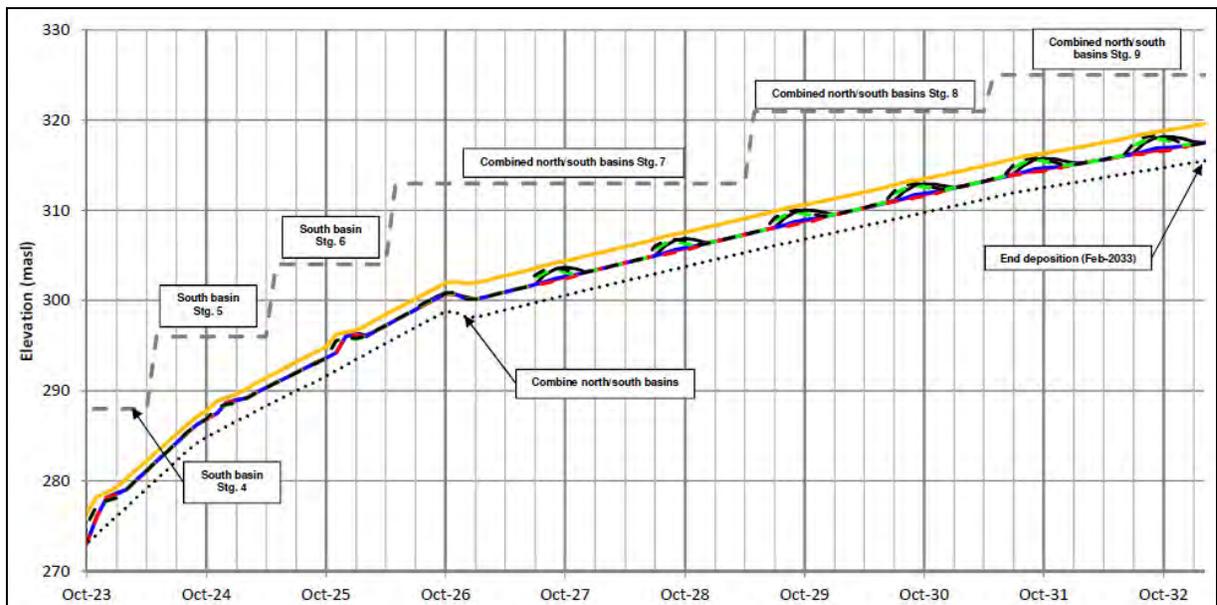
The following summarizes the site water management plan structures that were designed and the associated water routing, per facility/area:

- Maga waste rock dump (MWRD)

    – Two perimeter diversion channels and one culvert crossing will convey stormwater run-off to a single collection pond located north of the MWRD (denoted as the MWRD Collection Pond). The pond will be equipped with an emergency spillway that will discharge flows to the downstream environment in extreme wet conditions only.

- Block 3 low-grade stockpile (B3LGS)

    – Two perimeter diversion channels and three culvert crossings will convey stormwater run-off to the OCR, which is located to the southwest of the B3LGS.

- Block 9 low-grade stockpile (B9LGS), process plant perimeter, and North TSF dam perimeter

    – Two perimeter diversion channels around the B9LGS and two around the process plant perimeter will convey stormwater run-off to a single collector channel located along the northwest perimeter of the ultimate TSF dam. This collector channel and a channel located along the northeast perimeter of the ultimate TSF dam will convey stormwater to a single collection pond located north of the TSF (denoted as the TSF North Collection Pond). A total of eight culvert crossings will be required to convey flows beneath various roads for this system. The pond will be equipped with an emergency spillway that will discharge flows to the downstream environment in extreme wet conditions only.

- P11 waste rock dump (P11WRD), P11 low-grade stockpile (P11LGS), and South TSF perimeter

    – Two perimeter diversion channels around the P11LGS, three around the P11WRD, and one along the southeast perimeter of the TSF will convey stormwater to a single collection pond located east of the P11WRD (denoted as the TSF South Collection Pond). Two culvert crossings will be required for this system. The pond will be equipped with an emergency spillway that will discharge flows to the downstream environment in extreme wet conditions only.

The OCR will be located north of, and adjacent to, the Nobsin River. The OCR excavation was designed within the Saprolite layer, which is approximately one order-of-magnitude less permeable than the underlying Saprock layer, to limit the seepage to the groundwater system. Knight Piésold reviewed the Golder (2018) pit slope recommendations to design the OCR basin excavation. The inter-bench (i.e., bench face) slopes in this study were designed to 39 degrees, with 4 m wide benches every 10 m of height, which corresponds to an overall slope angle of 31 degrees. An access ramp was designed on the east side of the OCR with a total width of 16 m (1.5 m safety berm width, 14.5 m running width) and a longitudinal slope of 10%.

The Nobsin River offtake system to the OCR was assumed to have a flow-capture efficiency of 50% (i.e., 50% of the total simulated river flow was transferred to the OCR when needed). If this efficiency is significantly overestimated, it will take longer than expected to fill the OCR and there is a chance that the full water storage capacity will not be achieved. Further hydrologic and hydraulic analyses should be performed in detailed design to more accurately quantify the flow-capture efficiency and depending on these results modifications to the offtake infrastructure may be required. A backup pumping system and possibly a secondary offtake should be provided in the event that additional water transfer capacity is needed. Redundancy is recommended here to reduce risk because the Project will not be able to operate at the proposed production levels without the needed seasonal water harvesting from the Nobsin River.

The most likely reason for a reduced flow-capture efficiency is sediment accumulation in the offtake infrastructure. Further detailed sediment accumulation analyses are warranted for detailed design to confirm or modify the assumptions made in this study. River-flow scour analyses should also be conducted to verify that the structures will not be adversely affected by erosion.

Levees will be required to prevent the ingress/encroachment of river flows to the OCR and surrounding pits and roads in extreme wet conditions, per the design criteria. Knight Piésold designed levees around the OCR. The levees were designed with sufficient heights, extents, and erosion protection measures to prevent OCR flooding in the 1,000-year flood. The levees will be constructed of compacted fill from the OCR excavation and will be protected with riprap.

## 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 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 price 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.

- 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 Project is administratively part of the Mogtédo Department in the Province of Ganzourgou which itself is part of the Central Plateau Region of Burkina Faso.  The project site is located about 85 km east of Ouagadougou and is accessible by National Highway No. 4 from Ouagadougou.

The approach developed by Orezone throughout the various environmental and social studies that have been conducted since 2009, especially in the context of the ESIA conducted from 2014-2015, 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 would be required due to an eventual mining project.

### 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).

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[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

- Politique Nationale de Développement Durable[6] (PNDD).

- 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.

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[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 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.

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[11] Ministry of Mines and Quarries
[12] National Transitional Commission

*Authorizations*

- Prospecting Authorization.

- Traditional Artisanal Mining Authorization.

- Quarrying Exploration and Operation Authorization.

The Mining Code guarantees a stable tax and custom regime for the life of any mine developed. 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 in the amount, the base, the rate and the methods of recovery are 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 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).

- Direction Générale des Mines et de la Géologie[16] (DGMG).

- Bureau des Mines et de la Géologie du Burkina[17] (BUMIGEB).

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[13] Ministry of Environment, Green Economy and Climate Change
[14] National Office of Environmental Assessments
[15] Technical committee for environmental assessments
[16]General Management of Mines and Geology

- 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 MEDD.  The ESIA must be supported by a FS and include a RAP that has been accepted by all stakeholders.  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 1$^{st}$ 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.

In 2016, Orezone received the Industrial Operating Permit following the delivery and acceptance by the authorities of the ESIA and RAP, both supported by the 2015 FS.

The required permits and administrative procedures are presented in Table 20.1.

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[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.

**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":<br>• Submission of the ToR of the ESIA to the BUNEE<br>• Validation of the ToR<br>• Submission of the draft version of the ESIA to the BUNEE<br>• Public Inquiry<br>• COTEVE Session<br>• Final ESIA Report approved by the BUNEE | Completed | • Cost for review and validation of the ToR: 500,000 FCFA.<br>• Fees for the public inquiry and COTEVE session:<br>   • Public inquiry: 6,666,000 FCFA<br>   • Session: 2,870,500 FCFA.<br>• Cost for file processing-Project value of 50 billion FCFA and more:<br>   • Flat Fee: 25 million FCFA.<br>   • Proportional rights: 0.02% of the total investment cost.<br>   • Total: 56,344,805 FCFA |
| Industrial Operating Permit | • Delivered to Société Orezone Bomboré SA. on December 30, 2016 by "Décret n°2016-1266/ PRES/PM/MEMC/MINEFID/MEEVCC" | Completed | • Costs for the National Commission of Mines session: 2,500,000 FCFA.<br>• Fixed duties on mineral titles: 5 million FCFA.<br>• Proportional rights (area taxes):<br>   • First five years: 7.5 million FCFA/ km²/y.<br>   • Year 6 to10: 10 million FCFA/ km²/y.<br>   • From the year 11 onward: 15 million FCFA/ km²/y.<br>• Cost of proportional royalty on gold production: 5% of turnover if gold is USD 1,300 and higher, 4% if USD 1,000-1,300 and 3% if less than USD 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 and financed through World's Bank program (PADSEM[24])<br>• Environmental audit visits: every 3 years, by independent consultant and submitted to BUNEE, about 1 million FCFA per visit.<br>• ESMP monitoring program: protocol to be agreed between Orezone and BUNEE. |

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[24] PDASEM: Projet d'appui au développement du secteur des minéraux (Project to support the development of the minerals sector)

| Permit / Authorization | Main Requirements | Timeframe | Costs |
|---|---|---|---|
| Authorization for the Management of raw water | • Application to the Directorate of Legislation and regulations of the Ministry in charge of water.<br>• 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$^3$ [a] |
| Authorization for the Collection of raw water for civil work | • Application to the Nakanbé Water Agency<br>• 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.<br>• 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. |

*Note:*

*a) The Water Tax rate applicable to the mining sector in Burkina Faso is currently under review. The Minister of Mines and Quarries has declared recently that it should be reduced to 85 FCFA /m$^3$*

## 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 field studies to establish an appropriate baseline for measuring the overall environmental and social impacts of the Project.  In September 2009, an EIA 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 GMSI to prepare a NI 43-101 compliant 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 requires 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 FS 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 were in progress in adjacent areas favourable 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 have 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. The revised and final ESIA and RAP were reviewed by Burkina Faso authorities and Orezone addressed all the comments with revised versions of both documents submitted to the Ministry of Environment in April 2, 2016 and 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.

## 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 to 900 mm.  Monthly minimum and maximum temperatures recorded by Orezone on the property between October 2011 and April 2015 ranged from 18ºC in December-January for the average monthly minimum and 40ºC 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 344 m which corresponds to the Birimian rock and 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 hydrological 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 BF 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**
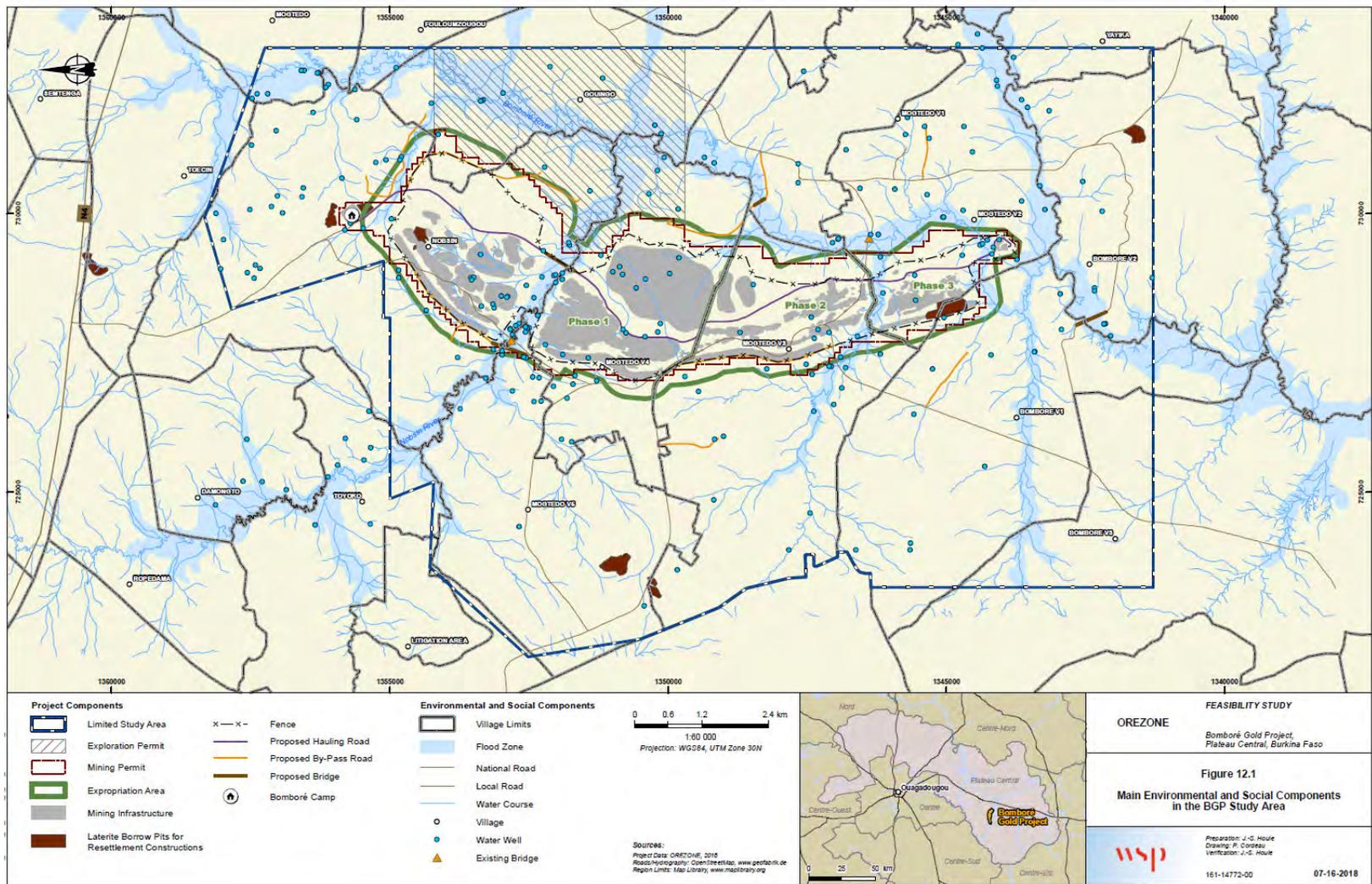
Deforestation is widespread over the permit area. Vegetation in uncultivated areas is mainly composed of savannah with denser vegetation, shrubby vegetation and riverine forests, growing only along the watercourses and the draining system. The main vegetation types comprised inside the study area can 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 AV Varea, 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.

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[1]Authority for the Development of the Volta Valleys.
[2] 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 (Kagtenga 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 but 63% also buy some chemical fertilizers.  Animal husbandry is practiced by most households; each owns 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 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 and to recharge mobile phones.

Inventories carried out in 2012 and 2014 identified 152 archaeological sites and 192 ethnographic sites (grave 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 many of these sites that are sacred places.

## 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 and meetings with administrative authorities, technical services, as well as representatives of the surrounding villages.

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. Three levels of the importance of the impact are determined: minor, medium, and 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 ground water during the construction and operational phases with 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 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, 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% as a result of the development of the project.

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.

The revenues generated by the mine operation will increase Burkina Faso's internal revenue through taxes and royalties charged by the local authorities, revenues which should have a beneficial impact at the local and regional levels through increased investments in social and health services, and educational facilities.

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 explorations licenses two economically viable deposits that miners from the two existing 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, 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 when any such incident or accident occurs.  The intent of the ERP is to define emergency situations 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 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 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 and local suppliers of goods and services as well as for 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, ground water, ambient noise, population and social cohesion, economy, and infrastructure, etc. will be maintained during the operational phase.  Several additional measures will include the following:

- Control 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 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, 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.

In regard to water quality, the monitoring will determine if arsenic is leaching from the weathered wastes and if it is present in the process water. The geochemical studies conducted to date suggest that arsenic leaching will be minimal.

## 20.6    Resettlement

### 20.6.1    People and Activities Affected by the Project

The resettlement of many people (about 655 households or about 3,915 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 in conjunction with the Feasibility Study.  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 about 410 households from traditional villages and about 915 households from the Sanam Yaar artisanal gold processing site.  The subsequent resettlement (Stage 2) of about 250 farming households and 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.

The reports issued by SOCREGE, BEGE and Cabinet Archi Consult, complemented by Orezone field validation and updated inventories as of February 2015 were used as a baseline for the RAP that was included in the final 2016 ESIA.  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    2015 Scope of Resettlement

The 2015 RAP requirements were 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 February 2015.

- Staged development of the project, i.e. Stage 1 in the North during the construction period (Year-1), to be followed by Stage 2 in the South during Year 1 and Stage 3 in the South and Southeast during Year 3.

- 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, plus financial compensation for granaries, sheds, ovens, parks, shops, roosts and similar small infrastructures.

- Cash compensation for farmlands located within the first 250 m of the buffer zone over five years was considered in the estimate and based on 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[1] 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 2018 Scope of Resettlement

The 2018 RAP requirements are based on the following assumptions:

- The area impacted is based on the site layout with residents within a 500 m buffer zone to be expropriated around the 2015 mining infrastructure (pits, pads, ponds, haul roads, etc.).

- 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 and Southeast during Year -1.

- 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, plus financial compensation for granaries, sheds, ovens, parks, shops, roosts and similar small infrastructures.

- Cash compensation for farmlands located within the first 250 m of the buffer zone over five years was considered in the estimate and based on 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 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. The revised 2018 mining infrastructure has a slightly reduced footprint resulting in a reduction of the expropriated farmland from 700 ha in the 2015 study to 656 ha in the revised 2018 study.

- Official data and property inventories as of April 2015 for the 2018 Stage 1 area, and June 2015 for the 2018 Stage 2 area. New official inventories were completed in June for the 2018 Stage 1 area and will be completed during the third quarter of 2018 for the 2018 Stage 2 area.

### 20.6.4 Staged Resettlement

The resettlement of villages is progressive and divided into three stages as a function of the mining plan (Cf. Figure 20.2).

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[1] Ministry of Agriculture, Water Resources, Sanitation and Food Security

**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 must start early in the development phase of the Project since this area is required for project construction; Orezone is planning to commence this program during the third quarter of 2018.

**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 2019.

The RAP budget is presented in Table 20.2.

**Table 20.2          RAP Budget ($)**

| Area | CAPEX | Sustaining Capital | Total |
|------|-------|--------------------|-------|
| Private Buildings | $ 15,556,787 | $ - | $ 15,556,787 |
| Public Buildings | $ 1,207,892 | $ 121,735 | $ 1,329,627 |
| Land Expropriation | $ 938,909 | $ - | $ 938,909 |
| Crop Compensation | $ 1,373,246 | $ 1,416,955 | $ 2,790,201 |
| Bridge and Bypass Roads | $ 2,141,841 | $ - | $ 2,141,841 |
| Misc. Allowances | $ 1,703,027 | $ - | $ 1,703,027 |
| RAP implementation | $ 1,424,955 | $ 1,594,684 | $ 3,019,639 |
| **Total** | **$ 24,346,656** | **$ 3,133,374** | **$ 27,480,030** |

**Figure 20.2          Existing Communities and Proposed Resettlement Sites for the Project**

## 20.7    Acid Rock Drainage

Samples of waste rock, potential construction materials and tailings were subjected to laboratory geochemical tests to assess their potential to generate acid rock drainage (ARD) and leachable 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.  A total of 57 waste 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 program to date are presented below.

All 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 BF 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 meta-sandstone (S3) unit at P8P9 and Siga South indicated some potential to generate ARD and this material will not be used for construction.  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 monitored prior to discharge.

### 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 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 while the process water chemistry comes from decanted water both cyanided but not treated and cyanided with cyanide destruction.

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 International Finance Corporation (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. 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. During extreme rainfall events, the current design is 1/100-year event, should the TSF storage capacity be exceeded and an effluent discharge is required there is the possibility to exceed both the Total CN and arsenic effluent discharge criteria.

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 insure the Burkinabe do not try to 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. Inert wastes will be cleaned and inspected, then disposed of in landfill sites at the toe of advancing waste dumps. 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. Waste oils will be recycled by the supplier, 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 or buried beneath waste dumps. 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 the 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 underground water 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$14.5M 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.

- 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 and rain and runoff water catchment basins. 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$)**

| ID | Description | Cost |
|---|---|---|
| **1. Environmental and Social Management Plan** | | |
| 1.1 | Environmental Management | $461,200 |
| 1.3 | Environmental and Social Clauses for Contractor Specifications | $19,600 |
| 1.4 | Hazardous Waste Management Plan | $29,400 |
| 1.5 | Environmental Monitoring Program During Construction | $94,000 |
| 1.6 | Environmental & Social Monitoring Program (During Operation and Closure) | $230,600 |
| 1.7 | Building Plan | $49,000 |
| | **Sub Total** | **$883,800** |
| 1.8 | Management Fees | $160,281 |
| 1.9 | Contingency | $352,618 |
| | **TOTAL Management Plan** | **$1,396,699** |
| **2. Dismantlement and rehabilitation** | | |
| 2.1 | General Mine Cleaning | $378,583 |
| 2.2 | Mine Pit Management (Security Fencing) | $39,160 |
| 2.3 | General Dismantling | $918,439 |
| 2.4 | Site Grading | $5,885,797 |
| 2.5 | Seeding & Reforestation of Dumps & Tailings | $1,823,653 |
| 2.6 | Seeding Labour | $74,575 |
| 2.7 | Post Closure Monitoring Over 5 Years | $671,236 |
| 2.8 | Supporting Employees and Suppliers | $97,900 |
| | **Sub Total** | **$9,889,344** |
| 2.9 | Management Fees (10%) | $988,934 |
| 2.10 | Contingency (20%) | $2,175,656 |
| | **TOTAL Site Reclamation & Closure** | **$13,053,933** |
| **3. Grand Total** | | **$14,450,632** |

*Note: calculation errors are due to rounding*

## 21.0    CAPITAL AND OPERATING COSTS

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.

## 21.1    Mining Costs

Four mining contractors currently operating in similar local mining operations in Burkina Faso were approached in May 2018 to provide quotes. Bidders were provided with the mine production schedule, including annual haulage distances for ore and waste and requested to estimate operating costs for the first five-years of production for loading, hauling, and dump rehandling activities for the following haulage routes:

- Ore: Mine to process plant.

- Waste: Mine to TSF.

- Waste: Mine to WRD.

The basis for the contractor estimates is as follows:

- Diesel fuel to be supplied by Orezone and charged back at cost, assuming US$1.00/L.

- Maintenance facilities to be supplied, built and staffed by the contractor.

- On-site accommodation camp to be provided by the contractor where deemed necessary.

- In-pit ramps and surface-haul roads to be maintained by the contractor.

- Dust-control water trucks to be provided by the contractor.

Contractor bids were analyzed, and the range of costs compared against the Reserve Estimate pit optimization inputs used by AMC. Contractor-quoted waste mining costs were in part higher than the costs used in the pit optimization due to the increased haulage distance associated with utilizing 82% of the waste for TSF construction. Mine operating costs were adjusted in the Minemax schedule to account for the increase.

## 21.2    Initial Capital Cost

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 Q2 2018 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 10), 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 checked 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 in the last quarter of 2020 (4Q2020) prior to achieving commercial production on January 1, 2021. The table also excludes the value of gold produced in 4Q 2020 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 to derive the Total Upfront Costs.

**Table 21.1          Project Capital Costs to January 1 2021 (US$, 2Q2018, ±15%)**

| Project Capital Area | US$ M |
|---|---|
| Construction In-directs | 13.2 |
| Treatment Plant | 34.7 |
| Reagents & Plant Services | 10.6 |
| Infrastructure | 16.2 |
| Management Costs (EPCM) | 11.6 |
| Resettlement Action Plan | 24.3 |
| Owner's Costs[1] | 22.4 |
| **Subtotal** | **133.0** |
| Contingency | 10.6 |
| **Sub-total** | **143.6** |
| Mine costs (2019/2020) | 37.5 |
| **Total** | **181.1** |

[1]includes $1.0M 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:

- RAP and other early works.

- Other Owner's costs.

- Process facilities.

- Site infrastructure.

- Stage 1 of the TSF.

- Initial surface water management facilities.

- Installation costs, EPCM costs and contractor distributable costs.

- Site earthworks, establishing the main access road and site roads and tracks.

- Project contingency.

Exclusions include the following:

- Project sunk costs.

- Import duties and taxes on the basis that the Project will be exempt.

- Escalation.

## 21.3    Process Plant & Infrastructure Capital Cost

The process plant and infrastructure capital estimate summarized by discipline is provided in Table 21.2.

**Table 21.2**        **Capital Estimate (Excluding Mine Development) Summary by Discipline (US$, 2Q2018, ±15%)**

| Discipline | Supply Cost US$ ('000) | Freight Cost US$ ('000) | Installation Cost US$ ('000) | Contingency US$ ('000) | Total US$ ('000) |
|---|---|---|---|---|---|
| General | 3,248 | 21 | 187 | 519 | 3,975 |
| Earthworks | 6,047 | 82 | 6,065 | 1,257 | 13,451 |
| Concrete | 4,032 | 15 | 2,229 | 744 | 7,020 |
| Steelwork | 3,030 | 283 | 1,261 | 495 | 5,069 |
| Platework | 4,114 | 444 | 2,523 | 696 | 7,777 |
| Mechanical | 16,493 | 2,279 | 1,293 | 1,953 | 22,018 |
| Piping | 5,949 | 526 | 3,983 | 1,416 | 11,873 |
| Electrical | 5,917 | 703 | 1,240 | 997 | 8,857 |
| Instrumentation | 1,510 | 216 | 442 | 218 | 2,385 |
| Buildings | 1,865 | 296 | 238 | 266 | 2,665 |
| Mining | - | - | - | - | - |
| Owners Cost (incl. RAP) | 44,705 | 166 | - | 2,056 | 46,928 |
| EPCM Costs | 344 | - | 11,259 | - | 11,604 |
| **Grand Total** | **97,254** | **5,031** | **30,720** | **10,617** | **143,622** |

## 21.4     Sustaining Capital and Closure Costs

Sustaining capital and closure costs are shown in Tables 21.3, 21.4 and 21.5 below. The reference to Stages in the sustaining capital estimate relates to the TSF development stages which, in the later years, spread over more than one year per stage.

**Table 21.3  Summary of Mine Capital Costs (US$, 2Q2018, ±15%)**

| Capital Cost | Units | Total | 2019 | 2020 | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| OCR Construction & Mine Development | US$M | 36.3 | 12.2 | 24.2 | | | | | | | | | | | |
| Surface-haul Roads | US$M | 1.7 | 0.2 | 0.4 | 0.2 | 0.2 | 0.1 | 0.1 | 0.1 | 0.1 | 0.1 | 0.0 | 0.0 | 0.0 | 0.1 |
| Pit dewatering | US$M | 0.6 | 0.4 | 0.1 | 0.1 | | | | | | | | | | |
| Total | US$M | 38.6 | 12.8 | 24.7 | 0.3 | 0.2 | 0.1 | 0.1 | 0.1 | 0.1 | 0.1 | 0.0 | 0.0 | 0.0 | 0.1 |

**Table 21.4  Sustaining Capital Excluding Mining (US$, 2Q2018, ±15%)**

| | Sustaining Capital Costs | Sustaining Total Cost (US$) | Stage 2 YEAR 2021 | Stage 3 YEAR 2022 | Stage 4 YEAR 2023 | Stage 5 YEAR 2024 | Stage 6 YEAR 2025 | Stage 7 YEAR 2026 | YEAR 2027 | YEAR 2028 | Stage 8 YEAR 2029 | YEAR 2030 | Stage 9 YEAR 2031 | YEAR 2032 | YEAR 2033 | Closure YEAR Closure |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Second Stage Tails Pump | $ 170,659 | | $ 170,659 | | | | | | | | | | | | |
| 2 | High Pressure Gland Water Pump | $ 72,762 | | $ 72,762 | | | | | | | | | | | | |
| 3 | TSF (excluding mine overhaul costs) | $ 55,877,331 | $ 4,261,041 | $ 5,394,079 | $ 5,001,271 | $ 2,505,010 | $ 4,330,950 | $ 4,485,997 | $ 4,485,997 | $ 4,485,997 | $ 6,301,371 | $ 6,301,371 | $ 2,774,749 | $ 2,774,749 | $ 2,774,749 | |
| 4 | Pipeline | $ 1,632,588 | $ 89,569 | $ 68,331 | $ 626,181 | $ 141,075 | $ 114,473 | $ 69,548 | $ 69,548 | $ 69,548 | $ 95,652 | $ 95,652 | $ 64,337 | $ 64,337 | $ 64,337 | |
| 5 | Valves | $ 37,406 | | | $ 37,406 | | | | | | | | | | | |
| | Total | $ 57,790,746 | $ 4,350,609 | $ 5,705,831 | $ 5,664,858 | $ 2,646,085 | $ 4,445,423 | $ 4,555,545 | $ 4,555,545 | $ 4,555,545 | $ 6,397,024 | $ 6,397,024 | $ 2,839,086 | $ 2,839,086 | $ 2,839,086 | $ - |

**Table 21.5  Closure Costs (US$, 2Q2018, ±15%)**

| | Reclamation and Closure | 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 | YEAR Closure |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | TSF Closure | $ 1,928,877 | | | | | | | | | | | | | | $ 1,928,877 |
| 2 | Remainder of Closure Cost | $ 12,521,755 | | | | | | | | | | | | | | $ 12,521,755 |
| | Total | $ 14,450,632 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ 14,450,632 |

## 21.5      Operating Costs

### 21.5.1     Overall

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

- The mine operating costs developed by AMC (refer 21.1).

- 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 21.6.

**Table 21.6          Life of Mine Operating Costs per Tonne and per Gold Ounce (US$, 2Q2018)**

|  | Total Cost ($M) | $/Tonne Processed | $/Ounce Au |
|---|---|---|---|
| Mining | 257.0 | 4.59 | 251 |
| Processing | 275.8 | 4.92 | 269 |
| G&A | 93.6 | 1.67 | 92 |
| Refining & Bullion Transport | 1.5 | 0.03 | 2 |
| Government Royalties & Dev Tax | 65.2 | 1.17 | 64 |
| **Total Cash Cost** | **693.1** | **12.38** | **677** |

### 21.5.2     Process Operating Costs

The process operating costs for the Project have been developed in accordance with industry practice for feasibility studies for gold ore processing plants.

Quantities and cost data were compiled from a variety of sources including:

- Metallurgical testwork.

- Consumable prices from suppliers.

- Advice from Orezone.

- Lycopodium data and estimating methodologies.

- First principle calculations.

The operating cost estimate includes the following major categories:

- Operating consumables.

- Plant maintenance costs.

- Power.

- Labour (operation and maintenance).

- Laboratory costs.

The process operating costs are summarized in Table 21.7.

**Table 21.7          Process Plant 4.5 Mtpa Operating Cost Summary**

| Cost Centre | Process Operating Cost | |
|---|---|---|
| | US$/year | US$/tonne Ore |
| **Plant Operating Costs** | | |
|    Operating Consumables | 9,751,011 | 2.17 |
|    Plant Maintenance | 845,070 | 0.19 |
|    Laboratory | 159,919 | 0.04 |
|    Power | 8,501,886 | 1.89 |
|    Labour (Plant Operations & Maintenance) | 2,904,311 | 0.65 |
| **Total** | **22,162,196** | **4.92** |

### 21.5.3    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$7.61M annually, or approximately US$1.69 per tonne ore over the LOM, not including preproduction.

These costs include insurance, permitting, office supplies, general management, accounting, communications, 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.

## 21.6    Operating Cost by Year

With varying annual mining rates in the pits, and a period at the end of the mine life where plant operations continue treating low-grade stockpiles, annual operating costs vary. The estimated operating costs by year are shown in Table 21.8.

**Table 21.8**  **Operating Costs by Year (USD, 2Q2018)**

| Cost | Total (US$/t) | Units | Total | 2020 | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mining | **4.59** | US$M | **257.0** | 36.3 | 25.6 | 27.4 | 24.4 | 20.5 | 19.1 | 19.2 | 19.8 | 18.0 | 16.1 | 15.9 | 11.9 | 1.4 | 0.4 |
| Processing | **4.92** | US$M | **275.8** | 3.3 | 22.2 | 22.2 | 22.2 | 22.2 | 22.2 | 22.2 | 22.2 | 22.2 | 22.2 | 22.2 | 22.2 | 22.2 | 6.5 |
| General & Administration | **1.67** | US$M | **93.6** | 0.0 | 7.8 | 8.2 | 8.1 | 7.8 | 7.5 | 7.4 | 7.6 | 7.7 | 7.6 | 7.7 | 7.5 | 6.6 | 2.0 |
| Refining & Bullion Transport | **0.03** | US$M | **1.5** | 0.02 | 0.23 | 0.19 | 0.19 | 0.14 | 0.12 | 0.11 | 0.10 | 0.10 | 0.09 | 0.09 | 0.08 | 0.05 | 0.02 |
| Government Royalties & Dev Tax | **1.17** | US$M | **65.2** | 1.6 | 9.9 | 8.1 | 7.9 | 5.9 | 4.9 | 4.8 | 4.4 | 4.2 | 3.7 | 3.6 | 3.5 | 2.3 | 0.7 |
| **Total** | **12.38** | **US$M** | **693.1** | **41.2** | **65.8** | **65.9** | **63.8** | **56.5** | **53.8** | **53.8** | **54.1** | **52.2** | **49.6** | **49.5** | **45.2** | **32.4** | **9.6** |

## 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. The model was structured using an EXCEL workbook.

Input data were 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 assessment was based upon:

- Capital cost estimates and expenditure schedules prepared by Lycopodium and AMC.

- Mine schedule and mining operation cost estimates based on contract mining operations, as developed by AMC for the study.

- Process operating and general and administration 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 WSP.

- 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 2Q 2018.

- The forecast gold price of $1,275/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.

- Payback.

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 $214.9 M, with an initial project construction cost of $143.8 M.

Table 22.2 shows the project cash flow summary. At a gold price of $1,275/oz the project is estimated to have an after-tax IRR of 42.4% and a pay-back period of 1.7 years. At a discount rate of 5% the after tax NPV is estimated at $223.5 M. The project economics are summarized in Table 22.3.

**Table 22.1          Production Summary**

|                                  | Value        |
|----------------------------------|--------------|
| Ore processed                    | 56.0 Mt      |
| Total tonnes mined               | 149.8 Mt     |
| Average head grade               | 0.64 g Au/t  |
| Contained gold in material       | 1.1 Moz      |
| Total gold produced              | 1.0 Moz      |
| Average gold recovery            | 89.1%        |
| Production life (processing)     | 13 years     |
| Nominal annual processing rate   | 4.5 Mtpa     |

**Table 22.2          Net Profit after Tax Summary (LOM Summary)**

|                                            | $M      | $/Ore Processed | $/oz Au |
|--------------------------------------------|---------|-----------------|---------|
| Revenue (99.93% payable)                   | $1,305  | $23.31          | $1274   |
| Mine Operating Cost                        | $257.0  | $4.59           | $251    |
| Processing Cost                            | $275.8  | $4.92           | $269    |
| G&A Cost                                    | $93.6   | $1.67           | $91     |
| Refining & Transport Costs                 | $1.5    | $0.03           | $2      |
| Government Royalties                        | $65.2   | $1.17           | $64     |
| **Total Cash Cost**                         | **$693.1** | **$12.38**   | **$677** |
| **EBITDA**                                  | **$611.9** | **$10.93**   | **$597** |
| Initial Capital                             | $143.8  | $2.57           | $140    |
| Sustaining Capital                          | $58.9   | $1.05           | $58     |
| Rehabilitation & Closure (net of salvage)   | $12.2   | $0.22           | $12     |
| **Gross Profit before tax**                 | **$396.9** | **$7.09**    | **$388** |
| Corporate Tax Payable                       | $104.4  | $1.86           | $102    |
| **Net Profit after tax**                    | **$292.6** | **$5.22**    | **$286** |

**Table 22.3          Financial Summary**

|  | Value |
| --- | --- |
| Revenue from gold (99.93% payable) | $1,305M |
| Adjusted Operating Costs (AOC) | $677/oz Au |
| Initial Capital | $143.8M |
| Sustaining capital | $58.9M |
| Closure costs/salvage | $12.2M |
| Pre-tax economics: |  |
| IRR | 59.3% |
| NPV (5%) | $315M |
| Payback | 1.3 years |
| After-tax economics: |  |
| IRR | 42.4% |
| NPV (5%) | $223M |
| Payback | 1.7 years |

## 22.2    Project Total Upfront Costs

The Total Upfront Costs shown in Table 22.4.

**Table 22.4          Total Upfront Costs**

|  | $ M |
| --- | --- |
| Process Plant | $45.3 |
| Infrastructure | $16.2 |
| Mining (Haul Roads & Pit Dewatering) | $1.2 |
| Construction In-directs | $13.2 |
| Management Costs (EPCM) | $11.6 |
| Resettlement Action Plan | $24.3 |
| Owner's Costs (less Opening Stocks) | $21.5 |
| **Subtotal** | **$133.3** |
| Contingency | $10.5 |
| **Total Initial Construction Costs** | **$143.8** |
| Working Capital | $33.7 |
| Pre-production Operating Costs | $8.5 |
| Pre-production Gold Sales | -$31.2 |
| **Total Upfront Costs** | **$154.8** |

The Total Upfront Costs represent the project capital estimate plus capitalized costs incurred to achieve commercial production (on January 1, 2021) less the value of gold recovered during the pre-commercial production period (October to December 2020 inclusive). For further detail on the project capital costs refer to Section 21.

## 22.3    Assumptions and Qualifications

*General*

- The cash flow model has been based on a 2-year project development period but all cash-outflows commence in Year -1 and that gold production commences Q4 2020. 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 $14.5 M has been included.

- An estimated asset residual sale value of $2.3 M has been included.

- The cash flow model assumes full equity funding.

- No provision has been made for interest on cost of capital.

- No provision has been made for corporate head office costs during operations.

- No provision has been made for escalation or inflation.

- The NPV calculation is based on payments occurring mid-year.

*Working Capital*

- Working capital has been calculated and included in the cash flow model.

- Working capital includes opening stocks.

- 18% TVA has been applied to operating expenses.

*Gold Price*

- A gold price of $1,275/oz has been applied in the cash flow model.

*Smelter Terms*

- Gold is assumed to be payable at 99.93%.

- Refining costs have been included at $0.5/oz.

- Transport costs have been included at $1.0/oz.

*Government Royalties and Taxes*

- Based on $1,275/oz gold price, 4% royalty has been applied.

- 1% development tax on revenues has been applied.

- No provision has been made for customs import duty.

- 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.

## 22.4    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 Tables 22.5 and 22.6 and in Figures 22.1 and 22.2.

**Table 22.5         NPV Sensitivity Analysis (Pre-tax)**

|  | Lower | Base Case | Higher |
|---|---|---|---|
| Gold Price (±10%) | $214,946,257 | $315,226,406 | $403,895,169 |
| CAPEX (±10%) | $334,599,885 | $315,226,406 | $295,877,441 |
| OPEX (±10%) | $367,998,784 | $315,226,406 | $262,454,028 |
| Recovery (±2%) | $295,195,231 | $315,226,406 | $335,257,581 |

**Table 22.6         IRR Sensitivity Analysis (Pre-tax)**

|  | Lower | Base Case | Higher |
|---|---|---|---|
| Gold Price (±10%) | 44% | 59% | 73% |
| CAPEX (±10%) | 68% | 59% | 52% |
| OPEX (±10%) | 65% | 59% | 54% |
| Recovery (±2%) | 56% | 59% | 62% |

**Figure 22.1        NPV Sensitivity Analysis (Pre-tax)**
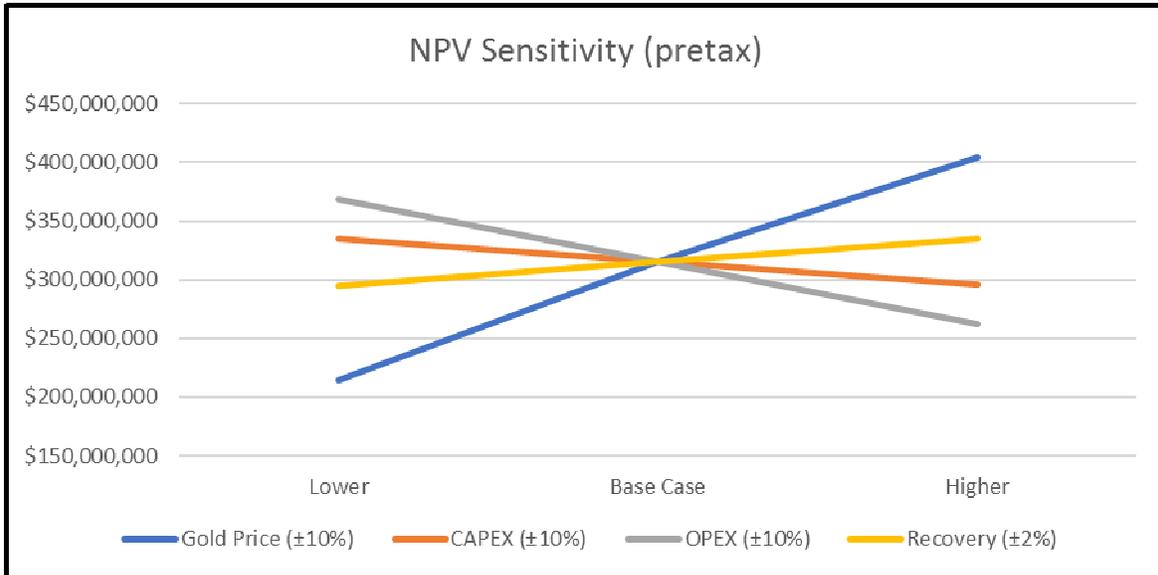


**Figure 22.2        IRR Sensitivity Analysis (Pre-tax)**
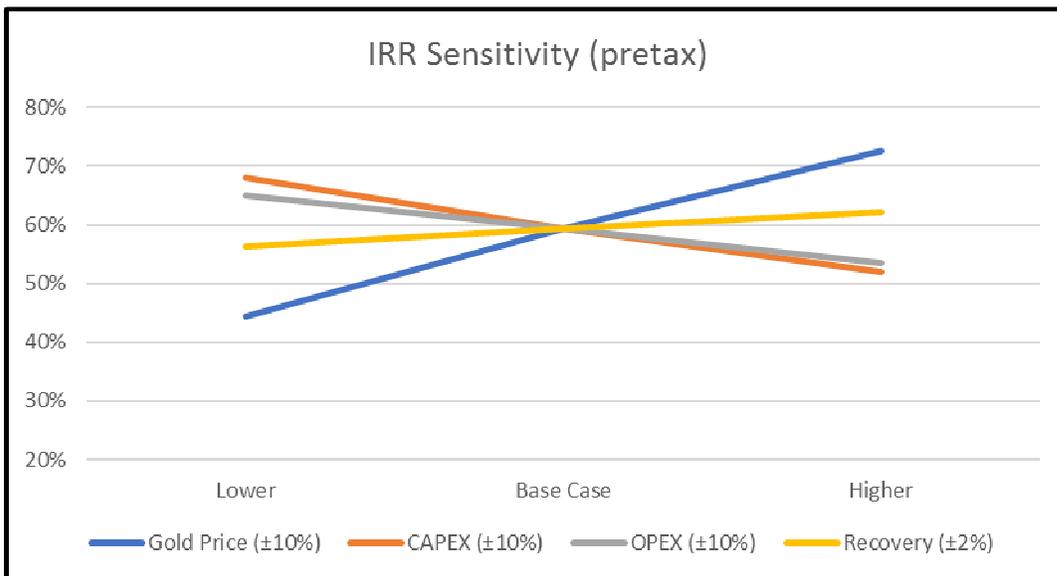
## 23.0    ADJACENT PROPERTIES

### 23.1    West African Resources Boulsa Project

Parts of the land tenements related to West African Resources' (TSXV: 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 only some 2 km northeast of the Bomboré tenements.

### 23.2    West African Resources Sanbrado Gold Project

In June 2018 WAF reported Mineral Reserves with an effective date of 25 April 2018 at their Sanbrado Project some 10 km to the southeast of the Bomboré tenements. Probable reserves are totalling 20.4 Mt at an average grade of 2.4 g/t for 1.57 Moz. These reserves are based on indicated resources totalling 39.60 Mt at an average grade of 1.8 g/t for 2.35 Moz, a combination of open cut (pit) resources of 1.73 Moz and underground resources of 0.62 Moz. WAF resource statement also shows inferred resources totalling 13.85 Mt at an average grade of 1.2 g/t for 0.55 Moz. In March 2018 their updated ESIA for the Project was approved.

### 23.3    B2GOLD Toega Gold Project

In January 2018 B2GOLD reported inferred resources, with an effective date of 8 January 2018, totalling 14.2 Mt at an average grade of 2.0 g Au/t for 0.92 Moz at their Toega Project some 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.

As these early works are being undertaken on site, the appointed Engineer will commence the Front End Engineering Design (FEED), the development of the various Project plans, procurement of certain long lead items, and the establishment of Project controls systems.  Once the FEED has been reviewed and accepted by Orezone, the Engineer may commence detailed engineering design and drafting.

Orezone is required to prepare, tender and award the mining contract to enable the successful contractor to mobilize and commence the OCR excavation works in June 2019. 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 2018 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 August 2019, leading to first gold pour on October 30, 2020.

A Level 1 schedule is provided in Figure 24.1.

Figure 24.1        Level 1 Schedule
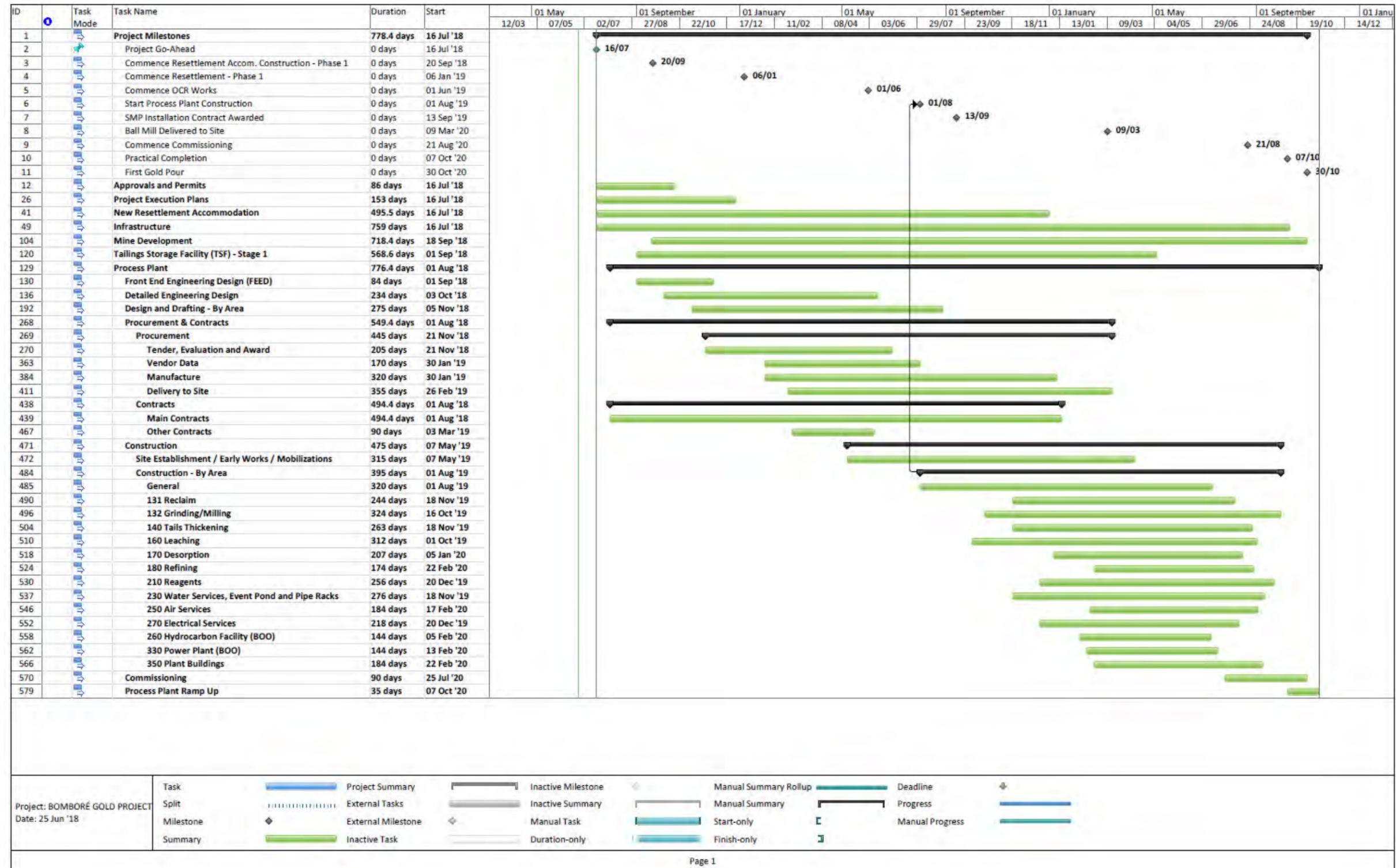
Key dates in the Project schedule are as follows:

- Project Go-Ahead — Jul 16, 2018

- Resettlement Accommodation — Jul 16, 2018 – Oct 15, 2019

- Infrastructure Works — Jul 16, 2018 – Jun 7, 2019

- Front End Engineering Design (FEED) — Sep 1, 2018 - Nov 30, 2018

- TSF Design, Tender & Construct — Sep 1, 2018 - May 5, 2020

- OCR Construct — Jun 1, 2019 – Jan 2, 2020

- Detailed Engineering — Oct 3, 2018 – Jun 12, 2019

- Design and Drafting — Nov 5, 2018 – Aug 28, 2019

- Procurement and Deliveries — Nov 21, 2018 – Mar 14, 2020

- Contracts Tender & Award — Aug 1, 2018 – Jan 15, 2020

- Earthworks — Aug 1, 2019 – Jan 31, 2020

- Concrete works — Oct 1, 2019 - Apr 27, 2020

- Tankage installation — Nov 30, 2019 – Jun 3, 2020

- Structural Mechanical Piping works — Jan 28, 2020 – Aug 28, 2020

- Electrical works — Apr 10, 2020 – Aug 28,2020

- Commissioning — Jul 25, 2020 – Oct 30, 2020

- Practical Completion — Oct 7, 2020

- First Gold Pour — Oct 30, 2020

The critical path for the Project schedule has been identified as follows:

- FEED, detailed engineering, design and drafting; process and mechanical/piping/platework.

- Tender and award of civil/concrete works contract.

- Mobilization of civil/concrete works contractor and establishment of concrete batch plant.

- Manufacture and delivery of ball mill.

- CIL tank concrete ring beams and ball mill foundation concrete.

- Grinding area mill access steel.

- Grinding area mechanical installation.

- Grinding piping installation.

- Grinding electrical and instrumentation installation.

- Commissioning.

## 24.2    Operational Readiness

Orezone will establish an experienced operational management team on site well in advance of commissioning.

Comprehensive policies will be developed and induction, training and operating procedures put in place as part of an overall Operational Readiness Plan that will be prepared 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 year broken down by department and Expatriate/National numbers is shown in Table 24.1.

**Table 24.1          Annual Headcount**

| | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Mining (Orezone employees only)** | | | | | | | | | | | | | |
| Ex-pats | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | - | - |
| Nationals | 51 | 51 | 51 | 51 | 51 | 51 | 51 | 51 | 51 | 51 | 26 | - | - |
| **Processing** | | | | | | | | | | | | | |
| Ex-pats | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 |
| Nationals | 98 | 98 | 98 | 98 | 98 | 98 | 98 | 98 | 98 | 98 | 98 | 98 | 98 |
| **G&A** | | | | | | | | | | | | | |
| Ex-pats | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 |
| Nationals | 56 | 56 | 56 | 56 | 56 | 56 | 56 | 56 | 56 | 56 | 56 | 56 | 56 |
| **Headcount total** | **213** | **213** | **213** | **213** | **213** | **213** | **213** | **213** | **213** | **213** | **187** | **160** | **160** |

### 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.  Initially 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.

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 direct operations personnel 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 56.0 Mt over a 13-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 is 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 | Ore tonnes Processed (Mt) | Gold Grade (g\t) | Recovery (%) | Gold Production ('000 ounces) |
|------|------|------|------|------|
| Pre-prod. | 0.68 | 1.20 | 94.2% | 24.5 |
| 1 | 4.50 | 1.14 | 93.9% | 155.4 |
| 2 | 4.50 | 0.95 | 92.6% | 126.7 |
| 3 | 4.50 | 0.92 | 92.5% | 123.5 |
| 4 | 4.50 | 0.71 | 90.2% | 92.0 |
| 5 | 4.50 | 0.60 | 88.5% | 77.3 |
| 6 | 4.50 | 0.59 | 88.1% | 74.7 |
| 7 | 4.50 | 0.54 | 87.2% | 68.7 |
| 8 | 4.50 | 0.53 | 86.8% | 66.1 |
| 9 | 4.50 | 0.47 | 85.1% | 57.7 |
| 10 | 4.50 | 0.46 | 84.9% | 56.7 |
| 11 | 4.50 | 0.44 | 84.3% | 54.2 |
| 12 | 4.50 | 0.32 | 78.3% | 36.2 |
| 13 | 1.32 | 0.32 | 78.3% | 10.6 |
| **Life of Mine** | **56.0** | **0.64** | **89.1%** | **1024.2** |

## 24.4    Closure

Disturbed areas and dump slopes will be stabilised and revegetated on an ongoing basis to reduce sediment loads in run-off, minimise 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, 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 of was derived from Knight Piésold'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 were 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 individual Qualified Persons conclusions listed below, the Feasibility Study has identified a viable and attractive development opportunity centred around the mining and processing of the oxide and upper transition zones of the mineralization on the Bomboré tenements.

## 25.1 Geology and Mineral Resources

RPA offers the following conclusions:

- 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 and Mineral Reserve estimation work.

- In RPA's opinion, the 5 January 2017 Mineral Resource estimation work is in accordance with the CIM definitions and the results are reasonable.

- At a cut-off grade of 0.2 g Au/t for oxide and transition material and 0.38 g Au/t for fresh material, M&I Mineral Resources are estimated to total 218.10 Mt at an average grade of 0.68 g Au/t for 4.8 million ounces of contained gold. At the same cut-off grades, Inferred Mineral Resources are estimated to total an additional 48.20 Mt at an average grade of 0.64 g Au/t for 994,000 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 Au/t.

- Data from 276 new holes totalling 15,387 m, located 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 resource model is still appropriate to be used as the basis for the 2018 FS and that the effective date should remain 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 a free dig open pit mining operation consisting of over 60 separate pits of variable size and depth across a mineralized zone 11 km long and 2.5 km wide. Mining is planned to span 12 years with ROM ore delivered to the plant followed by processing low-grade stockpiles at the end of mine life.

The key project life-of-mine highlights are:

- 149.8 Mt total material mined:

    – 56.0Mt of ore

    – Diluted gold grade 0.64g Au/t

    – 1.1 Million in situ ounces Au

    – 93.8Mt waste

    – 1.68 strip ratio

    – 13-year mine life

- Pre-production of 1 year for excavation of OCR for water storage and supply of waste for TSF construction.

- Process throughput of 4.5Mtpa:

    – 41.1Mt at 0.70g Au/t ROM ore

    – 14.9Mt at 0.47g Au/t low-grade ore rehandled from stockpiles

    – 1.024 Moz Au produced

The Reserve Estimate was generated by diluting the Resource Estimate block models. Contact dilution of 0.5 m was added to the footwall and hangingwall of the mineralization and internal waste blocks were included to reflect a 3 m minimum mining width appropriate to the selected mining equipment.

Pit optimizations were run using a base case gold price of US$ 1,250. The US$ 1,250 pit shells were selected as final pits to maximize project value and provide the required ore tonnages for the mine plan. The final pits formed the basis of the mine design incorporating the slope angle recommendations provided by Golder.

The mine will be contractor operated using 3-6 m$^3$ excavators and 30-50 tonne highway dump trucks which are appropriate for the scale of mining operations and provide the required selectivity for ore mining. An owner's team on site will be responsible for contract management, grade control and mine planning activities.

A mine production schedule was developed, including the excavation of the OCR, with the mine design divided into 48 mining areas and scheduled based on strip ratios and grades to maximise operating cash flows over life of mine. ROM ore will be hauled to the process plant and low-grade ore hauled to one of three low-grade stockpiles during the initial mining periods. The final two years of the mine plan consist of stockpile reclaim operations. Approximately 82% of waste produced will be used for TSF construction with the remainder being hauled to two WRD and four environmental barriers.

## 25.3    Tailings Disposal and Site Water Management

KP offers the following conclusions:

- The TSF will utilize mine waste in the construction of the dam without the need for additional borrow materials.

- By utilizing sub-aerial rotational tailings deposition methods the actual tailings density achieved will exceed the design tailings density thus creating more capacity within the TSF or will allowing the dams to be lowered in elevation to achieve the same storage tonnage

- The OCR will be a benefit to the local population, to some extend, as a long-term water resource.

## 25.4    Metallurgy and Process

Lycopodium offers the following conclusions from the metallurgical testwork with regards to a CIL process:

- Oxide and transition ores at Bomboré are readily amenable to whole ore cyanidation.

- Gold recoveries are expected to be 90% plus for head grades over 0.75 g Au/t, high 80%'s for head grades of 0.5 to 0.75 g Au/t, and low 80%'s for head grades of 0.3 to 0.5 g Au/t.

- Optimum grind size was determined to be $P_{80}$ 125 µm based on grind size and recovery relationship.

- Leach extraction rates are essentially complete within 24 hours based on the observed leach kinetics.

- Cyanide consumption rates are expected to be low, averaging about 0.19 kg NaCN/t ore.

- Lime consumption rates are expected to be moderate, averaging at 2 kg/t ore for 90% CaO quicklime.

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 oxide and transition ores.

## 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, both supported by the 2015 FS. 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.

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 655 households or about 3,915 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 in conjunction with the Feasibility Study.

This will require the initial (Stage 1) resettlement of about 410 households from traditional villages and about 915 households from the Sanam Yaar artisanal gold processing site. The subsequent resettlement (Stage 2) of about 250 farming households and 450 households from the Kagtanga artisanal gold processing site, all from the southern area of the Project, could occur after the initial Stage 1 resettlement as this area will not be immediately affected by the mine construction.

It is important to note that Orezone personnel have direct experience with resettlement on a large scale at a previous project in Burkina Faso.

## 25.7    Other Opportunities

There are several opportunities mentioned within the report. However, there are two key opportunities that should be investigated during Front-End Engineering and Design are outlined below.

### 25.7.1 TSF Redesign

At the commencement of the 2018 FS KP were briefed to keep the footprint of the TSF within the footprint of the combined TSF/heap leach pads from the 2015 study. At the time this was deemed prudent to avoid environmental permitting issues that may arise from an increase in the footprint of the HDPE lined area and to avoid displacing waste dumps from the area to the south of the liner.

Although the TSF constrained by this footprint is a viable option it became obvious towards the end of the study that:

- Limiting the area of the TSF resulted in a more expensive TSF design as the embankment volume to contained tailings volume is high.

- A larger footprint offered a more efficient design with significantly lower embankment volumes and costs.

- The waste dump and low-grade stockpile in the area immediately to the south of the TSF are modest in size and could be relocated.

- There is sufficient area elsewhere to relocate the dump and stockpile.

KP were requested to prepare a conceptual design for a TSF with a larger footprint extended to the south of its current location. This was reported in their memorandum DV-18-0756 of July 9, 2018.

The memorandum suggests that the quantity of compacted mine waste material required to contain the required +60 Mt of tailings could be reduced from 40 $Mm^3$ to 12.6 $Mm^3$ with a potential saving, life of mine, of over $25 million. Although some of the savings will be offset by an increase in the quantity of HDPE liner required the net savings are still expected to be of the order of $20 million.

In addition, the redesigned TSF will be lower in height and will be less visually intrusive and less prone to erosion or instability in the long term.

As the area impacted was previously overlain by a waste rock dump it is believed that obtaining a variation to the permits required to adopt the redesigned TSF will be straightforward. This opportunity will be further examined during Front End Engineering and Design.

### 25.7.2 Increase Plant Throughput

Although the process plant was designed to treat 4.5 Mtpa of ore the mining rate in the early years of the operation is in excess of this in order to feed the plant with material around 1 g Au/t. The lower-grade ore mined at this time is stockpiled for treatment later in the life of the operation.

The ball mill is designed to treat 4.5 Mtpa of a blend of 70% saprolite and 30% transition ores with the transition ores, although still soft, requiring more power to achieve the target grind size. A further 300 kW was also added to the ball mill drive to match the installed power with the shell dimensions as this will provide better operability and flexibility in the grinding circuit.

The life of mine mix of saprolite and transition ores is less than the design 70/30 blend meaning that the ore blend will be softer and there will be surplus capacity in the ball mill. The 3.2 MW ball mill has the power capacity to grind 5.72 Mtpa of 100% saprolite.

This surplus grinding capacity can be utilized in several ways:

- Increase the mill throughput in the early years of the operation using low-grade feed that is currently part of stockpile feed.

- Extending the mining further into the more competent transition zones of the orebody using perhaps a ripper-equipped large bulldozer.

- Delineating and mining the high-grade sulphide zones as a supplementary plant feed. This will require a small crushing plant to prepare the fresh rock as ball mill feed.

Although there is some surplus capacity in the ball mill the balance of the process plant is designed for 4.5 Mtpa. This will require redesign/debottlenecking before the plant 'nameplate' capacity can be increased. Alternatives are to undertake this at the front-end engineering and design stage, effectively building a larger plant, or wait until after commissioning and undertake a post-commissioning debottlenecking exercise, only making modifications where required. The obvious target for an increase in throughput is to 5.13 Mtpa as the addition of a seventh CIL tank will provide sufficient leach circuit retention time for this throughput. Provision for this seventh CIL tank has been made in the plant layout.

Lycopodium has been tasked with examining the capital cost impact of increasing the nameplate throughput to 5.13 Mtpa and undertaking an exercise to quantify the benefit of the resulting earlier treatment of stockpiled low-grade ore, or if permitted at that time the mining of the Restricted Zones as part of a higher mining rate as the grade of the material in the Restricted Zones is expected to be significantly higher than that of the lower-grade stockpiled ore. Orezone will then decide on whether to change the project design basis or whether to adopt a 'wait and see' approach and look at post-commissioning optimization options.

## 26.0    RECOMMENDATIONS

The Property hosts a significant gold deposit and merits considerable exploration, engineering, and development work.  The near-term primary objective is to advance engineering work.

## 26.1    Mineral Resource Model

- Incorporation of the in-fill drilling completed after the current resource model cut-off date, i.e. since November 2016, to 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 Au/t) 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 Au/t 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, environmental permitting of restricted (flood) zone resources, and a Preliminary Economic Assessment of the higher-grade sulphide resource.  The budget for this program is US$505,000.

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 | | 200,000 |
| | Restricted Zones ESIA | | 130,000 |
| | Open Pit Sulphide Project Preliminary Economic Assessment | | 75,000 |
| Toéyoko | P17S Resource Model Update | | 25,000 |
| | P13 Maiden Resource Model | | 25,000 |
| Bomboré II | KT Maiden Resource Model | | 25,000 |
| Bomboré III | P17S-NE Maiden Resource Model | | 25,000 |
| **TOTAL Phase I** | | | **505,000** |

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$1,242,000 (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 | 1,000 m | 125,000 |
| **TOTAL Phase II** | | | **1,242,000** |

## 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 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.

- A separate study to assess the feasibility of operating within the flood plains excluded from the Reserve Estimate in this study. This work has recently commenced and AMC are at present working on a mine plan.

- Continued negotiations with local 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 verses 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 discussed in Section 25.6.1, that the alternative TSF option described in that section be assessed and advanced to front end engineering and design if the stated conditions and criteria remain applicable. If this option is selected, which will not utilize the majority of the mine waste within the dam, then the mine plan for development of waste dumps may need to be reassessed.

Based on the final waste dump layouts the surface water management systems may need to be updated accordingly. The surface water management systems would then, at that time and commensurate with the final mine plan, be advanced to front end engineering and design.

In addition, a dam break assessment for the final TSF dam configuration should be completed.

## 26.4    Metallurgy and Process

It is recommended that the process plant and infrastructure elements proceed to Front-End Engineering and Design to further detail the project scope and facilitate commencing equipment procurement and site works once the Phase 1 RAP has been implemented.

It is recommended that during plant operations:

- Natural cyanide attenuation (free and WAD) be monitored in the tailings storage facility.

- Site water quality (raw and process) be monitored during the initial wet and dry seasons to document the seasonal impact of water quality

- Gold adsorption rate and equilibrium loading on carbon be monitored as the plant head grade varies during the life of the operation to ensure that carbon movement and management is optimized.

## 26.5    Environmental and Permitting

Given that Orezone already has already been granted the Industrial Operating Permit for the Project, resettlement efforts and planning should be pursued actively to allow for resettlement activities to begin immediately.

An Environmental and Social Management Plan (ESMP) should be prepared, aligned with the scope and schedule for the resettlement and project construction, to put in place the project environmental management protocols with the Bureau National des Évaluations Environnementales (BUNEE).

## 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:

- Commence execution of Phase 1 of the RAP. Tender documents for this work were issued and Orezone is currently reviewing bids for this work which is expected to be awarded in Q3 2018.

- Appointment of a lead EPCM Engineer.

- Preliminary site works to prepare for contractor mobilization. Work in this area has commenced and is at an early stage.

- Development of the FS schedule and budgets into detailed control tools for executing the project.

- Commencement of Front-End Engineering and Design across the Project scope.

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