REVISED PRELIMINARY ECONOMIC ASSESSMENT

NI 43-101 TECHNICAL REPORT

BLOCK 14 GOLD PROJECT

REPUBLIC OF THE SUDAN

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

The author has undertaken an extensive review of Orca Gold's Block 14 Gold Project's technical and economic data, and has reviewed Orca Gold's contributions to this report.

The overall report entitled Revised Preliminary Economic Assessment Technical Report, Block 14 Gold Project Republic of Sudan, was collated and signed by the following author:

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1. EXECUTIVE SUMMARY

1.1 Introduction

This report comprises a Revised Preliminary Economic Assessment (PEA) of Orca Gold's Block 14 Project (Project) located in Sudan.

The revised PEA has been prepared by Lycopodium Minerals Pty Ltd on behalf of Orca Gold Inc. (Company). This Technical Report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

1.2 Property Description and Ownership

1.2.1 Property Description

The Block 14 mineral exploration project covers an area of 2,170 km^2, located in the Nubian Desert in the north of the Republic of the Sudan, close to the international border with Egypt. The property is situated 700 km north of the national capital Khartoum and straddles the boundary between the Red Sea and Nile States. The nearest population centre is the town of Abu Hamad, located 180 km due south of the property.

1.2.2 Ownership

All exploration and mining projects in Sudan are subject to '*The Mineral Resources Development and Mining Act, 2007*', which sets forth the legal and fiscal framework for the administration of the country's mineral industry by the Ministry of Minerals (MoM). Industrial levels of exploration and mining rights are provided for in the Mining Code, defined by concession agreements and granted under Exclusive Prospecting Licence and Mining Leases. The Sudan Mineral Resources Company (SMRC) has been established by the Ministry of Minerals to oversee and monitor mineral exploration and development.

The Exclusive Prospecting License (EPL) for Block 14 was originally granted to Meyas Nub under a Concession Agreement dated 19 May 2010. A purchase agreement dated 1 March 2012 (the "Purchase Agreement"), granted Sand Metals Company Ltd (SMCL), a wholly owned subsidiary of Orca, the right to acquire a 70% interest in Meyas Sand Mineral Company Ltd (MSMCL), a Sudanese joint venture company incorporated to hold the Block 14 EPL.

Under the terms of the Purchase Agreement, SMCL have paid Meyas Nub US$9.5M and own 70% of MSMCL.

Under the Purchase Agreement, Orca agreed to fund all exploration, development, and construction costs to commercial production, and to fund all costs associated with maintaining the properties in good standing with respect to the mining code.

The Exclusive Prospecting Licence is granted under terms of the concession agreement for the exploration and exploitation of gold and associated metals and minerals in Lower Gabgaba Area ("Block 14"). The initial exploration period was for four years however, during 2014 the joint venture company MSMCL was granted an additional year extension to the Initial Exploration Period.

In May 2017, the Block 14 EPL was renewed for a period of one year with a 50% relinquishment, as laid out in the Concession Agreement.

Details of key dates and terms are detailed Table 1.1.

Table 1.1 Key Dates and Terms

Start Date	Expiry Date	Area (km^2)	Terms
19 May 2010	19 May 2014	7,046	Initial exploration period.
19 May 2014	19 May 2015	7,046	Additional year compensation period granted, extending the expiry date of the initial exploration period.
19 May 2015	19 May 2017	3,747	Formal notification of Mineral Discovery Areas, relinquishment of 50% of the Initial Exploration area.
19 May 2017	19 May 2018	2,170	Relinquishment of 50% of remaining area (excluding mineral discovery area).

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

A number of international airlines with routes to Europe, as well as other African centres, are scheduled daily from Khartoum. The airport at Port Sudan also has international flight handling capability, although services are limited to neighbouring countries. Air travel to other destinations in the country is limited.

The country is served by a deep water port at Port Sudan through which the bulk of the country's imports and exports pass.

The Project is accessible by an established network of roads and desert tracks, with the permanent camp located 180 km north of the town of Abu Hamad. A major paved route connects Port Sudan, Khartoum, and Abu Hamad from which point a well-used desert road capable of handling large loads connects to the project area. In general, vehicular access to the Project is good and is not affected by seasonal variations.

Since June 2015 a weather station has been recording data in Block 14. The climate is arid with a hot season from June to September, during which the maximum temperatures range from 45 - 49°C and minimum of 23° - 35°C. The coolest period covers the months of January and February, with daytime high temperatures of 28°C and cool nights approximately 5°C. During the hot weather the project area is subject to strong winds, predominantly from the north.

The northern desert of the Republic of the Sudan has infrequent precipitation and regional records indicate average annual precipitation to be nil. However, infrequent rainstorms do occur with records indicating events of up to 6.5 mm rainfall in a single event. The weather station in Block 14 has recorded five events of rain >1 mm and one event producing 17.5 mm since June 2015.

The local area of the Project is uninhabited with no significant population centres outside of Abu Hamad. People from the Ababda tribe have settled in small towns and villages close to the Nile and, wherever possible, Orca employs members of this community. There are a large number of artisanal miners working in the region whom Orca employs on a casual basis from time to time.

The infrastructure requirements for the development of the project are minimal. Operational requirements can be satisfied using regionally available materials and services, mainly from Abu Hamad, Atbara (where the company operates a small logistics office) and Khartoum (administration office).

Sudan has an established paved road network which is generally of good quality, however some small bridges in more remote areas have been washed away following flooding in 2013, but it is possible to bypass these locations using dry channels or wadis.

Access to the permanent camp in Wadi Gabgaba, Block 14 is via desert tracks from Abu Hamad that are used by the large artisanal mining community. Two fuel stations with telephone communications, known locally as “cafeterias”, are located along the route.

The landscape is characterised by rocky hills separated by wide, flat sand filled drainage channels known as wadis. Elevation varies between 198 and 810 meters above sea level (masl), with the bulk of the higher ground in the east of the project area. Wadi Gabgaba, which runs north-south through the centre of the project area, is the main drainage system in the area. The western part of Block 14 contains more subdued topography.

Vegetation is restricted to the larger wadis where water is available from the crystalline basement, and consists of Doum palms and sparse thorny shrubs. Much of the Project area is un-vegetated with occasional desert grasses and stunted trees.

1.4 Geology and Mineralization

In a number of locations, gold has also been identified in broad shear zones with sub-ordinate quartz veining. It is generally associated with narrow gash veins, shear type veins, and quartz veinlet swarms in well foliated schistose rocks within the volcano-sedimentry domains, which are intruded by stocks and sheets of diorites, syenites, and granitoids.

The mineralization types being targeted within the Block 14 project are broadly categorized into three groups, namely Orogenic Gold, Volcanogenic Massive Sulphide, and Rift Associated Epithermal.

Vein densities are seen to increase in and around contacts with intrusive that are focused within structural corridors. The veins form arrays that are often over 10 km long and transgress multiple lithologies. Broader shear zone hosted orogenic gold mineralization has been identified at a number of prospects in Block 14, however the current focus is on GSS where sheared host rocks, as well as discrete quarts veins, host broad zones of gold mineralization.

VMS style mineralization with well-developed base metal gossans have been identified at Tanashieb in Eastern Gabgaba, Block 14. The mineralization is associated with felsic volcanics (dacites) within a dominantly mafic package of arc related rocks.

The geological framework of Block 14 is dominated by the Keraf Shear Zone (KSZ) which underlies the main Wadi Gabgaba and separates two distinct geological terranes, namely Eastern Gabgaba and Western Gabgaba. Wadi Gabgaba masks the KSZ but it is apparent that the western edge of Eastern Gabgaba is represented by a complex tectonic interleaving of both terranes.

Eastern Gabgaba is dominated by thick sequences of andesitic volcanics with sub-ordinate felsic volcanics and some metasediments. The volcanic arc is intruded by multiple phases of syn-tectonic diorites and post collisional sub-alkalic intrusives that may represent the Bayudan phase of igneous activity. Western Gabgaba is characterised by a package of marine sediments containing numerous volcanic centres. The Western Gabgaba ‘donut’ represents an annular feature some 15 km across that contains mafic volcanics on its periphery, and is cored by an intermediate package of volcanics and calcareous metasediments.

The geological setting is varied but four main types of artisanal mining are apparent:

- Colluvial mining using metal detectors that have a reliable penetration of 20 - 30 cm. Tractors, front end loaders, and excavators are used to clear each swath of material after it has been screened.
- Colluvial mining using dry screening methods, often re-working gravel previously worked using metal detectors.
- Traditional artisanal mining on thin quartz veins which can be laterally extensive and are in places mined to 30 m below surface.
- Mechanised, open pit mining to depths of up to 30 m. These are often associated with vein swarms within distinct shear zones. Gold bearing rocks are selected using metal detectors both in situ and in broken material.

1.4.1 Galat Sufar South (GSS)

The deposit is located in the central portion of the Western Gabgaba anticlinorium (donut) on the southern flanks of a fold interference, culmination with an axial surface trace trending east-north-east.

The north east apices of this doubly plunging antiform hosts the GSS North prospect and other prospects are located on its northern flanks. A silicified dolomite is a marker to the core of the structure which contains an interleaved package of lower greenschist facies metamorphosed carbonates, marls and volcanics.

The GSS deposit is located within a package of intermediate volcanics, diorites, and syenites that have a penetrative schistosity that both controls and is cut by well-developed shearing, alteration, and mineralization. Scale mapping (1:25,000) has identified a chlorite epidote bearing sequence of monotonous andesitic volcanics distal to the sericite dominant core of GSS. Directly south of GSS the presence of chlorite–epidote is controlled by through going faults and, close to the mineralization, these fault zones contain quartz–epidote (epidosite).

The dominant trends of shearing at GSS are 110° (sub-parallel to the S1 cleavage) and 010°, they dominate alteration and vein development, are steep and display well developed C – S fabrics. The 010° set of shears are generally better developed in Main Zone and are mylonitic. An intersection lineation of the two main shears is defined by material shoots that plunge steeply to the NW. The N-S structural break between the eastern and western zones of GSS is poorly exposed and not completely understood.

Six domains of mineralization have been differentiated and resource modelled

- The 320 Zone: is present as discontinuous ribbons and plunging shoots located within a shear zone oriented 140° that is parallel with the local schistosity. The 320 Zone contains high grade shoots (+10 g/t) that plunge steeply to the NW in the same direction as the intersection lineation of 010° and 110° shears.
- The Main Zone: is a wide laminar zone oriented 020° that links in to the 320 Zone in the south and has a strike length of 150 m. The Main Zone represents a set of 010° trending shears that link two through-going 110° trending shears across a rigid body of syenite / k-feldspar altered diorite in the footwall. Shear fabrics in Main Zone are extreme and mylonites have been identified. The Zone is up to 90 m true width and is host to some of the best intercepts in the Project. Grade is strongest at the southern end, closest to the intersection of the 320 Zone.

- The 050 Zone is a small domain that links the East Zone with a north–south trending set of vein mineralization within a covered area termed The Gap. The 050 Zone is considered to be a compressional duplex that translates movement from the East Zone into a large through-going 010° trending shear within The Gap. Recent drilling has shown that in the south, the 050 Zone links into a 110° tending shear known as Target J.
- The East Zone: is hosted within a 100° oriented corridor within which several small dismembered intrusive stocks, often brecciated, have been mapped and logged. Shear fabrics wrap around the intrusive bodies although the kinematics of this fault system are not fully understood. True width again exceeds 90m in the central East Zone with subsidiary, parallel mineralization being present to the north and south with a similar, although not fully defined, trend. Shear fabrics in oriented core are dominated by a 110° orientation dipping steeply to the west. Sub-ordinate 010° trending shear is also present, although these north going areas of mineralization are often obliterated by silicification. A third sub-ordinate fabric oriented 030° is related to steep, irregular, coarse grained, gold bearing cataclasite that is interpreted to be a late stage deformation. A barren plug of alteration in East Zone is adjacent to high grade tectonic breccias and may be associated with the quartz blow seen at surface.
- The Far East Zone has a similar trend to the East Zone and the mineralization is hosted within a sheared sericitised microdiorite. Steep to vertical mineralized trends are open to the north and west, where discontinuous grade extends under cover.
- The Shareg Zone is a north / south trending steep trend of mineralization hosted within a sericite / carbonate altered diorite / microdiorite.

Alteration is pervasive and deposit-scale units are defined by their alteration assemblages, which are variably zoned outward from the gold mineralization. The first phase of mineralization is a milky white, fused quartz blow that is randomly oriented and cross cut by shear zones and associated veining. On the eastern margin of GSS the k-feldspar alteration clearly overprints a Foliated Diorite (MDI) and Porphyritic Diorite (IDI) forming the Potassium Altered Diorite (KDI). The KDI forms a rigid body that is sheared on its contacts and overprinted by the Quartz-Sericite Schist (QSS) and Quartz-Sericite-Pyrite schist.

Where the k-feldspar alteration is intense, a texture destructive Black Red Diorite (BRD) with incipient hornfelsing is preserved within highly sheared QSP altered high grade mineralization, but generally the k-feldspar alteration contains weak, variable mineralization. Altered tectonic breccias are common in the East Zone, where they are often pervasively overprinted by potassium feldspar alteration and subsequent shear foliation.

Vein quartz is present within all of the mineralized intersections but is rarely mapped at surface due to its exploitation by artisanal miners, and is best represented by the position of artisanal workings.

The quartz veins show multiple generations of development and are themselves mylonitised and brecciated in Main Zone. Quartz blow is present throughout the region and outside of GSS carries no gold. However, at GSS it is brecciated, has sheared contacts, and can contain significant gold.

Gold at GSS is associated primarily with intense sericite – carbonate alteration, moderate silicification, pyrite content and quartz veining from a millimetre scale to a maximum of 1.5 m.

1.4.2 Wadi Doum (WD)

The main, high grade mineralization at WD outcrops at the base of the hill and is hosted by a strongly sulphidic volcaniclastic unit, which is in contact with a distinct rhyolite unit to the immediate

east. The volcanoclastic unit dips at an angle of 20° to the south west. This rhyolite is bounded to the east by a dacitic unit intruded by syn-tectonic Syenite / potassium altered diorite body which forms the summit of the main hill.

These lithologies are cut by thin (<0.75 cm), late, un-mineralized felsic and mafic dykes. In contrast to the volcaniclastics, the rocks on the hill dip 75° to the east. Mineralization on the hill is associated with stringer zones within the syenite and in places smaller shears.

The high grade mineralization is hosted within the volcanoclastic units which are confined by late felsic and mafic dykes. The mineralization is divided into three distinct units:

- A western volcaniclastic unit characterized by a dark colour caused by very fine grained sulphides (>10 - 15%), which contains some of the best intercepts.
- A central unit of paler, sulphide rich felsic volcaniclastics, which contain deformed sulphide veinlets.
- A lower grade footwall unit of largely un-deformed felsic volcaniclastics.

The dominant sulphide is pyrite (85% in Qemscan analysis) with the remainder comprising a mix of sphalerite, galena, chalcopyrite, and Freibergite.

Alteration is confined to sericitisation within the felsic volcanics and a wider halo of carbonate alteration. Silicification is noticeably absent or weak within the high grade part of the deposit (hence its location at the base of the hill.

The area is dominated by a strong and pervasive, north-south trending schistosity, which is largely followed by the late dykes. The high grade mineralization often appears un-affected by structure, whereas the mineralization hosted by the syenite on and around the summit of the hill does appear structurally controlled.

1.5 Mineral Resource Estimate

In February 2017, MPR Geological Consultants Pty Ltd (MPR) estimated gold Mineral Resources for the GSS and WD deposits. MPR estimated recoverable resources by Multiple Indicator Kriging with block support correction to reflect open pit mining selectivity, a method that has been demonstrated to provide reliable estimates of gold resources recoverable by open pit mining for a wide range of mineralization styles

The current estimates utilized Reverse Circulation (RC) and diamond drilling data supplied by Orca in December 2016. Modifications to the supplied sampling information included adjusting down-hole survey entries which showed unrealistic down-hole deviations, such as azimuth changes of more than 50° in five metres.

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

The Mineral Resource estimates have been classified and reported in accordance with NI 43-101 and classifications adopted by CIM Council in November 2004. The table below shows the estimate at 1.0 g/t cut-off subdivided by oxidation type. The figures in these tables are rounded to reflect the precision of the estimates and may include rounding errors.

The Mineral Resources are reported using supplied topographic surfaces with no allowance for depletion by currently active artisanal mining, which is considered to have a minor impact on the reported estimates.

The GSS estimates extend to around 350 m depth. The WD estimates extend to around 210 m depth.

For GSS the combined oxidized and transitional material hosts around 37% and 13% of the Indicated and Inferred resources respectively, with the remainder lying in fresh rock. For WD, the combined oxidized and transitional material hosts around 17% and 13% of the Indicated and Inferred resources respectively.

Table 1.2 Resource Estimate

GSS

Material	Indicated			Inferred		
	Mt	**Au g/t**	**Au koz**	**Mt**	**Au g/t**	**Au koz**
Oxide	5.6	1.85	334	0.6	1.7	32
Transition	4.9	1.76	279	0.6	1.5	27
Fresh	18.0	1.72	996	7.8	1.7	426
TOTAL	**28.6**	**1.75**	**1,609**	**9.0**	**1.7**	**485**

WD

Material	Indicated			Inferred		
	Mt	**Au g/t**	**Au koz**	**Mt**	**Au g/t**	**Au koz**
Oxide	0.2	2.97	24	0.1	2.5	7
Transition	0.1	2.87	9	0.0	1.5	0.5
Fresh	1.7	2.76	150	0.6	2.2	44
TOTAL	**2.0**	**2.79**	**183**	**0.7**	**2.2**	**52**

Combined

Material	Indicated			Inferred		
	Mt	**Au g/t**	**Au koz**	**Mt**	**Au g/t**	**Au koz**
Oxide	5.9	1.90	358	0.7	1.8	39
Transition	5.0	1.78	288	0.6	1.5	27
Fresh	19.7	1.81	1,146	8.4	1.7	470
TOTAL	**30.6**	**1.82**	**1,792**	**9.7**	**1.7**	**536**

1.6 Metallurgy

The flowsheet development has been focused on the following concept:

- Single Stage crushing followed by SAG mill – ball mill ("SAB") circuit.
- Whole Rock Leaching in hybrid CIL mode, with pre-leaching of gold to speed up carbon kinetics.
- No cyanide detoxification with recycling of free cyanide back to the circuit from tailings thickener.
- Conventional tailings storage.

The main thrust of the recent flowsheet development has focused upon the optimization of the primary grind size over the life of mine. Twenty-three variability samples from the four zones (East, Main, Wadi Doum, and North East) and domains (Oxides, Transition, and Fresh) were selected to

represent a range of gold grades. All samples were tested at four different P_{80} grind sizes (53, 75, 106, and 125 µm). Six SMC (SAG mill) hardness tests from dominant domains in each zone and 57 Modified Bond Ball mill tests calibrated with eight full Bond Ball mill tests were performed.

The following metallurgical gold dissolutions and silver dissolutions are summarized in Table 1.3.

Table 1.3 Summary of Metallurgical Data

	Grind (µm)	Au Recovery	Ag Recovery
East Zone Oxide	53	89.70%	32%
Main Zone Oxide	53	91.80%	35%
NE Zone Oxide	53	91.80%	33%
WD Oxide	53	91.80%	33%
East Zone Transition	75	84.90%	69%
Main Zone Transition	75	81.30%	47%
NE Zone Transition	75	83.10%	58%
WD Transition	75	83.10%	58%
East Zone Fresh	94	81.00%	68%
Main Zone Fresh	94	83.70%	59%
NE Zone Fresh	94	82.40%	63%
Wadi Doum Fresh	94	85.70%	57%

The next stage of work will focus upon:

- Further hardness variability testing to refine SAB circuit physical properties for dominant lithologies in each domain.
- Further bottle roll variability testing to increase the understanding of gold dissolution and cyanide consumptions in each zone, domain and dominant lithology.
- Carbon Modelling on oxide, transition and fresh composite blends of material to replicate the mining sequence and capture the effects that these different mixes of material types have on the overall leach circuit design.
- The tailings generated from carbon modelling will be used to provide thickener sizing data.
- The potential to reduce cyanide consumptions by recycling non detoxified tailings water back to the circuit will be verified.
- Performing effect of site water from post leach thickener overflow to optimize the process water circuit.

1.7 Mining

A review of existing Block 14 gold concession geotechnical data was conducted by SRK Consulting (UK) Limited, which was used as a basis for the subsequent optimization and pit design.

Nine specific geotechnical boreholes have been drilled at Galat Sufar South and Wadi Doum for the geotechnical study in order to gather rock mass and structural information to derive slope parameters. This data complimented the basic geotechnical data from 18 cored boreholes which was provided for the 2016 PEA study. A structural database was created from the logging of downhole geophysics OTV / ATV images, in addition to foliation and joint orientation measurements made in 2016. Structural domains were identified, presenting their own joint set orientations and attributes. Point load testing and laboratory strength testing of drill core was carried out to characterise the rock mass strength. Inter-ramp pit slope angles range from 37° in the near surface

weathered oxide rock mass to between 58° and 65° in the fresh rock, depending on structural geology controls and vertical inter-ramp height.

Given the gold grades and proximity to surface, the deposits will be mined via a conventional truck and excavator open pit mining method. The WD deposit will be exploited through a single pit approximately 150 m deep. The GSS deposit will be exploited by ten separate pits, eight of which are shallow oxide / transitional pits only. The other two are deeper than the WD pit at 210 and 250 m.

There is scope for larger pits under improved geotechnical or financial conditions.

It is assumed that mining is conducted by a mining contractor, utilizing a mining fleet comprised of Caterpillar 777 rigid body haul trucks (90 t) with suitably sized loading unit, although there may be merit in using a mix of equipment sizes to improve recovery and minimise mining dilution. Mining costs were broken into base and incremental mining costs. Costs were built from first principles using knowledge of several mining contracts operating under similar conditions in West Africa.

Optimization scenarios were primarily centred around flexing the annual processing rate and processing cost associated with volume of water supply and tailings deposition strategy. Optimizations examined Indicated material, and Indicated and Inferred material for each scenario. A gold price of US$1,200 was used for the optimization.

A processing rate of 3.4 Mtpa was selected and Run 32 for GSS and Run 17 for WD were used for the work. Less than 10% of the crusher feed material included in the pits was Inferred and was included in the optimization.

Using the selected optimization shells as reference, open pits were designed to develop a more realistic mining scenario. Ramps and berms were included in these designs.

Batter, berm, and wall angle configurations varied for the different material types and ramp widths were varied between double lane and single lane to reflect likely mining practice. Where possible, a "goodbye cut" was designed at the base of each pit to maximise extraction of crusher feed.

Based on the assumed mining equipment, a bench height of 5 m was used, although geotechnical conditions allowed for up to four benches to be excavated between safety berms, depending on the material. There may be some opportunity to mine higher bench heights in areas of bulk waste.

The GSS deposit contributes 94% of the total crusher feed and 89% of the contained ounces. The deposit is exploited through 10 individual pits extending along strike for over 3.2 km. The overall strip ratio for the pits is 2.22:1.

Although much smaller than the GSS pits to the west, the WD deposit contains mineralization at 1.8 times the grade of GSS. The deposit is exploited through a three stage 150 m deep pit. The overall strip ratio for the WD pit is 4.3:1.

Waste dumps were designed for both GSS and WD. Given the terrain and lack of other land use, there is very little restriction on waste dump capacity so dump location was chosen to best suit the extraction requirements.

A ROM pad for GSS was also designed and can cater for six separate ROM fingers. Blending will occur on the ROM pad. A skyway has been designed at the back of the ROM pad which allows the construction of 10 m high fingers, allowing approximately 20,000 t of rock to be stockpiled on each finger.

While space has been allocated for a ROM pad at WD, no specific design has been completed. It is unlikely that ROM material will accumulate at WD to such quantities as to require multiple lifts being constructed.

Surface haul roads have been designed in a preliminary form although detailed cut and fill analysis has not yet been completed. The topography and climate will mean that relatively simple haul road construction will be sufficient. Surface haulage distances were estimated for the various deposits to allow for the calculation of mining costs.

The WD pit was divided into three stages to assist with waste stripping and crusher feed optimization. The two larger GSS pits were also divided into several cutbacks while the rest of the pits were treated as single pits for the purposes of scheduling.

A ramp up period of 2.5 years was assumed at the start of the schedule. Pre-stripping and minor feed stockpiling (approximately 250,000 t) was scheduled six months prior to the start of processing (Pre-production). The target for Year 1 was 85% of the ultimate 3.4 Mtpa rate, while the second year was increased to 91%. From Year 3, the full processing rate of 3.4m Mtpa was achieved until the final year.

A combined mining schedule was produced which preferentially treated the higher grade material from WD, while meeting the annual production targets with additional material from GSS. The mining schedule shows a 13.2 year mine life, with WD being completed in Year 7. Mining dilution and mining recovery were not included in the schedule, which are accounted for in the Resource Estimate.

The Owner will not undertake any mining activities directly. All mobile maintenance will be the responsibility of the contractor. A second contract may be required for the crusher feed haulage from WD to GSS.

1.8 Recovery Methods

Metallurgical testwork conducted to date (standard bottle roll and diagnostic leach testwork), indicate that the GSS and WD material types are amenable to gold recovery via cyanidation. The most economically effective process scheme identified as comminution followed by adsorption of gold onto activated carbon, through a hybrid (pre-leaching to speed up carbon kinetics) carbon-in-leach (CIL) process. The design of the process plant has been based on a nominal capacity of 3.4 Mtpa.

The process flowsheet incorporates a single stage primary jaw crusher with a crushed feed surge bin and an emergency stockpile, providing surge capacity between the crushing and grinding circuits. Feed from the emergency stockpile will be reclaimed by front end loader (FEL) to feed the mill during periods when primary crushing is off-line. The grinding circuit will be configured as a two stage circuit with a SAG mill and ball mill, both with the ability to operate in closed circuit. The circuit will produce a P_{80} of 53 µm for oxide dominant, approximately 75 µm for transition dominant material and between 80 and 106 µm for fresh dominant blends. Grinding circuit product flows to a pre-leach thickener and then a Leach and CIL circuit incorporating three dedicated leach tanks ahead of six stages of CIL for gold adsorption. A split AARL elution circuit followed by electrowinning, mercury retorting, and smelting to recover gold and silver from the loaded carbon will produce doré, whilst safely removing mercury. Tailings will be thickened to recover and recycle process water and then pumped to the tailings storage facility (TSF).

1.9 Project Infrastructure

1.9.1 Water Supply

The estimated water demand is 7,000 m^3/day (81 L/s), resulting in total aquifer requirement of 34.0 Mm^3 over the 13.2 year life.

Two alternatives sources of water were identified:

- **Alternative 1:** Establishing a pump station and pipeline from the Nile River in Abu Hamed, located to approximately 200 km south of the project area.
- **Alternative 2:** Establishing a number of production bores, a pump station and pipeline from a groundwater resource ("Area 5"), located approximately 80 km south-west of the project area.

In order to verify the hydrogeological viability of Alternative 2, regional and local scale groundwater resource related investigations, involving remote sensing, ground and air geophysics, drilling and pump testing, have been carried out in selected areas of the Project area.

Based on the air-geophysical surveys an anomaly of 50 km^2 to 100 km^2 (Area 5), hosted within the Nubian Sandstone basin, was identified and drilled with preliminary pump tests conducted. Based on available drilling observation and the pump test data in Area 5, it can be conservatively assumed that adequate aquifer storage exists to supply the water demand for the entire LOM.

Based on the preliminary data and comparisons between Alternative 1 and 2, Alternative 2 was selected as the preferred alternative for the purposes of the Revised PEA design and economic assessment. Further drilling and testwork is planned to confirm aquifer hydraulic parameters.

1.9.2 Electric Power

Multiple power supply options for the project including grid connection and onsite generation were considered. Based on high capital costs and availability risk, a grid connection has been deemed unviable to the project at this stage. Budgetary tenders for onsite generation from reputable international vendors including Inglett & Stubbs International, Aggreko and KPS Africa were evaluated during the study. Tenders were solicited based on a "build, own, operate" (BOO) cost model, with a staged capital transfer (BOOT) or optional buy-out of the station in Year 5 of operation. This presents the least capital and sovereign risk to the project.

Subsequent evaluation of the tenders determined that an onsite high speed diesel (LFO) power station provides the lowest net present cost (NPC) in the first seven years of operation.

The overall calculated energy cost to the project based on this solution is US$0.168 / kWh.

1.9.3 Roads and Transport

The main supply route to the site will be either via Khartoum or Port Sudan. Roads are tarred between these cities and Abu Hamad (the closest main town to the project area).

Access to the Project area from Abu Hamad is via a site access road, comprising a track through the desert. This track is also used by artisanal and small scale miners, water delivery vehicles, and nomadic travellers. The routes are not clearly defined but are heavily used.

The GSS site is located 100 km east of the No. 6 railway station, North Sudan, and is accessible from site via desert tracks.

1.10 Environmental

There are currently no objections to the development of the Project. The current Exploration Project has been mentioned as an example of good practice by the SMRC, the National authority.

There are few receptors in the area, with no human settlements in proximity. The Project has commenced a number of environmental studies, with a view to developing a detailed database. The remoteness and arid conditions mean that it is hard for wildlife, and those animals present tend to avoid human activity. The use of remote cameras provides the opportunity to record these fauna. Other wildlife records are captured through daily observations. Climate data and weather data has been collected and compared, to provide reliable data for the EIA and design teams. Water data from the existing boreholes and Talat Abda well has been collected, even though there are no known sources of potable water and few potential water users in the vicinity. Social data is also being collected during the Exploration stage, with continuous engagement with artisanal miners resulting in the collection of information that will be used in managing social and environmental impacts arising from the Project.

From a legal perspective, the Project is authorized under the Concession Agreement, which gives the right to establish a mining operation in a responsible manner. The Company has the responsibility to manage the effects of the mining activity, including exploration, in such a way as to mitigate the negative impacts. Environmental measures have been initiated through an Exploration Statement, which is specific to exploration work and includes an Environmental Protection and Management Programme (EPMP) to mitigate impacts associated with the work. In accordance with the Concession Agreement, the EPMP is a dynamic document that will be revised and updated as the Project progresses.

The development of the Project is likely to give rise to a range of environmental and social impacts. However, assuming the implementation of mitigation measures proposed in the EIA, these impacts are considered manageable and controllable. Therefore the development, operation and closure of the Project could be undertaken in an effective environmental and social manner.

1.11 Capital and Operation Costs

1.11.1 Capital Cost

Capital and operating costs have been estimated for the proposed project. These costs were developed in support of a projected cash flow for the operation, which would assess the financial viability of the project.

The capital cost estimate was developed to an accuracy level range ±30% and addresses the engineering, procurement, construction, and start-up of the mine and processing facilities, as well as the ongoing sustaining capital costs. The operating cost estimate includes the cost of mining, processing and related general and administration (G&A) services.

The capital and operating cost estimates were developed for a conventional open pit mine, CIL process plant, and supporting infrastructure for an operation capable of treating 3.4 Mtpa of material. For the purpose of this PEA, a contract mining scenario has been assumed.

The estimate covers the direct and indirect costs of purchasing and constructing the CIL facility and infrastructure components of the project and an allowance for mining related infrastructure.

The estimated cost, shown in the capital estimate summary below, was benchmarked against similar sized projects and was found to be within the cost range of the various projects.

Table 1.4 Capital Estimate Summary (1Q17, ±30%)

Main Area	US$'000s
Mine Costs	$8,332
Treatment Plant Costs	$122,392
TSF Initial cost	$7,902
Engineering	$15,810
Owners Costs	$15,077
Subtotal	**$169,514**
Contingency	$41,113
Grand Total	**$210,627**

1.11.2 Operating Cost - Mining

The mine operating costs were derived from three existing mining contracts using similar equipment awarded in West Africa in the last three years.

Unit costs were determined for the following items:

- Loading.
- Fixed hauling component.
- Drill & Blast.
- Ancillary.
- Mine Administration.

Unit mining costs averaged US$2.64 /t.

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

Incremental mining costs were determined for the fleet and included a fuel and non-fuel component. The non-fuel component covered costs such as operator salary, maintenance costs and other running costs associated with the time spent on ramps. The Incremental Mining Cost was determined to be US$0.032 /t/10 m vertical lift.

It was assumed that crusher feed material from WD would be re-handled into road trucks and hauled to the processing plant at GSS. A haulage cost was calculated based on physical parameters of the haul route and costs from a similar project in West Africa. The material haulage cost applied in this study was US$7.74 /t for crusher feed contributed by WD.

1.11.3 Operating Cost – Process Plant and Infrastructure

The Operating Cost Estimate (OPEX) for the plant and infrastructure has been divided into multiple cost centres with Fixed and Variable costs calculated for each cost centre for each different material type. The operating cost estimate is presented in Section 21.6 and is deemed to have an accuracy of ±30% based on pricing as at 2Q2017. The process operating cost includes all direct costs to produce gold bullion for the Project and is shown for each material type below.

Table 1.5 Process Operating Costs

Cost Centre	Fixed US$'000/y	Main Fresh	Main Trans	Main Oxide	East Fresh	East Trans	East Oxide	Wadi Fresh	NE Oxide
		Variable US$ /t	Variable US$ /t	Variable US$ /t	Variable US$ /t	Variable US$ /t	Variable US$ /t	Variable US$ /t	Variable US$ /t
Power (excluding grinding)	3,127,	1.60	1.60	1.60	1.60	1.60	1.60	1.60	1.60
Grinding Power	-	2.72	2.97	3.05	3.47	3.35	3.15	3.92	2.43
Operating Consumables	-	6.03	5.46	4.13	5.19	6.29	5.10	6.27	4.21
Maintenance Materials	2,921	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Laboratory	581	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06
Process & Maintenance Labour	2,633	0.00	0.00	0.00	0.000	0.00	0.00	0.00	0.00
Administration Labour	2,647	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
General & Administration	6,106	0.00	0.00	0.00	0.00	00.00	0.00	0.00	0.00
TOTAL	**18,015**	**10.65**	**10.33**	**9.09**	**10.56**	**11.55**	**10.15**	**12.10**	**8.54**

1.12 Economic Analysis

A preliminary economic analysis has been carried out for the project using a cash flow model. The model has been constructed using annual cash flows taking into account annual processed tonnages and grades for the CIL feed, process recoveries, metal prices, operating costs and refining charges, royalties and capital expenditures (both initial and sustaining).

The financial analysis used a base price of US$1,200 /oz. The financial assessment of the project is carried out on a "100% equity" basis and the debt and equity sources of capital funds are ignored. No provision is made for the effects of inflation. Current Sudan tax regulations are applied to assess the tax liabilities. All amounts in this section are presented in US$. Discounting has been applied from the first year of operation.

On a pre-tax basis, the project has a Net Present Value $(NPV)_{7\%}$ of US$278M, an Internal Rate of Return (IRR) of 27%, with a payback of 2.6 years following commencement of production; on a post-tax basis, the $NPV_{7\%}$ is US$228M, the IRR is 23%, and the payback is 3.0 years following commencement of production.

1.13 Recommendations

The PEA has demonstrated a strong project with several opportunities for improvement.

1.13.1 Environmental

The final Project design still has to be finalised, but by initiating the EIA process early, results can be used to improve the design, as well as maximising the benefits of the EIA without incurring excessive costs. There are a few improvements that the Project should undertake in the near future, namely:

- Examine the potential for renewable power supplies.
- Maintain a grievance procedure to identify and pre-empt potential tensions with artisanal and small scale mining operations, as well as distant communities that may feel that the Project affects them.

1.13.2 Mining and Geology

Follow up work is required as shown below:

- Extend geotechnical investigations to cover full depth of open pits.

- Complete waste dump and haul road design to allow for more accurate estimate of haulage requirements.
- Consider using a blending algorithm to improve the scheduling based on material blending to provide a more consistent grade and material composition through the processing plant.

1.13.3 Metallurgical Testwork

The next stage of work will focus upon:

- Completion of comminution testing to determine the variability within each dominant zone, domain, and lithology.
- Confirmation of the effect of primary grind on gold recovery and cyanide consumption rates.
- Evaluation of the effect of recycling post leach thickener overflow on cyanide consumption and gold dissolution.
- Carbon modelling in order to confirm the required retention time and carbon advancement rate.

1.13.4 Water Supply from Borehole Well Field

Follow up work is required as shown below:

- Drilling of five larger diameter test boreholes within 50 m from the existing observation / exploration boreholes.
- Detailed pump tests at an optimal constant discharge rate for at least 24 - 48 hrs per borehole.
- It is also recommended to drill an additional four exploration boreholes to determine the full extents of the aquifer to the north and south.
- Detailed assessment of the data, including a preliminary numerical groundwater model to illustrate the zone of depression and aquifer boundaries.

2. INTRODUCTION

Orca Gold's Block 14 Project (Project) is an advanced exploration project, located in Sudan. It is located in the Nubian Desert in the north of the Republic of the Sudan, close to the international border with Egypt. The property is situated 700 km north of the national capital Khartoum and straddles the boundary between the Red Sea and Nile States. The nearest population centre is the town of Abu Hamad, located 180 km due south of the property.

The Project is envisaged to comprise open pit mining operations at Galat Sufar South (GSS) and Wadi Doum (WD), situated approximately 55 km due east of GSS, with the process plant situated at GSS. Material will be trucked from WD for processing at the GSS plant facility.

2.1 Basis of Technical Report

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

The qualified persons (QPs) who signed off on each item in this Technical Report are as follows:

- QP: Michael Hallewell (MPH Minerals Consultancy Ltd), responsible for report Sections: 1.6, 13, 25.7.2, and 26.2.3.
- QP: Nicholas Johnson (MPR Geological Consultants Pty Ltd), responsible for report Sections: 1.5 and 14.
- QP: Carl Nicholas (Mineesia Ltd), responsible for report Sections: 1.10, 20, 25.5, and 26.2.1.
- QP: Chris Reardon (Deswik), responsible for report Sections: 1.7, 16, 21.5, 25.2, 25.7.1, 25.7.2 and 26.2.2.
- QP: Pieter Labuschagne (GCS), responsible for report Sections: 1.9.1, 18.1, and 26.2.4.
- QP: Geoff Duckworth (Lycopodium Minerals Pty Ltd), responsible for report Sections: 17, 18, 21, 22, 25, 26, and 27.
- Other sections have been provided by the Company.

This document provides a Technical Report on the Orca Gold Block 14 Project, prepared according to NI 43-101 guidelines.

2.2 Property Inspections by Authors

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

Table 2.1 Summary of QP Site Visits

Qualified Person	Site Visit	Report Sections of Responsibility (or Shared Responsibility)
Michael Hallewell	-	Sections 1.6, 13, 25.7.2, 26.2.3
Nicholas Johnson	17/01/14 – 21/01/14	Sections 1.5, 14
Carl Nicholas	24/11/14 – 03/12/14	Sections 1.10, 20, 25.5, 26.2.1
Chris Reardon	05/10/16 – 07/10/16	Sections 1.7, 16, 21.5, 25.2, 25.7.1, 25.7.2, 26.2.2
Pieter Labuschagne	23/11/16 – 02/12/16	Sections 1.9.1, 18.1, 26.2.4
Geoff Duckworth	05/10/16 – 07/10/16	Sections 17, 18, 21, 22, 25, 26, 27

2.3 Effective Dates

The Effective Date of this report is 30 May 2017. There were no material changes to the scientific and technical information of the Project between the Effective Date and signature date of this Report.

2.4 Abbreviations

AAS	Atomic Absorption Spectrometry
As	Arsenic
Au	Gold
B14WC	Block 14 Water Concession
BOO	Build Own Operate
BOOT	Build Own Operate Transfer
CA	Concession Agreement
CAE Fusion	Geological Data Management System
CIL	Carbon-in-Leach
CRM	Certified Reference Material
˚C	Degree Celsius
EPL	Exclusive Prospecting Licence
EPMP	Environmental Protection and Management Programme
F_{80}	80% of a unit process feed particle size is below a given size , based on particle size distribution (PSD)
g	grams
g/L	grams per litre
g/t	grams per tonne
HA8	Hydrological Area for potential project water supply (not preferred)
HQ	Exploration drill size (96 mm OD / 63.5 mm ID)
GAT	Gravity Amenability Test
GDMS	Geographical Data Management System
GED	General Exploration Drilling
GPS	Global Positioning System
GSS	Galat Sufar South
HARD	Half Normal Distribution
HLS	Heavy Liquid Separation
hr	Hour / hours
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
IRR	Internal Rate of Return
km	kilometres
km^2	square kilometres
kV	kilovolt
kWh	kilowatt hour
L	litre
M	million
masl	Metres above sea level
MDA	Mineral Discovery Area
Meyas Nub	Previously known as Emdehan Multi-activities Company Ltd
MIK	Multiple Indicator Kriging
Min	minutes
Mm^3	Million cubic metres
MoM	Ministry of Minerals
MSMCL	Meyas Sand Mineral Company Ltd

Mtpa	Million tonne per annum
Mt	Million tonne
NPV	Net Present Value
NQ	Exploration drill size (75. 5mm OD / 47.6 mm ID)
oz	31.10348 grams
PDS	Particle Size Distribution
PFS	Pre-Feasibility Study
ppm	Parts per million
PQ	Exploration drill core size (122.6 mm OD / 85 mm ID)
PVR	Present Value Ratio (NPV/PV of Net Negative Cash Flow)
P_{80}	80% of a unit process product particle size is below a given size , based on particle size distribution (PSD)
RC	Reverse Circulation
RMR	Rock Mass Rating
ROM	Run-of-Mine
RQD	Rock Quality Designation
SMCL	Sand Metals Company Ltd
SMRC	Sudan Mineral Resource Company
SD	Standard Deviation
SG	Specific Gravity
Station 6	Hydrological Area for potential project water supply (preferred)
t	Metric tonne (1,000 kg)
TDS	Total Dissolved Solids
TSF	Tailings Storage Facility
VMS	Volcanogenic Massive Sulphide
WD	Wadi Doum
µm	micron

3. RELIANCE ON OTHER EXPERTS

The author of this report is not qualified to provide comment on the legal issues associated with the Project, including any agreements, joint venture terms, and the legal status of the exploration permits and mining tenure included in the Project.

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

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

Geology: the Author has relied on information provided by Orca Gold Inc for Sections 1.4, 7, 8, 9, 10, 11, and 12.

Mining: the Author has relied on information provided by Deswik Europe Limited for Sections 1.7, 16, 21.5, 25.2, 25.7.1, 25.7.2, and 26.2.2.

Metallurgical Testwork: the Author has relied on information provided by MPH Minerals Consultancy Ltd for Sections 1.6, 13, 25.7.2, and 26.2.3. Lycopodium has reviewed the metallurgical testwork results and concurs with their interpretation.

Hydrogeology: the Author has relied on information provided by GCS Water and Environmental Consultants for Sections 1.9.1, 18.1, and 26.2.1.

Environment and Social: the Author has relied on information provided by Mineesia Limited for Sections 1.10, 20, 25.5, and 26.2.1.

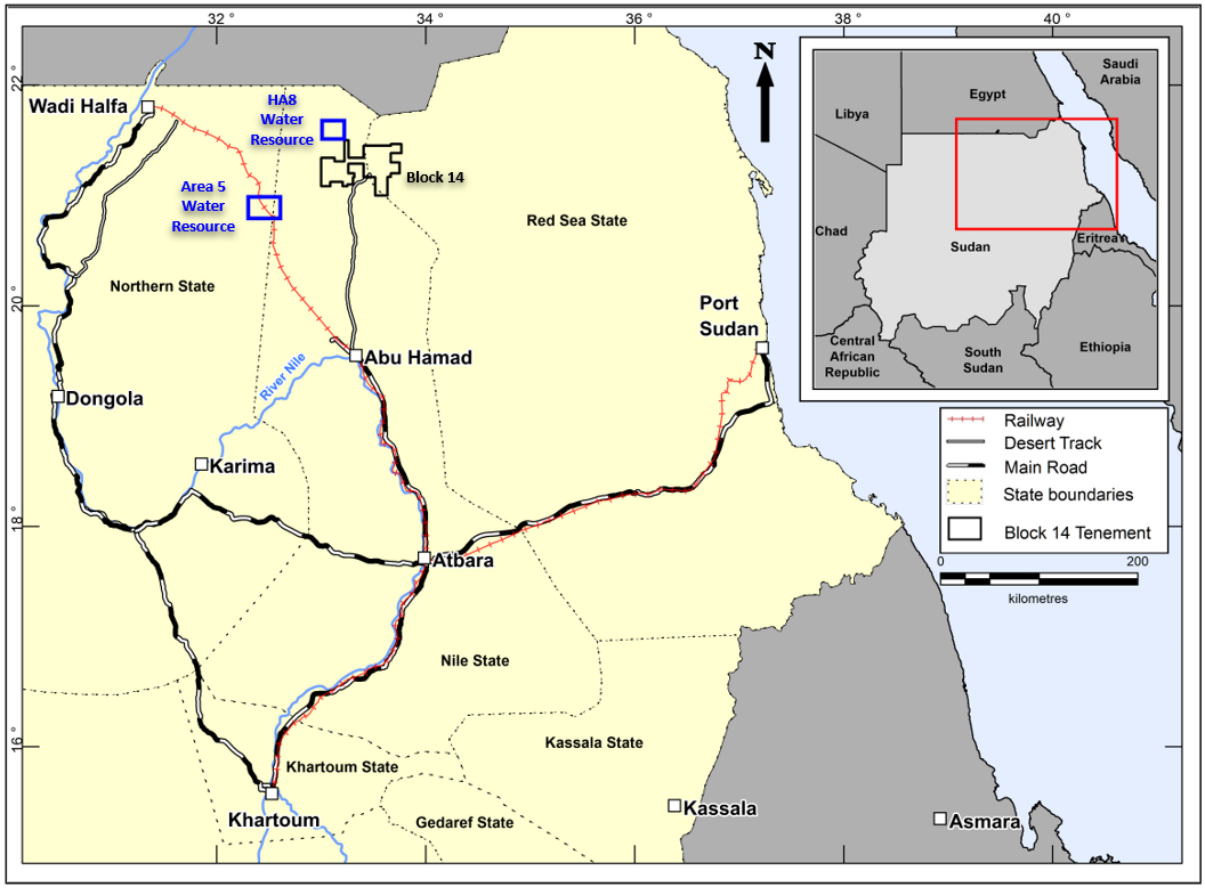
Financial: the Author has relied upon the financial analysis by Orca Gold in Sections 1.12 and 22 of this report. Lycopodium has reviewed the inputs and basis for the financial analysis.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Block 14 mineral exploration project covers an area of 2,170 km^2. It is located in the Nubian Desert in the north of the Republic of the Sudan, close to the international border with Egypt (Figure 4.1). The property is situated 700 km north of the national capital Khartoum and straddles the boundary between the Red Sea and Nile States. The nearest population centre is the town of Abu Hamad, located 180 km due south of the property.

Figure 4.1 Block 14 Current Tenement Boundary



Source Orca Gold Inc.

4.2 Sudan Mineral Tenure and Fiscal Frame Work

All exploration and mining projects in Sudan are subject to '*The Mineral Resources Development and Mining Act, 2007*', which sets forth the legal and fiscal framework for the administration of the country's mineral industry by the Ministry of Minerals (MoM). Industrial levels of exploration and mining rights are provided for in the Mining Code, defined by concession agreements and granted under Exclusive Prospecting Licence and Mining Leases.

The Sudan Mineral Resources Company (SMRC) has been established by the Ministry of Minerals to oversee and monitor mineral exploration and development.

4.2.1 Exclusive Prospecting Licence (EPL)

An EPL is granted by the Minister of Minerals and gives the holder the exclusive right to explore for specified minerals, both on the surface and at depth, within a certain parcel of land described by a

set of co-ordinates. The Block 14 EPL was issued for an initial period of four years renewable for two subsequent extension periods of two years and one year provided the holder has complied with the terms and conditions of the permit and the general provisions of the Concession Agreement.

There is no limit to the maximum surface area of an EPL, but there is a requirement to reduce the size of the EPL by 50% after the first Exploration period, and by a further 50% at the end of the first extension period. The EPL also gives the holder the exclusive right, at any time, to convert the licence into a Mining Lease in accordance with the Concession Agreement.

4.2.2 Mining Lease

A Mining Lease is granted upon application by an existing EPL holder, subject to the provision of a feasibility study, a development and operating plan and an environmental management plan.

Under the terms of the concession agreement, after consultation with the MoM, MSMCL shall incorporate a new company for the purpose of holding the issued mining lease. The permit is granted for an initial 30 year period and may be renewed for subsequent 10 year terms until the mining deposits are exhausted.

A Mining Lease is granted by the Minister of Minerals and gives the holder the exclusive right to explore for and mine mineral deposits within a certain parcel of land for which the permit is granted. The Mining Lease gives the holder the right to construct mineral processing and support facilities and to operate these facilities to produce saleable mineral products that the permit holder will be entitled to sell on world markets.

Upon grant of the Mining Lease, the permit holder is required to give the Republic of the Sudan an un-dilutable free-carried 20% interest in the company holding the title to the permit.

Exploitation Permits are treated as real property rights with complete right of mortgage and liens. Both exploration and mining permits are transferable rights.

4.2.3 Ownership

The EPL for Block 14 was originally granted to Meyas Nub under a Concession Agreement dated 19 May 2010.

A purchase agreement dated 1 March 2012 (the "Purchase Agreement"), granted SMCL, a wholly owned subsidiary of Orca, the right to acquire a 70% interest in MSMCL, a Sudanese joint venture company incorporated to hold the Block 14 EPL.

Under the terms of the Purchase Agreement, SMCL have paid Meyas Nub US$9.5M in three instalments, in exchange for an increasing ownership interest. Payments have been made, as shown in Table 4.1, and Meyas Nub now retains the remaining 30% interest in MSMCL.

Table 4.1 Meyas Nub Purchase Agreement Schedule

Date	Payment	Orca Ownership Interest
01/03/2012	US$3.5M	35.0%
01/09/2013	US$3.0M	52.5%
01/09/2014	US$3.0M	70.0%

Under the Purchase Agreement, Orca agreed to fund all exploration, development, and construction costs to commercial production and to fund all costs associated with maintaining the properties in good standing with respect to the mining code.

Under the Concession Agreement, the MoM has a right to a 20% free-carried interest in any mining operation developed in Block 14. Under the purchase agreement between SMCL and Meyas Nub, the MoM's 20% interest will come solely from Meyas Nub's ownership interest in MSMCL.

4.2.4 Applicable Fiscal Elements

The holders of mineral titles are subject to the provisions of the Concession Agreements which set out the application of mining royalties, taxes, and fees for authorisations issued pursuant to the Mining Act. The key provisions of the terms of the concession agreement for Block 14 are shown in Table 4.2.

Table 4.2 Fiscal Elements

Tenement Name	Block 14
Current Area	2,170 km^2
Concession Type	Gold and associated minerals
Annual Surface Rental (US$ /km^2)	10
Annual training Fund Contribution (EUR)	36,000
Initial Exploration Expiry Date	19/05/2015
First Extension Period Expiry Date	19/05/2017
Second Extension Period	1 year
Royalty	7%
Corporate Tax	15%
Government Free Carry (%)	20%
Partner Free Carry (%)	10%
Mining Lease Period	30 years
Mining Lease Extension Period	10 years

4.3 Mineral Tenure of the Block 14 Project

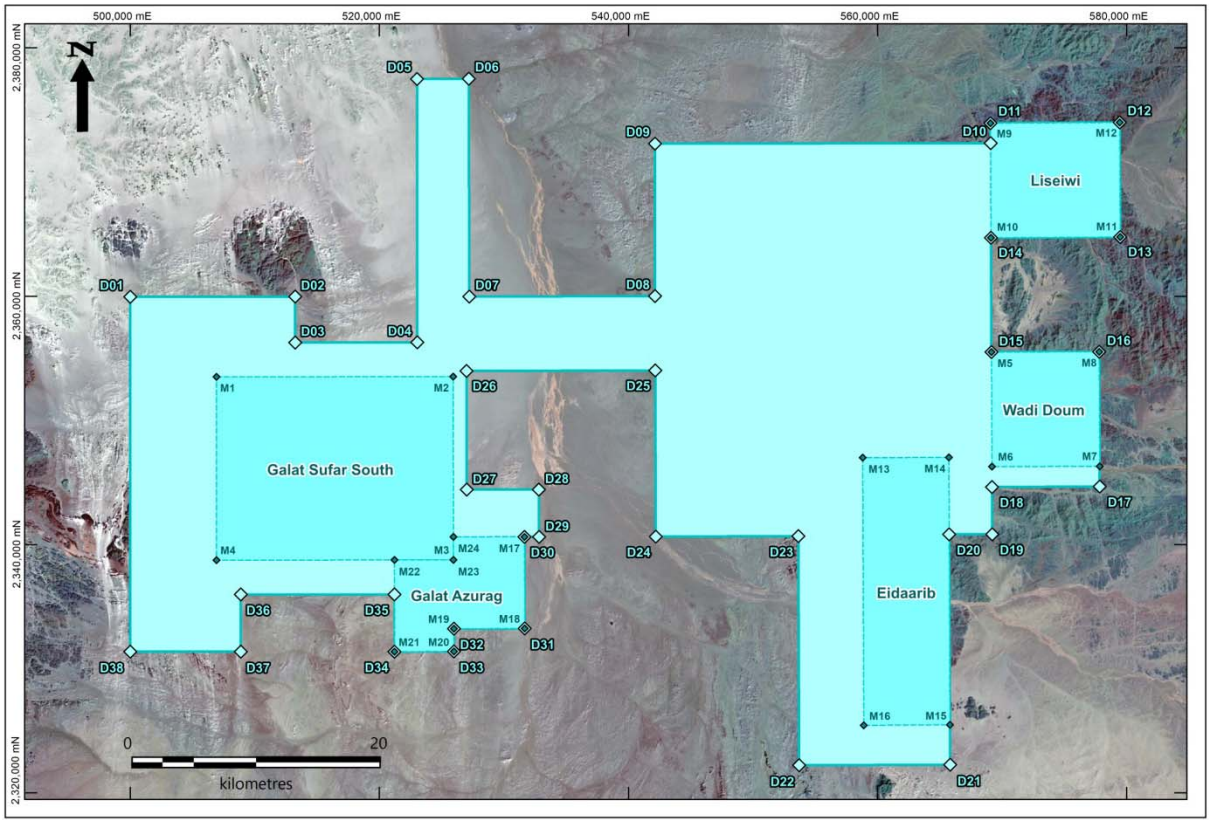
The Exclusive Prospecting Licence is granted under terms of the concession agreement for the exploration and exploitation of gold and associated metals and minerals in Lower Gabgaba Area ("Block 14"). The initial exploration period was for four years, however during 2014, the joint venture company MSMCL, was granted an additional year extension to the Initial Exploration Period. Details of key dates and terms are detailed in Table 4.3.

Table 4.3 Summary of Concession Agreement Obligations and Key Dates

Start Date	Expiry Date	Area (km^2)	Terms
19 May 2010	19 May 2014	7,046	Initial exploration period.
19 May 2014	19 May 2015	7,046	Additional year compensation period granted, extending the expiry date of the initial exploration period.
19 May 2015	19 May 2017	3,747	Formal notification of Mineral Discovery Areas, relinquishment of 50% of the Initial Exploration area.
19 May 2017	19 May 2018	2,170	Relinquishment of 50% of remaining area (excluding mineral discovery area).

On 27 February 2015, MSMCL submitted a declaration of Mineral Discovery Areas (Figure 4.2) and requested the first two year extension period in line with the concession agreement. On 19 February 2017, MSMCL formally requested the second extension period (May 2017 to May 2018) and relinquished 50% of the permit area (less than declared Mineral Discovery Areas). The permit extension has been granted, and the licence is currently valid until 19 May 2018.

Figure 4.2 Map Showing Block 14 EPL



Source: Orca Gold Inc.

4.4 Environmental responsibilities

To the extent known, the Project is not subject to any environmental liabilities. Artisanal mining within the project area is illegal and, with respect to ground disturbances caused by artisanal operations, the licence owners have no liabilities.

All permits and permissions to conduct mineral exploration have been granted by central government under the terms of the Concession Agreement.

4.5 Other Factors and Risks

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

Under the terms of the Concession Agreement, the company has the right to access all areas for the purpose of mineral exploration. The area is uninhabited and there are no areas that are held by individuals. There are a large number of illegal artisanal miners, who do not have the right to mine; however, the company works alongside the miners at this stage of exploration in a friendly and co-operative manner.

Orca has secured all necessary permits to conduct the planned exploration programmes and to continue economic and engineering studies on the project.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE PHYSIOGRAPHY

5.1 Accessibility

A number of international airlines with routes to Europe as well as other African centres are scheduled daily from Khartoum. The airport at Port Sudan also has international flight handling capability, although services are limited to neighbouring countries. Air travel to other destinations in the country is limited.

The country is served by a deep water port at Port Sudan through which the bulk of the countries imports and exports pass.

The Project is accessible by an established network of roads and desert tracks, with the permanent camp located 180 km north of the town of Abu Hamad. A major paved route connects Port Sudan, Khartoum, and Abu Hamad from which point a well-used desert road capable of handling moderate to large loads connects to the project area (Figure 4.1 and Figure 5.1). In general, vehicular access to the Project is good and is not affected by seasonal variations.

Figure 5.1 Transporting 20 t Reverse Circulation Drill Rig

Source: Orca Gold Inc.

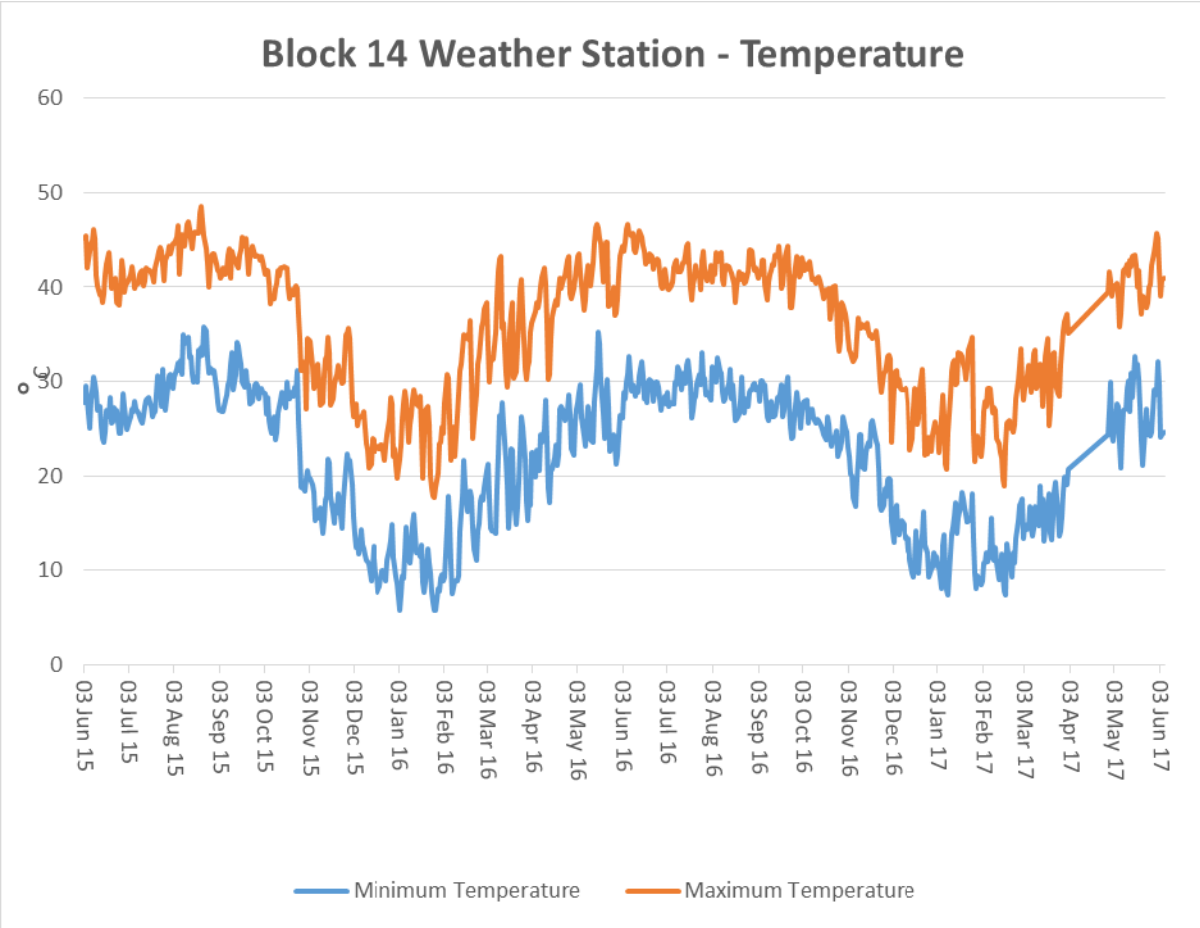
5.2 Climate

Since June 2015 a weather station has been recording data in Block 14 (Figure 5.2). The climate is arid with a hot season from June to September, during which the maximum temperatures range

from 45° - 49°C and minimum of 23° - 35 °C. The coolest period covers the months of January and February, with daytime high temperatures of 28°C and cool nights ~5°C. During the hot weather, the project area is subject to strong winds, predominantly from the north (Figure 5.3).

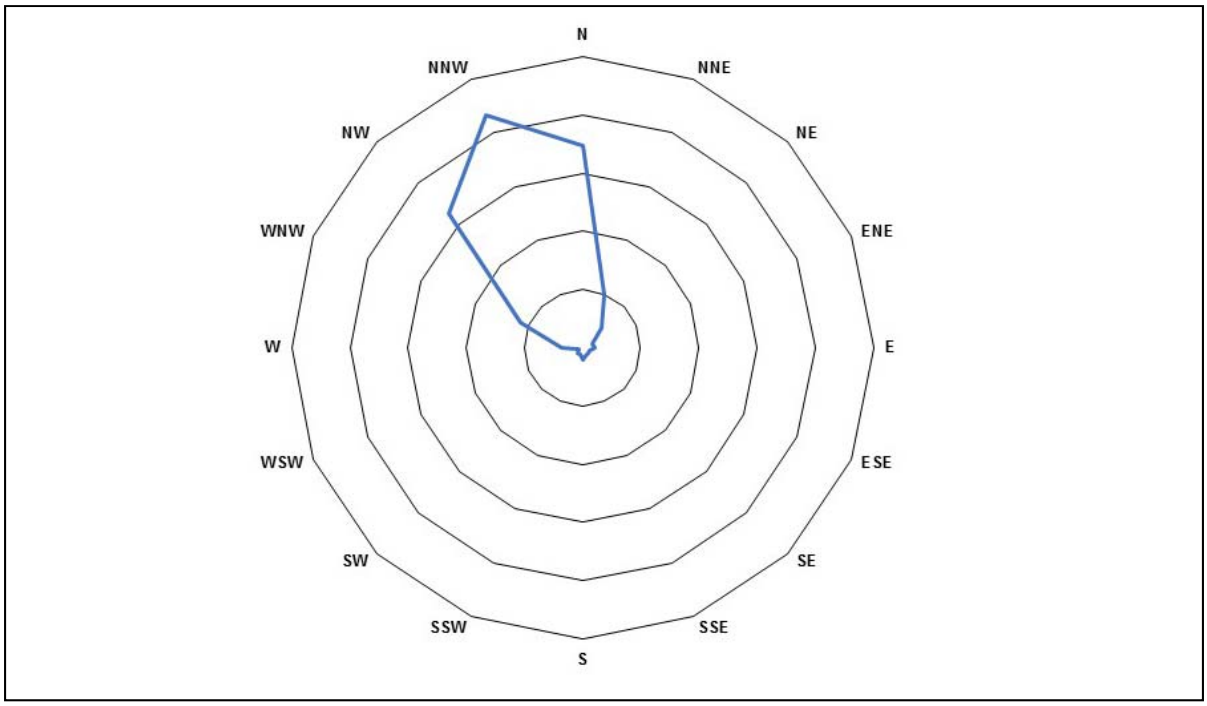
The northern desert of the Republic of the Sudan has infrequent precipitation and regional records indicate average annual precipitation to be nil. However, infrequent rainstorms do occur with records indicating events of up to 6.5 mm rainfall in a single event. The weather station in Block 14 has recorded five events of rain >1 mm and one event producing 17.5 mm since June 2015 (Figure 5.4).

Figure 5.2 Site Temperature Records from Block 14



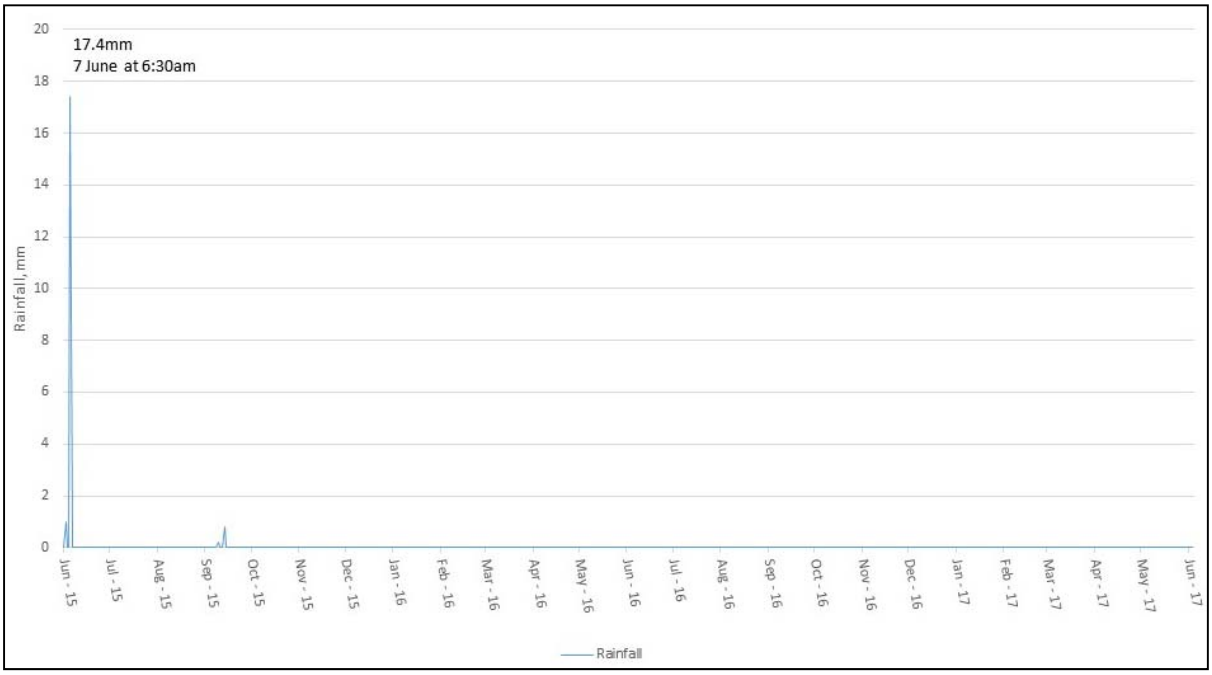
Source: Orca Gold Inc.

Figure 5.3 Wind Direction (Origin) Recorded at Block 14



Source: Orca Gold Inc.

Figure 5.4 Rainfall Recorded at Block 14



Source: Orca Gold Inc.

5.3 Local Resources

The local area of the Project is uninhabited with no significant population centres outside of Abu Hamad. People from the Ababda tribe have settled in small towns and villages close to the Nile and, wherever possible, Orca employs some members of this community as staff or as casual labourers.

There are a large number of artisanal miners working in the region whom Orca employs on a casual basis from time to time.

To the extent relevant to the mineral project, the infrastructure requirements for development of the project are minimal. However, operational requirements can be satisfied using regionally available materials and services, mainly from Abu Hamad, Atbara (where the company operates a small logistics office) and Khartoum (administration office).

5.4 Infrastructure

The Company operates a 100 person exploration camp in Block 14, comprising converted sea containers and brick built buildings that house exploration and support staff (Figure 5.5). Power is self-generated from fuel delivered as needed in 25,000 L tankers. Fresh water is supplied by tanker from the Abu Hamad.

Sudan has an established paved road network which is generally of good quality; however, some small bridges in more remote area were washed away following flooding in 2013. It is possible to bypass these locations using dry channels or wadis.

Access to the permanent camp in Wadi Gabgaba, Block 14 is via desert tracks from Abu Hamad that are used by the large artisanal mining community (Figure 5.1). Two fuel stations with telephone communications, known locally as cafeterias are located along the route.

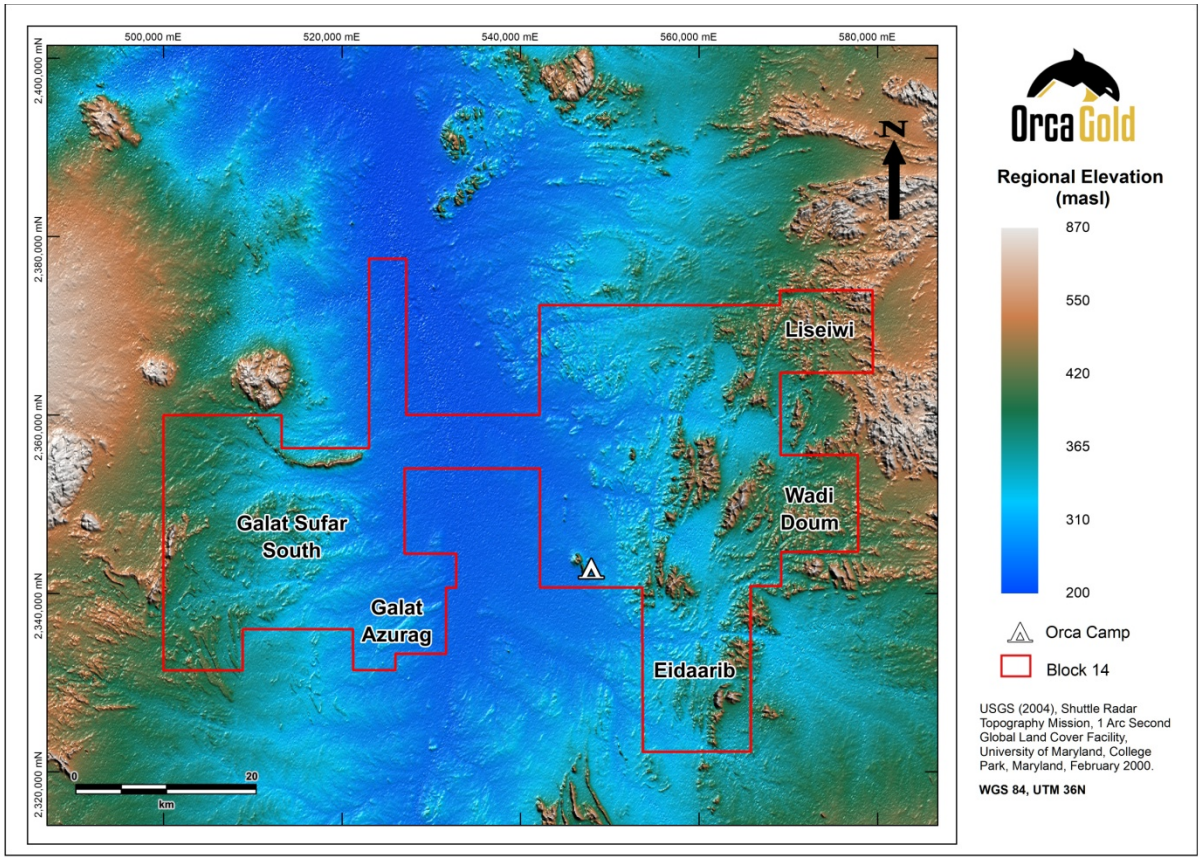
Figure 5.5 MSMCL Exploration Camp

Source: Orca Gold Inc.

5.5 Physiography

Elevation varies between 198 and 810 masl with the bulk of the higher ground in the east of the project area (Figure 5.6). Wadi Gabgaba, which runs north-south through the centre of the project area, is the main drainage system in the area. The western part of Block 14 contains more subdued topography.

Figure 5.6 Physiography of Block 14 Tenement



Source: Orca Gold Inc.

Figure 5.7 Photograph of Wadi Gabgaba

Source: Orca Gold Inc.

The landscape is characterised by rocky hills separated by wide, flat sand filled drainage channels known as wadis (Figure 5.5 and Figure 5.7).

Vegetation is restricted to the larger wadis where water is available from the crystalline basement and consists of Doum palms and sparse thorny shrubs. Much of the Project area is un-vegetated with only occasional desert grasses and stunted trees.

6. HISTORY

The Red Sea Hills and Nubian Desert of the Sudan have seen gold mining since 3,000 BC, and the Block14 EPL contains numerous documented sites with dilapidated stone huts and historic mining infrastructure. In the early 19th century colonial gold mining occurred across the Red Sea Hills. There was some renewed interest in these mining operations in the 1970's and 80's that culminated in a study carried out by Robertson's Research International (RRI) on behalf of a British company, Minex in 1981. The work revisited many of the colonial gold mines and working in conjunction with GRAS, compiled all of the historical information focussing on high grade remnants and low grade tailings.

The first grass roots modern exploration programme ran from 1977 to 1981, led by a French-Sudanese team who conducted several reconnaissance programmes for various metals (W, Pb, Zn, Cu, Ag, Au, Cr, etc.) over five large areas throughout the Sudan under the framework of a co-operation agreement. One of these programmes identified the gossans at Ariab and in 1981 a joint venture was signed between the Bureau de Research Geologique et Minieres (BRGM) and the Sudanese government. In 1983 the discovery of gold in silica-kaolinite±barite gossans at Ariab shifted all of the JV's focus towards gold and the Ariab oxide mine production started in 1991.

From 1983 to 1984, Geosurvey International conducted an Egypt–Sudan integration study using satellite imagery and structural interpretation. Only the summary interpretative maps are currently available.

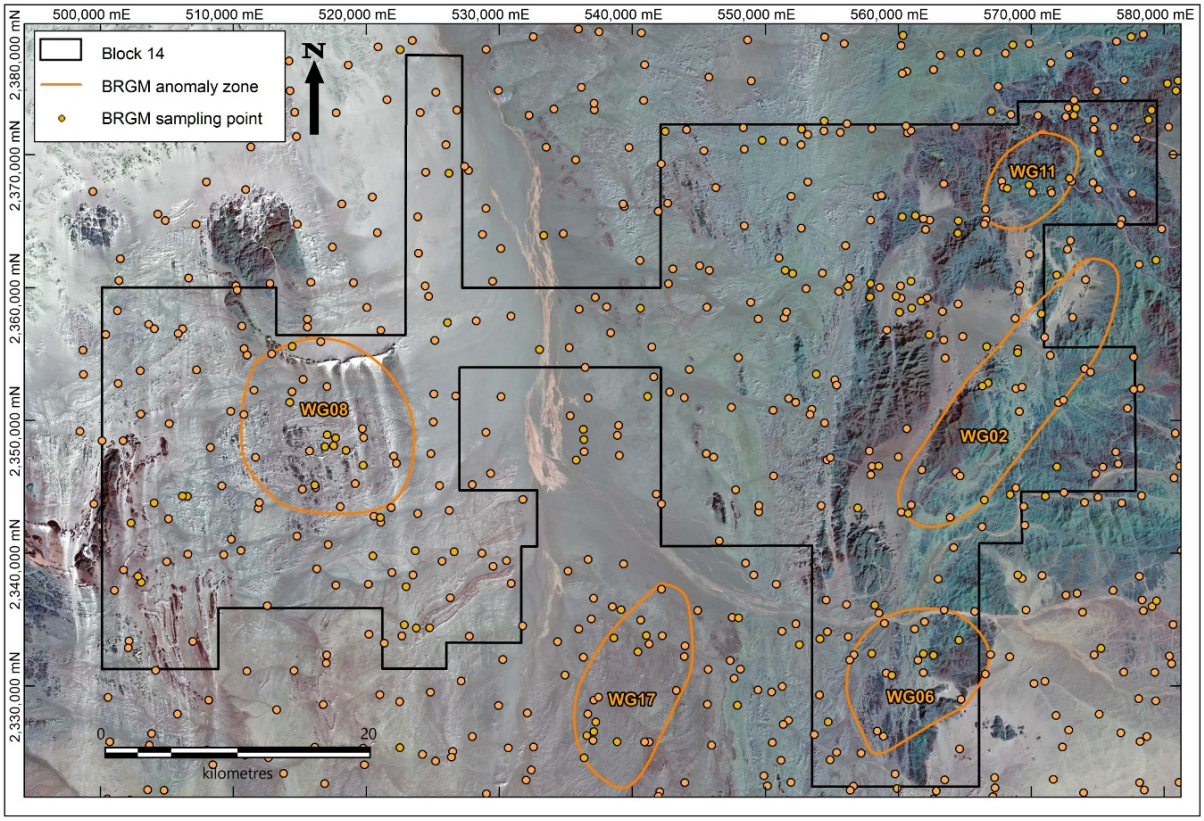
In January 1996, an agreement was signed between the Government of the Republic of Sudan and Cominor to explore for gold in the Nubian Desert, and in 1997 the BRGM affiliate La Source acquired 90% of Cominor's 90% in the Nubian Desert Gold Project. In 1996, the Nubian Desert Gold Study initiated grass roots exploration in northern Sudan.

In 1998, RRI completed a 1:1,000,000 scale geological map of the Red Sea Hills. The map drew from the prospecting work carried out by RRI for Minex with field support and collaboration from GRAS. Large areas of the mapping coverage were in extremely remote areas and had not been visited in the modern era.

In 1998, a 58,300 km^2 EPL was granted to the La Source JV and gold anomalies derived from the regional drainage results were ranked for detailed follow up (Figure 6.1). Several of these anomalies lie within the Block 14 Project area. La Source focussed on the principal anomaly located 80 km south of Block 14, and follow up on targets in the Block 14 project area was limited to one area, where soil sampling covered a 1 km long shear corridor adjacent to an uplifted diorite block and of 20 trenches dug across the largest of the defined anomalies only one intersect returned 1.8 m at 2.26 g/t gold. Ten RC drillholes for 720 m returned only one significant intersect of 4 m at 14.58 g/t gold on a quartz vein estimated to be only 200 m long.

In 2004 the Bundesanstalt fur Geowissenschaften un Rohstoffe (BGR) completed a 1:1,000,000 scale geological map of the Sudan, drawing most of the geological elements for northern Sudan from the RRI 1:1,000,000 scale geological map.

Figure 6.1 BRGM Nubian Gold Project Drainage Anomalies and Sampling Points



Source: Orca Gold.

Gold exploration in the Sudan has attracted significant international interest over recent times, encouraged by the rapid rise in artisanal gold mining in the Nubian Desert and Red Sea Hills over the last four years. Artisanal mining is concentrated in Block 14 and in the Block 15 EPL adjacent to the south (held by Managem). The Government estimates that more than 1,000,000 people are now involved in artisanal mining in the Sudan, and reported gold production of 93 t in 2016 making the country the third largest gold producer in Africa.

Both colluvial and hard rock mining has intensified with the addition of metal detectors and mechanised mining and nuggets up to 8 kg have been discovered in the recent past. Companies such as Managem (Moroccan, Block 15), Tahe Mining (Turkish, Block 17), and Rida Mining (Sudanese) have begun operation of pilot plant scale (700 - 1,500 tpd) mines. In addition, over the last three years, a number of heap leach and CIL processing plants have been constructed to process tailings material from the artisanal mining process.

Orca commenced exploration activities in January 2012, following agreement on terms for the JV of Block 14 with Meyas Nub in 2011, Orca began due diligence sampling on Meyas Nub's principal targets at Tanasheib and Mussieye. In July 2012 the Block 14 EPL was transferred to the Meyas Sand Minerals Company Ltd (MSMCL) and in August 2012, channel sample results from a new prospect at GSS confirmed the presence of gold in wide intersections of a major shear zone at the prospect. In 2014, a second deposit was discovered at WD through systematic follow up and evaluation of artisanal mining sites.

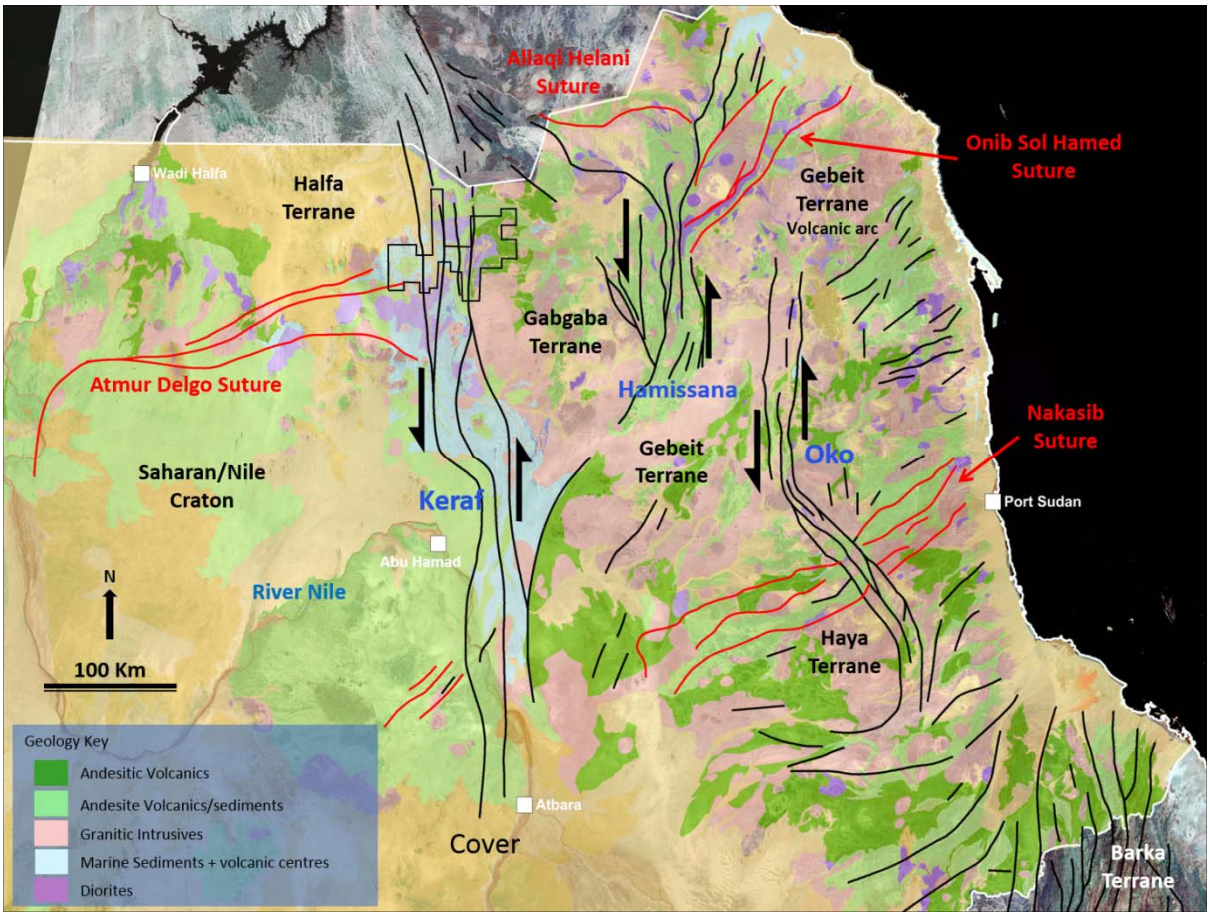
7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Red Sea Hills of the Sudan hosts in excess of 250,000 km^2 of the Arabian-Nubian Shield (ANS), an assemblage of green schist metamorphosed, dominantly arc-accretionary belts, of Neoproterozoic age that was accreted to the African craton during the East African Orogen.

The GSS and WD deposits are located within the Block 14 EPL at the boundary of the Halfa and Gabgaba terranes on the eastern edge of the African craton (Figure 7.1). The geological framework of Block 14 is dominated by the Keraf Shear Zone (KSZ) which formed during the Neoproterozoic consolidation of Gondwana. It is ~500 km long, 50 km wide and represents a north trending suture dominated by sinistral transpressive shear zones (Abdelsalam et al. 1998). The KSZ is considered to represent the collision between East and West Gondwana after the consumption of the 'Mozambique Ocean', with remnants preserved as intra-oceanic island arc / back arc / ophiolite assemblages that define the ANS (Burke and Sengor 1986). Within Block 14, the Wadi Gabgaba represents the axis of the KSZ separating Western Gabgaba from Eastern Gabgaba (Figure 7.1).

Figure 7.1 Regional Schematic Geology

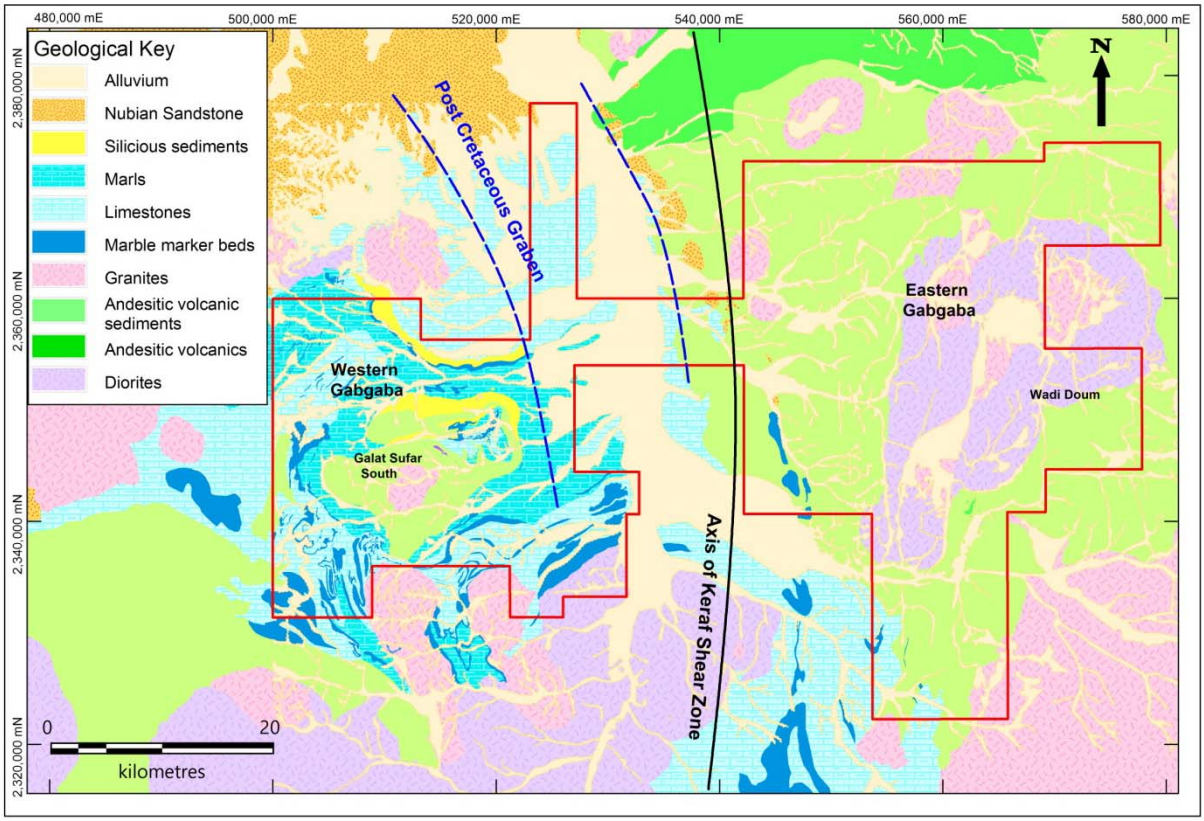


Source: Orca Gold Inc.

Western Gabgaba (Figure 7.2) is part of the Halfa terrane and is characterised by a package of shallow, calcareous, marine sediments containing iron formations and numerous discrete volcanic centres, and is classified as an immature continental tectonic setting (Galley et al. 2007). Abdelsalam et al. 1998 and Johnson et al. 2011 describe Western Gabgaba in Block 14 as a possible aulacogenic oceanic re-entrant as evidenced by the Atmur-Delgo suture.

One such area of volcanics is termed the West Gabgaba 'donut', Figure 7.2, which is represented by an annular feature some 15 km across that contains mafic volcanics on its periphery and is cored by a package of intermediate–subalkalic volcanics and calcareous meta-sediments that host the GSS deposit. The donut can be termed a culmination / anticlinorium that is best defined by Orca's airborne magnetic survey (Figure 7.2).

Figure 7.2 Regional Geology of Block 14 EPL



Source: Orca Gold Inc.

Eastern Gabgaba (Figure 7.2) is dominated by thick sequences of andesitic volcanics with sub-ordinate felsic volcanics and some metasediments of the Gabgaba terrane. The mafic dominated bi-modal volcanic arc is classified according to Galley et al. 2007 as an immature oceanic tectonic setting. The arc sequence is intruded by multiple phases of collisional syn-tectonic diorites and post collisional sub-alkalic intrusives The subalkalic intrusives are seen across both the East and West Gabgaba areas within Block 14 and may represent the Bayudan phase of igneous activity (<500 Ma).

Regional deformation of the ANS within Block 14 is poorly understood and little if any regional mapping has been done throughout the northern limits of Sudan. In Western Gabgaba, the thick sequences of metasediments and the contrast between volcanics and sediments allows more detailed study of the regional deformation. Prior to assembly of eastern and western Gondwana at least two phases of isoclinal folding are present and a continuum of regional steep folding and segmentation is evidenced by the juxtaposition of coaxial and non-coaxial interference fold patterns across shear zones. In Eastern Gabgaba the regional structural setting is less clear due to the dominance of andesitic volcanics that do not readily display folding and penetrative cleavage.

The Keraf Shear zone dominates the geological framework of Block 14 where it both controls and is controlled by diorite intrusives and discrete alkalic intrusive bodies. Post collisional syenites and granitoids are present in both Eastern and Western Gabgaba and these late plutonic bodies have truncated margins along regional scale shear zones representing the final phase of suture. The

Cenozoic rift event in Block 14 is evidenced by NNW trending extensional faults that form the margins of Wadi Gabgaba (Figure 7.2).

7.2 Property Geology and Mineralization

7.2.1 Geology

The geological framework of Block 14 is dominated by the Keraf Shear Zone (KSZ) which underlies the main Wadi Gabgaba and separates two distinct geological terranes, namely Eastern Gabgaba and Western Gabgaba (Figure 7.2). Wadi Gabgaba masks the KSZ but it is apparent that the western edge of Eastern Gabgaba is represented by a complex tectonic interleaving of both terranes.

Eastern Gabgaba is dominated by thick sequences of andesitic volcanics with sub-ordinate felsic volcanics and some metasediments. The volcanic arc is intruded by multiple phases of syn-tectonic diorites and post collisional sub-alkalic intrusives that may represent the Bayudan phase of igneous activity. Western Gabgaba is characterised by a package of marine sediments containing numerous volcanic centres. The Western Gabgaba 'donut' represents an annular feature some 15 km across that contains mafic volcanics on its periphery, and is cored by an intermediate package of volcanics and calcareous metasediments.

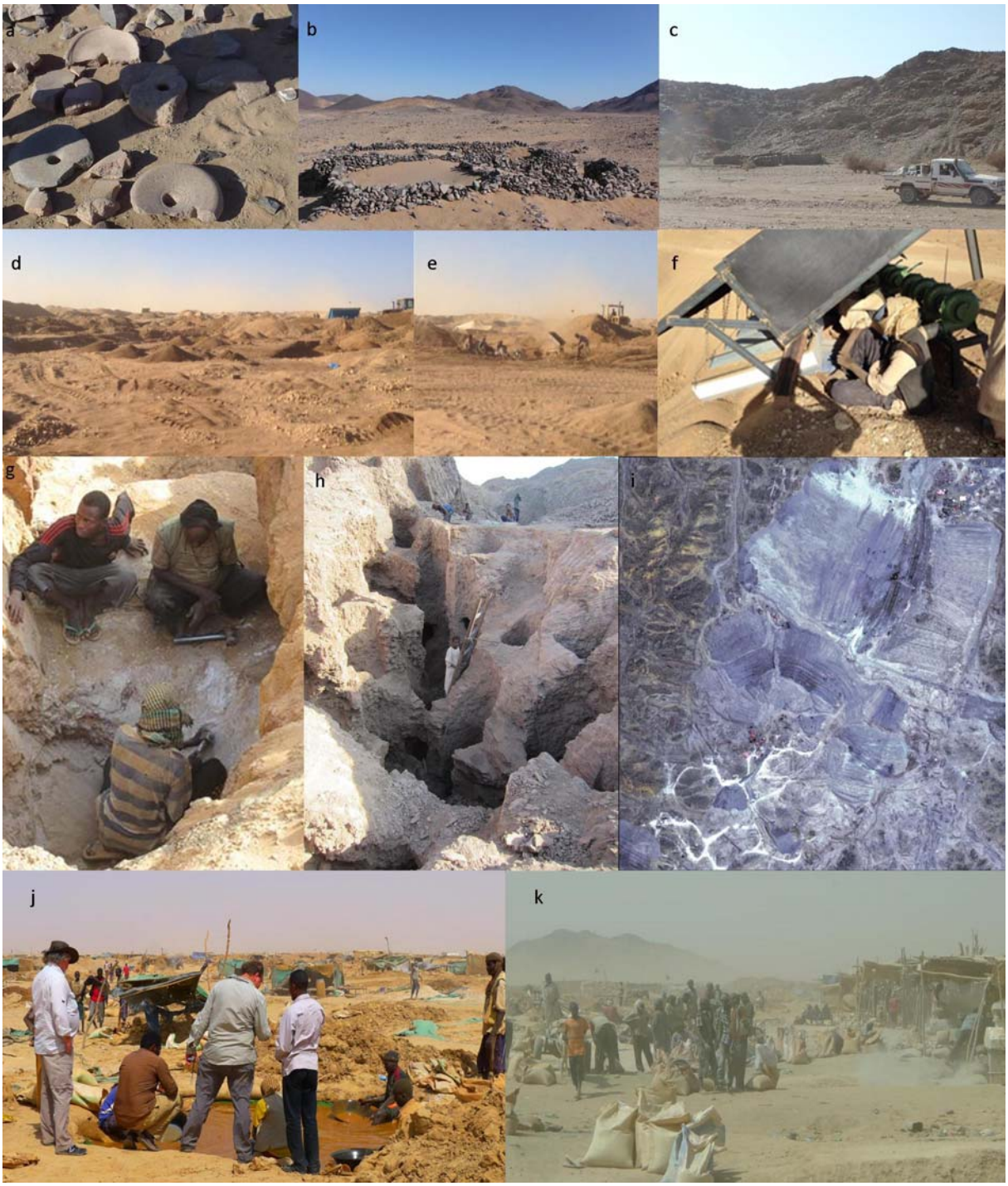
7.2.2 Mineralization

Gold mineralization is being exploited by artisanal miners throughout Block 14. The geological setting is varied but four main types or artisanal mining are apparent:

1. Colluvial mining using metal detectors that have a reliable penetration of 20 - 30 cm. Tractors, front end loaders, and excavators are used to clear each swath of material after it has been screened.
2. Colluvial mining using dry screening methods, often re-working gravel previously worked using metal detectors.
3. Traditional artisanal mining on thin quartz veins which can be laterally extensive and are in places mined to >30 m below surface.
4. Mechanised, open pit, hard rock mining to depths of up to 30 m. These are often associated with vein swarms within distinct shear zones. Gold bearing rocks are selected using metal detectors both in situ and in broken material.

In a number of locations gold has also been identified in broad shear zones with sub-ordinate quartz veining for example at GSS and the surrounding area.

Figure 7.3 Artisanal Workings



a: Pharaonic Grinding stones at SE Gabgaba, b: Pharaonic settlement at SE Gabgaba, c: Colonial settlement at Onib Mine. d: Large scale colluvial mining, GSS, e: Front end loader feeding air screens, GSS, f: Artisanal miners adjusting dry gravity separators, GSS. g: small scale hand mining of discrete quartz carbonate lode vein, NW Gabgaba, h: large scale hand mining of shear zone hosted gold, WD, i: Mechanised artisanal mining at Big Pits, Western Gabgaba (worldview image of open pit trending NNE, showing spoil heaps where coarse gold is identified and sorted using metal detectors, the southern half of the image shows extensive mechanised colluvial mining) j: Hard rock mining product is hand cobbed and sent to small scale grinding sites close to the river Nile. K: 2 –stage ground material is panned. Source: Orca Gold.

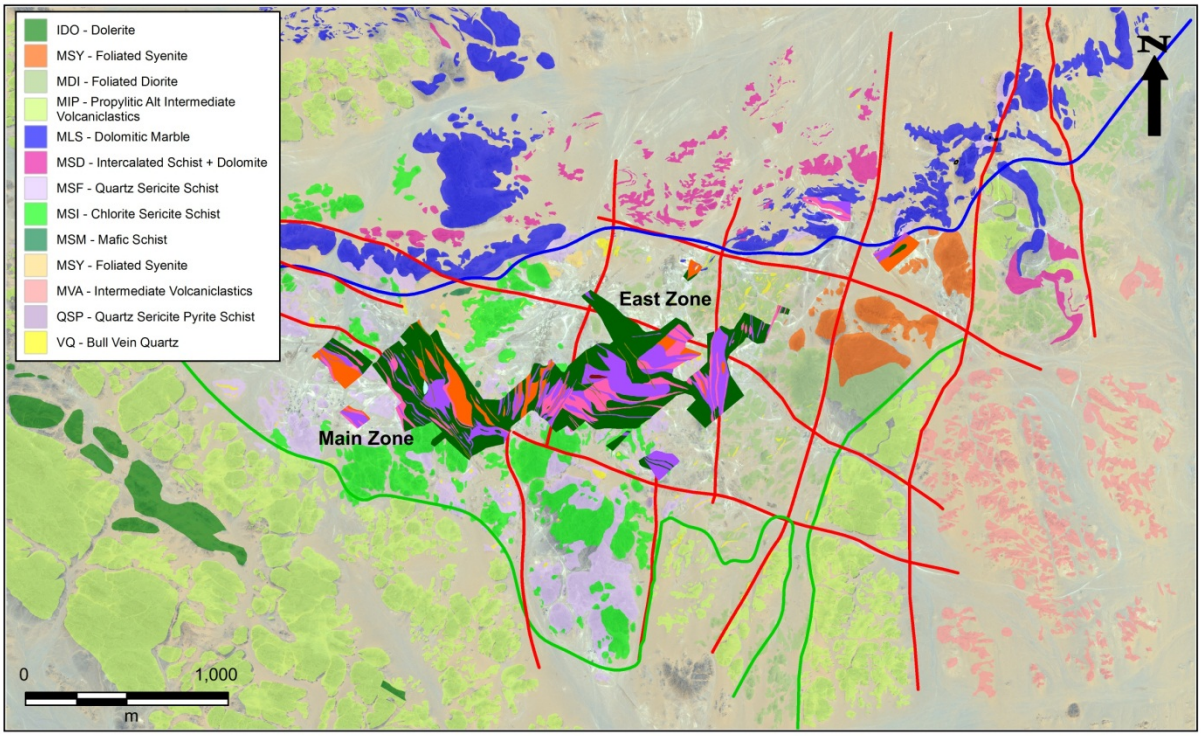
7.2.3 The Galat Sufar South (GSS) Prospect

Geology

The deposit is located in the central portion of the Western Gabgaba anticlinorium (donut) on the southern flanks of a fold interference culmination with an axial surface trace trending east-north-east.

The north east apices of this doubly plunging antiform hosts the GSS North prospect and other prospects are located on its northern flanks. A silicified dolomite is a marker to the core of the structure which contains an interleaved package of lower greenschist facies metamorphosed carbonates, marls and volcanic.

Figure 7.4 GSS Local Geology



Source: Orca Gold Inc.

The GSS deposit is located within a package of intermediate volcanics, diorites, and syenites that have a penetrative schistosity that both controls and is cut by well-developed shearing, alteration and mineralization. Scale mapping (1:25,000) has identified a chlorite epidote bearing sequence of monotonous andesitic volcanics distal to the sericite dominant core of GSS. Directly south of GSS the presence of chlorite–epidote is controlled by through going faults and, close to the mineralization, these fault zones contain quartz–epidote (epidosite).

The dominant trends of shearing at GSS are 110° (sub-parallel to the S1 cleavage) and 010°, they dominate alteration and vein development, are steep and display well developed C – S fabrics. The 010° set of shears are generally better developed and in Main Zone and are mylonitic. An intersection lineation of the two main shears is defined by material shoots that plunge steeply to the NW. The N-S structural break between the eastern and western zones of GSS is poorly exposed and not completely understood.

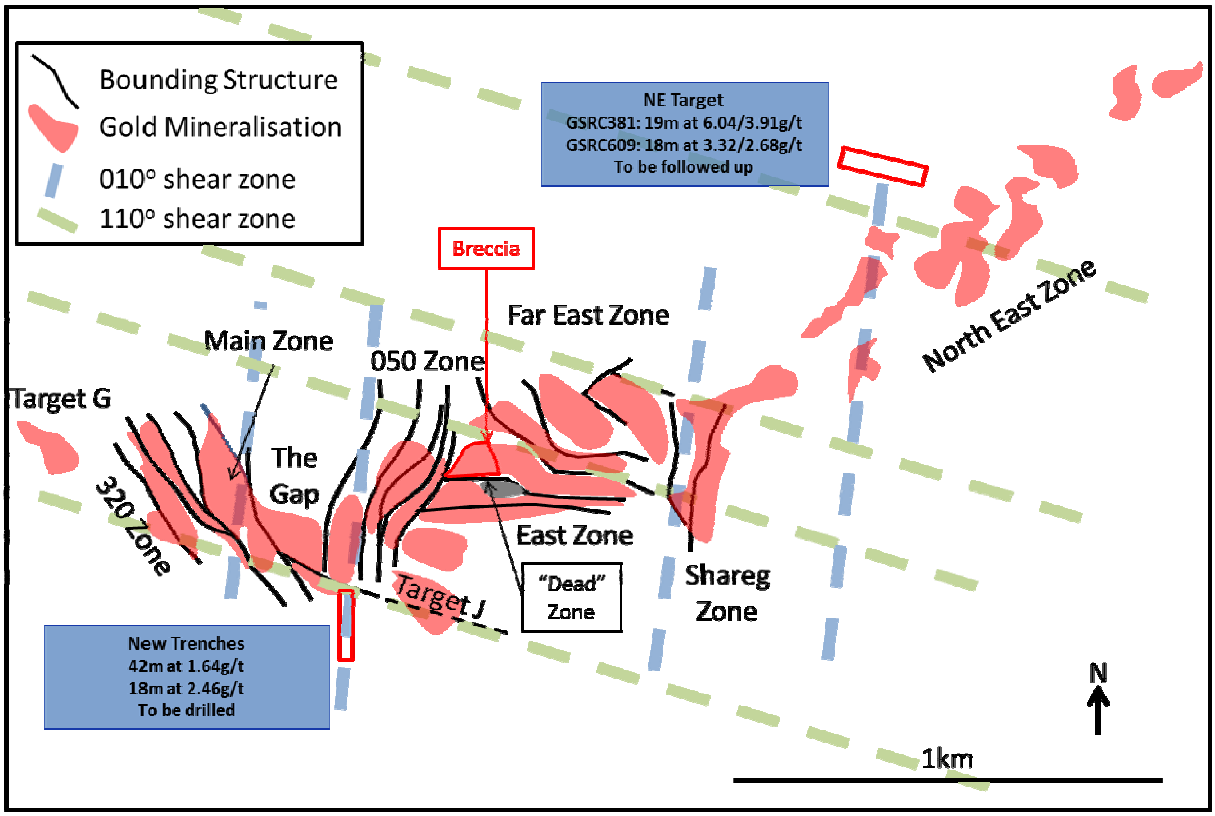
Mineralization

Six domains of mineralization have been differentiated and resource modelled:

- The 320 Zone: is present as discontinuous ribbons and plunging shoots located within a shear zone oriented 140° that is parallel with the local schistosity. The 320 Zone contains high grade shoots (+10 g/t) that plunge steeply to the NW in the same direction as the intersection lineation of 010° and 110° shears.

- The Main Zone: is a wide laminar zone oriented 020° that links in to the 320 Zone in the south and has a strike length of 150 m. The Main Zone represents a set of 010° trending shears that link two through-going 110° trending shears across a rigid body of syenite / k-feldspar altered diorite in the footwall. Shear fabrics in Main Zone are extreme and mylonites have been identified. The Zone is up to 90m true width and is host to some of the best intercepts in the Project. Grade is strongest at the southern end, closest to the intersection of the 320 Zone.
- The 050 Zone is a small domain that links the East Zone with a north–south trending set of vein mineralization within a covered area termed The Gap. The 050 Zone is considered to be a compressional duplex that translates movement from the East Zone into a large through-going 010° trending shear within The Gap. Recent drilling has shown that in the south, the 050 Zone links into a 110° tending shear known as Target J.
- The East Zone: is hosted within a 100° oriented corridor within which several small dismembered intrusive stocks, often brecciated, have been mapped and logged. Shear fabrics wrap around the intrusive bodies although the kinematics of this fault system are not fully understood. True width again exceeds 90 m in the central East Zone with subsidiary, parallel mineralization being present to the north and south with a similar, although not fully defined, trend. Shear fabrics in oriented core are dominated by a 110° orientation dipping steeply to the west. Sub-ordinate 010° trending shear is also present although these north going areas of mineralization are often obliterated by silicification. A third sub-ordinate fabric oriented 030° is related to steep, irregular, coarse grained, gold bearing cataclasite that is interpreted to be a late stage deformation. A barren plug of alteration in East Zone is adjacent to high grade tectonic breccias and may be associated with the quartz blow seen at surface.
- The Far East Zone has a similar trend to the East Zone and the mineralization is hosted within a sheared sericitised microdiorite. Steep to vertical mineralized trends are open to the north and west, where discontinuous grade extends under cover.
- The Shareg Zone is a north / south trending steep trend of mineralization hosted within a sericite / carbonate altered diorite / microdiorite.

Figure 7.5 GSS Deposit Model



Source: Orca Gold Inc.

Alteration

Alteration is pervasive and deposit-scale units are defined by their alteration assemblages which are variably zoned outward from the gold mineralization (Table 7.1). The first phase of mineralization is a milky white, fused quartz blow that is randomly oriented and cross cut by shear zones and associated veining. On the eastern margin of GSS the k-feldspar alteration clearly overprints a Foliated Diorite (MDI) and Porphyritic Diorite (IDI) forming the Potassium Altered Diorite (KDI). The KDI forms a rigid body that is sheared on its contacts and overprinted by the Quartz-Sericite Schist (QSS) and Quartz-Sericite-Pyrite schist.

Table 7.1 Progressive Alteration Assemblage at GSS

Alteration	Code	Description	Mineralization
	QSP	Quartz sericite pyrite schist	Strong
↑	QSS	Quartz sericite schist	Weak
	MD	Foliated diorite	Weak, variable
	KD	Potassium altered diorite	Weak, variable
	BRD	Black red diorite	Weak, variable
	IDI	Porphyritic diorite	None

Where the k-feldspar alteration is intense a texture destructive Black Red Diorite (BRD) with incipient hornfelsing is preserved within highly sheared QSP altered high grade mineralization, but generally the k-feldspar alteration contains weak, variable mineralization. Altered tectonic breccias are common in the East Zone where they are often pervasively overprinted by potassium feldspar alteration and subsequent shear foliation.

Vein quartz is present within all of the mineralized intersections but is rarely mapped at surface due to its exploitation by artisanal miners and is best represented by the position of artisanal workings.

The quartz veins show multiple generations of development and are themselves mylonitised and brecciated in Main Zone. Quartz blow is present throughout the region and outside of GSS carries no gold. However, at GSS it is brecciated, has sheared contacts, and can contain significant gold.

Gold at GSS is associated with primarily with intense sericite–carbonate alteration, moderate silicification, pyrite content and quartz veining from 1 mm scale to a maximum of 1.5 m.

7.2.4 Wadi Doum (WD)

The main, high grade mineralization at WD outcrops at the base of the hill and is hosted by a strongly sulphidic volcaniclastic unit, which is in contact with a distinct rhyolite unit to the immediate east. The volcanoclastic unit dips at an angle of 20° to the south west. This rhyolite is bounded to the east by a dacitic unit intruded by syn-tectonic syenite / potassium altered diorite body which forms the summit of the main hill.

These lithologies are cut by thin (<0.75 cm), late, un-mineralized felsic and mafic dykes. In contrast to the volcaniclastics the rocks on the hill dip 75° to the east. Mineralization on the hill is associated with stringer zones within the syenite and in places smaller shears.

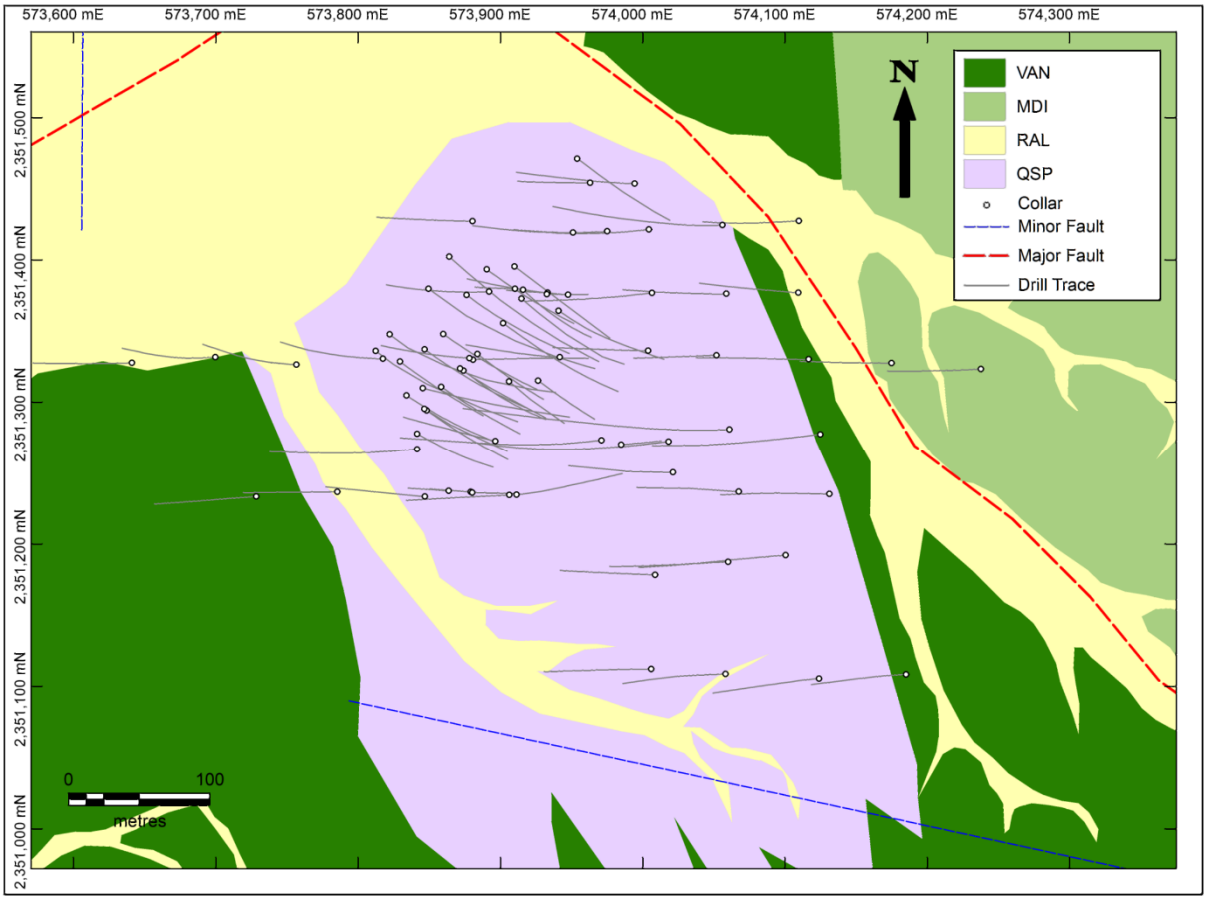
The high grade mineralization is hosted within the volcanoclastic units which are confined by late felsic and mafic dykes. The mineralization is divided into a western volcaniclastic unit characterized by a dark colour caused by very fine grained sulphides (>10 - 15%), which contains some of the best intercepts and a central unit of paler, sulphide rich felsic volcaniclastics which contain deformed sulphide veinlets and a lower grade footwall unit of largely un-deformed felsic volcaniclastics.

The dominant sulphide is pyrite (85% in Qemscan analysis) with the remainder comprising a mix of sphalerite, galena, chalcopyrite and Freibergite.

Alteration is confined to sericitisation within the felsic volcanics and a wider halo of carbonate alteration. Silicification is noticeably absent or weak within the high grade part of the deposit (hence its location at the base of the hill).

The area is dominated by a strong and pervasive, north-south trending schistosity which is largely followed by the late dykes. The high grade mineralization often appears un-affected by structure whereas the mineralization hosted by the syenite on and around the summit of the hill does appear structurally controlled.

Figure 7.6 WD Local Geology



Source: Orca Gold Inc.

8. DEPOSIT TYPES

The mineralization types being targeted within the Block 14 Project are broadly categorised into three groups, namely Orogenic Gold, Volcanogenic Massive Sulphide, and Rift Associated Epithermal.

The Arabian Nubian Shield is an underexplored emerging gold province. Orogenic gold mines within the region include the world-class Sukari mine in Egypt (23.2 Moz gold, Centamin Plc 2015), the Ad Duwayhi Project in Saudi Arabia (2.4 Moz gold, Saudi Arabia Mining Co., 2015), and the Zara Mine in Eritrea (0.9 Moz gold, Chalice Gold Mines Ltd., 2012). A number of advanced exploration projects in various degrees of development are present in the northern Red Sea Hills, Sudan, with Tahe Mining in the early mining stages of its 0.5 Moz gold Abu Sari mine, and Managem in its pilot mining stage at the 3.0 Moz gold Gabgaba Project.

Evidence of orogenic gold mineralization is present throughout the project area. It is generally associated with narrow gash veins, shear type veins and quartz veinlet swarms in well foliated schistose rocks within the volcano-sedimentary domains which are intruded by stocks and sheets of diorites, syenites and granitoids. Vein densities are seen to increase in and around contacts with intrusives that are focused within structural corridors. The veins form arrays that are often over 10 km long and transgress multiple lithologies. Broader shear zone hosted orogenic gold mineralization has been identified at a number of prospects in Block 14, however the current focus is on GSS, where sheared host rocks as well as discrete quartz veins host broad zones of gold mineralization.

The Nubian volcanic sequences are also prospective for Volcanogenic Massive Sulphide (VMS) mineralization and examples of economic VMS deposits within the Arabian Nubian Shield include the Bisha Project in Eritrea (1.2 Moz gold, 3,500 Mlb copper, and 16,300 Mlb zinc, Nevsun Resources Ltd., 2014), the Jabal Sayid Mine in Saudi Arabia (1,400 Mlb copper, Barrick Gold Corp., 2014) and the Ariab Project in Sudan (5.9 Moz gold, 3,000 Mlb copper, La Mancha Resources Inc., 2014).

VMS style mineralization with well-developed base metal gossans have been identified at Tanashieb in Eastern Gabgaba, Block 14. The mineralization is associated with felsic volcanics (dacites) within a dominantly mafic package of arc related rocks.

9. EXPLORATION

Orca Gold's exploration strategy has been to identify historic and artisanal gold mining activity and carry out reconnaissance mapping and sampling, followed by systematic continuous chip and trench sampling in the search for broad zones of shear zone hosted gold in or around lode gold veins.

The systematic approach has included the analysis of satellite imagery, geological mapping, rock chip and chip-channel sampling, trenching and both reverse circulation and diamond core drilling.

Detailed exploration methodologies and procedures for the project are described in detail in a previous Technical Report "*NI43-101 Independent Technical Report, Block 14 Project, Republic of the Sudan*" dated 11 March 2014.

Table 9.1 below shows the work completed since the commencement of exploration on the Block 14 Project.

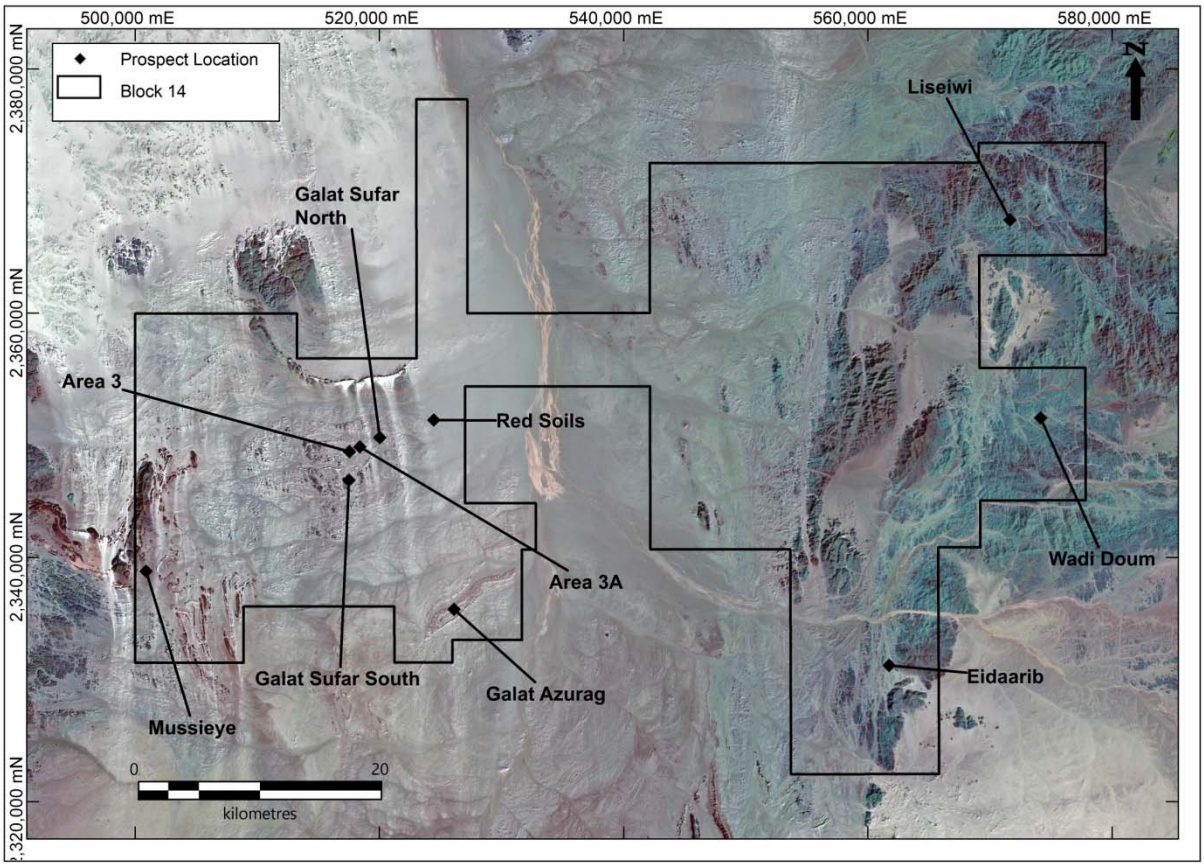
Table 9.1 Summary of Exploration Work Completed on Block 14, 2012 - 2017

Surface Sampling	
Rock Chip samples	4,753
Soil Samples	2,682
Chip Channels (m)	58,427
Trenching (m)	40,954
Airborne Geophysics	
SKYTEM Electromagnetic survey (2017) (km^2)	2,415
VTEM Electromagnetic Survey (km^2)	835
Magnetic and radiometric Survey (km^2)	3,407
Satellite Imagery Acquired	
Landsat TM/Aster satellite Imagery (km^2)	7,046
SPOT Imagery	3,250
Worldview Imagery	895
Drilling	
Reverse circulation drilling (m)	85,694
Diamond core drilling (m)	7,560
Water Borehole Drilling (m)	5,496
Ground Geophysics	
Ground Magnetics (km^2)	7.01
Ground Radiometrics (km^2)	7.01
Time domain electromagnetic (TEM) profiling (km)	261.6

10. DRILLING

Reverse Circulation (RC) and diamond core drilling in the Block 14 project area commenced in November 2012. During the period to June 2013, drilling was focussed at GSS however several other prospects were also drilled (Figure 10.1). From July to December 2013 all drilling was focussed at GSS. In 2014 several other targets were drilled including the discovery holes at WD with subsequent infill drilling. In 2015 drilling was undertaken at a number of prospects and included two diamond core holes at WD. In 2016, further resource drilling was carried out at GSS and Wadi Doum.

Figure 10.1 Drilled Prospect Areas



Source: Orca Gold Inc.

A summary of the drilling completed is shown in Table 10.1.

Table 10.1 Drilling Completed in the Block 14 Project Area

Prospect	Core Drilling		RC Drilling	
	Holes	Metres	Holes	Metres
GSS	31	6,770	494	62,285
WD	6	789	104	12,220
Galat Azurag	-	-	36	3,944
Mussieye	-	-	4	447
Area 3	-	-	6	750
Area 3A	-	-	8	1,000
GSS North	-	-	4	354
Red Soils	-	-	14	1,840
FE Ox 07	-	-	3	160
Liseiwi	-	-	18	2,055
EG2.2	-	-	9	525
Target	-	-	3	114
Total	**37**	**7,560**	**703**	**85,694**

Figure 10.2 RC Drilling at Main Zone, GSS

Source: Orca Gold Inc.

10.1 Drilling and Sampling Procedures

All drilling undertaken on the Block 14 project has been carried out by General Exploration Drilling (GED) using multipurpose KL400, KL500, and GED 850 riges (Figure 10.1).

10.1.1 Drilling and Sampling Method

RC samples are collected at 1 m intervals from the base of the RC cyclone with new plastic bags, which are clearly labelled with the hole number and metre interval. Drill chips in the bags are geologically logged and the information recorded on a paper drill log sheets by the attending geologist. The bags are then sealed. Below is a systematic procedure from the collection at the cyclone to the laboratory dispatch stage:

- Each metre sample is collected from the cyclone into a plastic sample bag measuring 100 cm x 55 cm and weighed at the rig with the weight recorded on the drill log sheet.
- The bulk sample is then passed through a three tier riffle splitter with two sub-sample ports, one to produce a ~3 kg sub-sample in a 30 x 40 cm plastic bag.
- The bulk sample is passed through riffle splitter a second time to produce a ~3 kg archive sample with the remaining sample stored in the original bag.
- When a duplicate is required, the bulk bag is passed through the riffle splitter a third time to produce a ~3 kg duplicate sample.
- Samples tags are added to each 3 kg sample from numbered ticket books, with the hole number and interval clearly written on the ticket stub for reference.
- The 100 x 55 cm plastic bags containing the bulk reject sample are then numbered and left at the drill site in ordered lines.
- The riffle splitter is cleaned thoroughly with compressed air prior to the next sample being split.
- All samples (original, archive and duplicate) are then transported to the Block 14 camp at the end of the shift, where the archive sample is stored and original and duplicates prepared for despatch to the sample preparation facility.

Diamond drill core is collected from the core barrel in up to 3 m drilling intervals (in places reduced runs were undertaken to maximise core recovery) and placed directly in purpose-built plastic core trays. All on-site core handling was supervised by a company geologist. Core quality and recovery data are collected at the drill site prior to delivery of the core to the camp. All core drilling on this program was oriented where possible using a spear.

- Drill core is transported to the camp at the end of every shift regardless of how many metres have been drilled.
- Once the geologists have finished logging the orientated drill core it is cut using a core saw and half core sampled based on intervals defined by the logging geologist, generally 1 m (minimum sample size is 0.45 m).
- Sampled PQ and HQ half core is placed into 40 x 50 cm plastic bags and NQ half core in to 30 x 40 cm plastic sample bags in sequence to await batch assignment and sample organisation.

10.1.2 Drill Sample Quality

Due to problems with the weighing scales during the first phase of drilling sample weights were not recorded (16,910 samples). Samples for all drilling since July 2013 were weighed routinely (66,576 samples).

Sample recovery for the RC samples was estimated based on a 127 mm hole size and densities of assumed density of 2.45 g/cm^3 for oxide, 2.65 g/cm^3 for transition ,and 2.93 g/cm^3 for fresh samples. Sample recovery for RC samples was generally good, averaging 95.9%. Statistics relating to poor recoveries are shown in Table 10.2.

Table 10.2 Samples with Poor RC Recoveries

Oxidation State	Samples with Weights (Total Samples)	Recovery <60%	% of Measured Samples
Oxide	12,023 (16,121)	486	4.04
Transition	16,077 (20,117)	327	2.03
Fresh	38,476 (47,248	1,072	2.79
Total	**66,576 (83,486)**	**1,885**	**2.83**

Water is recorded in isolated intervals. Table 10.3 shows that 582 samples were logged as having contained some water present, often at rod changes, and these were not saturated with water.

Samples logged as wet (125 samples) returned grades >0.50 g/t representing 0.9% of the 13,872 original RC samples which returned >0.50 g/t.

All RC drilling was conducted in dry conditions and if at any time samples became saturated with water, the hole in question was stopped and diamond tailed at a later date. No smearing of grades is apparent and the hole is routinely blown dry before the re-commencement of drilling.

Table 10.3 Wet Sample Statistics

Oxidation State	Samples	Wet Samples	%
Oxide	16,121	35	0.22%
Transition	20,117	20	0.10%
Fresh	47,248	527	1.12%
Total	**83,486**	**582**	**0.70%**

Through the use of PQ sized core in the upper part of the hole, reducing to HQ in fresh rock and NQ in deeper holes (>200 m), core recovery for diamond drilling was generally good with an average of 95.6%. Of a total 6,040 Diamond Drilling samples with recovery data, 164 (2.72%) were within runs with <80% recovery. Forty-two samples (0.70%) were within runs of <60% recovery. Of 2,025 Diamond Drilling samples with grades >0.5 g/t, 81 (1.34%) were within runs with <80% recovery. Nineteen (0.31%) were within runs of <60% recovery).

10.1.3 Drillhole Surveying

Collar Location Surveying

Drillhole locations were initially set out using a handheld GPS and marked with a painted rock. Upon completion of the drilling, a cement marker, inscribed with the drillhole name, was placed at the collar (Figure 10.3). After drilling, all collars were surveyed using differential GPS (DGPS) equipment.

Figure 10.3 Drillhole Collar

Source: Orca Gold Inc.

Azimuth and Dip Surveying

The drill rigs were aligned to the design azimuth for each hole using compasses that were corrected for magnetic declination. A line of pegs, approximately 6 m long and oriented to the design azimuth, is first pegged adjacent to the planned hole collar. The drill rig is then brought into position such that the tracks are approximately parallel to the pegged line. Offset distances from the pegs to the tracks are then monitored by tape measure during a final adjustment to fine-tune the rig's position. The rig is then regarded as being aligned to the design azimuth and drilling commences.

A Reflex Ez-Trac single-shot survey tool (Reflex) was used for all drilling, surveying at 50 m intervals during the drilling of the hole. Due to significant azimuth variation seen in the RC drilling up to the end of June 2013, a Reflex Gyro multi-shot survey tool (Gyro) was used to carry out surveys on completion of all new holes (in addition to the reflex used during drilling). It was also used to re-survey all mineralized holes from the first phase of drilling. The Gyro model used is not a north seeking Gyro and only measures variations in the azimuth and dip from the reference point of the collar.

The results of the Gyro survey of pre-June 2013 holes validated the reflex survey results but for clarity the survey data from the Gyro instrument was used in the Mineral Resource Estimate.

10.1.4 Geological Logging

RC Drilling

RC drill chips were geologically logged at 1 m intervals, recording rock types, structures, quartz veining type and percentages, sulphide occurrence and alteration type and intensity. Sample weight, estimated recovery and quality were also noted during logging (Figure 10.4).

Figure 10.4 Example of an RC Geological Log

RC Drilling Log Sheet

Hole ID:GSRC 181 Licence:B14 Project:MEYAS SAND Target:GS5 Geo:MMS Page: 4 of 5

From (m)	To (m)	Weight (KG)	Recovery	Wet_Dry	Contamination	Lithology	Colour	Weathering	Fabric	Fab Int	Alt_1	Degree	Alt_2	Degree	Alt_3	Degree	Min_1	Min_1%	Min_2	Min_2%	Min_3	Min_3%	VQ %	VQC %	VS %	Comments	Date
75	76	22.5	100	D	L	MSF	LG	W1	Fo	2	SE	3					Py	4									13/06/13
76	77	16	100	D	L	MSF	LG	W1	Fo	2	SE	3					Py	4									
77	78	22.5	100	D	L	MSF	LG	W1	Fo	2	SE	3					Py	4									
78	79	22	100	D	L	MSF	LG	W1	Fo	2	SE	3					Py	4									
79	80	18	100	D	L	MSF	LG	W1	Fo	2	SE	3					Py	4									
80	81	22.5	100	D	L	MSF	LG	W1	Fo	2	SE	2	SI	2			Py	4									
81	82	15	100	D	L	MSF	LG	W1	Fo	2	SE	2	SI	2			Py	4									
82	83	19	100	D	L	MSF	LG	W1	Fo	2	SE	2	SI	2			Py	4									
83	84	20	100	D	L	MSF	LG	W1	Fo	2	SE	2	SI	2			Py	4									
84	85	19	100	D	L	MSF	LG	W1	Fo	2	SE	2	SI	2			Py	4									
85	86	24	100	D	L	MSF	LG	W1	Fo	2	SE	2	SI	2			Py	4									
86	87	22.5	100	D	L	MSF	LG	W1	Fo	2	SE	2	SI	2			Py	4									
87	88	20	100	D	L	MSF	LG	W1	Fo	2	SE	2	SI	2			Py	4									
88	89	17.5	100	D	L	MSF	LG	W1	Fo	2	SE	2	SI	2			Py	4									
89	90	22.5	100	D	L	MSF	LG	W1	Fo	2	SE	2	SI	2			Py	4									
90	91	13	100	D	L	MSF	LG	W1	Fo	2	SE	2	SI	2			Py	4									
91	92	25	100	D	L	MSC	G	W1	Fo	2	SE	1	Cl	3			Py	2									
92	93	16	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	2									
93	94	20.5	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	3					20				
94	95	21.5	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	3					20				
95	96	24	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	3					20				
96	97	10.5	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	2									16/06/13
97	98	21	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	2									
98	99	22	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	2									
99	100	24	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	2									

Source: Orca Gold Inc.

RC drill chip samples were sieved at 1 m intervals to produce a sub-sample to act as a visual reference material. These samples are stored in plastic chip trays as shown in Figure 10.5.

Figure 10.5 1 m Interval Sieved RC chip Sample

Source: Orca Gold Inc.

Diamond Drilling

To facilitate geotechnical and structural logging, diamond drillholes are orientated using a spear marking the down-side of the core. Orientation lines are marked on the core by the driller and checked by the supervising geologist.

Structural logging of the oriented core is conducted using a kenometer on whole core prior to it being cut and sampled. Foliation, cleavage, faulting, veining, and geological contacts are logged.

Geological logging is undertaken by the Company's geologists on cut core after it has been sampled. The cut surface is preferred for logging as it provides better detail on texture, lithology, alteration and sulphide mineralization. Rock type, stratigraphic subdivisions, alteration, oxidation, and mineralization are routinely recorded.

Diamond core is logged according to geological domains which are identified by geologists. Intervals varied between 0.20 m and 1.6 m. The core was generally sampled in 1 m sample intervals, with a minimum interval used of 0.45 m.

Graphic geological and geotechnical logging is used to record recovery, rock quality designation (RQD), rock strength, and weathering. It is undertaken on the drill site prior to the core being transported to the core yard.

All core is digitally photographed at the drill site in a wet and dry state.

Diamond core is stored in plastic core trays (each holding approximately 4 m of core) at the Company's exploration camp.

In preparation for assaying, the core is cut down its axis with a diamond saw, with half of the core being returned to the core trays for future reference.

10.1.5 Density Measurements

All core holes are systematically sampled for the purpose of density measurements, 31 holes from GSS, and six holes from WD. A total of 956 samples were selected for density measurements based on lithology and oxidation state, with the aim of selecting a representative suite of samples for each lithology / oxidation condition (oxidized, transition, and fresh).

The density measurements were carried out on core samples, 10 - 15 cm long, full or half core (PQ, HQ or NQ size) selected at 5 m intervals. The lithology, depth and oxidation condition are recorded.

The samples are dried in an oven for 24 hr (at 100°C) and weighed. The dried samples are then waxed and weighed, before being submerged in water to record the volume of water displaced (hence density through the appropriate formula, applying a wax density of 0.9, to correct for the volume of wax). The bulk density is then calculated as: Bulk density = [Mass core] / [(Mass in air – Mass in water) – (Mass wax / 0.9)]. Table 10.2 below shows a summary of the density measurements made.

Table 10.4 Density Sampling

Zone	Lithology	GSS		WD	
		g/cm^3	Samples	g/cm^3	Samples
Oxide	Potassium Altered Diorite	2.66	6	-	-
	Diorite - Foliated	2.60	26	2.96	1
	Quartz Sericite Pyrite Schist	2.45	57	2.49	3
	Quartz Sericite Schist	2.58	11	-	-
	Vein Quartz	2.50	4	-	-
	Oxide Average	**2.52**	**104**	**2.61**	**4**
Transition	Breccia - Volcanic	2.72	19	-	-
	Diorite - Foliated	2.73	36	2.66	2
	Dolerite Dyke	2.69	2	-	-
	Quartz Sericite Pyrite Schist	2.63	30	-	-
	Quartz Sericite Schist	2.70	14	-	-
	Vein Quartz	2.51	4	-	-
	Transition Average	**2.69**	**105**	**2.66**	**2**
Fresh	Black Red Diorite	2.68	29	-	-
	Breccia - Hydrothermal	2.76	29	-	-
	Breccia - Volcanic	2.87	15	-	-
	Diorite	2.80	18	-	-
	Potassium Altered Diorite	2.80	119	-	-
	Diorite - Foliated	2.80	202	2.95	1
	Dolerite Dyke	2.59	18	2.78	9
	Quartz Sericite Pyrite Schist	2.81	216	2.95	23
	Quartz Sericite Schist	2.85	45	-	-
	Vein Quartz	2.68	17	-	-
	Fresh Average	**2.79**	**708**	**2.90**	**33**

10.2 Drilling Completed and Significant Results

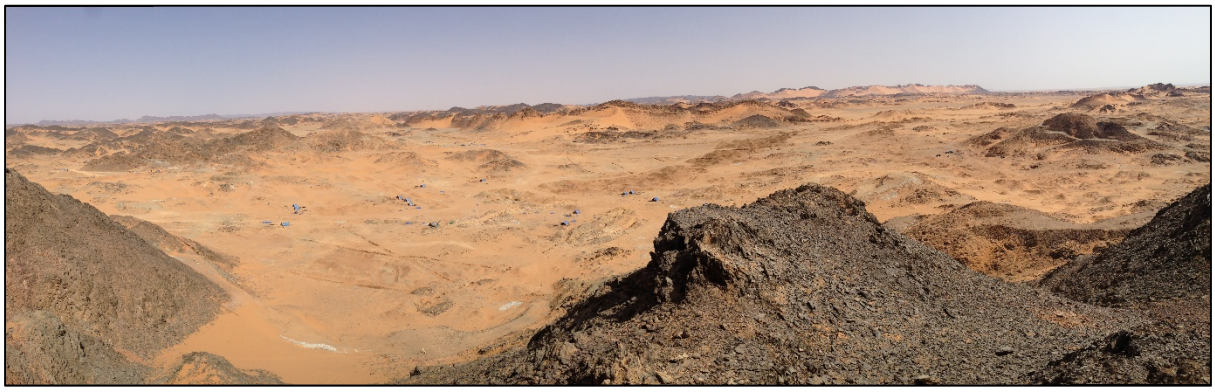
Since the discovery of GSS, it has been the focus of drilling completed on the project with 63,267 m completed (75% of project total) comprising 58,207 m of RC drilling in 449 holes and 5,060 m of core drilling in 17 holes. A further 10,312 m has been completed at WD (12% of project total), comprising 10,062 m of RC and 250 m of core drilling.

10.2.1 Galat Sufar South

Drilling and other exploration work at GSS (Figure 10.6) has identified a significant mineralized system and Mineral Resource over an area of 2 x 1 km (Figure 10.7). Mineralization is hosted within

variably sheared intermediate intrusive, and is associated with intense quartz-sericite-carbonate alteration and pyrite. Drilling at GSS has identified significant mineralization, with true widths in excess of 80 m intersected (Figure 10.8 and Figure 10.9).

Figure 10.6 View over GSS Resource Area



Source: Orca Gold Inc.

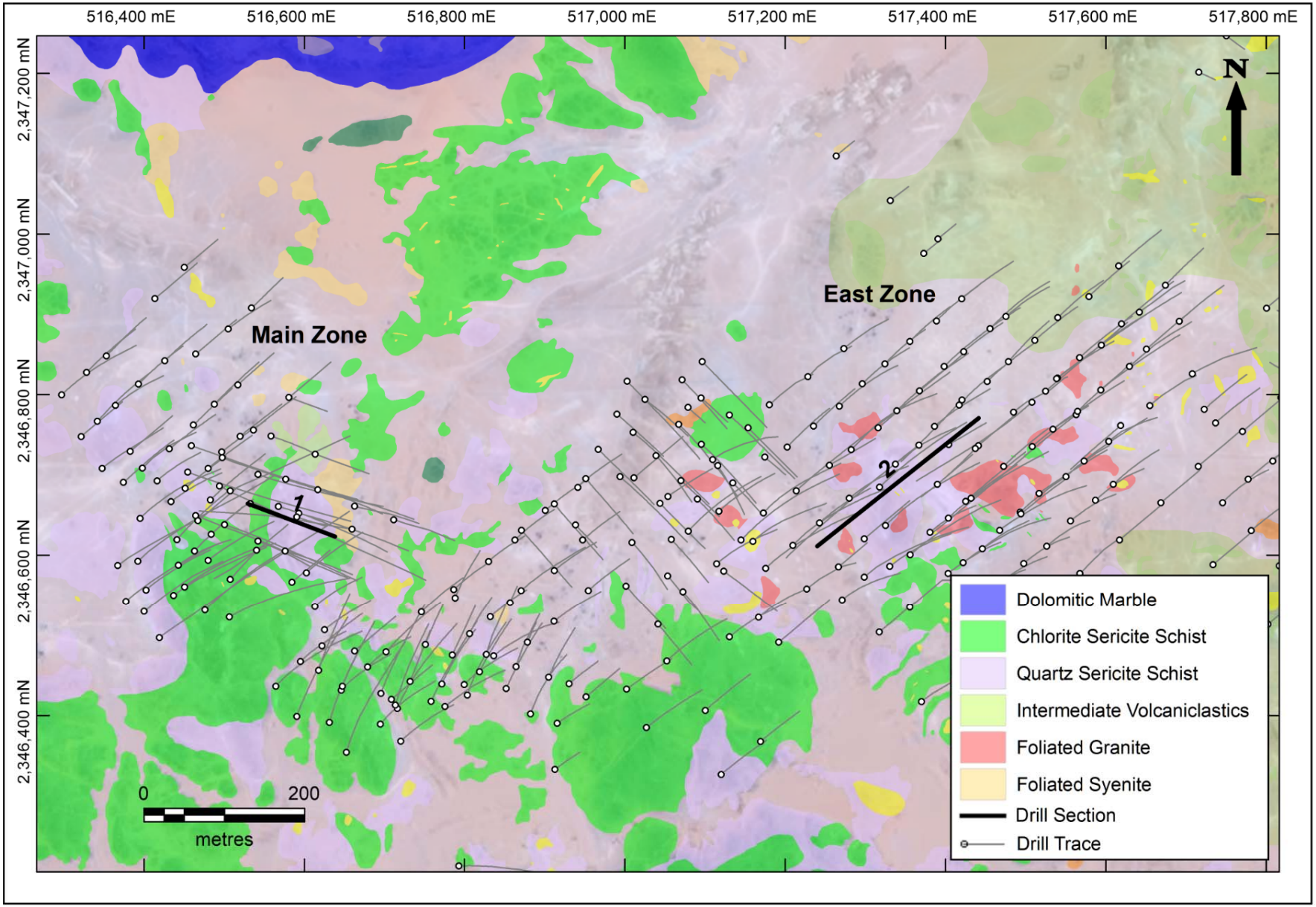
A selection of drill intercepts is shown in Table 10.5 below.

Table 10.5 Selected Intercepts from GSS

Hole ID	Depth From	Depth To	Interval	No Top Cut	Top Cut 10 g/t
				Au g/t	Au g/t
GSRC205	1	73	72	2.97	2.67
GSDD013	21	149	128	1.49	1.47
GSRC177	17	122	105	1.73	1.73
GSRC201B	135	233	98	1.83	1.83
GSRC081	162	187	25	7.60	6.30
GSRC001	26	88	62	2.55	2.52
GSRC203	0	44	44	3.68	3.48
GSRC219	45	83	38	3.76	3.73
GSRC012	0	63	63	2.39	2.03
GSRC011	0	35	35	3.65	3.60
GSRC201B	61	132	71	1.96	1.77
GSDD011A	143	161	18	14.66	6.94
GSRC003	98	120	22	11.80	5.57
MET002DD	0	96	96	1.26	1.26
GSRC081	52	71	19	15.29	6.24
GSRC024	19	36	17	7.46	6.90
GSRC138	35	49	14	11.20	8.32
GSRC108	83	112	29	4.03	3.98
GSRC194	1	49	48	2.36	2.35
GSDD008	69	117	48	2.26	2.26

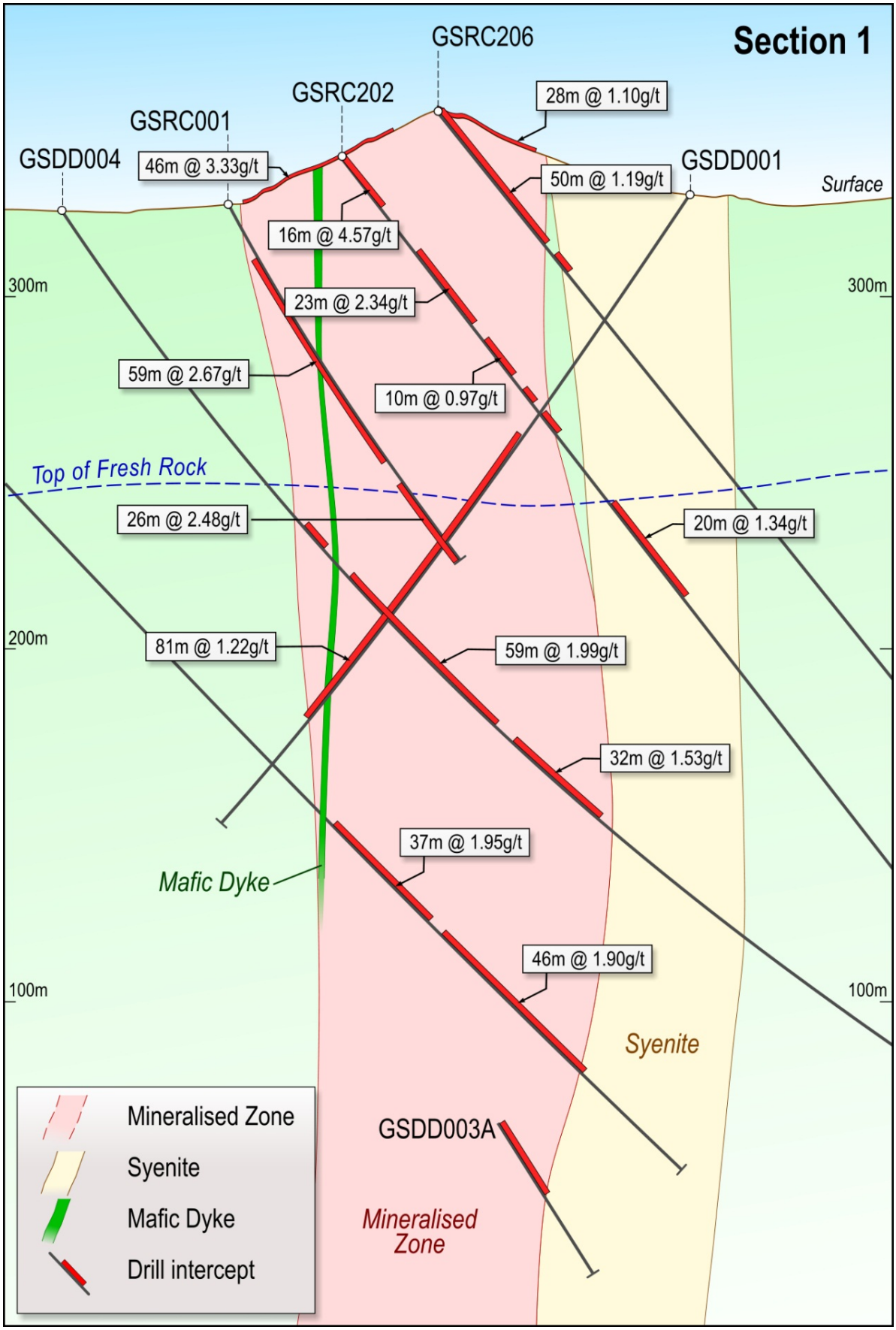
True widths are 65 - 75% of intercept widths.

Figure 10.7 GSS: Geology, Mineralization and Drill Traces



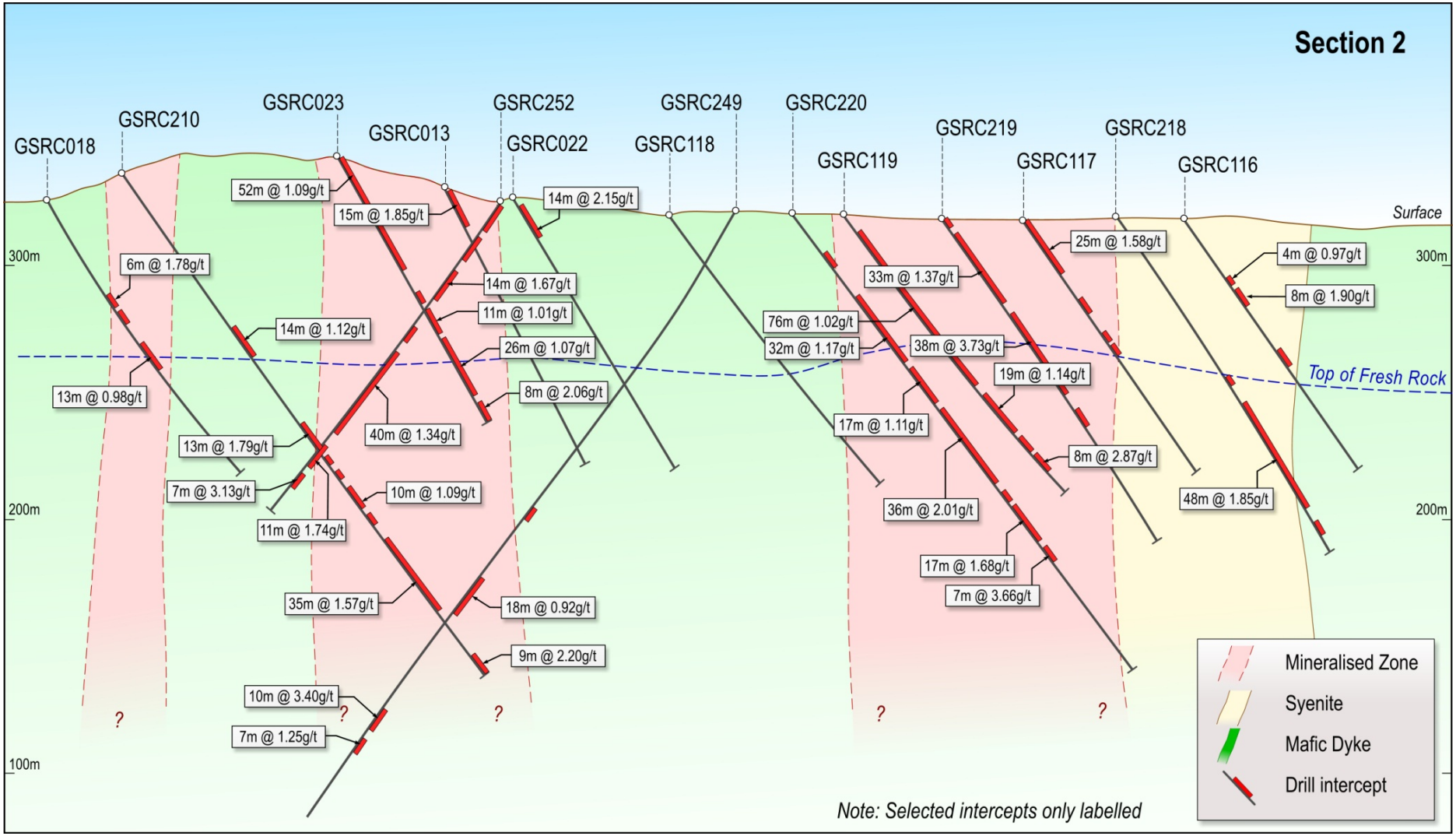
Source: Orca Gold Inc.

Figure 10.8 Main Zone Drill Section 1



Source: Orca Gold Inc.

Figure 10.9 East Zone Drill Section 2



Source: Orca Gold Inc.

10.2.2 WD

Drilling at WD has identified significant mineralization and a Mineral Resource over an area of 300 m x 300 m and centred on a small hill as shown in Figures 10.9 and 10.10 below.

Figure 10.10 View over WD Resource Area

Source: Orca Gold Inc.

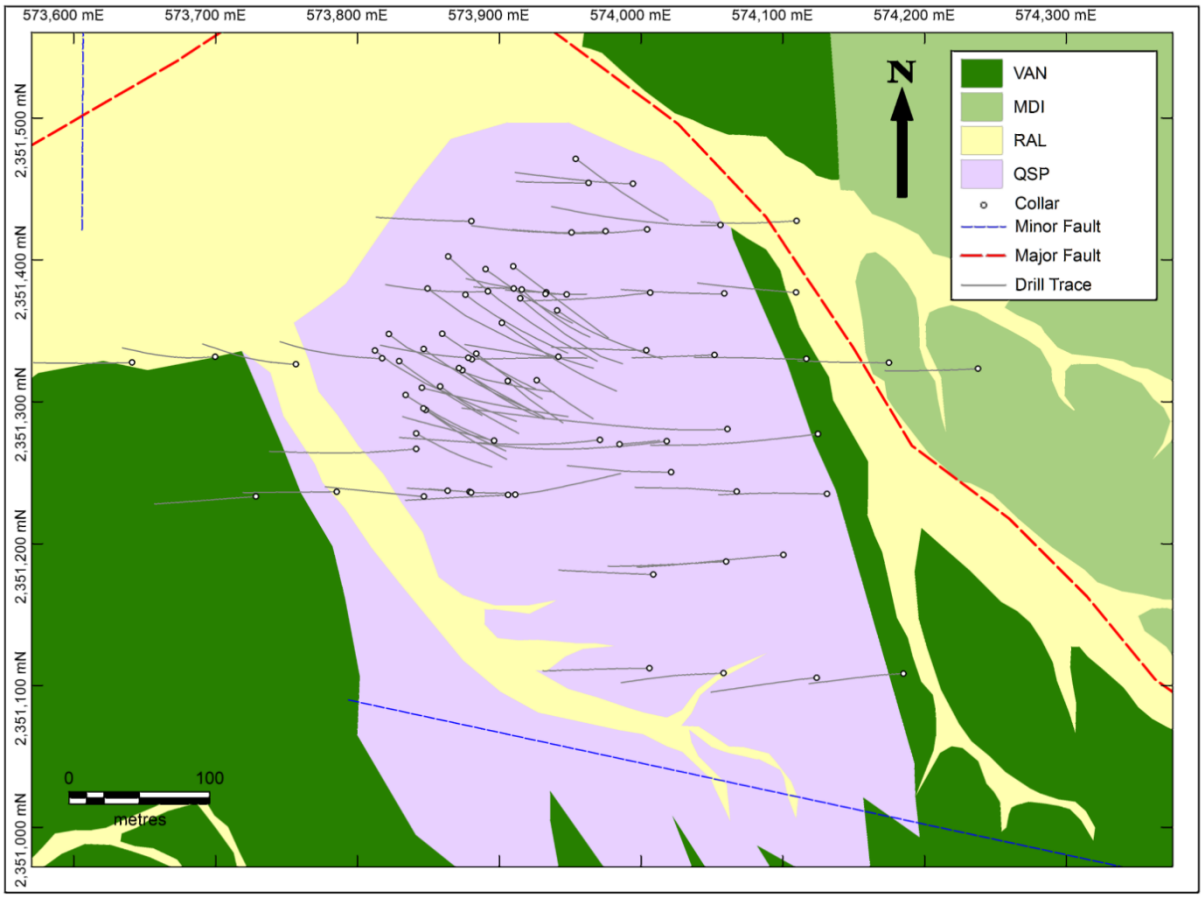
High grade mineralization is hosted by volcaniclastic units outcropping at the base of the hill and is characterised by significant levels of fine grained pyrite and a lack of silicification. Further mineralization is located below the main part of the hill with wide areas of anomalous to low grade containing several higher grade sections. Table 10.6 below shows a selection of intercepts from drilling at WD.

Table 10.6 WD Drill Intercepts

Hole	From	To	Metres	Au g/t Uncut	Au g/t Cut to 20 g/t
GSRC339	9	23	14	65.79	10.99
	70	79	9	6.76	6.76
GSRC341	29	50	21	19.35	5.93
GSRC342	6	13	7	7.07	4.56
GSRC401	35	57	22	7.17	6.17
GSRC402	84	103	19	2.63	2.63
GSRC413	32	41	11	21.47	7.65
	115	143	28	16.52	3.95
GSRC532	57	75	18	8.06	5.61
GSRC542	90	103	13	13.09	9.90
	107	120	13	5.46	5.46
GSRC543	55	73	18	6.10	3.94
GSRC545	40	47	7	13.45	6.98
GSRC547	66	97	31	6.08	5.63
	119	125	6	3.79	3.79
GSRC548	6	22	16	13.88	8.14
	35	53	18	5.04	3.67
	72	100	28	3.38	3.30
GSRC549	114	124	10	7.40	7.40
GSRC550	11	38	27	5.30	5.30
	47	62	15	5.54	5.54
	83	121	38	5.30	2.67

True widths are 60 - 75% of intercept.

Figure 10.11 Location of Targets Drilling within the Block 14 Project

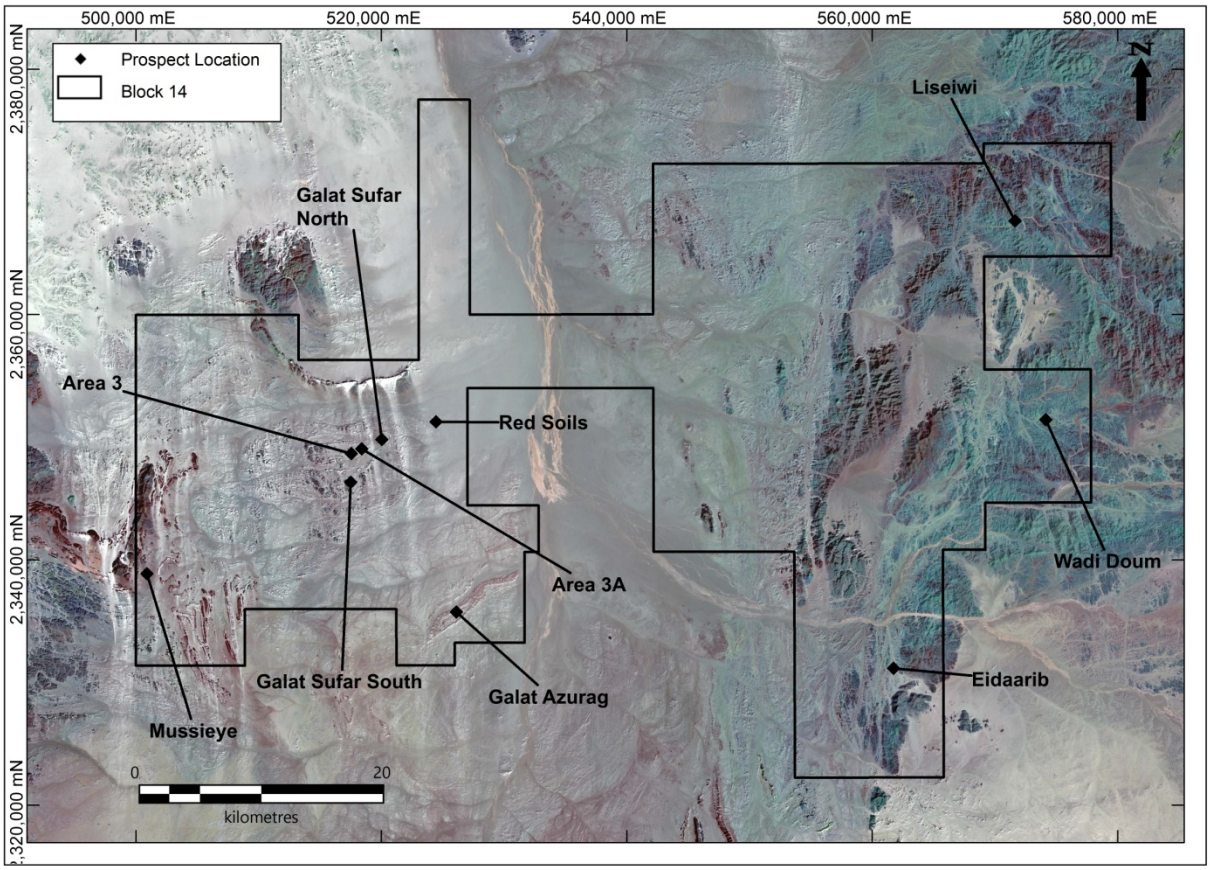


Source: Orca Gold Inc.

10.2.3 Other Prospects Drilled

RC drilling of 11,189 m (105 holes) has been completed on targets outside GSS and WD, within the Block 14 project area (Table 10.1); locations are shown in Figure 10.12.

Figure 10.12 Location of Targets Drilling within the Block 14 Project



Source: Orca Gold Inc.

Several of these targets have yielded significant intervals (Table 10.7) however further exploration work is required to determine whether mineral resources can be developed at these prospects.

Table 10.7 Significant Intercepts from Other Prospects

Prospect / Target	Hole ID	Depth From	Depth To	Interval	No Top Cut	Top Cut 10
					Au g/t	Au g/t
A3A	GSRC043	22	45	23	8.65	4.17
A3A	GSRC120	80	86	6	4.33	4.33
Liseiwi	GSRC596	38	51	13	11.85	5.76
Liseiwi	GSRC595	0	24	24	3.89	2.13
Liseiwi	GSRC599	16	24	8	12.80	5.78
Liseiwi	GSRC599	29	37	8	5.21	4.57
Liseiwi	GSRC598	30	45	15	2.46	2.38
Target P	GSRC632	11	15	4	3.44	3.44
Target P	GSRC633	4	10	6	1.88	1.88
EG2.2	GSRC397	4	15	11	2.54	2.54

True widths are 60 - 75% of intercept

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

All samples collected on the project by Orca were subject to quality control procedures which ensured the use of industry best practice in respect of the handling, sampling, transport, analysis, storage, and documentation of sample materials and their analytical results.

In October 2011 Orca commissioned a sample preparation facility at a project site in Sudan. The commissioning and training of operators was supervised by ALS Chemex (ALS) and all rock chip, trench and diamond core samples were crushed before splitting and shipping to the ALS Rosia Montana laboratory in Romania. Samples from RC drilling (1 kg) were shipped direct to the laboratory.

On behalf of Orca, ALS commissioned a containerised sample preparation facility in the town of Atbara, Sudan in March 2013 (Figure 11.1). Since that time, all samples have been prepared under the supervision of ALS at this sample preparation facility, followed by analysis at the ALS Rosia Montana laboratory in Romania.

ALS is a global independent provider of assaying and analytical testing services for the mining and mineral exploration industry with consistent quality standards implemented across all regions. Both of the ALS facilities used are certified to ISO 17025. The laboratory participates in group-wide round robin assay work to ensure internal quality performance.

11.1 Sample Submission Procedures

When samples are dispatched to the laboratory, a completed sample submission form accompanies the samples. The submission form details the sample number sequences and also instructs the laboratory on the elements required for analysis and the analytical methods to be used. Below is a step by step procedure in the sample submission process to the preparation facility in Atbara:

- Batch assignment of samples is organised by a geologist in communication with the data entry personnel at the exploration camp and with reference to batch assignment paperwork / excel sheet.
- All samples are arranged in order at the exploration camp. Reference / archive samples are separated and stored. Triplicate samples are separated and stored for third party assay at a later date.
- QA/QC samples comprising standards (~60 g sealed packet of certified reference material) and blank material are inserted into the sample sequence by a geologist.
- All samples are packaged in sequence into plastic drums and sealed with plastic ties for transit to Atbara by company vehicle (Figure 11.2).
- The sample submission form is prepared and checked for each batch. A hardcopy accompanies the samples and an electronic copy is emailed to the manager of the sample preparation facility.

11.2 Sample Preparation and Analysis

11.2.1 Rock Chip, Trench, RC Drill, and Drill Core Samples

Up to March 2013

- All drill core, trench, channel, and rock chip samples were crushed to a nominal crush size of 80% passing <2 mm. Samples are weighed before and after crushing.

- A single tier splitter was used to produce a split of the original sample for dispatch to the assay laboratory. For drill core samples, the split is approximately 1 kg. For other forms of sampling this split was 250 – 300 g. Compressed air is used to clean the crusher and splitter between samples.
- RC drill samples were not crushed due to the nature of the drilling and a 1 kg split representing each metre is submitted for analysis.
- Laboratory samples were placed in new plastic bags, with the sample ticket included, and the sample number written on the outside of the bag. The plastic bag containing the assay sample is then sealed with a cable tie. Plastic drums with sealable lids are used to transport the completed samples to the Shark office in Khartoum.
- For drill core, trench and rock chip sample batches a crusher flushing sample of barren vein quartz was used to clean the crusher plates after 20 samples and at the end of individual sample batches.
- All drill core, RC chip, trench, and rock chip sample were analysed for gold by 50 g fire assay with lead collection, solvent extraction and AAS finish (Au-AA26).
- Selected samples were shipped to the ALS facility in Vancouver for analysis using a multi-element package comprising 51 elements by ICP-MS and ICP-AES.

March 2013 Onwards

Samples are dispatched in individual batches from the exploration camp to the sample preparation facility in Atbara. Below is a step by step approach of how the samples are then handled:

- Each batch received is laid out in sequence, weighed and checked in to the ALS system. Missing (not received or insufficient sample) or extra samples (not listed on sample submission form) are flagged and brought to the attention of personnel submitting the batch documentation. Once problems are rectified (submission of replacement sample from the exploration camp, or submission of corrected sample submission forms) preparation may commence.
- The entire ±3 kg sample is crushed to >80% passing -2 mm using a jaw Crusher, with 5% sieve tests. The crushed sample is then riffle split to produce 2 x 1.5 kg samples (Figure 11.3 and Figure 11.4).
- For drill core, RC chips, trench, and rock chip sample batches a crusher flushing sample of barren vein quartz is used to clean the crusher plates after 20 samples and at the end of individual sample batches.
- A 1.5 kg sample is pulverised in a Labtechnics LM2 ring mill to 85% passing 75µ, 5% sieve tests.
- A split of 250 g pulp is shipped to the Rosia Montana laboratory and a second 500 g sample is taken for archive (Figure 11.5).
- Pulps are shipped in batches in sealed boxes via the Orca office in Khartoum and international airfreight to Bucharest in Romania, where they are collected by ALS staff and delivered to the laboratory at Rosia Montana.
- All drill core, RC chip, trench and rock chip sample are analysed for gold by 50 g fire assay with lead collection, solvent extraction, and AAS finish (Au-AA26).
- Selected samples are shipped to the ALS facility in Vancouver for analysis using a multi-element package comprising 51 elements by ICP-MS and ICP-AES.

11.3 Sample Security

All aspects of the sample collection, their organisation and transport is supervised by Orca geological staff. Samples are transported from the exploration camp to the sample preparation facility in plastic drums sealed with numbered plastic security ties.

All samples for assay are stored securely at the sample preparation facility prior to processing and are transport to Khartoum in company vehicles. In Khartoum the sealed boxes of pulps are stored within the Orca offices prior to dispatch. Commercial airfreight with Turkish Airlines is used to transport the samples from Khartoum to the ALS Chemex laboratory in Romania.

Figure 11.1 Sample Preparation Facility in Atbara

Source: Orca Gold Inc.

Figure 11.2 Sealed Drums of Samples after Arrival at Sample Preparation Facility

Source: Orca Gold Inc.

Figure 11.3 Jaw Crusher

Source: Orca Gold Inc.

Figure 11.4 Riffle Splitter

Source: Orca Gold Inc.

Figure 11.5 Storage of Archived Pulp Samples

Source: Orca Gold Inc.

11.4 Quality Control and Quality Assurance

Orca has instigated external QA/QC processes to monitor the reproducibility of geochemical, trenching, and drilling data. The QA/QC programs have been rigorously employed during the exploration programs to monitor assay sample data for contamination, accuracy and precision.

For all sampling programmes since exploration commenced Orca routinely inserts blanks and Certified Reference Materials (CRM), in addition to taking duplicate samples.

Orca's quality control regime is shown in Table 11.1.

Table 11.1 Quality Control Regime

Sample Medium	QA/QC Sample Type	QA/QC Sample Spacing
Drainage Samples Rock Chip Samples	Field Duplicates	1 in 20
Chip Channel Samples Trench Samples	Standards	1 in 20
RC Samples Core samples	Blanks	1 in 20

In addition, the laboratory (ALS Chemex) has its own internal quality performance processes. These follow best practice guidelines required for qualification under International Organisation for Standardisation ("ISO") standards. The standard QA/QC protocols for the laboratories includes the insertion of CRMs, blank, duplicates, and repeat assaying to monitor the quality of the preparation and analytical processes of the laboratory.

The results of the internal laboratory quality control are reported regularly to Orca on a batch by batch basis, and the results are closely monitored by Orca personnel.

11.4.1 Certified Reference Materials

Various CRM standards are used by Orca to monitor the accuracy and precision of the assay laboratory. The CRMs selected by Orca adequately cover the expected grade ranges likely to be encountered for the style of mineralization being targeted. CRMs are supplied by Geostats Pty Ltd of Perth, Australia in sealed 80 g plastic bags. CRM samples are submitted in sequence with sample batches.

11.4.2 Blank Material

From mid-2013 all blank material has been obtained from an outcrop of barren dolomite located adjacent to Orca's B14 field camp. For drill sampling and surface sampling methods, the blank material is inserted in sequence as coarse fragments. Material from the same source is also used by the sample preparation facility as a coarse crusher flush. Prior to mid-2013, blank material was sourced from a quartz vein located outside the Project area. All submitted blank material is treated as a normal sample and the same sub-sample size is submitted for further preparation and analysis by ALS in Romania.

11.4.3 Duplicate Samples

The term 'duplicate' is a generic name for any repeat assay measurement or a second sample of the same sample interval or location. Duplicate samples check on the quality of the sample collection, sample preparation, and analytical precision. The inclusion of duplicate sample and their

comparative analysis is essential in determining the level of precision, or reproducibility of the assay using a particular sampling method and analytical method.

11.4.4 Field Duplicates

Field duplicates are one specific type of duplicate consisting of a sample, sampled by the same primary sampling method as the original. In the case of drainage and rock chip samples (surface samples) a second sample is collected from the same immediate locality as the original.

For trenching a second channel is cut along the line of the first sample. For RC drilling field duplicates, the sample is split through a riffle splitter to produce an original sample and a duplicate sample. For diamond drilling, a quarter core is submitted as the duplicate sample.

11.4.5 Umpire Lab, Pulp Duplicates

Umpire Lab

A batch of 1,069 pulp duplicates was submitted to ACME Analytical Laboratories in Vancouver in December of 2013 for umpire assay by 50 g fire assay with AAS finish (with multi element suite). These pulp samples represent five complete drillholes (1 x diamond, 4 x RC), and consisted of 125 g pulp splits taken from the reference samples stored at the Atbara preparation facility.

Bottle Roll

A batch of 978 pulp duplicates was submitted to ALS Chemex in Ireland in April of 2013 for bulk cyanide leach with AAS finish (with fire assay on tails and a multi element suite). These pulp samples represent seven drillholes (all RC) and consisted of 500 g pulp splits taken at ALS Romania from the original samples' pulp rejects.

11.5 Monitoring of QC Samples

Results from the sample control programmes are scrutinised for each assay batch by Orca personnel for any obvious gross errors. In addition, the final laboratory QA/QC certificates are also examined by Orca and no problems have been detected for any of the control sample data. The author has reviewed and independently assessed all available QA/QC sample data for the sampling completed on the project by Orca. Overall, the QA/QC samples have performed within the control limits, indicating that the sample data is of a high standard and appropriate for the reporting of exploration results. The results of the analysis are summarized below.

11.5.1 Blank Material

Blanks are assessed by graphical representation of the assay value and the maximum control value. Blank samples for drill sampling are displayed in Figure 11.6. A summary of all blank samples is reported in Table 11.2.

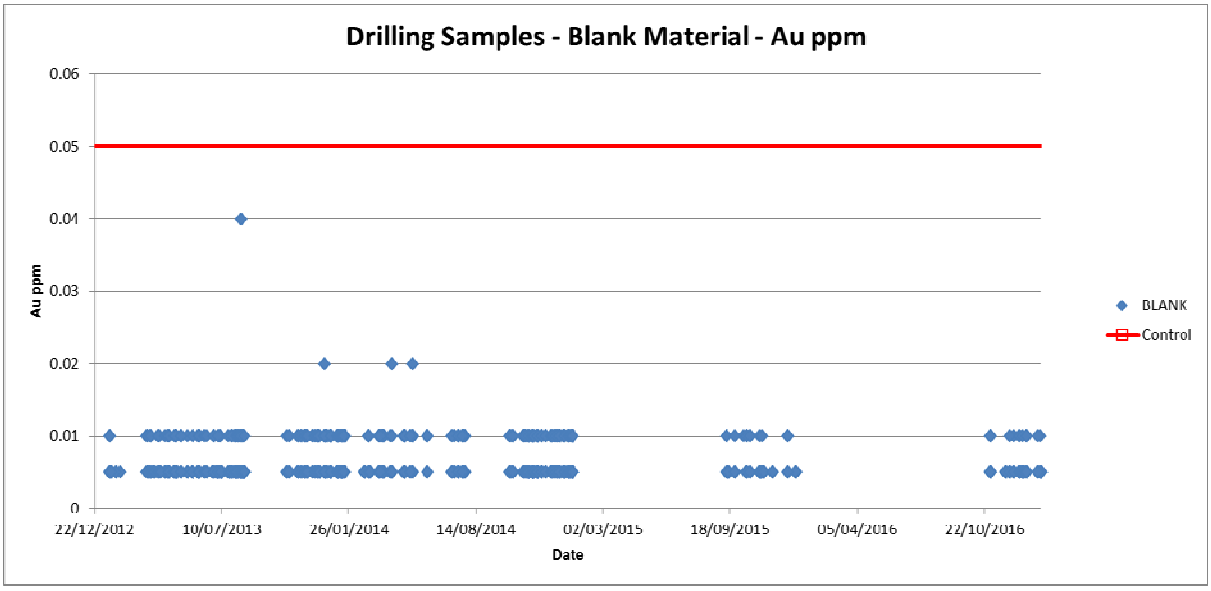
Table 11.2 Summary of Blank Samples for all Sampling Methods

Sampling	STD	Count	Control	Result	STDEV of Results	% Fail
Drilling (DD, RC)	Blank	4,548	0.05	0.006	0.002	0%
Surface (RCHIP, Drainage)	Blank	89	0.05	0.005	0.005	0%
Trench (TR, HC)	Blank	936	0.05	0.005	0.001	0%

Note: all samples assaying below detection limit of 0.01 ppm (DL) are rounded to ½ DL or 0.005 ppm.

Results show that for drilling samples and surface samples, all blank results report below 0.05 ppm. For trench sampling, one sample plots marginally outside the threshold limit of 0.05 ppm. In total, 99.9% of all blank material assays return grades within control limits (<0.05 ppm). Both trench sampling and surface sampling blanks indicate an improvement in blank quality in late-2012, coincident with a change of preparation facility. This change occurred before the first submission of drill samples and subsequently drill sample blanks demonstrate a high standard of preparation quality throughout.

Figure 11.6 Blanks Samples for Drilling Samples (50 g Fire Assay)



11.5.2 Field Duplicates

Duplicate data can be statistically analysed in a number of different ways, either by direct comparison of the duplicate sample to the original sample or by calculating different relative differences between duplicate pairs. The quantity of duplicate pairs is detailed in Table 11.3.

Table 11.3 Quantity of Duplicate Pairs Collected by Different Sampling Methods

Type	Quantity of Pairs
Diamond Drill Core	269
Reverse Circulation Drilling	4,065
Trench	1,108

X-Y Scatter Plots

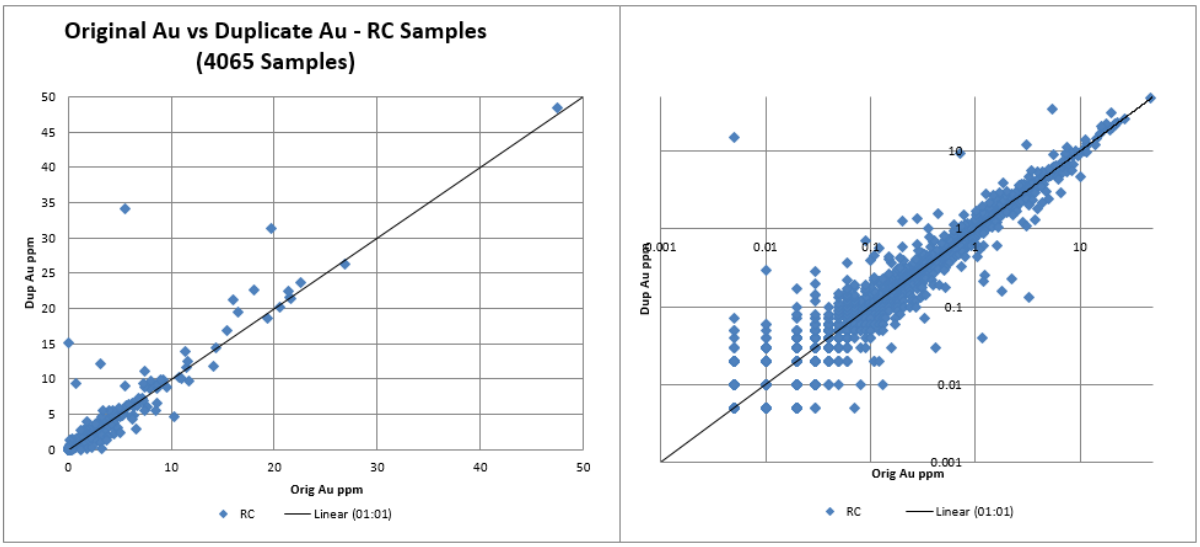
The simplest initial analysis is accomplished using an X-Y scatter plot to gain a general view of the repeatability of results and to identify obvious errors in samples; these are generated both with normal axis and log axis. The normal scatter plot aptly demonstrates correlation above 1 ppm, however due to the skew of the data set towards <1 ppm values the Log scatter plot is required to assess values at the lower grade end of the distribution.

Scatter plots have been generated for DD duplicates, RC duplicates, channel and trench / channel duplicates. Diamond and RC duplicates demonstrate the best quality distributions, with less than 1% of samples identified as outliers for both data sets. The remainder of duplicate samples conform well to the 1:1 correlation line, with some spread of results at values near / below the detection limit (0.01 ppm for fire assay). See Figure 11.7 for an example of the normal and log scatter plots for the

RC field duplicates. Five outliers are identified reporting higher grades in the field duplicate sample, suggesting potential 'nuggety' gold distribution in these samples.

Duplicate results for trenches and channels demonstrate the greatest spread and least coherence to the 1:1 correlation line, it should be considered however that a larger majority of this trench and channel data set is of waste material (near / below detection) and that the method for producing a field duplicate of this sample type is much more sensitive to human error than for RC and DD sampling. The four largest outliers for this data set may be attributable to sampling of coarser 'nuggety' gold.

Figure 11.7 Normal and Log Scatter Plots of RC Field Duplicates

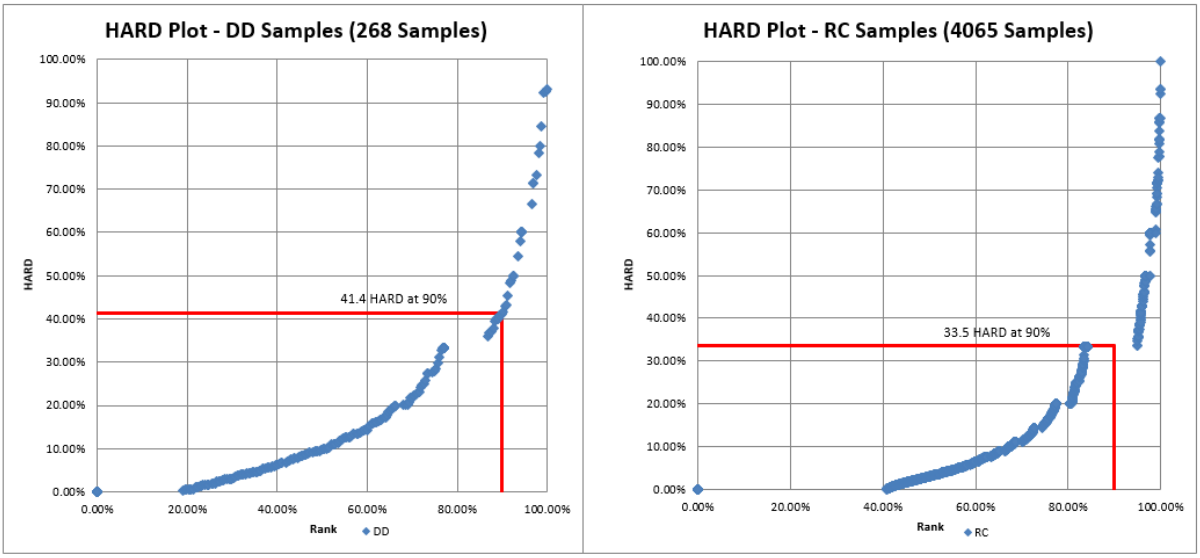


Ranked Half Absolute Relative Difference (HARD)

Ranked HARD plots demonstrate the precision of a data set for a given proportion of the samples; this is typically reported at 90%. Due to the absolute nature of HARD it cannot be used to reveal bias in duplicate data and is only used to assess relative magnitude of differences (the precision).

Ranked HARD plots have been generated for DD duplicate samples and RC duplicate samples (Figure 11.8). These plots demonstrate that RC duplicates have a better precision at the 90% proportion (33.5 HARD at 90%, note that the HARD only increases to 33.7 at 95%) compared to the DD duplicates (41.4% HARD at 90%). This difference between DD and RC may be related to the quarter-coring of DD duplicates compared to the riffle-splitting of RC samples however it should also be noted that the RC data set contains almost 20 times as many duplicate pairs.

Figure 11.8 Ranked Hard Plots for DD Duplicate Samples and RC Duplicate Samples

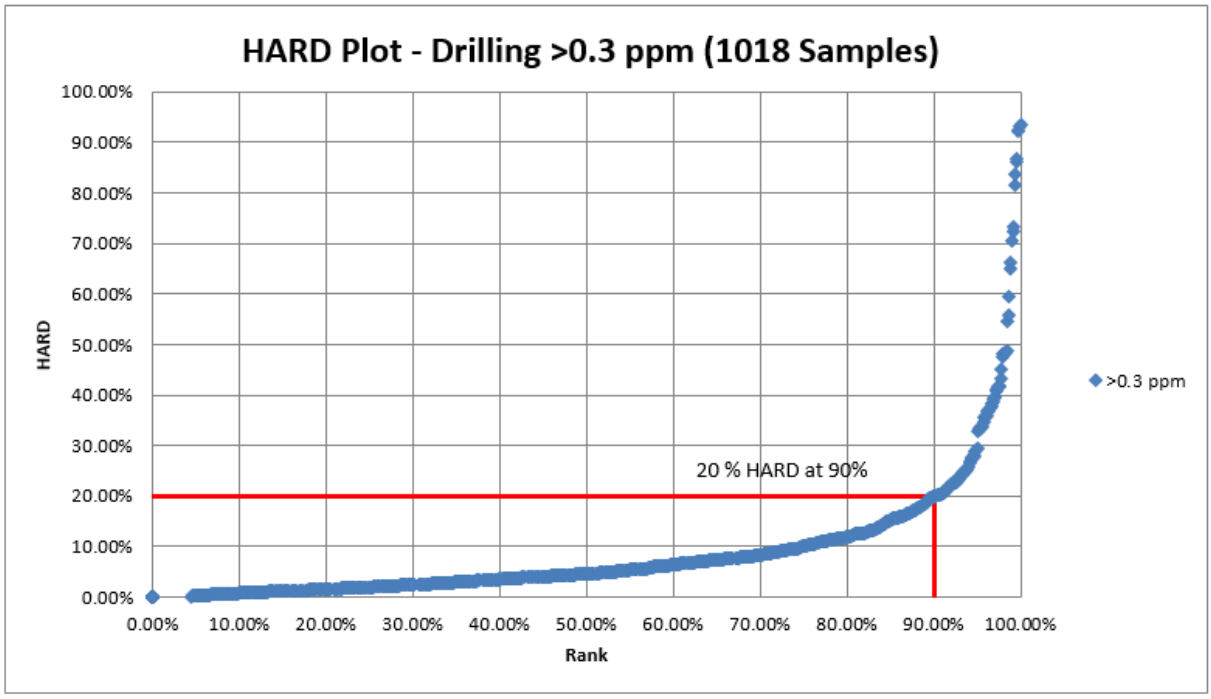


Waste / near detection limit assays compose a significant proportion of both data sets. When assessing relative differences, these near detection results can have significant influence. For example, an original assay of 0.03 ppm and a duplicate of 0.01 ppm with a pair mean of 0.02 ppm have a %RD of 100% and a HARD of 50%. A large number of these results in a population will produce a significant skew to %RD and HARD results. Subsequently a ranked HARD plot has also been generated for all drilling duplicates (DD & RC) above 0.3 ppm (Figure 11.9).

This data set consists of only 1,018 duplicate pairs compared to the total of 4,333 for the previous two sets. Therefore ¾ of the previous data set were <0.3 ppm, this is in part due to the utilization of a fixed sequence duplicate sampling procedure and could be improved by increasing the frequency of duplicates in zones of expected grade.

For the >0.3 ppm data set the 90% proportion HARD is much lower than in the previous two data sets at 20.3% HARD. This plot demonstrates that for assay grades of economic significance the precision of duplicates is very good, and that the majority of poor precision in the previous two data sets was a result of waste grade duplicates skewing the data.

Figure 11.9 Ranked Hard Plot for All Drilling (DD and RC) Duplicates



Relative Difference (%RD)

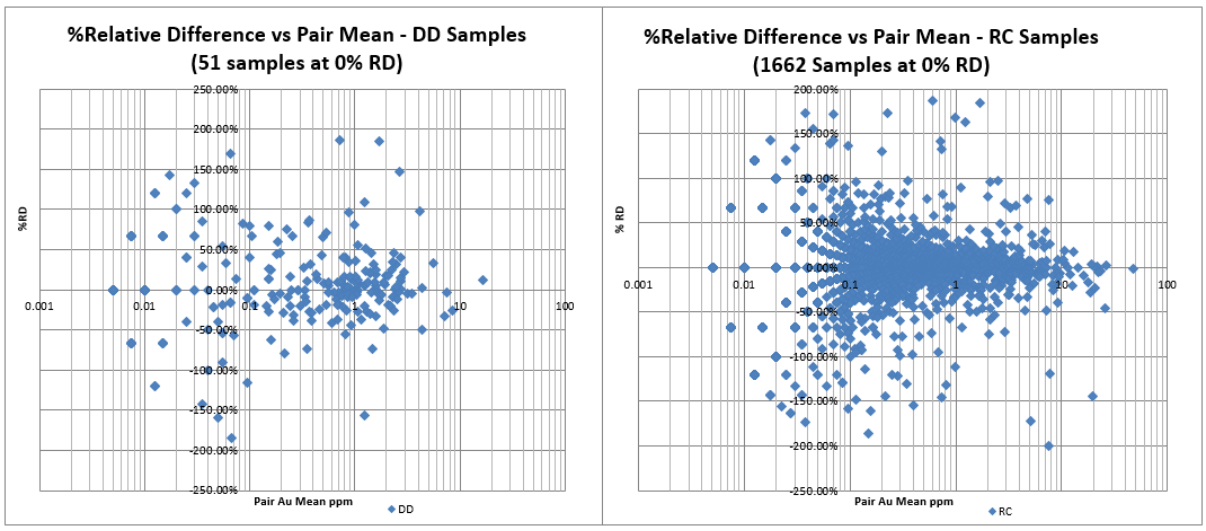
As discussed, the Ranked HARD plot is not able to be used to asses bias therefore a Relative Difference plot is used. If there is no bias the %RD plot is expected to be near symmetrical, forming a funnel shape with larger %RD at lower grades approaching the detection limit, due to the lower precision encountered at such ranges.

For both diamond and RC duplicates this funnel shape is evident (Figure 11.10). DD duplicates demonstrate a small positive bias (original > duplicate), with more pairs plotting >+50% than below <-50%. This bias may be due to the quarter-coring of DD duplicates.

The %RD, although well spread and demonstrating a high degree of difference, is evenly distributed with no clear positive or negative bias. There are both positive and negative outliers however the bulk of duplicates plot within ±40% of the 0% RD line. On both plots the %RD can be seen to improve above 1 ppm, indicating a high degree of precision for higher grade samples.

Negative outliers in the RC duplicates data above 5 ppm suggest possible 'nuggety' gold occurring in those samples. This is not demonstrated in the majority of DD duplicates where the effect of 'nuggety' gold would be expected to have been enhanced by the quarter coring practice thus suggesting it is not a widespread occurrence.

Figure 11.10 %Relative Difference Plot for DD and RC Duplicates



11.5.3 Pulp Duplicates (Umpire Assay)

Pulp duplicates (973) were sent to an umpire lab and have been assessed by duplicate statistical methods. The submission included Blank and CRM materials which have been assessed separately and has passed QA/QC thresholds.

As can be seen the X-Y scatter plots conform well to the 1:1 linear (Figure 11.11). There are no extreme outliers, however five points are evident below the 1:1 distribution of the majority of the population.

The assessment of %Relative Difference indicates no distinct bias, with values spread evenly (positive and negative). Figure 11.12 shows the ranked HARD assessment for all original values >0.3 ppm. By removing these samples, the 90% population HARD decreases from 25% (all data 973) to 8.47% (>0.3 ppm data, 483 samples). This result indicates good quality precision of umpire lab duplicates. The five outliers identified only account for 0.5% of the data set and may be the result of nugget gold or poor sub-sampling of the pulp either from the pulp reject or from the submitted pulp.

Figure 11.11 Normal and Log X-Y Scatter Plots for Pulp Duplicate Umpire Assays

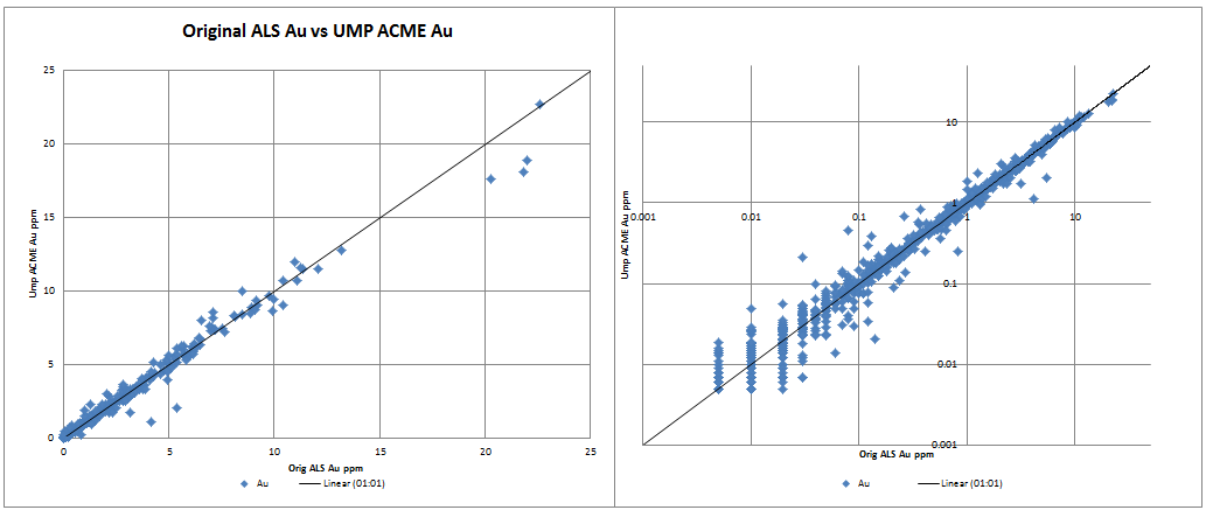
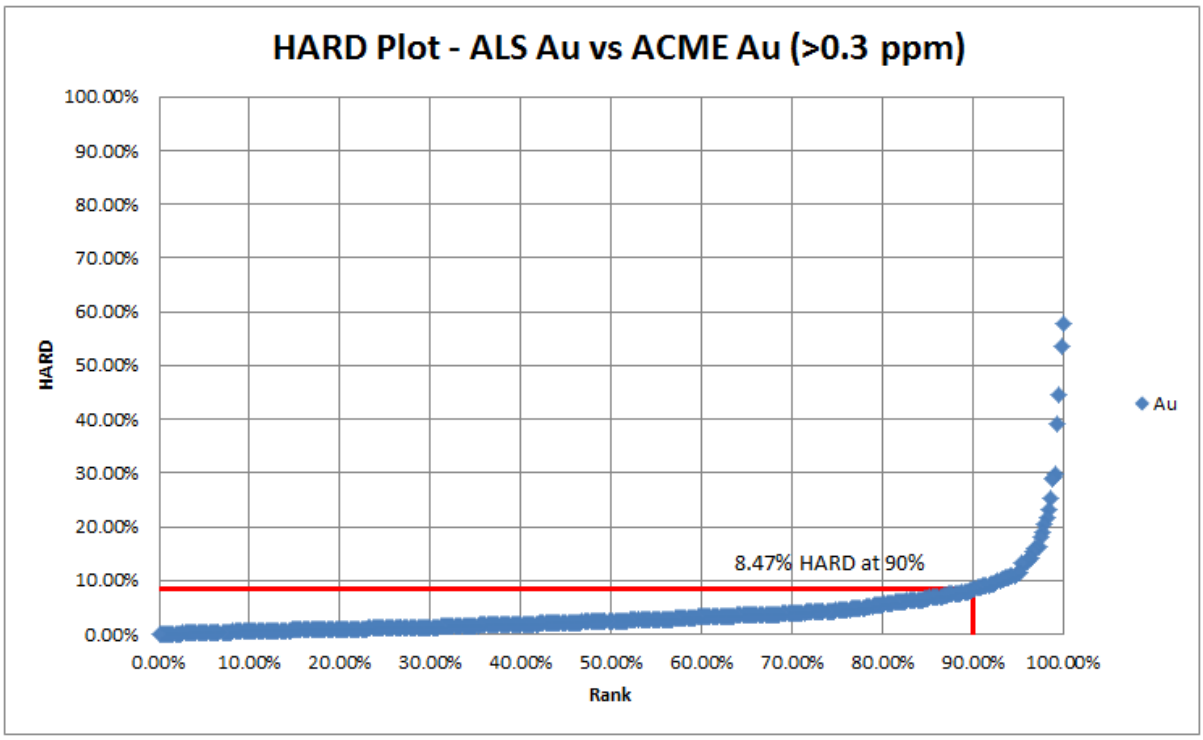


Figure 11.12 Ranked HARD Plot of All Duplicates with Original Result >0.3 ppm (483 Samples)



11.5.4 Bottle Roll Pulp Duplicate

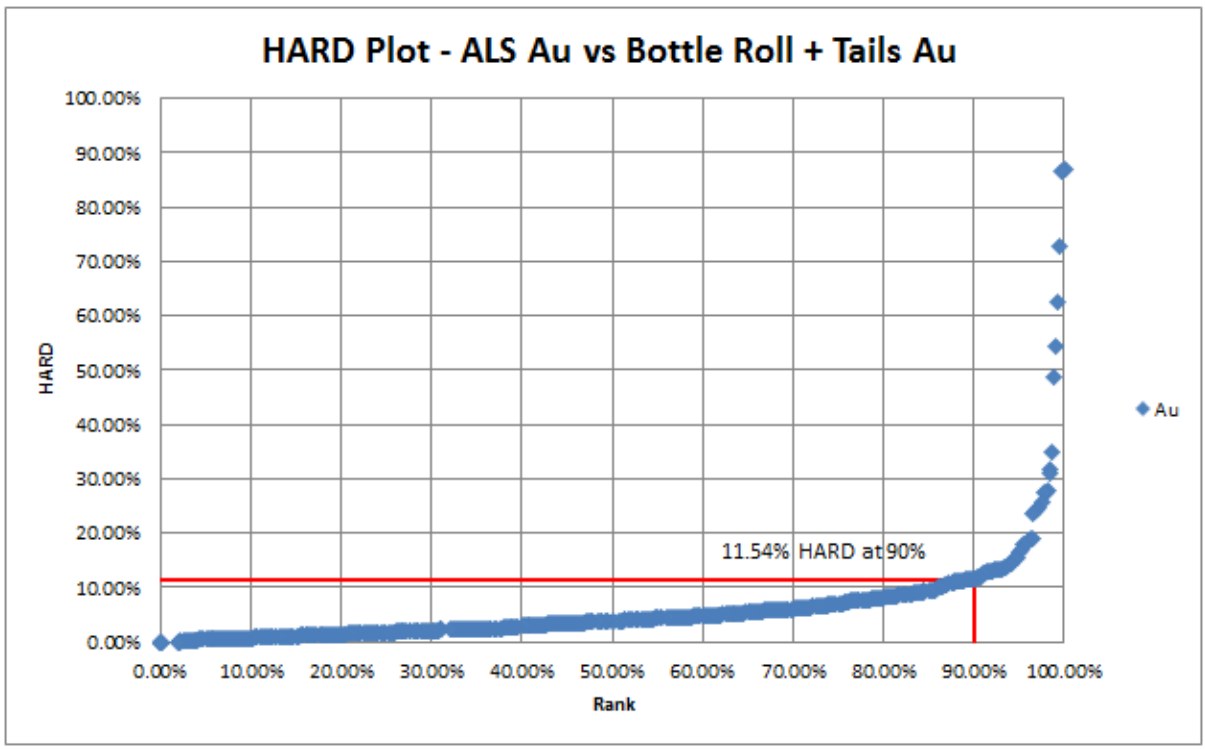
A number of samples (481) of 500 g were assayed by bottle roll. For the QA/QC assessment the leach assay and the fire assay of the leach tails have been summed and compared to the original fire assay result.

Despite the considerable sample size variance between the 500 g bottle roll with tails fire assay and the original 50 g fire assay, the duplicate data performs well. The normal X-Y scatter plot shows a strong coherence to the 1:1 linear with the only significant deviations evident on the log X-Y plot at low grades.

The assessment of % relative difference plot demonstrates no clear bias with the majority of all points plotting between +50% and -50. There are two areas of interest, firstly between 0.2 and 0.3 ppm pair mean where there are six points demonstrating positive relative difference above 50%, and secondly six points plotting below -50% between 0.6 and 10 ppm. The first set of positive values is not unexpected at low original grades, taking into account the larger sample size of the bottle roll assay. The second set of negative values could suggest poor sub-sampling of the original pulp however these only constitute 1% of the data set.

The % HARD at 90% of the population is 11.54% which is particularly good considering the difference in the two assay methods applied (Figure 11.13). No results have been removed from this data set as none of the original assay results of the pulps submitted were below 2.1 ppm.

Figure 11.13 Ranked HARD Plot of Bottle Roll Duplicate Pairs



11.5.5 Certified Reference Material (CRM)

CRM Assessment Overview

The performance of CRM samples has been assessed regularly throughout the period of sample assay. Each individual batch is assessed upon receipt of assay results to identify any immediate errors; of particular interest are sample swaps or transcription errors in the recording of CRM identities. Periodically, all to-date CRMs are assessed by plotting against the certified value of the CRM on control charts. This periodic assessment is intended to identify any bias or trends in the data set which cannot be easily distinguished on a single batch basis.

Control limits are assigned based on the certified reference value and certified standard deviation (SD) as reported by the CRM issuing laboratory. Common limits are the ±2 SD and ±3 SD ranges (95% and 97.7% confidence limits respectively). Standard QA/QC practice is to monitor results plotting > ±2 SD and to action results plotting > ±3 SD (re-assay or re-submit batch). In order to maintain the highest standard of QA/QC, ±2 SD has been chosen as the primary control limit for this project, however with the consideration that 5% of samples are expected to fall legitimately outside of these limits. For results within ±2 SD the control charts have been scrutinised to ensure that no other bias or trends exist, which despite plotting within limits can still indicate errors or malpractice in the assay process.

For simplicity the CRM assay results have been tabulated and summarized for drilling below (Table 11.4).

Drilling CRM

A total of 4,441 CRM samples have been included in drilling sample sequences as summarized below in Table 11.5.

The least frequently used (GLG310-5, GLG901-2, and GBMS304-5) were all identified in an early assessment to be unsuitable standards. These three CRMs are primarily base metal standards and

were identified to be unsuitable for the fire assay methodology of gold analyses used most frequently. The below detection limit, certified value of GLG901-2 also discredits this CRM as a suitable gold standard. The use of these three CRMs was ceased in mid-2013.

Table 11.4 Summary Table of all Drilling CRM Results

STD	Count	Cert Value	Result	STDEV	% 3SD Fail (#)	% 2SD Fail (#)	% 1SD Fail (#)
G307-7	370	7.87	7.90	0.167	0%	0%	10.8% (40)
G303-2	486	4.11	4.24	0.106	0%	0%	5.3% (26)
G900-7	359	3.22	3.23	0.081	0%	0%	2.8% (10)
G308-3	344	2.47	2.50	0.055	0%	0%	2% (7)
GBMS304-5	4	1.62	1.52	0.022	0%	0%	75% (3)
G300-9	606	1.53	1.51	0.028	0%	0%	1% (6)
G901-7	22	1.53	1.48	0.030	0%	0%	0%
G908-3	229	1.03	1.03	0.023	0%	0%	2.2% (5)
G907-1	617	0.79	0.77	0.018	0%	0%	0%
G909-6	15	0.56	0.54	0.013	0%	0%	6.7% (1)
G311-7	1,053	0.40	0.39	0.024	0.1% (1)	0%	0.2% (2)
GLG303-1	281	0.164	0.16	0.008	0%	0%	4.3% (12)
GLG310-5	44	0.07983	0.08	0.011	2.3% (1)	11.4% (5)	47.7% (21)
GLG901-2	11	0.00992	0.01	0.002	0%	0%	63.6% (7)
Total	**4,441**			**Number**	**2**	**5**	**140**
				%	**0.0%**	**0.1%**	**3.2%**

11.6 Conclusions

The sample collection and preparation, analytical techniques and security protocols implemented are consistent with standard industry practice, and are suitable for the reporting of exploration results. QA/QC monitoring of assay results is consistent with standard industry practice with the application of robust standards, blank material and duplicate assay analyses. Blank material submissions demonstrate a high quality of sample preparation with no observed contamination.

Implementation of field duplicate sampling and the analysis thereof demonstrates a suitably high level of precision in primary sampling techniques. The high precision of umpire assays provides confidence in primary laboratory results.

Regular submission of certified reference materials across a wide range of expected grades provides further confidence in the quality (precision and accuracy) of the primary laboratory assays and has been assessed to very stringent standards. The sampling procedures, analyses and QA/QC are adequate for and consistent with the style of mineralization.

12. DATA VERIFICATION

Mr Nic Johnson, the author responsible for the project Mineral Resource Estimates visited the Block 14 project area between the 17 and 21 January 2014. The visit included discussions in Khartoum and on site with the exploration personnel who have managed and supervised the exploration programmes to date.

The site visit involved comprehensive data verification, inspections, and reviews of the following:

- Geology and exploration history of the permit areas.
- Exploration model and strategy.
- Current exploration data and exploration procedures.
- Geochemistry and geophysics results.
- QA/QC procedures and control data.
- Data and database management systems.
- Sample handling and storage.
- The sample preparation facility in Atbara.
- Review of procedures, drill data and QA/QC sampling data undertaken as part of the exploration programmes on the Block 14 Project.

12.1 Data Validation

All geochemical and drilling data relating to the Block 14 project is managed through a customised CAE Mining Fusion GDMS database system. Geological and sampling data is validated and uploaded into the database and assay data is merged electronically into the database with QA/QC data checked during import.

Orca provided exports of the data required and has supplied all relevant data to Mr Nic Johnson, the author responsible for the Project Mineral Resource Estimates to undertake various validation checks of the stored information:

- An audit of selected geochemical and drilling samples from original field sampling sheets through to the database was completed and involved cross validation of sample numbers from the field sampling forms through sample submission, laboratory job number generation, and data entry.
- Cross validation of certified reference material numbers between the original field sampling form and the data spread sheets.
- Cross validation of the digital ALS assay certificates against the data spread sheets.
- Confirmation of sampling details and geological logging from field copies to the data spread sheets. Cross validation of sample numbers from the field sampling forms through sample submission, laboratory job number generation, and data entry with cross validation of the digital ALS assay certificates against the data spread sheets.

12.2 Conclusions

Mr Nic Johnson assessed the geological work undertaken, the surface geochemical, and drilling data for the Block 14 project and concluded that all logging, sampling and QA/QC procedures implemented by Orca to date were undertaken to a high standard when compared to industry practice.

Mr Nic Johnson reviewed available QA/QC data relating to the drilling completed and concluded that the quality control samples are unbiased and show a very good level of precision and accuracy, indicating that the sample data is of a high standard and appropriate for use in the mineral resource estimation.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

Various phases of metallurgical testwork have been completed since 2014 which are summarized below. Further detail can be found in the July 2016 Preliminary Economic Assessment.

13.1 ALS Metallurgy Testwork 2014

Seven composite samples were prepared by Orca by combining RC drill chips and half drill core to provide samples of between 20 - 36 kg. The following work was conducted on the seven composite samples:

- Main Zone Fresh 1 & 2, East Zone Fresh 1 & 2, East Zone Oxide 1, East Zone Transition 1 and GSS High Grade Zone.
- Bond Ball mill Work Index and Abrasion Work Index.
- Gravity followed by Cyanide Leach using P_{80} = 75 µm:
  - the gravity recovery was only 10 - 15% and no free gold was observed
  - the gold leached readily in cyanide
  - overall recovery was found to range between 78% and 87%.
- Diagnostic Leaching (Main Zone Fresh 1 and 2):
  - 16 - 18% of the gold was locked in sulphides
  - 2 - 4% of the gold was locked in carbonate minerals.
- QEMScan Mineralogical Analysis Main Zone Fresh 1 at P_{80}=75µm:
  - confirmed that the gold is primarily pyrite hosted
  - the average gold grain size is between 1 and 10 µm i.e. extremely fine and the reason why gravity concentration is not applicable.

13.2 SGS Mineral Services UK Testwork 2015

SGS Minerals Services UK conducted three phases of work for Orca. Samples for GSS comprised quarter HQ and PQ core, while samples from WD were made up of RC chips.

The following testwork was carried out on the East Zone Oxide 2 sample:

- Heap Leach Amenability Bottle roll tests at 25, 12, 6, 3.35, 1.18 mm and 75 µm:
  - gold recoveries ranged between 63% for the coarsest size tested and 92% at the finest size tested at 0.5 g/L CN concentration
  - cyanide consumptions were low and ranged between 0.19 – 0.52 kg/t.

The following testwork was carried out on the East Zone Fresh 3 and Main Zone Fresh 3:

- Direct Whole Rock Leaching at P_{100} = 150, 106, 75, and 53 µm at pH 10.5 – 11.0, 45% solids and 0.5 g/L CN concentration with no carbon additions:
  - gold recoveries on East Zone Fresh 3 were found to increase with decreasing grind size and ranged between 73% at 150 µm and 83% at 53 µm at low CN consumptions rates of between 0.21 - 0.37 kgs/t

- gold recoveries on Main Zone Fresh 3 were found to increase with decreasing grind size and range between 70% at 106 µm and 78% at 53 µm at low CN consumptions rates of between 0.13 - 0.28 kgs/t.

- Carbon in Leach bottle roll tests at P_{100} = 106, 75 and 53 µm:
  - gold recoveries on East Zone Fresh 3 were found to fluctuate and not show the expected trend observed above. The gold dissolution ranged between 73% and 79% at CN consumptions rates of between 0.63 - 0.69 kgs/t
  - gold recoveries on Main Zone Fresh 3 were found to increase with decreasing grind size and range between 74% at 106 µm and 80% at 53 µm at relatively low CN consumptions rates of between 0.43 - 0.56 kgs/t
  - the difference in gold recovery rates was negligible and within experimental error, and so it was concluded that the samples tested did not have "preg robbing" tendencies.

- Gravity (Falcon) followed by cyanide leaching of gravity tailings:
  - East Zone Fresh sample gravity recovery was 15% and 83% overall
  - Main Zone Fresh sample gravity recovery was 28% and 82% overall
  - no free gold was observed, all gravity concentrates were visibly sulphidic material.

As a consequence of the gold association with sulphides (pyrite), the program advanced into a Phase 2 testing using Main Zone Fresh 4 and East Zone Fresh 4.

The following testwork was carried out on the Main Zone Fresh 4 and East Zone Fresh 4 samples:

- Rougher kinetic flotation scoping testwork using 50 g/t Aerofloat 208, 100 g/t PAX and 32 g/t MIBC at P_{100} = 180, 150, 125, 106, & 75 µm:
  - gold recoveries of 93-94% associated with consistent total sulphur recoveries of 98% were observed to a 12 - 15% mass yield to flotation concentrate on all grind sizes tested for East Zone Fresh 4 sample
  - gold recoveries of 90 - 91% at sizes above 125 µm and 93 - 94% below 125 µm were associated with consistent total sulphur recoveries of 98 - 99% were observed to a 12 - 15% mass yield to flotation concentrate
  - the 125 µm grind size was adopted for subsequent reagent optimization testwork

    A simple flotation regime was adopted, no copper sulphate was required and no pH modifier

    Cleaner testing was executed at optimal rougher reagent conditions

    5 - 6% gold loss was consistently incurred to cleaner tailings during cleaning whilst weight deportment to final concentrate as reduced to 7 - 9%.

  - regrinding of cleaner and rougher flotation concentrates to 20 µm using intensive cyanide leach conditions still did not improve overall gold recovery

    Overall gold recovery was 78% for East Zone Fresh 4 and 83% for Main Zone Fresh 4.

Attempts were made to Float East Zone Oxide 2 sample using sulphidizing conditions but the gold recovery was consistently low at between 47 - 58%. It was therefore concluded that oxide domain material is not conducive to froth flotation.

The following testwork was carried out on the Wadi Doum Fresh 1, 2 and 3:

- Whole Rock Leaching at P_{100} = 75 µm at pH 10.5 – 11.0, 45% solids and 0.5 g/L CN concentration:
  - gold recoveries on Wadi Doum Fresh 1 & 2 were very similar at 77 - 78% whereas the much higher grade Wadi Doum Fresh 3 gave rise to elevated gold recovery at 93%
  - Cyanide consumptions were found to range between 0.66 – 0.97 kg/t.
- Rougher Kinetic Froth Flotation using optimized conditions above at P_{100} = 106, 75 and 53 µm grind size:
  - high gold recoveries of 91 - 97% were obtained and consistent total sulphur recoveries of 97 - 99% were observed to a 10 - 32% mass yield to flotation concentrate.

13.3 SGS Mineral Services South Africa Testwork 2016

Samples representing each dominant zone and domain were subjected to the following testwork:

- Comminution Testwork:
  - this again highlighted that the oxide materials were the softest of all the various domains and transitional and fresh were harder, East Zone Fresh being the hardest
  - a similar trend was observed with abrasivity
  - all materials were classified as soft to moderate soft.
- Gold Mineralogy at P_{80} = 75 µm:
  - the majority (>95%) of gold present is native gold (Au-Ag), the remaining minor quantity of gold is present as Petzite (Ag_3AuTe_2) which has much slower leach kinetics than native gold
  - petzite was slightly more prevalent in the East Zone Fresh 4 Sample when compared with Main Zone Fresh 4 sample
  - the typical quantity of Ag in the native gold was 20% in East and Main Zone Fresh 4 samples but was found to be as high as 35% in Wadi Doum sample tested
  - the majority of gold particles were associated with sulphides but Wadi Doum appears to contain more liberated gold particles
  - the gold grain sizes are extremely fine (<40 µm) and Wadi Doum gold grain size is slightly coarser when compared with East and Main Zone Fresh 4 samples
  - gold is either present as free grains or more predominantly as fine (1 – 10 µm) associations and/or inclusions in pyrite
  - the predominant gold carrier in East and Main Fresh 4 samples is pyrite whereas in Wadi Doum, Arsenopyrite, Spahlerite and Galena were also observed carrying gold.

- Diagnostic leaching Testwork at various grind sizes:
  - the diagnostic leaching results highlighted that the refractory component of East and Main Zone Fresh 4 samples was mainly occluded with pyrite and a small amount with host rock
  - this refractory component was found to be around 15 - 20% at P_{80} = 75 µm, decreasing to 10 - 15% at P_{80} = 25 µm and increasing to 25% at P_{80} = 125 µm
  - the Wadi Doum composite typically has 5% less refractory component than East and Main Zone Fresh 4 sample.
- Flash Flotation and CVD Knelson Gold-Sulphide Recovery Pre-concentration testing on samples of material showed that Flash Flotation out performed CVD Knelson Concentration:
  - this probably due to the fact that the gold is very fine grained and associated with pyrite. Flash Flotation is better at recovering fine grained (-20 µm) gold than Knelson / Falcon technology
  - flash flotation was selected as a technology that may provide "upward potential" for selectively pre-concentrating the pyrite carrying gold.
- Heap Leaching of the Oxide domains was further investigated along with associated agglomeration and percolation testwork:
  - typical gold dissolution rates were 60% using 11 – 15 kg/t cement to agglomerate the feed material crushed to minus 12 mm.

13.4 SGS Mineral Services South Africa 2016 and 2017

Four specific areas of metallurgical study have been executed and reported as follows:

- Further Hardness Testing on variability samples from Main, East, Wadi Doum and North East and domains (Oxides, Transition, and Fresh) to further understand physical competence variability and provide input data for grinding modelling studies:
  - six SMC tests were carried out, a sample coming from East Zone Fresh, Transition and Oxide domains, Main Zone Fresh, Transition and Oxide domains and Wadi Doum Zone Fresh domain
  - 56 Modified Bond Ball mill Tests were carried out and eight "Calibration" Full Bond Ball mill Tests as follows:

Domain	Number
Main Zone Fresh	10
Main Zone Transition	6
Main Zone Oxide	6
East Zone Fresh	9
East Zone Transition	6
East Zone Oxide	7
Wadi Doum Zone Fresh	7
North East Zone Oxide	5

- Optimization Flowsheet Development Testwork of Flash Flotation Concept on Composite Samples (Main Zone Fresh Domain 6 and East Zone Fresh Domain 6):
  - variability testing of the optimized Flash Flotation Operating Parameters on 33 samples selected from East Zone Fresh Domain (eight) and Transition Domain (five), Main Zone Fresh Domain (nine)and Transition Domain (five) and Wadi Doum Zone Fresh Domain (six)
  - this data was utilized by Lycopodium to provide a Trade off Study comparing Whole Rock Leaching with and without Flash Flotation.

13.4.1 SGS South Africa – Hardness Testing

Six SMC tests were conducted and the results are presented in Table 13.1.

Table 13.1 Summary of SMC Test Results

ID	DWi	DWi	M_{ia}	M_{ih}	M_{ic}	A	b	SG	t_a	SCSE
	kWh/m	*%*	*kWh/t*	*kWh/t*	*kWh/t*					*kWh/t*
East Zone Fresh	7.58	64	20.9	15.9	8.2	75.9	0.48	2.77	0.34	10.47
East Zone Oxide	2.40	7	9.2	5.5	2.9	68.6	1.53	2.52	1.08	6.73
East Zone Trans	6.04	43	17.7	12.8	6.6	69.6	0.65	2.73	0.43	9.39
Main Zone Fresh	4.38	22	13.0	8.8	4.6	61.0	1.07	2.87	0.59	8.22
Main Zone Oxide	2.78	9	10.8	6.7	3.5	66.9	1.30	2.41	0.93	7.22
Wadi Doum Fresh	5.53	36	16.0	11.4	5.9	65.2	0.78	2.81	0.47	9.05

Table 13.2 Classification of SMC Test A and B Parameters

Classification	Very Hard	Hard	Mod. Hard	Medium	Mod. Soft	Soft	Very Soft
A·b	<30	30 - 38	38 - 43	43 - 56	56 - 67	67 - 127	>127
ta	<0.24	0.24 - 0.35	0.35 - 0.41	0.41 - 0.54	0.54 - 0.65	0.65 - 1.38	>1.38

- East Zone Fresh is the only domain type that is ranked as hard.
- East Zone Transitional and Wadi Doum Zone Fresh are ranked as medium hardness.
- Main Zone Fresh is ranked as moderate to soft hardness.
- Main Zone Oxides and East Zone Oxides are ranked as soft.

Figure 13.1 shows the rank percentile plot of all the Modified Bond Ball mill Work Index results. Figure 13.2 shows good correlation of the Modified Bond Ball mill Work Index and the Full Bond Ball mill Work index, so that the Modified Bond Ball mill results can be used for design inputs by Lycopodium.

Figure 13.1 Rank Percentile Plot of all Modified Bond Ball Mill Work Index Results

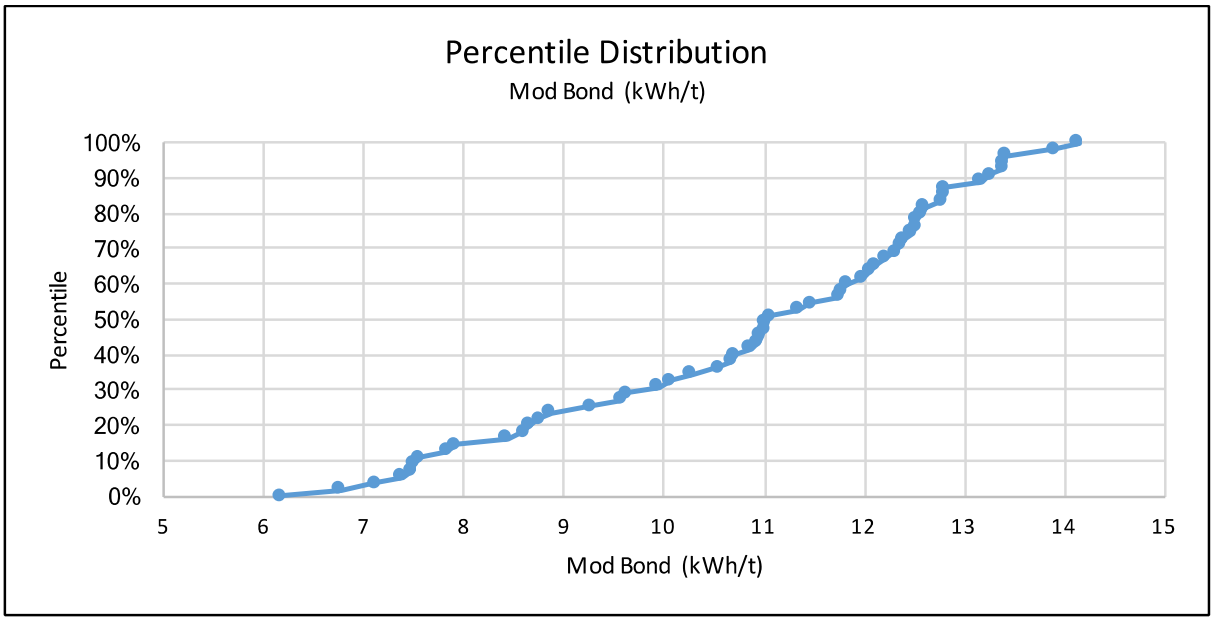


Figure 13.2 Modified Bond BWi Vs Full Bond BWi Calibration

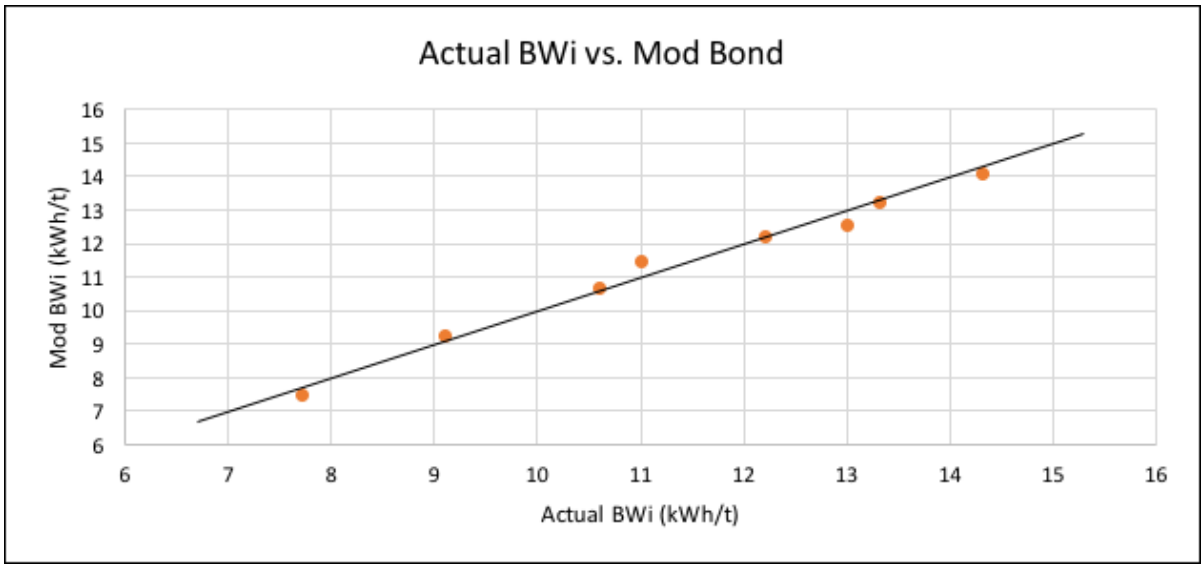
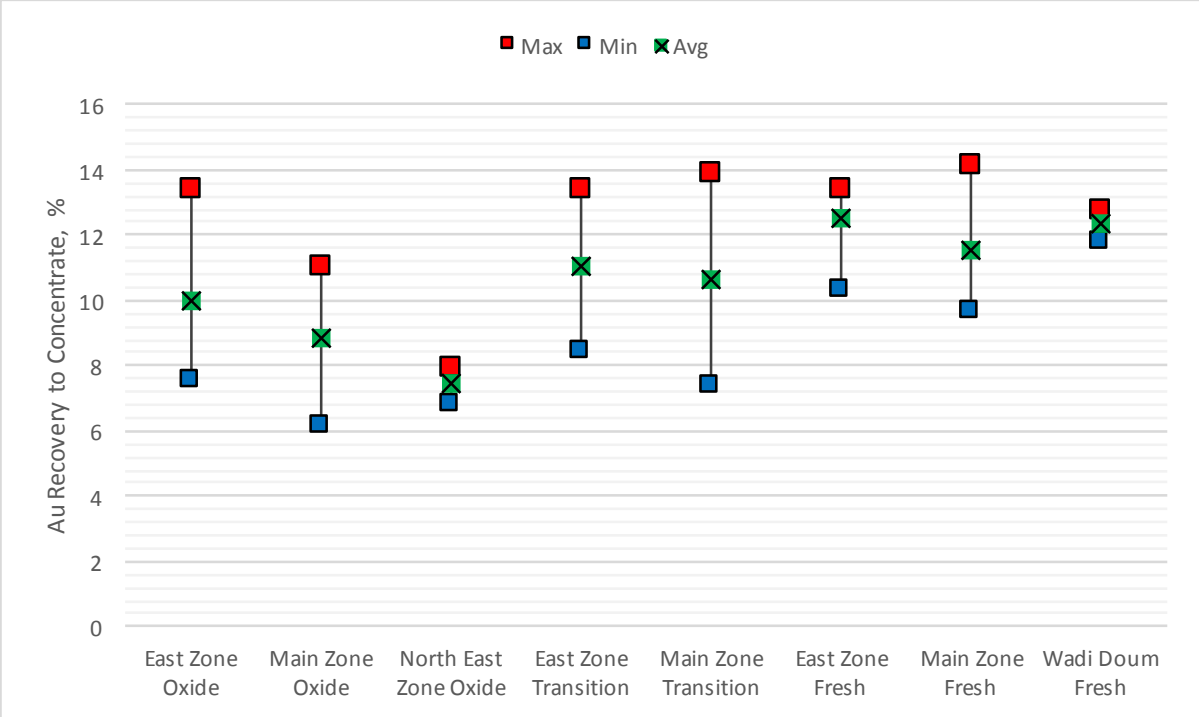


Figure 13.3 shows the summary of the range of Bond Ball mill Hardness values observed:

- It is clear that Main Zone Transition Domain has the greatest variability followed closely by East Zone Oxide Domain.
- North East Oxide Domain and Wadi Doum Zone Fresh Domain vary the least.
- East Zone Fresh Domain is on average the hardest domain, and North East Zone Oxide domain on average the softest domain.

Figure 13.3 Summary of Bond Ball Mill Work Index Variability by Domain



13.4.2 SGS South Africa – Flash Flotation Investigation

In order to evaluate the effectiveness of flash float pyrite in the grinding circuit metallurgical testwork was conducted on two new samples "East Zone Fresh 6" and "Main Zone Fresh 6".

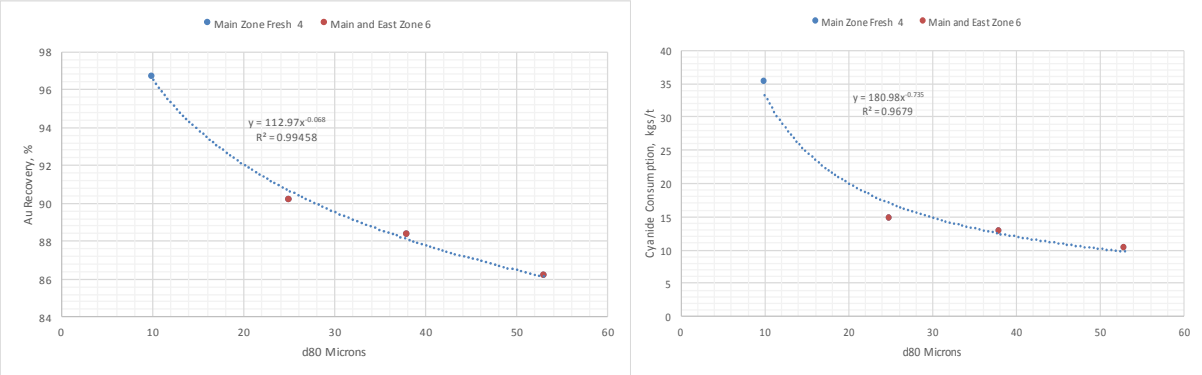
The optimization testwork was conducted on East Zone Fresh 6 Sample and then verified using Main Zone Fresh 6 sample that was ground to P_{80} = 425 µm which was used as a size to simulate typical grinding circuit circulating load and the maximum potential size at which particles are receptive to froth flotation.

The conditions for Flash Flotation tested were:

- 200 g/t PAX, 40 g/t MIBC and using 3 min residence time was found to provide the optimum operating conditions.
- Copper sulphate additions have no effect of flotation performance and so no copper sulphate was subsequently used.
- The flash flotation tailings were reground from P_{80}= 425 µm to P_{80} = 75 µm to simulate regrinding of the flash flotation tailings in the plant.

Figure 13.4 shows a good relationship between gold dissolution rates and cyanide consumption rates using intensive (20 g/L CN) concentration with 1 kg/t Lead Nitrate dosing.

Figure 13.4 Effect of Regrind Size on Gold Dissolution and Cyanide Consumptions



Subsequent to this optimization work, 32 variability samples were tested to ascertain the variability of the fresh domain material types to Flash Flotation and the secondary objective was to understand whether transition samples were amenable to flash flotation. Oxide samples are not amenable to simple flash flotation techniques.

Figure 13.5 Flash Flotation Variability Plots of all Samples Tested

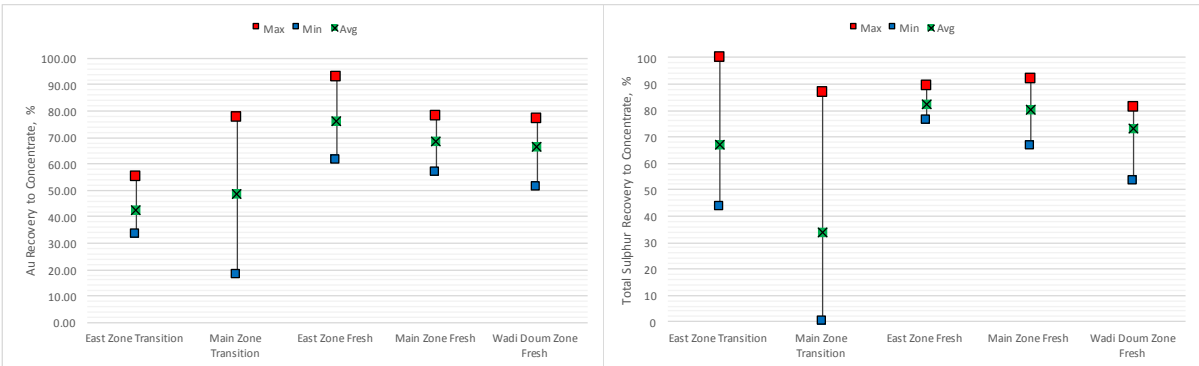


Figure 13.5 above clearly shows that East Zone responds the best of the fresh domains, Wadi Doum is the worst. The East and Main Transition zones do not respond as well, and Main transition zones appear to vary the most.

The implementation of Flash Flotation in conjunction with regrinding of the flash flotation concentrate and subsequent intensive leaching of the flash flotation concentrates can clearly have potential to increase overall recoveries on fresh material types. However, the incremental capital and operating costs, especially sodium cyanide consumption rates at the much finer grind sizes renders flash flotation commercially unviable.

13.5 SGS Vancouver – Whole Material Leach Optimization by Size

Twenty-four variability samples, representing a range of gold grades, were selected from each zone and dominant domain to improve the understanding and basis for primary grind selection.

13.5.1 Head Assays

A full chemical Analysis is shown below in Table 13.3 and Table 13.4 below.

Table 13.3 Head Assays

Sample ID	Assays (g/t)						Assays (%)		
	Au*	Ag	Hg	As	Zn	Te	S	Cu	Fe
Main Fresh 10	1.93	1.70	1.65	196	400	0.33	2.85	0.01	6.80
Main Fresh 11	3.29	1.80	1.49	110	500	0.44	2.40	0.02	5.74
Main Fresh 13	2.73	2.50	0.19	138	<300	1.28	3.94	0.01	5.91
Main Oxide 2	1.42	3.10	3.58	197	1200	1.31	0.10	0.01	4.25
Main Oxide 3	1.52	7.40	0.61	221	<300	0.58	0.13	0.00	5.80
Main Oxide 4	1.48	0.90	0.40	90	<300	0.47	<0.01	0.01	4.68
Main Transition 2	1.97	7.50	10.2	229	2800	3.35	3.92	0.02	4.41
Main Transition 4	0.70	0.90	0.54	55	400	0.31	1.25	0.01	5.61
Main Transition 6	2.94	7.00	1.29	237	1400	4.65	0.19	0.01	6.67
NEZ Oxide 1	3.02	2.50	0.31	226	400	2.40	1.28	0.01	8.73
NEZ Oxide 2	2.06	0.80	0.72	168	500	0.58	0.23	0.02	5.65
NEZ Oxide 3	1.30	1.40	0.57	84	600	0.54	0.16	0.02	5.77
Wadi Fresh 7	2.12	14.8	0.25	360	6400	0.44	8.00	0.01	8.17
Wadi Fresh 11	2.39	6.90	0.16	256	4100	<0.05	4.08	0.01	3.80
East Fresh 11	1.09	3.60	1.99	27	2200	2.07	3.37	0.01	3.83
East Fresh 13	1.87	1.90	3.07	21	1700	1.38	2.45	0.02	3.48
East Fresh 16	2.40	2.90	2.00	34	900	2.01	2.82	0.01	4.06
East Oxide 8	0.72	1.20	1.07	29	500	1.24	0.23	0.01	4.43
East Oxide 9	2.81	0.70	0.18	52	600	0.47	<0.01	0.02	4.79
East Oxide 11	1.44	1.70	0.26	18	400	1.78	0.20	0.00	3.52
East Transition 4	1.21	1.60	1.09	37	800	1.31	1.76	0.01	3.94
East Transition 5	0.78	1.10	0.55	58	<300	1.35	0.55	0.00	4.66
East Transition 7	1.61	2.70	1.12	31	700	2.44	2.44	0.01	3.73

***Average of 3 to 5 analyses**

Table 13.4 Head Assays – Whole Rock Analysis

Sample ID	Assays (%)													
	LOI	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	K_2O	Na_2O	TiO_2	MnO	P_2O_5	Cr_2O_3	V_2O_5	Sum
Main Fresh 10	8.45	45.5	15.3	9.57	3.59	6.87	5.00	0.09	0.93	0.63	0.24	<0.01	0.04	96.2
Main Fresh 11	9.00	50.5	13.4	8.00	3.71	6.46	3.48	0.98	0.79	0.44	0.21	0.01	0.04	97.0
Main Fresh13	6.67	52.3	11.4	8.46	2.70	6.79	3.04	1.78	0.85	0.46	0.37	<0.01	0.03	94.8
Main Oxide 2	5.22	67.7	10.5	6.11	1.48	3.12	3.90	0.12	0.54	0.29	0.17	0.02	0.03	99.2
Main Oxide 3	3.41	70.9	11.5	8.15	0.53	0.28	3.36	0.16	0.73	0.46	0.15	0.02	0.04	99.6
Main Oxide 4	4.34	62.2	14.7	6.66	1.28	2.40	5.66	0.87	0.77	0.39	0.30	0.01	0.02	99.6
Main Transition 2	4.81	65.9	8.8	6.19	0.85	4.21	2.43	1.01	0.53	0.39	0.18	0.02	0.03	95.3
Main Transition 4	14.4	45.1	11.0	7.85	3.71	10.3	3.57	0.13	0.67	0.48	0.15	0.02	0.03	97.4
Main Transition 6	5.81	57.1	13.9	9.52	1.36	3.15	4.88	0.63	1.07	1.23	0.38	<0.01	0.05	99.1
NEZ Oxide 1	8.07	54.9	15.2	12.3	0.79	0.63	4.70	0.64	1.00	0.21	0.29	<0.01	0.04	98.8
NEZ Oxide 2	5.78	56.4	15.9	8.08	0.92	3.59	3.89	2.39	0.87	0.70	0.37	<0.01	0.03	98.9
NEZ Oxide 3	4.85	58.9	15.5	8.25	0.77	2.69	4.70	1.29	0.92	0.45	0.37	<0.01	0.04	98.8
Wadi Fresh 7	7.69	46.9	15.8	11.2	3.07	3.20	3.76	1.52	0.76	0.61	0.18	0.01	0.03	94.8
Wadi Fresh 10	7.77	54.3	14.0	6.36	3.27	5.70	3.33	0.19	0.45	0.60	0.10	0.01	0.02	96.2
Wadi Fresh 11	5.18	64.0	14.1	5.24	1.02	1.91	3.74	0.79	0.27	0.50	0.08	<0.01	<0.01	96.9
East Fresh 11	3.94	62.4	12.6	5.40	0.96	3.38	1.67	4.76	0.61	0.35	0.21	<0.01	<0.01	96.3
East Fresh 13	4.28	61.8	13.0	4.91	1.65	2.68	5.37	2.16	0.63	0.32	0.20	0.02	<0.01	97.0
East Fresh 16	5.15	58.6	14.4	5.74	1.52	3.61	3.43	3.12	0.68	0.33	0.22	0.01	0.03	96.8
East Oxide 8	6.06	59.1	13.2	6.30	0.84	5.84	3.13	2.82	0.70	0.47	0.24	<0.01	0.03	98.8
East Oxide 9	3.12	63.6	15.9	6.68	0.84	0.36	6.45	1.02	0.78	0.43	0.24	0.01	0.01	99.5
East Oxide 11	3.25	72.5	12.0	4.99	0.43	0.61	2.56	2.11	0.59	0.06	0.17	0.01	0.01	99.3
East Transition 4	5.16	62.0	13.6	5.61	0.80	3.66	2.99	2.43	0.68	0.35	0.22	0.01	0.01	97.5
East Transition 5	4.08	63.3	14.1	6.64	1.05	2.42	4.30	1.30	0.72	0.26	0.30	<0.01	0.01	98.5
East Transition 7	2.89	61.8	14.5	5.31	0.36	2.29	3.44	5.76	0.70	0.32	0.25	<0.01	<0.01	97.6

13.5.2 Cyanide Leach Variability Bottle Roll Testwork

- Grind Optimization Whole Rock Cyanide Leach Variability Testing on 23 samples coming from East Zone Fresh, Transition and Oxide domains, Main Zone Fresh, Transition and Oxide domains, and Wadi Doum Zone Fresh domains:
  - each of the 23 samples were tested at four different grind sizes (P_{80} 53, 75, 106, and 125 µm) and the main objective of this testwork was to provide data to optimize the primary grind size
  - a total of 87 whole rock cyanide leach variability tests were conducted as follows:

Domain	Number
Main Zone Fresh	10
Main Zone Transition	14
Main Zone Oxide	12
East Zone Fresh	9
East Zone Transition	11
East Zone Oxide	11
Wadi Doum Zone Fresh	6
North East Zone Oxide	14

  - the output data from this study was utilised in the grind optimization study.

The variability testwork was conducted using the following parameters which were kept the same to analyse the effect of primary grind size:

- 40% solids.
- pH of 10.5.
- 0.5 g/L NaCN (maintained).
- Retention time of 48 hr.
- Vancouver tap water.
- Dissolved oxygen kept above 3 mg/L DO with air as needed.

Intermediate solution samples were extracted at 2, 4, 8, 24, and 32 hr and submitted for gold analysis. At completion of each test, the filtrate was titrated using silver nitrate and oxalic acid to establish the cyanide and lime consumption respectively. The residue and filtrate were reserved for analysis of gold, silver, copper, and mercury.

13.5.3 Gold Dissolution Trend with Grind Size and Variability

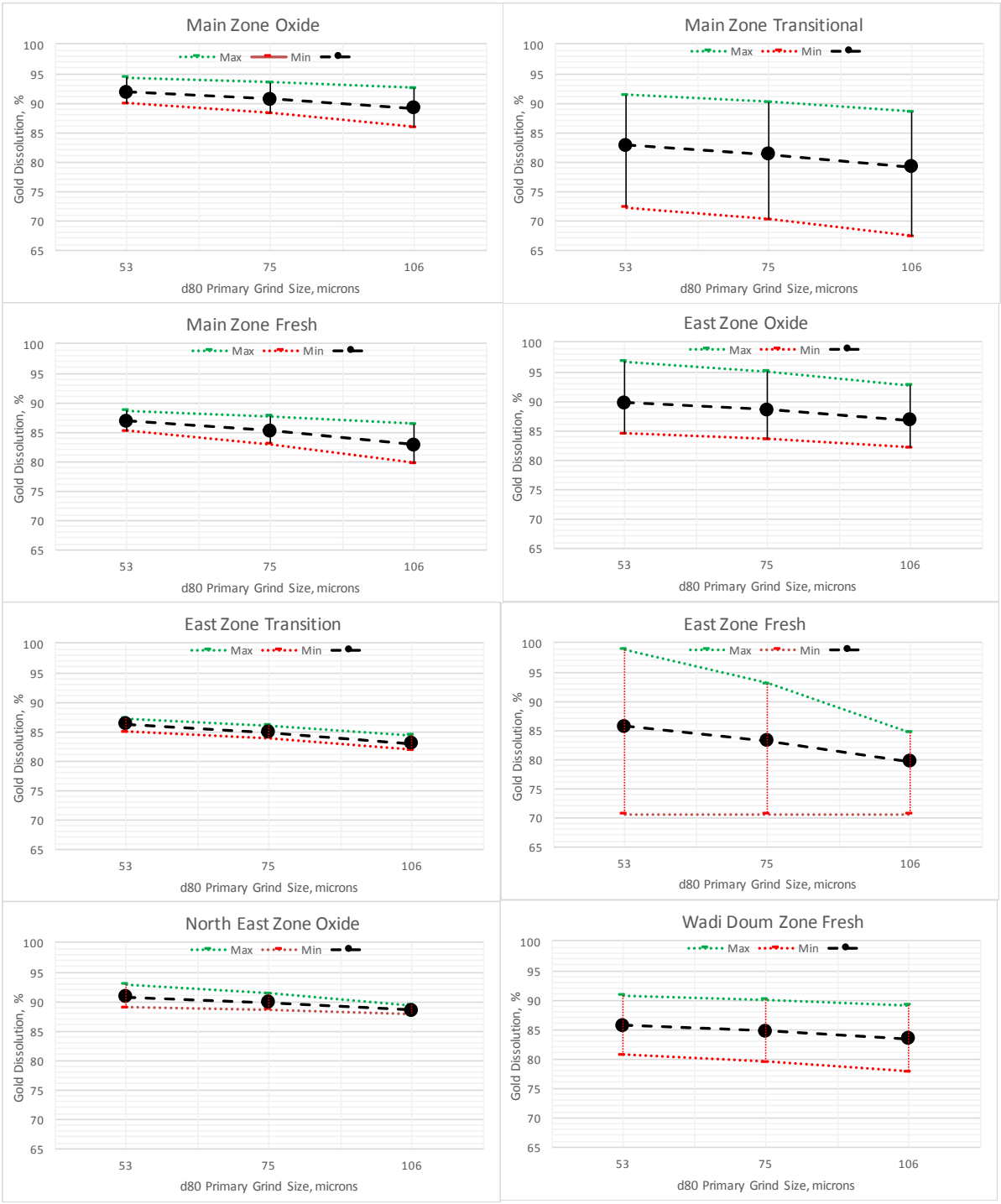
The individual sample plots of gold dissolution versus grind size for the 48 hr gold dissolution response for each of the eight domains tested as a function of grind size and shows maximum, minimum, and average data are shown in Figures 13.6.

The trend line equations between gold dissolution and grind size P_{80} for each individual samples tested was used to produce gold dissolution data for each sample at whatever size selected. If the trend line equation r^2 was poor, the average value was taken.

In two domains, East Zone Transition and North East Zone Oxide the gold dissolution variability between the three samples tested was found to be minimal and because this data provided better r^2 data, the trend line between overall samples was taken as opposed to the average as was the case

with all other six domains. This strategy ensured that the strongest trend relationships were employed at all times.

Figure 13.6 Gold Dissolution Trends and Variability



Conclusions - Gold Dissolution Trend with Grind Size and Variability

- All 23 samples from the eight domains tested showed a generic trend for gold dissolution to increase with decreasing grind size.

- The fresh domains in all three zones appear on average to be the most sensitive to grind size (typically 5% gold dissolution loss between 106 and 53 µm) and some samples demonstrated quite extreme sensitivity. For example, in East Zone Fresh 13 sample, 14% gold dissolution was observed:
  - this phenomenon is likely to be due to inherent more tightly bound fresh fine gold mineralogy when compared with the more oxidized gold in oxides, which will increase fine gold grain exposure.
- The highest gold dissolutions (typically >90%) were obtained on the oxide domains.
- East Zone Transition and North East Zone Oxide vary the least, whereas Main Zone Transition and East Zone Fresh vary the most.

13.5.4 Cyanide Consumption Trend with Grind Size and Variability

In three domains, Wadi Doum Zone Fresh, North East Zone Oxide and Main Zone Fresh the cyanide consumption was found to be insensitive to grind size and variability was minimal. However, in three domains, Main Zone Transition, East Zone Fresh, and East Zone Oxides the variability was greatest and the sensitivity of grind size greatest.

- All 23 samples from the eight domains tested showed a generic trend for cyanide consumptions to increase with decreasing grind size.
- The highest cyanide consumptions (typically 1.40 – 1.60 kg/t) were observed on Wadi Doum Zone Fresh and Main Zone Fresh.
- The East Zone Oxide domain was by far the most grind sensitive domain tested in terms of cyanide consumption, the average difference between 106 and 53 µm being 0.50 kg/t. Most other domains ranged between virtually zero and 0.30 kg/t difference.
- The domains with the greatest variability in cyanide consumptions were Main Zone Oxide, Main Zone Transition and East Zone Oxide.
- The domains with the least variability in cyanide consumptions were coincidentally mainly the highest consuming domains of main Zone Fresh, Wadi Doum Fresh, along with North East Zone Oxide.

13.5.5 Gold Dissolution Relationship with Rock Chemistry

Referring to Figure 13.14 below, there is no relationship between gold dissolution and gold g/t in feed for the two main zones where sufficient samples have been tested to formulate a statistical view.

This suggests that the gold dissolution rate is potentially mineralogically constrained by fine association and inclusion with pyrite irrespective of gold grade.

Figure 13.7 Relationship between Gold Dissolution and g/t Au in Feed

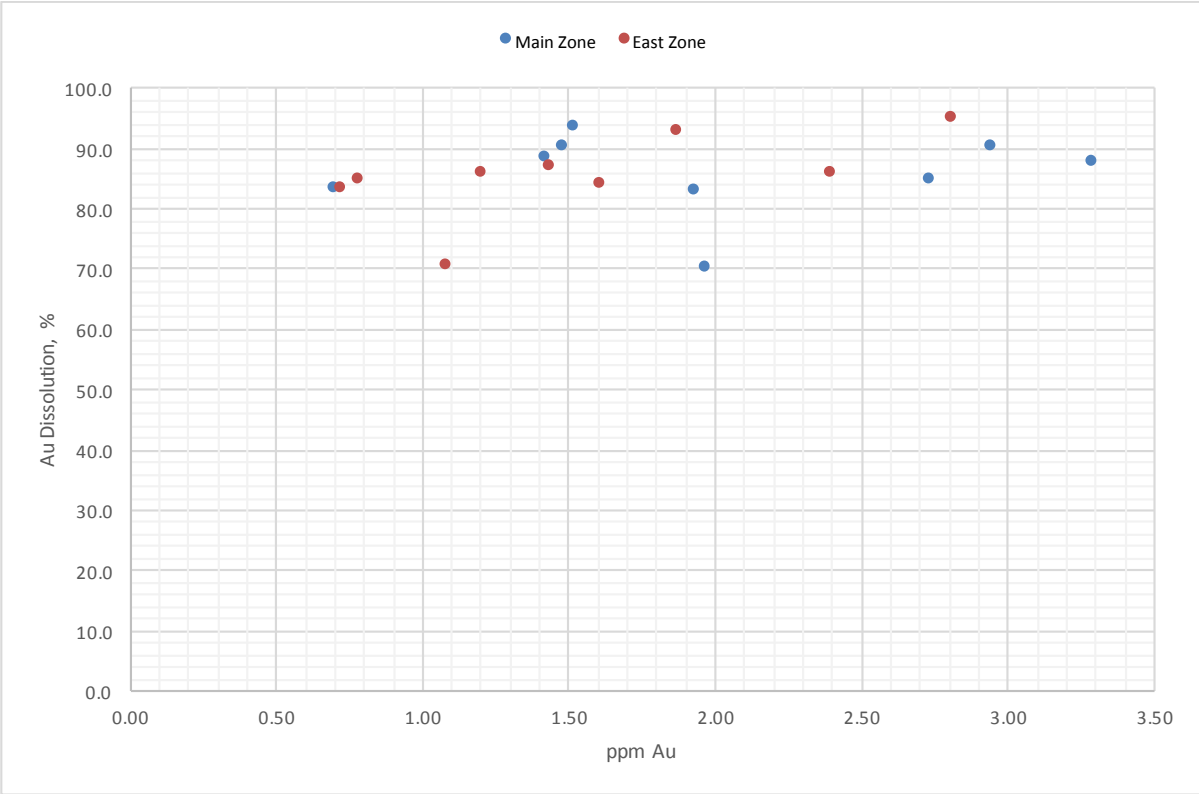
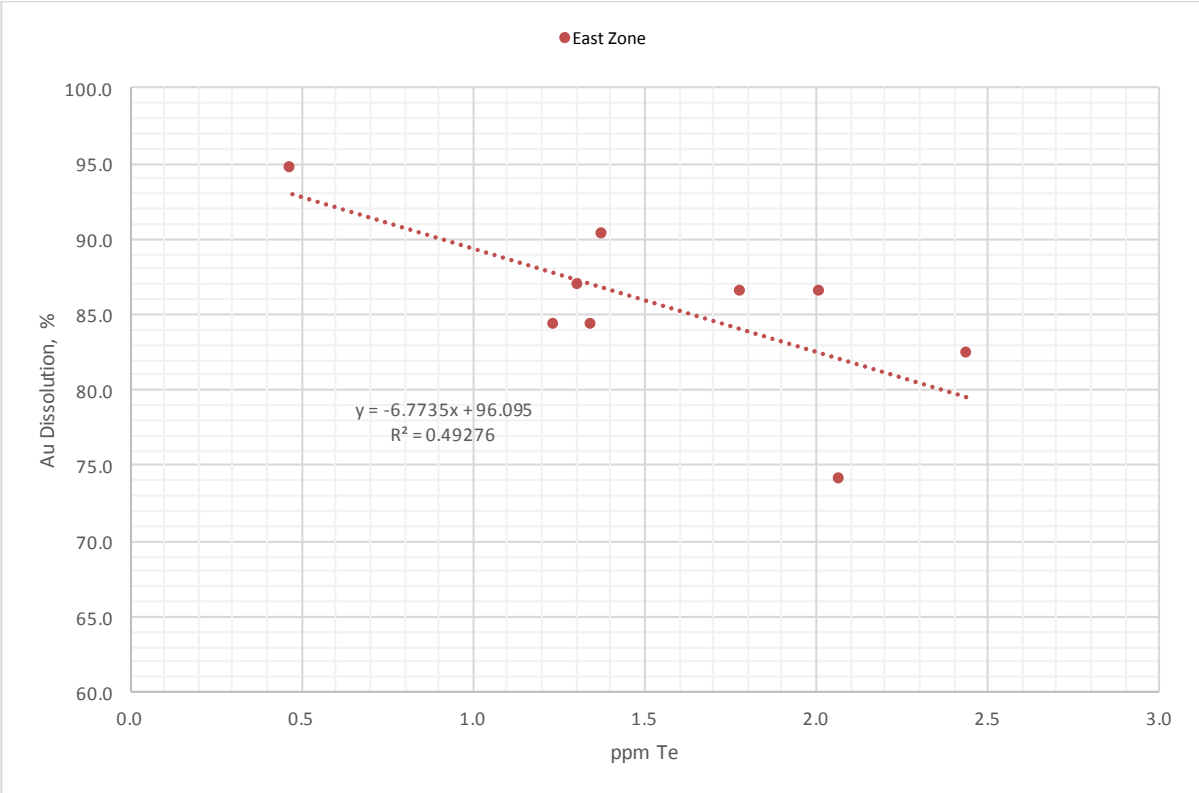


Figure 13.8 Relationship between Gold Dissolution and ppm Te for East Zone Types



Referring to Figure 13.15 above, there is a moderate relationship between gold dissolution and ppm Tellurium in the feed in East Zone material only. This suggests that the East Zone gold dissolution may be partially constrained by the presence of Petzite as highlighted by studies conducted previously and reported above in Section 13.3.

No other relationships were found between gold dissolution and material chemistry.

Conclusions – Relationships between Gold Dissolution and Material Chemistry

- The lack of any relationship between gold dissolution and g/t Au in the feed suggests that the metallurgy is driven more by mineralogical constraints associated with gold sulphide associations.
- The East Zone materials clearly have some constraining Tellurium mineralogical feature in the form of Petzite.
- The Main Zone materials do not appear to have any relationship with material chemistry.
- These relationships will be further evaluated as part of the DFS Study.

13.6 SGS Vancouver – Bulk Leach Tailings Sample Production and Effect of Site Water

- Bulk Whole Rock Leaching followed by a carbon adsorption stage to produce simulated leach residues for tailings thickener and filtration testing:
  - this testwork was conducted on "Domain composite samples" generated from the various variability samples from each domain of oxides, transitional, and fresh
  - this testwork was conducted with a sample of saline water sourced from site
  - this site water was also used to conduct comparative cyanide bottle roll tests to ascertain the effects on gold recovery, cyanide and lime consumption rates.

The head assays reported in Table 13.5 below.

Table 13.5 Head Assays of the Master Domain Composite Samples

Composite	Assays (g/t)			Assays (%)		
	Au*	Ag	Hg	Cu	Fe	S
Oxide	2.80	2.0	0.75	<0.01	4.51	0.14
Transition	1.31	2.6	3.02	<0.01	4.66	1.55
Fresh	2.02	2.4	1.29	<0.01	4.48	2.99

***Average of triplicate assays**

Charges of 11 kg were leached at a target grind size P_{80} = 106 µm at 40% solids, pH 10.5, 0.5 g/L CN maintained, for 32 hr leach residence time followed by 4 hr carbon residence time at 15 g/L carbon concentration. DO was tracked and kept above 3 mg/L with air if required. Kinetic samples were taken at 2, 4, 8, and 24 hr.

Table 13.6 Composite Bulk Leach Gold Results

Composite No. / Test No.	P_{80} (µm)		Au Extraction (%)					Carbon Au Extraction (%)		Residue Au (g/t)	Head Grade		Reagent Cons (kg/t)	
	Target	Actual	2 h	4 h	8 h	24 h	32 h	Loaded Carbon	Barren PLS		Au (calc.) (g/t)	Au (dir.) (g/t)	NaCN	CaO
Oxide – BR1	106	78	80	82	84	90	**93.2**	**78.6**	**10.5**	0.18	1.83	2.00	0.52	1.81
Transition – BR2	106	68	69	72	77	84	**86.1**	**66.0**	**13.0**	0.26	1.24	1.31	0.58	2.01
Fresh – BR3	106	73	73	74	77	87	**88.2**	**73.0**	**8.7**	0.36	1.94	2.02	0.63	2.04

Table 13.7 Composite Bulk Leach Silver, Copper and Mercury Results

Composite No. / Test No.	Ag			Cu Extraction (%)	Hg Extraction (%)
	Extraction (%)	Residue (g/t)	Head (calc.) (g/t)		
Oxide – BR1	17.1	1.5	1.8	2.1	7.0
Transition – BR2	47.6	1.1	2.1	8.8	7.8
Fresh – BR3	53.2	0.8	1.7	5.1	3.7

The resultant adsorption stage pulp was used for thickener and filter size testwork and Geochemistry.

The same Master Domain Composite samples were used for effect of site water testing. The site water that was used emanated from HA8 aquifer.

Table 13.8 Summary of Results using Site Water

Composite No. / Test No.	P^{80} (µm)		Water Source	Au Extraction (%)					Residue Au (g/t)	Head Grade		Reagent Cons (kg/t)	
	Target	Actual		2 h	4 h	8 h	24 h	32 h		Au (calc.) (g/t)	Au (dir.) (g/t)	NaCN	CaO
Oxide – CN-1	106	70	Van Tap	76	84	86	90	**92.2**	0.15	1.86	2.09	0.41	0.46
Oxide – CN2	106	109	Site	74	83	81	81	**88.5**	0.20	1.70		0.32	1.80
Transition – CN-1	106	96	Van Tap	20	68	73	79	**80.6**	0.24	1.24	1.31	1.06	0.42
Transition – CN-2	106	89	Site	59	67	70	73	**82.2**	0.23	1.29		0.56	2.23
Fresh – CN-1	106	68	Van Tap	47	70	73	81	**81.8**	0.37	2.01	2.02	0.89	0.28
Fresh – CN-2	106	119	Site	59	71	72	76	**81.6**	0.39	2.09		0.70	1.94
Fresh – CN-3	106	133	Van Tap	57	70	74	78	**82.5**	0.36	2.03		0.93	0.96
Fresh – CN-4	103	127	Van Tap	71	79	80	84	**81.5**	0.37	1.97		0.52	0.46
Fresh – CN-5	103	103	Van Tap	72	75	75	80	**83.8**	0.36	2.19		0.45	0.41

Full chemical analysis and Cyanide speciation was performed on the resultant final barren solution.

13.6.1 Conclusions of the Bulk Cyanide Leach and Effect of Water Testwork

- The effect on the residue assays is within experimental error for the Master Transition and Fresh Composites and there is therefore concluded that the effect of site water has no effect on the gold dissolution on these domain samples. However, the Master Oxide Composite results suggest that gold dissolution may be lower.
- The cyanide consumptions were lower using site water.
- The lime consumptions were significantly higher and this due to the fact that saline waters tend to buffer pH.

13.7 Thickener Sizing Testwork

The pulp produced from the testwork described in Section 13.6 above, were used for thickener and filter sizing testwork.

The thickener sizing testwork results are summarized below in Table 13.9.

Table 13.9 Summary Results of Thickener Sizing Testwork

High Rate Thickener Sizing	FrMC Tailings	TrMC Tailings	OxMC Tailings
Tested Material d(80), microns	52	68	43
Feed Suspended Solids Conc., wt%	12	12	10
Flocculant Dosage, g/t	20	20	30
Design Unit Area, m^2/tpd	0.04	0.04	0.05
Min. Mud Residence Time Required, hrs	2.0	1.5	1.5
Design Underflow Solids, wt%	65	65	63
Recommended Yield Stress for Rake Design, Pa	50	50	50
Design Overflow Clarity, ppm	<50	<50	<50

13.7.1 Conclusions Thickener Sizing Testwork

- The Oxide Master Composite requires the largest unit area and drives the design sizing criteria.
- An anionic polyacrylamide flocculant with a medium molecular weight and very low charge density (SNF AN-905SH) produce the best overflow clarity and settling velocities for all three tailings samples.
- Recommended flocculant dosage for the FrMC and TrMC tailings samples is 20 g of flocculant per metric tonne of dry solids.
- The flocculant dosage for the OxMC sample is 30 g of flocculant per metric tonne of dry solids.
- Flux testing showed the optimum feedwell suspended solids concentration for flocculation to be 12 wt% for the FrMC and TrMC samples and 10 wt% for the OxMC sample.
- The highest underflow solids concentration that can be produced in the FrMC and TrMC thickeners is approximately 72 wt%.
- The highest underflow solids concentration that can be produced in the OxMC thickener is approximately 68.5 wt%.

13.8 Filter Sizing Testwork

The filter sizing testwork results are summarized below in Table 13.10.

Table 13.10 Summary of Filter Sizing Testwork

Process Parameter	FrMC Tailings	TrMC Tailings	OxMC Tailings
Tested Material d(80), microns	52	68	43
Chamber Type	Recessed	Recessed	Recessed
Feed Solids Concentration, wt%	64	64	60
Filter Media	24 oz Felt	24 oz Felt	24 oz Felt
Feed Pressure, bar	10	10	10
Drying Pressure, bar	6.9	6.9	6.9
Chamber Thickness, mm	65	65	65
Cake Consolidation Time, min	0.4	0.4	0.7
Cake Blow, min	4.8*	3.5*	6*
Design Cake Moisture, wt%	15	15	15
Filtration Rate, $kg/m^2/hr$	320	375	280
Dry Cake Density, kg/m^3	1,550	1,560	1,580

*** Cake blow times will vary depending on the filter cake moisture requirements. The filtration rate will be affected by any change to this parameter.**

13.8.1 Conclusions Filter Sizing Testwork

- The Oxide Master Composite has the lowest unit filtration rate and drives the design criteria.
- There was only marginal benefit cake moisture content and filtration rate when high filter feed pressure was simulated and is not recommended in full scale design.
- Membrane squeeze simulations did not result in a reduction in cake moisture content and significantly reduced the filtration rates of each material.

14. MINERAL RESOURCE ESTIMATES

14.1 Introduction

In February 2017, MPR Geological Consultants Pty Ltd (MPR) estimated gold Mineral Resources for the GSS and WD deposits. The estimates incorporate results from RC and diamond drilling up to December 2016.

As with previous estimates for GSS and WD, for the current study MPR estimated recoverable resources by Multiple Indicator Kriging with block support correction to reflect open pit mining selectivity, a method that has been demonstrated to provide reliable estimates of gold resources recoverable by open pit mining for a wide range of mineralization styles.

The current estimates are based RC and diamond drilling data supplied by Orca in February 2017. Details of this sampling and assay are described in previous sections of this report. Modifications to the supplied sampling information included adjusting down-hole survey entries which showed unrealistic down-hole deviations, such as azimuth changes of greater than 50° in 5 m.

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

The Mineral Resource estimates have been classified and reported in accordance with NI 43-101 and the classifications adopted by CIM Council in November 2004.

The Qualified Person responsible for the Mineral Resources is Mr Nic Johnson, who is a full time employee of MPR Geological Consultants Pty Ltd and a member of the Australian Institute of Geoscientists. Mr Johnson visited the project site between 17 and 21 January, 2014.

14.2 Estimation of GSS Resources

14.2.1 Resource Dataset

Relative to the February 2016 dataset, the GSS sampling database available for the current review includes an additional 45 RC holes and eight diamond cored holes drilled between August and December 2016.

The 2016 diamond holes were primarily drilled to provide samples for metallurgical test-work and generally targeted areas previously tested by relatively close spaced drilling.

The 2016 RC drilling in-filled several areas of previously tested by comparatively broad-spaced drilling, increasing confidence in estimated resources for these areas. Approximately 44% of these holes were drilled in the far north-east zone substantially increasing the amount of drilling in this area. Although locally significant, for the other resource areas the 2016 RC drilling did not significantly increase the size of the estimation dataset.

The current estimates are based on two metre down-hole composited gold grades from RC and diamond drilling with un-sampled intervals generally assigned gold grades of 0.00 g/t. Surface rock chip and trench samples were excluded from the resource dataset, along with peripheral drillholes not relevant to the current estimate.

The compiled resource dataset comprises 33,306 composites with gold grades ranging from 0.00 to 122.3 g/t, and averaging 0.50 g/t. The dataset is dominated by composites from RC holes which represent 90% of mineralized domain composites. Holes drilled since completion of the February 2016 resource estimates provide around 7% of the combined dataset including 9% of composites within mineralized domains.

14.2.2 Geological Interpretation and Domaining

Drilling to date has delineated several distinct bodies of gold mineralization at GSS. In general, the transition from gold mineralization to barren host rock is characterised by diffuse grade boundaries. The interpreted spatial continuity and tenor of the gold varies markedly throughout the resource area.

The current estimates are based on 12 mineralized domains interpreted by MPR on the basis of composited gold grades with reference to surface mapping and trench sampling. Domain boundaries were digitised on cross-sections, snapped to drillhole traces where appropriate, then wire-framed into three-dimensional solids.

The mineralized domains are designated as Domains 2 to 13, with Domain 1 representing generally un-mineralized composites not captured by the mineralized domain wire-frames. In addition to the mineralized domains, the resource area was subdivided into sub-areas of varying drilling orientation and spacing to facilitate assignment of estimation parameters.

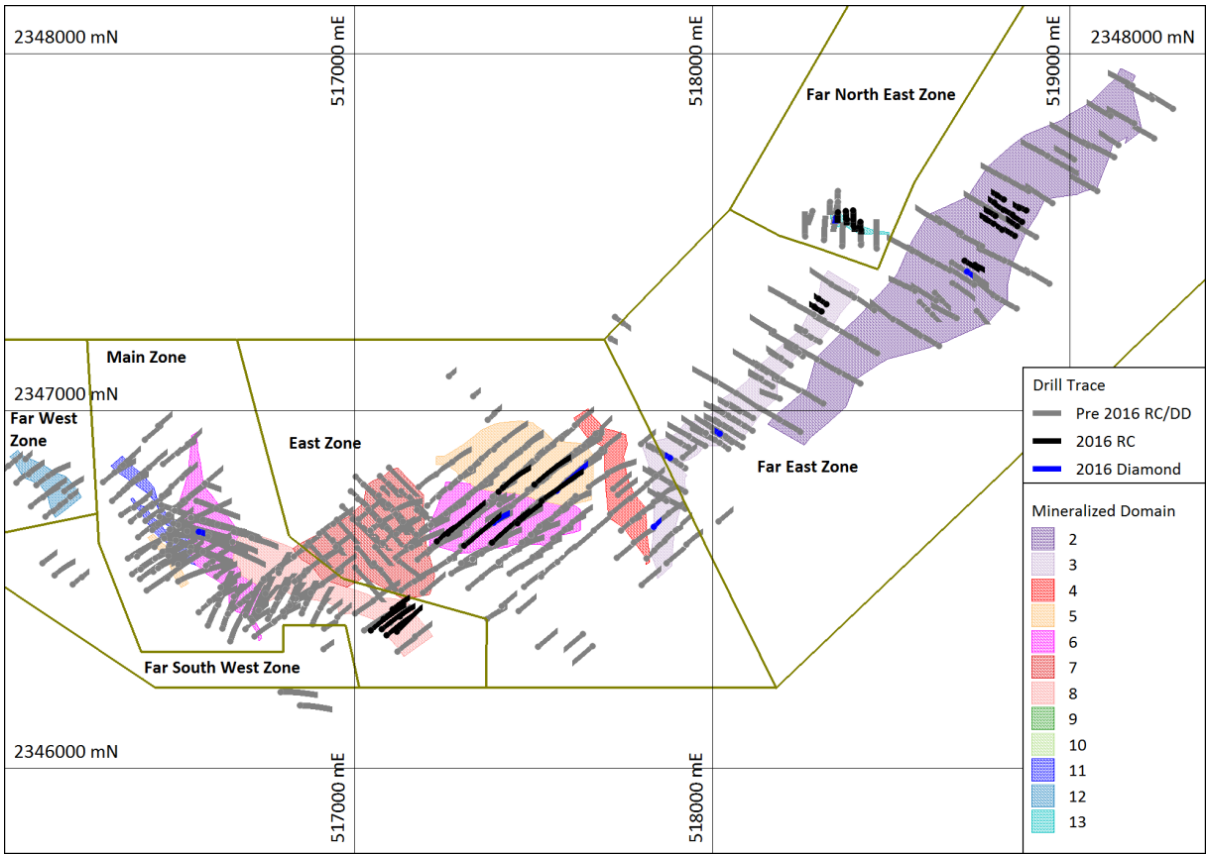
Figure 14.1 presents a plan-view of the surface expression of the mineralized domains relative to drillhole traces coloured by drilling type and phase.

For the 2016 resource estimation, Orca supplied surfaces representing the base of oxidation and the top of fresh rock interpreted from drillhole logging. The comparatively limited infill drilling completed since the 2016 study has not significantly changed this interpretation, and as specified by Orca, the 2016 oxidation surfaces were used for the current estimates.

The oxidation surfaces were used for flagging of the resource composites into oxide, transition and fresh sub-domains, density assignment and partitioning final resources by oxidation type.

Depth to the interpreted base of complete oxidation ranges from locally zero where transitional material outcrops to around 75 m and averages approximately 26 m. The interpreted depth to fresh rock ranges from around 3 - 113 m depth and averages approximately 55 m.

Figure 14.1 GSS Mineralized Domains and Drill Traces



14.2.3 Exploratory Data Analysis

Table 14.1 shows univariate statistics of composite gold grades for the resource dataset subdivided by mineralized domain. Notable features of these statistics include the following.

At 0.08 g/t, the mean gold grade for Domain 1 composites is notably lower than for the mineralized domains demonstrating that the domaining has been effective in assigning most mineralized composites into the mineralized domains.

Typical of many gold deposits, all populations of gold grades show strong positive skewness with coefficients of variation of generally greater than 2.5 for mineralized domains indicating that MIK is an appropriate estimation technique and that selective mining above elevated cut-off grades will be difficult.

Table 14.1 GSS Composite Statistics (Au g/t)

	Domain						
	1	**2**	**3**	**4**	**5**	**6**	**7**
Number	9,722	2,980	3,039	750	3,002	3,844	3,200
Mean	**0.08**	**0.32**	**0.51**	**0.43**	**0.71**	**0.90**	**0.51**
Variance	0.19	1.54	4.38	0.85	1.63	4.72	2.72
Coef. Var.	**5.50**	**3.87**	**4.14**	**2.15**	**1.80**	**2.41**	**3.26**
Minimum	0.00	0.00	0.00	0.00	0.00	0.00	0.00
1st Quartile	0.01	0.01	0.01	0.04	0.11	0.07	0.03
Median	0.01	0.03	0.07	0.13	0.27	0.37	0.14
3rd Quartile	0.04	0.18	0.42	0.40	0.86	1.10	0.45
Maximum	19.1	30.9	100.5	9.05	32.7	78.2	61.2
	Domain						
	8	**9**	**10**	**11**	**12**	**13**	**Total**
Number	1,978	3,029	256	938	363	205	33,306
Mean	0.57	0.92	2.02	1.07	0.57	1.84	0.51
Variance	10.0	5.70	35.8	7.28	5.53	7.57	3.40
Coef. Var.	5.55	2.60	2.97	2.53	4.10	1.50	3.65
Minimum	0.00	0.00	0.00	0.01	0.00	0.00	0.00
1st Quartile	0.02	0.04	0.06	0.06	0.01	0.21	0.01
Median	0.07	0.19	0.26	0.33	0.11	0.90	0.07
3rd Quartile	0.31	0.93	1.08	0.97	0.52	2.25	0.37
Maximum	122.3	49.6	48.7	41.7	33.3	24.7	122.3

14.2.4 Indicator Thresholds and Bin Mean Grades

For Domains 10 and 13 which contain relatively few composites, composites from all three oxidation sub-domains were combined for determination of indicator thresholds and class mean grades. This approach was taken to provide sufficient composites to generate robust conditional statistics. Composites from the larger, more populous domains were subdivided by oxidation sub-domain for determination of indicator threshold and bin grades.

For each dataset, indicator thresholds were defined using a consistent set of percentiles representing probability thresholds of 0.1, 0.2, 0.3, 0.4, 0.5, 0.6, 0.7, 0.75, 0.8, 0.85, 0.9, 0.95, 0.97, and 0.99 for data in each data subset.

All class grades were determined from bin mean grades with the exception of the upper bins, which were reviewed on a case by case basis and bin grades selected on the basis of bin mean, or median with or without exclusion of high grade composites. This approach was adopted to reduce the impact of a small number of outlier composites. In the author's experience this approach is appropriate for MIK modelling of highly variable mineralization such as GSS.

Table 14.2 summarizes upper bin thresholds and bin mean grades and describes with the methodology used to determine upper bin grades.

Table 14.2 GSS Upper Bin Thresholds and Class Grades

Domain	Sub-domain	Upper bin (>99%ile) Au g/t			Source of bin grade
		Threshold	Maximum	Bin grade	
1	Oxide	0.750	4.995	1.085	Median
	Transition	1.350	4.735	2.125	Median exclud. comps. > 5.0 g/t
	Fresh	1.075	3.445	1.580	Median exclud. comps. > 5.0 g/t
2	Oxide	5.090	18.200	9.659	Mean
	Transition	3.960	13.975	6.315	Median
	Fresh	3.785	8.100	5.936	Mean exclud. comps. >10 g/t
3	Oxide	4.730	8.735	4.975	Median
	Transition	5.505	9.705	7.170	Mean exclud. comps. >10 g/t
	Fresh	5.185	11.525	6.780	Median
4	Oxide	2.200	4.295	3.507	Mean
	Transition	2.000	3.445	2.635	Median
	Fresh	5.480	9.050	8.162	Mean
5	Oxide	3.230	6.415	3.805	Median exclud. comps. >7 g/t
	Transition	5.740	8.755	7.045	Median exclud. comps. >10 g/t
	Fresh	5.280	13.095	6.470	Median
6	Oxide	5.500	9.960	6.590	Median
	Transition	5.190	6.425	5.675	Median exclud. comps. >15 g/t
	Fresh	6.290	24.580	9.300	Median exclud. comps. >25 g/t
7	Oxide	7.900	12.425	10.492	Mean
	Transition	4.145	17.500	9.861	Mean
	Fresh	4.710	13.735	6.905	Mean exclud. comps. >15 g/t
8	Oxide	6.490	9.080	7.919	Mean exclud. comps. >10 g/t
	Transition	7.180	11.230	9.345	Mean exclud. comps. >12 g/t
	Fresh	6.585	14.175	9.110	Median
9	Oxide	11.340	20.625	14.325	Median
	Transition	9.725	48.700	17.475	Median
	Fresh	6.675	49.550	10.855	Median
10	All	18.380	20.445	19.623	Median exclud. comps. >25 g/t
11	Oxide	7.005	10.265	9.685	Mean
	Transition	4.995	7.140	5.515	Median
	Fresh	15.675	19.475	17.828	Mean exclud. comps. >20 g/t
12	Oxide	2.200	2.825	2.825	Median
	Transition	3.250	3.490	3.427	Mean exclud. comps. >4 g/t
	Fresh	2.985	8.315	8.315	Median
13	All	9.260	9.830	9.260	Mean exclud. comps. >10 g/t

14.2.5 Variogram Models

The variogram models developed for the January 2016 estimates were used for the current estimates. This approach reflects the relatively small changes in the resource dataset for most domains.

Some less populous domains were combined for the purpose of variogram analysis. Table 14.3 summarizes domain grouping for modelling of the variograms used or the MIK estimation.

For each dataset, indicator variograms were modelled for thresholds defined using a consistent set of percentiles representing probability thresholds of 0.1, 0.2, 0.3, 0.4, 0.5, 0.6, 0.7, 0.75, 0.8, 0.85, 0.9, 0.95, 0.97, and 0.99 for data in each data subset. For determination of variance adjustment factors a variogram model of composite gold grades was also developed for each dataset.

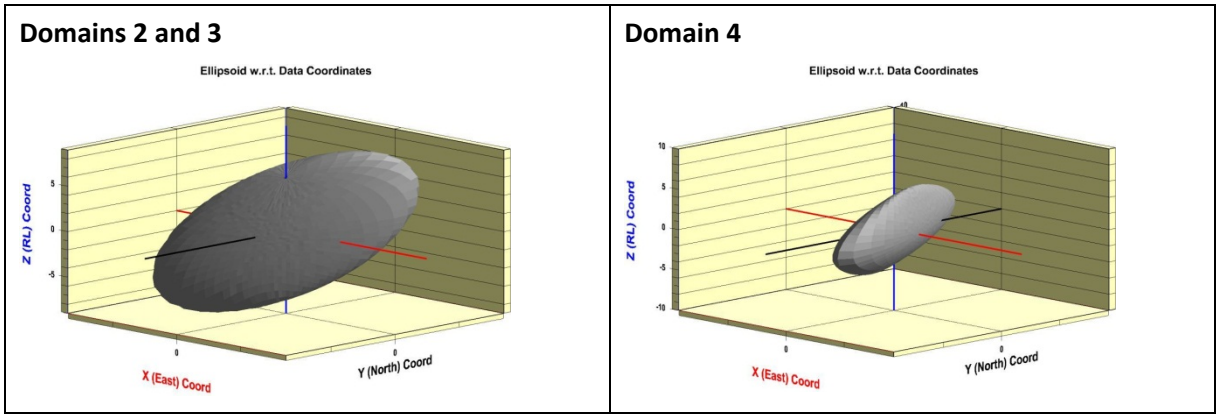
The spatial continuity observed in the variograms is consistent with geological interpretation and trends shown by resource composite gold grades. The fitted models generally have a fairly large short range structure and a smaller long range structure consistent with the strike and dip of dominant mineralized orientation.

As examples of the variogram models, Figure 14-2 presents three dimensional variogram surface maps of the median indicator variogram model for selected domains at a variogram value of 0.5.

Table 14.3 Variogram Models used for Estimation

Domain	Zone	Variogram models
1	Far West, Far SW, Main, East,	Use Dom 5&6
	Far East	Use Dom 2&3
	Far North East	Use Dom 8,10,11,12
2,3	All	Combine for modelling
4	All	Model
5,6	All	Combine for modelling
7	All	Model
8,10,11,12	All	Combine for modelling
9	All	Model
13	All	Use Dom 8,10,11,12

Figure 14.2 GSS 3D Variogram Plots



14.2.6 Estimation Parameters

The block model frame work used for the MIK modelling covers the full extents of the composite dataset. It includes panels with dimensions of 15 m east-west x 25 m north-south x 5 m vertical. The plan-view panel dimensions are consistent with drillhole spacing in more closely drilled portion of the deposit.

The four progressively more relaxed search criteria used for MIK estimation are presented in Table 14.4. For each domain the search ellipsoids were aligned with dominant domain orientation. Search pass four was used only for panels within the Far East Zone reflecting the commonly broader spaced drilling in this area.

Table 14.4 GSS Search Criteria

Search	Radii (m) (x, y, z)	Minimum Data	Minimum Octants	Maximum Data
1	15,30,25	16	4	32
2	30,60,50	16	4	32
3	30,60,50	8	2	32
4 (Far East only)	30,90,50	8	2	32

The resource estimates include a variance adjustment to give estimates of recoverable resources for selective mining (SMU) dimensions of 5 m east x 5 m north x 2.5 m in elevation. The variance adjustments were applied using the direct lognormal method and the adjustment factors listed in Table 14.5.

Table 14.5 GSS Variance Adjustment Factors

Domain	Zone	Block / Panel	Information Effect	Total Adjustment
1	Far West, Far SW, Main, East, Far East	0.177	0.633	0.112
	Far North West	0.185	0.457	0.085
2, 3	All	0.149	0.500	0.075
4	All	0.149	0.500	0.075
5, 6	All	0.177	0.633	0.112
7	All	0.255	0.271	0.069
8, 10, 11, 12	All	0.185	0.457	0.085
9	All	0.325	0.565	0.184
13	All	0.185	0.457	0.085

14.2.7 Resource Classification

Resource model panels have been classified as Indicated or Inferred on the basis of search pass and the zone boundaries shown in Figure 14.2.

All panels within the Far West, Main, East, and Far East zones informed by search pass one and two are classified as Indicated. For the Far North East zone, panels within the mineralized domain (Domain 13) informed by search pass one are classified as Indicated. All other panels, including all panels informed by search passes three and four, and all panels within the Far South West zone are assigned to the Inferred category.

14.3 Estimation of WD Resources

14.3.1 Resource Dataset

Relative to the February 2016 dataset, the WD sampling database supplied for the current estimates includes 30 additional holes drilled since October 2015 comprising the following:

- Three diamond holes drilled in the west of the main mineralized domain. These holes were primarily intended to provide samples for metallurgical testing and target areas previously tested by relatively close spaced drilling.
- 27 RC holes which infill western and less commonly southern portions of the main mineralized domain. These holes in-fill areas of previously comparatively broad-spaced drilling, increasing confidence in estimated resources for these areas.

The current estimates are based on two metre down-hole composited gold grades from RC and diamond drilling.

The resource dataset comprises 5,598 composites with gold grades ranging from 0.005 to 388.6 g/t and averaging 0.94 g/t. The dataset is dominated by composites from RC holes which represent 96% of mineralized domain composites. Data compiled since completion of the 2016 resource estimates represents around 25% of composites within mineralized domains.

14.3.2 Geological Interpretation and Domaining

Drilling to date has delineated two bodies of gold mineralization at WD comprising a main central zone and a subsidiary eastern zone. In general the transition from gold mineralization to barren host rock is characterised by diffuse grade boundaries.

The current estimates are based on two mineralized domains interpreted by MPR on the basis of composited gold grades. Domain boundaries were digitised on cross-sections, snapped to drillhole traces where appropriate, then wire-framed into three-dimensional solids.

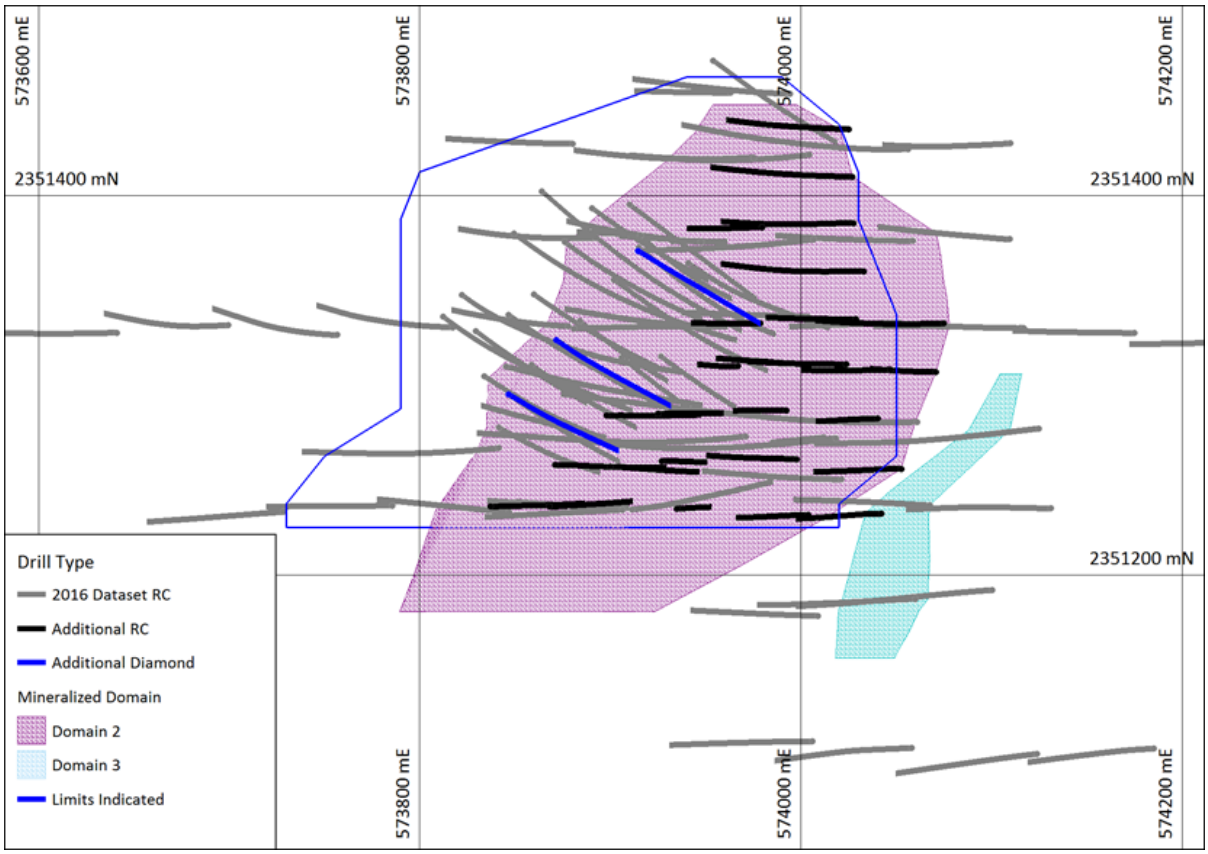
For flagging of composite grades and resource estimation the mineralized domains are designated as Domains 2 and 3, with Domain 1 representing generally un-mineralized composites not captured by the mineralized domain wire-frames.

Figure 14.3 presents a plan-view of the surface expression of the mineralized domains relative to drillhole traces coloured by drilling type and phase.

Orca supplied surfaces representing the base of oxidation and the top of fresh rock interpreted from drillhole logging. With minor adjustments to remove overlaps these surfaces were used for flagging the resource composite into oxide, transition and fresh sub-domains, density assignment and partitioning final resources by oxidation type.

Depth to the interpreted base of complete oxidation ranges from around 4 m to approximately 50 m and averages approximately 16 m. The interpreted depth to fresh rock ranges from around 13 to 58 m depth and averages around 25 m.

Figure 14.3 WD Mineralized Domains and Drill Traces



14.3.3 Exploratory Data Analysis

Table 14.6 shows univariate statistics of composite gold grades for the resource dataset subdivided by mineralized domain. Notable features of these statistics include the following:

- At 0.10 g/t, the mean gold grade for Domain 1 composites is notably lower than for the mineralized domains demonstrating that the domaining has been effective in assigning most mineralized composites into the mineralized domains.
- Typical of many gold deposits, all populations of gold grades show strong positive skewness with a coefficient of variation of around 6.9 for the mineralized domains indicating that MIK is an appropriate estimation technique and that selective mining above elevated cut-off grades will be difficult.

Table 14.6 WD Composite Statistics (Au g/t)

	Domain				
	1	**2**	**3**	**2 & 3**	**Total**
Number	1,143	4,311	144	4,455	5,598
Mean	**0.10**	**1.18**	**0.40**	**1.15**	**0.94**
Variance	0.05	64.8	0.21	62.7	50.1
Coef. Var.	**2.18**	**6.83**	**1.17**	**6.87**	**7.54**
Minimum	0.01	0.01	0.01	0.01	0.01
1st Quartile	0.03	0.10	0.11	0.10	0.07
Median	0.06	0.25	0.26	0.25	0.18
3rd Quartile	0.11	0.57	0.50	0.56	0.46
Maximum	5.27	388.6	3.41	388.6	388.6

14.3.4 Indicator Thresholds and Bin Mean Grades

For each domain, composites from all three oxidation sub-domains were combined for determination of indicator thresholds and class mean grades. This approach was taken to provide sufficient composites to generate robust conditional statistics.

For each dataset, grade thresholds were defined using a consistent set of percentiles representing probability thresholds of 0.1, 0.2, 0.3, 0.4, 0.5, 0.6, 0.7, 0.75, 0.8, 0.85, 0.9, 0.95, 0.97, and 0.99 for data in each data subset.

All class grades were determined from bin mean grades with the exception of the upper bins, which were reviewed on a case by case basis and an appropriate grade selected to reduce the impact of small numbers of outlier composites. In the author's experience this approach is appropriate for MIK modelling of highly variable mineralization such as WD.

Table 14.7 summarizes upper bin thresholds and bin grades and describes with the methodology used to determine upper bin grades.

Table 14.7 WD Upper Bin Thresholds and Bin Grades

Domain	Sub-domain	Upper Bin (>99%ile) Au g/t			Source of Bin Grade
		Threshold	Maximum	Bin Grade	
1	Combined	0.660	5.270	0.430	97th percentile
2	Combined	17.335	388.60	29.775	Bin median
3	Combined	1.635	3.410	2.767	Bin median

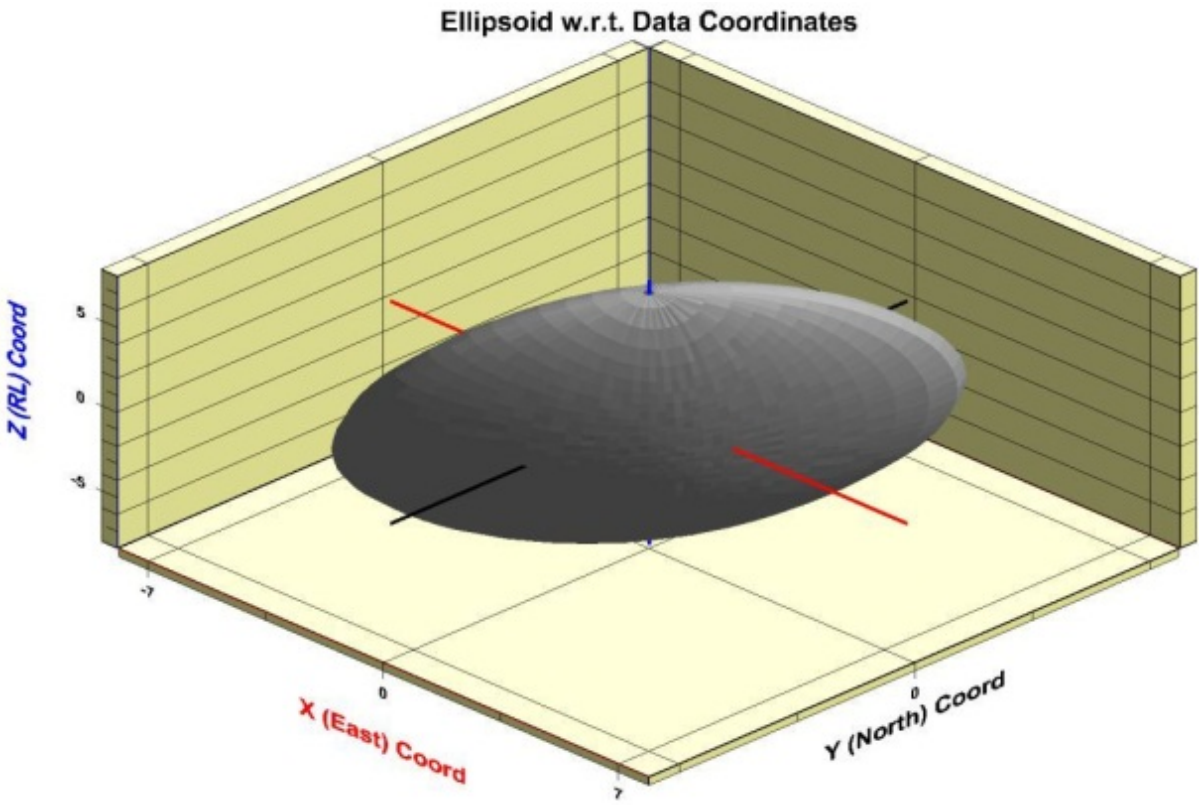
14.3.5 Variogram Models

Domains 1 and 3 contain too few mineralized composites for reliable variogram modelling and one set of indicator variograms modelled from Domain 2 composites was used for the MIK modelling. Indicator variograms were modelled for thresholds defined for percentiles representing probability thresholds of 0.1, 0.2, 0.3, 0.4, 0.5, 0.6, 0.7, 0.75, 0.8, 0.85, 0.9, 0.95, 0.97, and 0.99 for the dataset. For determination of variance adjustment factors, a variogram model of composite gold grades was also developed for Domain 2.

The spatial continuity observed in the variograms is consistent with geological interpretation and trends shown by resource composite gold grades. The fitted models generally have a fairly large short range structure and a smaller long range structure consistent with the strike and dip of dominant mineralized orientation.

As examples of the variogram models as Figure 14.4 presents a three dimensional variogram surface map of the median indicator variogram model at variogram value of 0.5.

Figure 14.4 WD, Domain 2 3D Variogram Plot



14.3.6 Estimation Parameters

The block model frame work used for the MIK modelling covers the full extents of the informing composites. It includes panels with dimensions of 10 m east-west x 25 m north-south x 5 m vertical. The plan-view panel dimensions were selected on the basis of drillhole spacing in more closely drilled portion of the deposit.

The three progressively more relaxed search criteria used for MIK estimation are presented in Table 14.8. Search ellipsoids were aligned with dominant domain mineralization orientation.

The resource estimates include a variance adjustment to give estimates of recoverable resources for selective mining (SMU) dimensions of 5 m east x five m north x 2.5 m in elevation. The variance adjustments were applied using the direct lognormal method and the adjustment factors listed in Table 14.9.

Table 14.8 WD Search Criteria

Search	Radii (m) (x, y, z)	Minimum Data	Minimum Octants	Maximum Data
1	37.5,37.5,10	16	4	48
2	50,50,13	16	4	48
3	50,5013	8	2	48

Table 14.9 WD Variance Adjustment Factors

Domain	Block / Panel	Information Effect	Total Adjustment
1	0.241	0.504	0.121
2	0.241	0.504	0.121
3	0.241	0.504	0.121

14.3.7 Resource Classification

Resource model panels have been classified as Indicated or Inferred on the basis of search pass and a wire-frame outlining more closely drilled portions of the mineralization. Figure 14.3 shows the surface projection of this wire-frame.

Panels within the classification wire-frame, informed by passes one and two, are classified as Indicated. All other panels, including all panels informed by search pass three and all panels outside the classification wire-frame are assigned to the Inferred category.

14.3.8 Bulk Densities

Bulk densities assigned to the GSS and WD estimates were obtained from the site generated density data base and are shown in Table 14.10.

Table 14.10 Bulk Densities

Oxidation Type	Bulk Density (t/bcm)
Oxide	2.45
Transition	2.65
Fresh Rock	2.83

14.4 Model Reviews

14.4.1 Plots of the Models

Figure 14.6 and Figure 14.7 shows representative cross-sections of the GSS and WD resource models respectively. These plots show the resource model panels scaled by the estimated proportion above 1.0 g/t cut-off, and coloured by the estimated gold grade above this cut-off relative to the resource domains and drillholes traces coloured by two metre composited gold grades. Indicated panels are shown as solid colour and Inferred blocks are hatched. Figure 14.5 shows the location and orientation of each section.

It should be noted that when viewing the vertical sections through the resource model there are situations where the model blocks appear to be un-correlated to the mineralized intercepts in the neighbouring drillholes. This is occurring because of the way the resource model blocks have been presented. The model blocks plotted are only those that contain an estimated resource above 1.0 g/t Au cut-off, and the proportion above cut-off has been used to scale the east and north dimension of the model block for presentation purposes. The scaling occurs about the model block centroid co-ordinate and therefore introduces the apparent miss-match between data and the resource model blocks.

Figure 14.5 Plan View of Section Traces

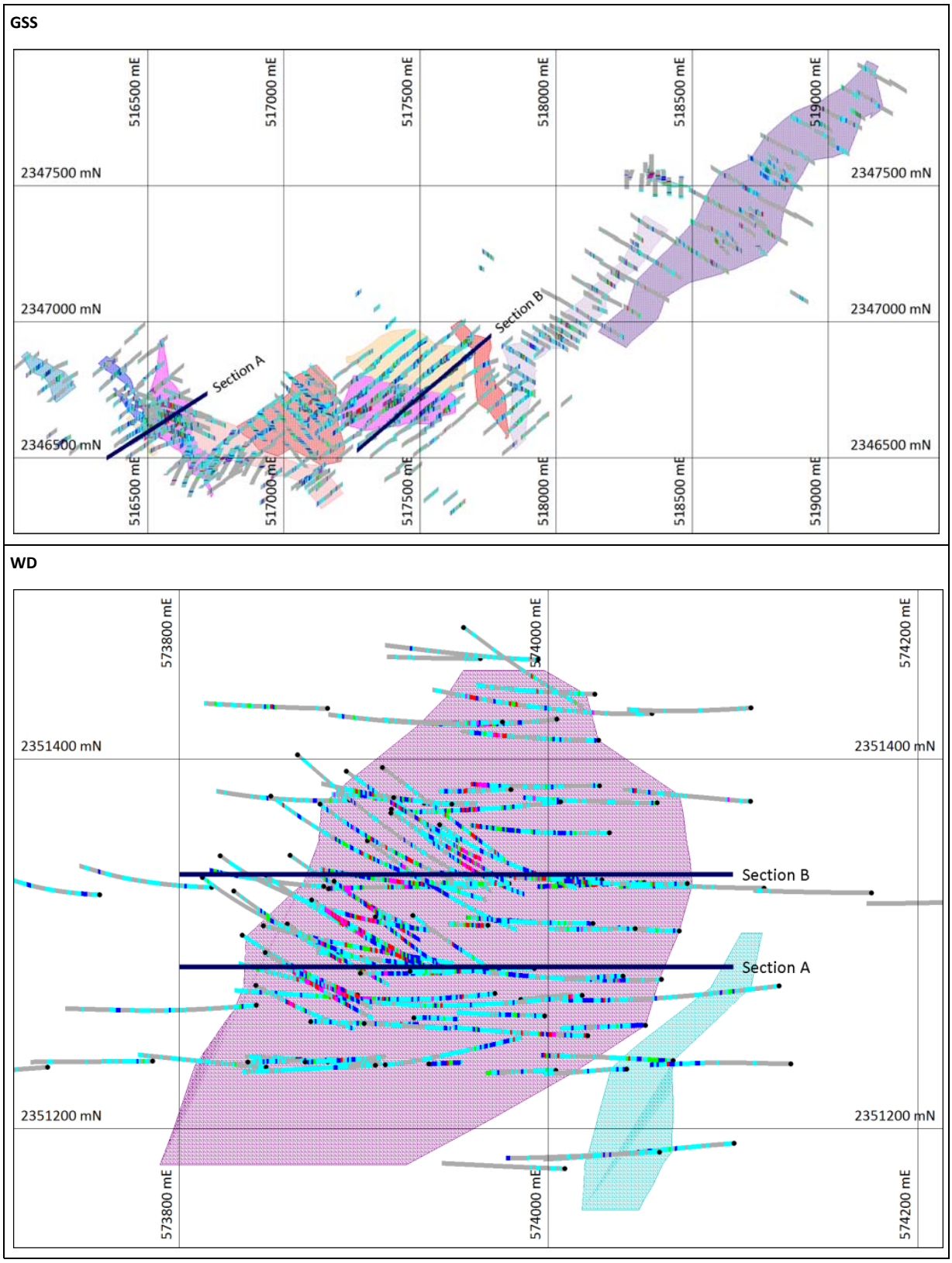
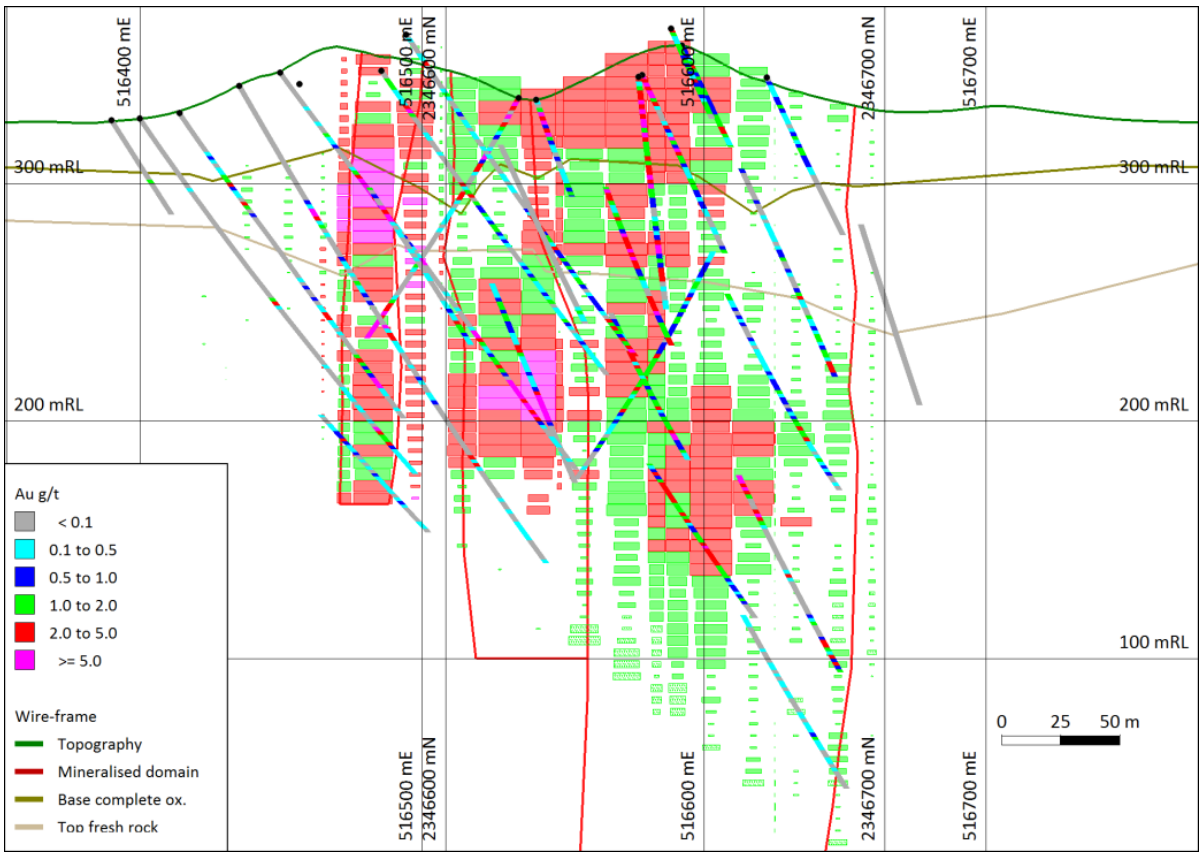


Figure 14.6 GSS Resource Model Blocks at 1.0 g/t Cut-off

Section A



Section B

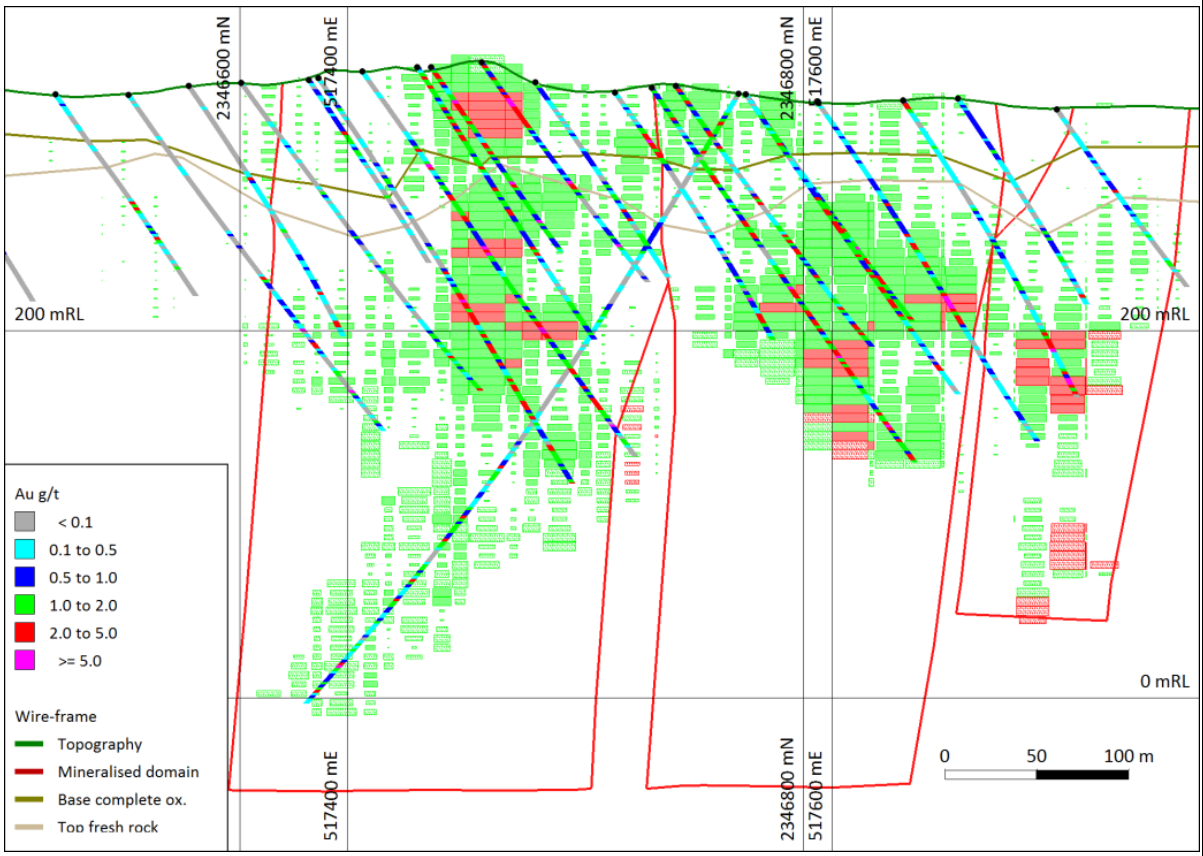
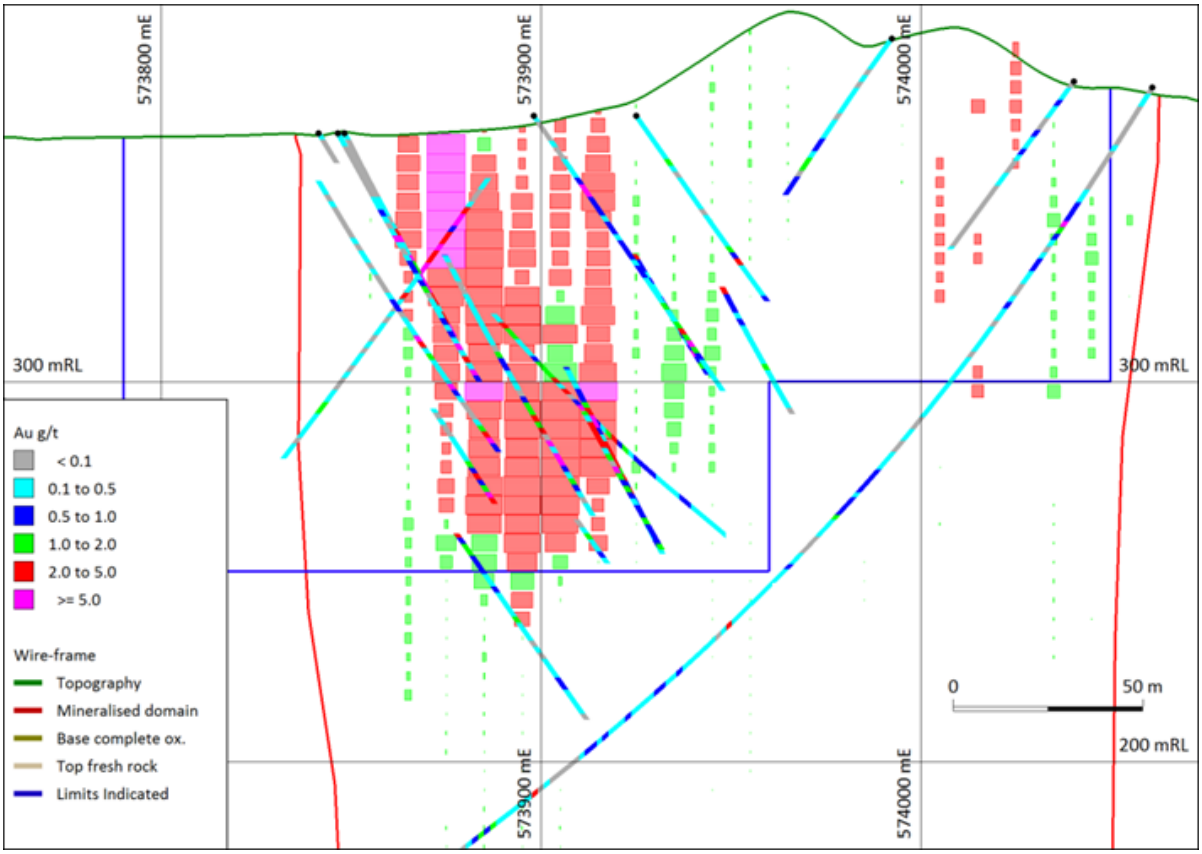
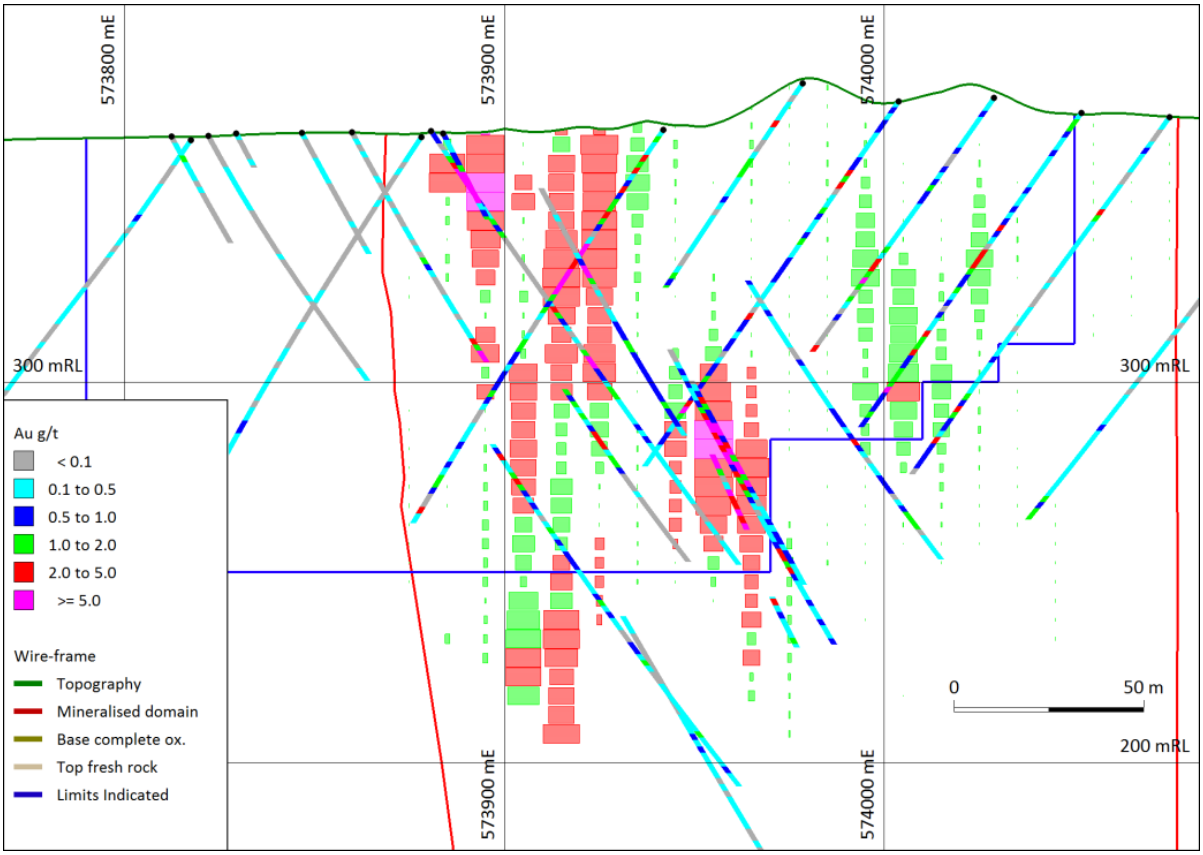


Figure 14.7 WD Resource Model Blocks at 1.0 g/t Cut-off

Section A



Section B



14.4.2 Model Validation

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

The swath plots in Figure 14.8 and Figure 14.9 compare average estimated panel grades for Indicated Resources and average composite grades by easting, northing and elevation for GSS and WD respectively.

The plots shown for GSS exclude the Far South West area, where all estimates are classified as Inferred, and the Far North East area where drillhole coverage is inconsistent with the general resource area coverage. Average composite grades include an upper cut of 20 g/t which represents the 99.9th percentile of the estimation dataset and reduces the impact of a small number of outlier composite grades of up to 122.3 g/t.

For WD, average composite grades include an upper cut of 80 g/t which represents the 99.9th percentile of the estimation dataset and reduces the impact of a small number of outlier composite grades of up to 388.6 g/t.

The plots in Figure 14.8 and Figure 14.9 show that although the average block grades estimated by MIK are smoothed compared to the average composite grades, they generally closely follow the trends shown by the composite mean grades with the exception of areas of variably spaced or limited sampling. There are minor local deviations between the model and composite trends seen on the plots and these are influenced the following features:

- Excluding the highest composite grades have reduced the amount of metal (grade) estimated in the resource model.
- The use of an octant search strategy in the MIK estimation has a de-clustering effect on the estimates.
- The data used in the estimation of the MIK block grades are coming from a greater volume than the vertical or horizontal slices being compared which are consistent with model panel dimensions.
- Variability in drillhole spacing, such as clustering of drillholes in areas of higher grade mineralization.

Figure 14.8 GSS Average Estimated Panel Grades vs. Composite Grades

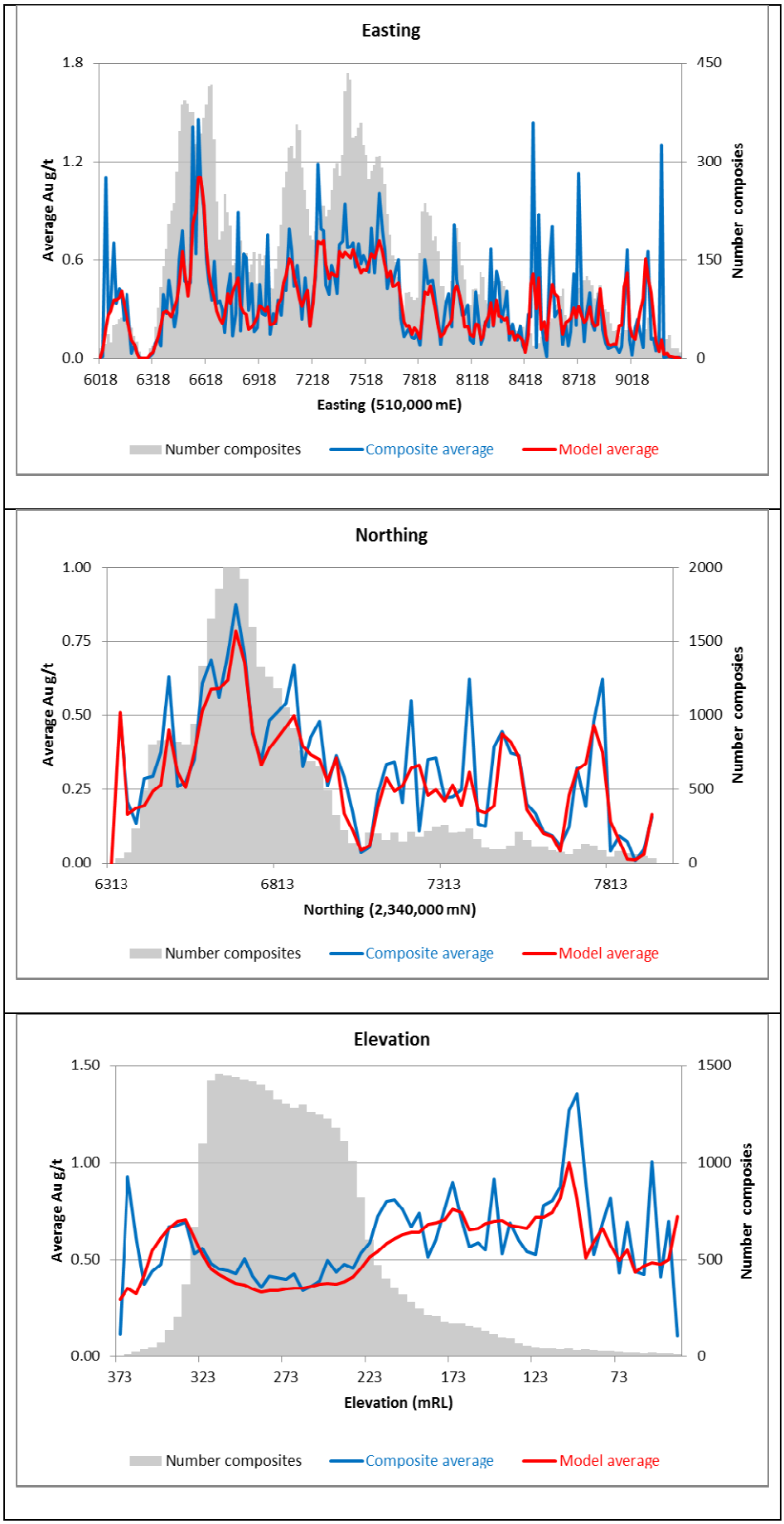
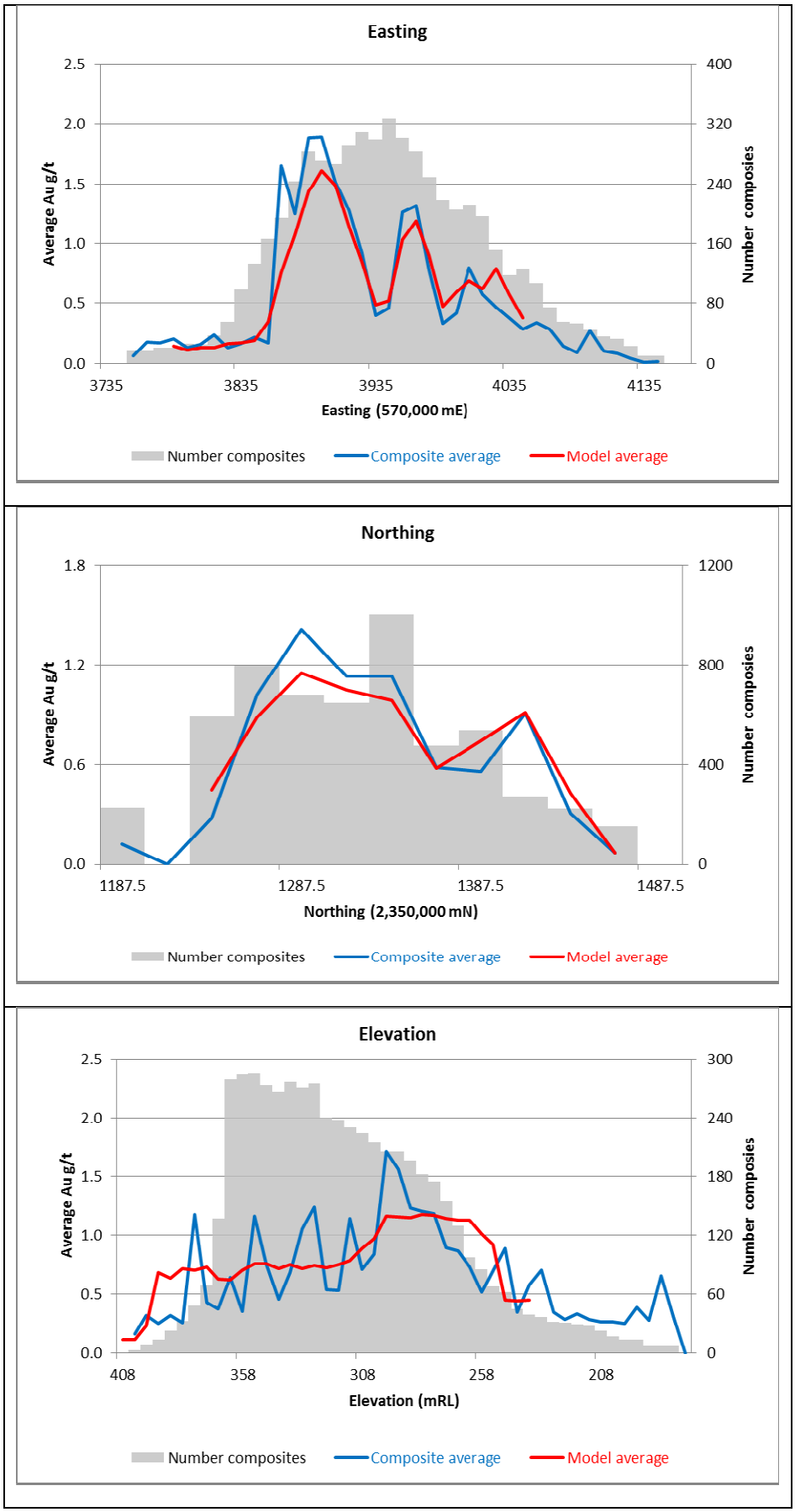


Figure 14.9 WD Average Estimated Panel Grades vs. Composite Grades



14.5 Resource Estimates

Table 14.11 shows the current Mineral Resource Estimates for GSS and WD for a range of cut off grades. Table 14.12 shows the estimates at 1.0 g/t cut off subdivided by oxidation type. The figures in these tables are rounded to reflect the precision of the estimates and include rounding errors.

The Mineral Resources are reported below using supplied topographic surfaces with no allowance for depletion by currently active artisanal mining, which is considered to have a minor impact on the reported estimates.

The GSS estimates extend to around 350 m depth. The WD estimates extend to around 210 m depth.

For GSS the combined oxidized and transitional material hosts around 37% and 13% of the Indicated and Inferred resources respectively, with the remainder lying in fresh rock. For WD the combined oxidized and transitional material hosts around 17% and 13% of the Indicated and Inferred resources respectively.

Table 14.11 February 2017 Mineral Resource Estimates

GSS

Cut-off	Indicated			Inferred		
Au g/t	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz
0.60	54.1	1.29	2,246	18.3	1.2	716
0.80	39.0	1.52	1,909	12.6	1.5	591
1.00	28.6	1.75	1,609	9.0	1.7	485
1.20	21.1	1.99	1,347	6.4	1.9	395

WD

Cut-off	Indicated			Inferred		
Au g/t	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz
0.60	3.2	2.04	213	2.1	1.3	84
0.80	2.5	2.44	196	1.2	1.7	64
1.00	2.0	2.79	183	0.7	2.2	52
1.20	1.7	3.10	172	0.5	2.6	44

Combined

Cut-off	Indicated			Inferred		
Au g/t	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz
0.60	57.3	1.33	2,459	20.3	1.2	800
0.80	41.5	1.58	2,105	13.8	1.5	654
1.00	30.6	1.82	1,792	9.7	1.7	536
1.20	22.8	2.07	1,518	6.9	2.0	439

Table 14.12 February 2017 Mineral Resource Estimates at 1.0 g/t Cut-off by Oxidation Type

GSS

Material	Indicated			Inferred		
	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz
Oxide	5.6	1.85	334	0.6	1.7	32
Transition	4.9	1.76	279	0.6	1.5	27
Fresh	18.0	1.72	996	7.8	1.7	426
Total	**28.6**	**1.75**	**1,609**	**9.0**	**1.7**	**485**

WD

Material	Indicated			Inferred		
	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz
Oxide	0.2	2.97	24	0.1	2.5	7
Transition	0.1	2.87	9	0.0	1.5	0.5
Fresh	1.7	2.76	150	0.6	2.2	44
Total	**2.0**	**2.79**	**183**	**0.7**	**2.2**	**52**

Combined

Material	Indicated			Inferred		
	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz
Oxide	5.9	1.90	358	0.7	1.8	39
Transition	5.0	1.78	288	0.6	1.5	27
Fresh	19.7	1.81	1,146	8.4	1.7	470
Total	**30.6**	**1.82**	**1,792**	**9.7**	**1.7**	**536**

15. MINERAL RESERVE ESTIMATES

There is no mineral reserve estimate for the Block 14 Project.

16. MINING METHODS

16.1 Geotechnical Considerations

Two reviews of existing geotechnical data were conducted by SRK Consulting (UK) Limited. The reports from these reviews have been used as a basis for the subsequent optimization and pit design work.

Nine specific geotechnical boreholes have been drilled at Galat Sufar South and Wadi Doum for the geotechnical study in order to gather rock mass and structural information to derive slope parameters. This data complimented the basic geotechnical data from 18 cored boreholes which was provided for the 2016 PEA study. A structural database was created from the logging of downhole geophysics OTV/ATV images, in addition to foliation and joint orientation measurements made in 2016. Structural domains were identified, presenting their own joint set orientations and attributes. Point load testing and laboratory strength testing of drill core was carried out to characterise the rock mass strength.

Slope design recommendations were generated based upon the merging of findings from kinematics analyses and slope stability modelling. Analyses show that the maximum overall slope angles from fresh rock is constrained by the bench and berm geometry, designed to minimise kinematic instability and trap potential rock fall. The recommended slope parameters are presented in Table 16.1.

Table 16.1 Slope Design Recommendations

Pit	Domain	Slope Direction	Bench Height (m)	Bench Face Angle (°)	Bench Width (m)	Inter-ramp Angle (°)
Overburden		000 – 360	10	50	5	36.7
Galat Sufar South – West Pits	1	000 – 030	20	75	7	58.3
	2	030 - 310	20	75	5	62.6
	1	310 – 360	20	75	7	58.3
Galat Sufar South – East Pits	3	000 – 050	20	75	6	60.4
	4	050 – 250	20	75	4	64.9
	3	250 – 360	20	75	6	60.4
Galat Sufar South – Far East Pits	2	000 – 360	20	75	5	62.6
Wadi Doum	4	000 – 360	20	75	4	64.9

16.2 Hydrogeology and Mine Dewatering

Groundwater has been encountered in five holes during exploration drilling at either deposit. With only rare annual precipitation in the region, there is no requirement to consider routine mine dewatering. Further work will be carried out to evaluate the impact of this water on the pit slope stability and operations.

There may be a need to manage rare localised rain events, but the volumes of precipitation involved suggest that this will be a minor (< 1 day) disruption to operations and present minimal flood risk.

16.3 Mining Method Selection

Given the gold grades and proximity to surface, the deposits will be mined via a conventional truck and excavator open pit mining method. The WD deposit will be exploited through a single pit approximately 150 m deep. The GSS deposit will be exploited by ten separate pits, eight of which are shallow oxide / transitional pits with minimal fresh material. The other two pits (East and Main) are deeper than the WD pit at 210 and 250 m respectively.

While the mineralization below the pits is not of a sufficiently high grade to support underground development, there is scope for larger pits under improved geotechnical or financial conditions, or an increase in defined mineralization below the base of the pits.

16.4 Mine Optimization

16.4.1 Cut-off Grade Determination

The block model is a partial percentage or proportional model. Grade bins were spaced at 0.05 g/t intervals from 0.5 g/t to 1.1 g/t with additional bin divisions at 1.2 g/t and 1.5 g/t. In order to process this model it was necessary to determine cut-off grades prior to running the optimizations.

Cut-off grades were calculated for oxide, transitional and fresh material for WD, Main, East, and Far East pits. This division was based on the availability of different processing costs and processing recoveries for these pits. One of the primary areas of investigation was the effect that annual throughput had on processing costs (due to the effect of the distribution of fixed costs). The second was the effect of different methods of tailings deposition related to the available water supply, again affecting processing costs.

Numerous scenarios were developed which explored the effects of varied processing costs on the optimization results. As more certainty around the processing costs was obtained, primarily through metallurgical testwork and the sourcing and delineation of the Area 5 aquifer, the optimization parameters were refined.

Table 16.2 to Table 16.4 below show the parameters applied to the cut-off grade calculation used in the chosen optimization, while Table 16.5 shows the calculated cut-off grades and associated bins used within the model to determine tonnes and grade of potential crusher feed.

Table 16.2 Selling Parameters

Item	Value (US$)
Gold Price	1,100 /oz
Refining & Selling Cost and Royalty	81 /oz

Table 16.3 Processing Recoveries

Pit	Material	% Recovery
Main	Oxide	91.8
	Transitional	81.3
	Fresh	82.8
East	Oxide	89.7
	Transitional	84.9
	Fresh	79.7
Far East	Oxide	90.8
	Transitional	83.1
	Fresh	84.2
WD	Oxide	90.8
	Transitional	83.1
	Fresh	85.7

Table 16.4 Processing Costs (US$ /t Processed)

	Main	East	Far East	WD
Oxide	$15.86	$16.57	$15.71	$15.87
Transitional	$17.71	$18.38	$18.05	$17.10
Fresh	$18.03	$17.58	$18.51	$18.33

Haulage for WD material to the GSS ROM pad has been estimated at US$7.74 /t of crusher feed and subsequently applied to the above processing cost.

Table 16.5 Cut-off Grades and Grade Bins

	Main		East		Far East		WD	
	Cut-off	Model	Cut-off	Model	Cut-off	Model	Cut-off	Model
Oxide	0.53	0.50	0.56	0.55	0.53	0.50	0.79	0.75
Transitional	0.67	0.65	0.66	0.65	0.66	0.65	0.91	0.90
Fresh	0.67	0.65	0.67	0.65	0.67	0.65	0.93	0.90

16.4.2 Cost Data for Optimization

For the purposes of the optimization it was assumed that mining is conducted by a mining contractor. Mining costs were broken into base and incremental mining costs. Costs were built from first principles using knowledge of several mining contracts operating under similar conditions in West Africa.

For cost calculations, the mining fleet for WD and GSS was assumed to comprise of Caterpillar 777 rigid body haul trucks (90 t) with suitably sized loading unit.

Unit costs were determined for the following items:

- Loading.
- Fixed hauling component.
- Drill & Blast.

- Ancillary.
- Mine Admin.

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

Incremental haulage costs were also determined for the fleet and applied during the optimization process to account for vertical haulage. See Section 21.5 for a breakdown of the cost estimates.

16.4.3 Optimization Scenarios

All optimizations were performed in the Deswik software using the Pseudoflow tool.

GSS

Optimization scenarios were primarily focussed on flexing the annual processing rate and associated cost changes. The processing rate was in turn driven by water supply and tailings deposition strategy. Optimizations examining Indicated material, and Indicated and Inferred material for each scenario were undertaken.

A processing rate of 3.4mtpa was selected and optimization (Run 32) was used. 10% of the crusher feed material included in the pits was Inferred.

Table 16.6 shows the comparison between the 2016 PEA results and the 2017 update, considering the pit shell with revenue factor 1 for each scenario.

Table 16.6 Comparison of GSS (Main, East, and Far East) Optimization Results

	2016 PEA	2017 Revised	Difference
Oxide tonnes (million)	10.5	11.1	0.6
Oxide grade	1.37	1.31	-0.06
Transitional tonnes (million)	8.2	8.2	0.0
Transitional grade	1.33	1.35	0.02
Fresh tonnes (million)	12.2	23.2	11.0
Fresh grade	1.61	1.45	-0.16
Total crusher feed tonnes (million)	30.8	42.5	11.7
Total crusher feed grade	1.46	1.40	-0.06
Waste tonnes (million)	55.7	85.9	30.2
Total mined tonnes (million)	86.5	128.4	41.9
Strip Ratio	1.81	2.02	0.21
Total cost (million)	**$851**	**$1,079**	**$228**
Revenue (million)	**$1,261**	**$1,626**	**$365**
Value (million)	**$410**	**$547**	**137**
Recovered Ounces ('000)	**1,233**	**1,596**	**363**

The 2017 Revision for GSS shows a significant increase in material processed with a decrease in average grade and increase in ounces. This is due to a number of reasons including:

- A lower diesel price reducing the mining costs.
- Reduced processing costs.
- An expanded geological resource.
- Additional geotechnical data supporting the use of steeper wall angles in the optimization.

It is interesting to note that there is minimal increase in oxide or transitional material. This is due to the bulk of the material being mined in both studies, due to the presence of economic unweathered material below these horizons.

Wadi Doum

The WD optimization also considered the change to the pit shells based on changing economics around throughput rates. No consideration was given to treating material locally. The higher grade WD feed will be blended with the GSS feed for the first seven years of the operation.

Table 16.7 provides a comparison of the WD optimizations (examining the Revenue Factor = 1 shells).

Table 16.7 Comparison of WD Optimization Results

	2016 PEA	2017 Revised	Difference
Oxide tonnes ('000)	348.1	452.9	104.8
Oxide grade	2.72	2.23	-0.49
Transitional tonnes ('000)	172.7	114.2	-58.5
Transitional grade	2.59	2.60	0.01
Fresh tonnes ('000)	1530.8	2,038.7	507.9
Fresh grade	2.74	2.64	-0.1
Total feed tonnes ('000)	2,051.7	2,605.9	554.2
Total crusher feed grade	2.73	2.56	-0.17
Waste tonnes ('000)	7,796.4	8,807.4	1,011
Total mined tonnes ('000)	9,848.0	11,413.3	1565.3
Strip Ratio	3.80	3.38	-0.42
Total cost (million)	**$85**	**$97**	**$12**
Revenue (million)	**$152**	**$189**	**$37**
Value (million)	**$67**	**$92**	**$25**
Recovered Ounces ('000)	**148**	**186**	**38**

The study has shown a significant increase in crusher feed tonnes and ounces from the 2016 PEA. This is due to a number of reasons, including:

- A lower diesel price reducing both the mining costs and feed haulage cost, which reduced from US$8.48 /t hauled to US$7.74 /t hauled.
- Improved processing costs.
- An expanded and refined geological resource.
- Additional geotechnical data supporting the use of steeper wall angles in the optimization.

16.5 Preliminary Scheduling

A preliminary round of scheduling was undertaken to assess the viability of producing a mine plan for different processing rates. Mine schedules were created which were based on the optimization shells for the 2.6 Mtpa, 3.0 Mtp, and 3.4 Mtpa cases

The selected shells were scheduled using current equipment assumptions to ensure:

- Crusher feed requirements could be met.
- Enough separate mining areas could be maintained.
- Crusher feed and waste extraction could be balanced sufficiently to keep a relatively consistent number of fleets in operation throughout the life of the project.

This exercise demonstrated that all three rates could be supported with the selected optimization shells. Based on this and the associated financial considerations around mine life, the throughput rate of 3.4 Mtpa was selected.

16.6 Mine Design and Sequencing

16.6.1 Pit Shell Selection

Based on the final parameters provided, it was decided to use the following optimization runs for the pit design:

- GSS: Run 32 – Indicated and Inferred material at a nominal 3.4 Mtpa processing rate.
- WD: Run 17 – Indicated and Inferred material at a nominal 3.4 Mtpa processing rate.

In order to account for the inclusion of ramps and other design features, the RF=0.95 shells for all pits were used as the basis of design for the pits.

16.6.2 Mine Design Parameters

Using the selected optimization shells as reference, open pits were designed to develop a more realistic mining scenario. Ramps and berms were included in these designs.

Batter, berm and wall angle configurations varied for the different material types and ramp widths were varied between double lane and single lane to reflect likely mining practice. Where possible, a "goodbye cut" was designed at the base of each pit to maximise extraction of crusher feed.

Based on the assumed mining equipment, a bench height of 5 m was used, although geotechnical conditions allowed for up to four benches to be excavated between safety berms, depending on the material. There may be some opportunity to mine higher bench heights in areas of bulk waste where grade control and mining dilution do not need to be considered.

16.6.3 Pit Statistics

GSS

The GSS pits contribute over 94% of the total crusher feed and 89% of the contained ounces. The ten GSS pits will be exploited along a strike of over 3.2 km (Figure 16.1).

The total insitu economic material for the GSS pits is 41.68 Mt at 1.40 g/t with 92.46 Mt of waste.

The overall strip ratio for the pits is 2.22:1. Table 16.8 shows the material contained within the GSS pits while Figure 16.2 shows the relative proportions of resource and material categories for the crusher feed. The two larger pits are exploited through a series of cutbacks, while the small pits are mined as single entities.

Figure 16.1 3D View of the GSS Pits

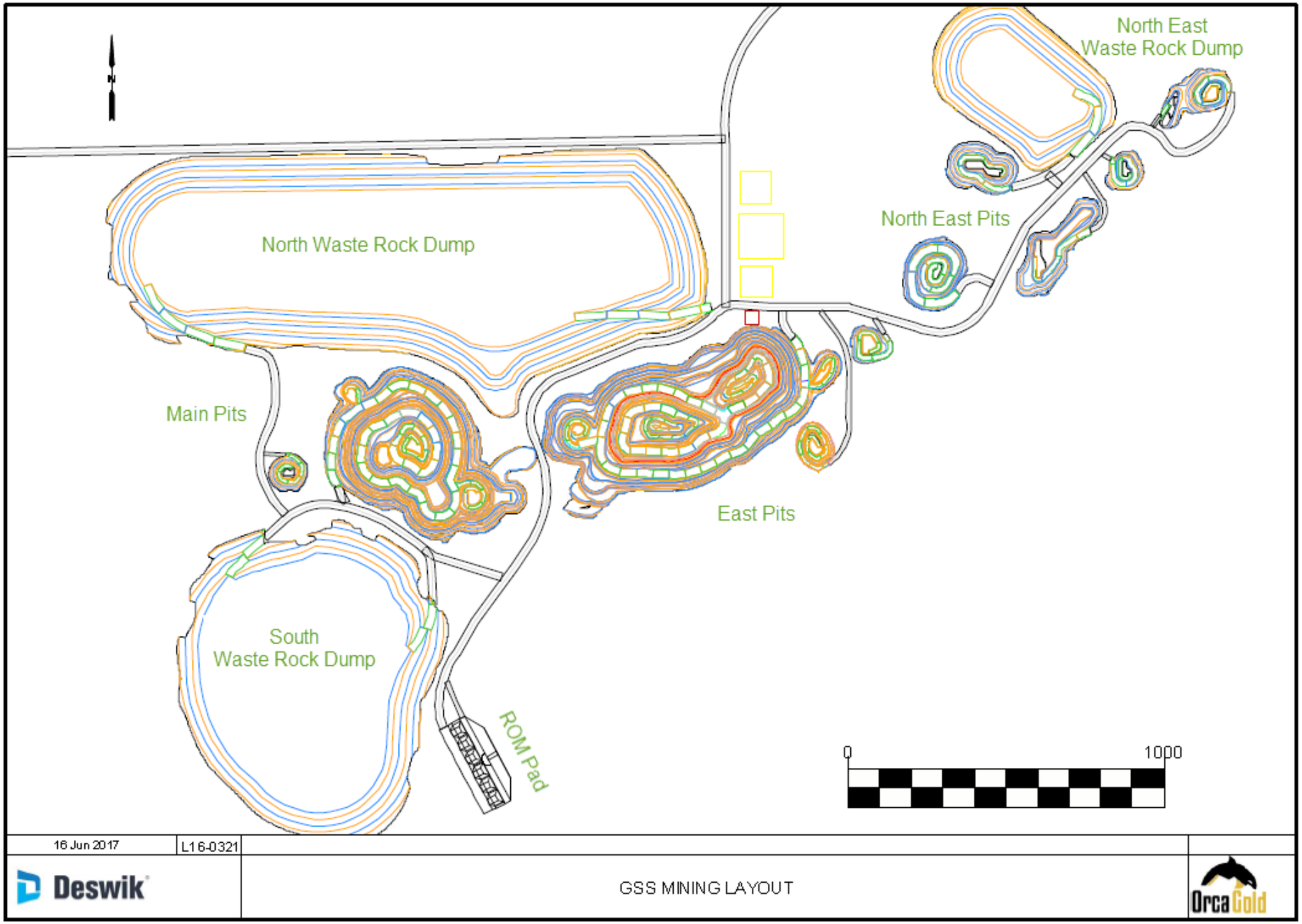
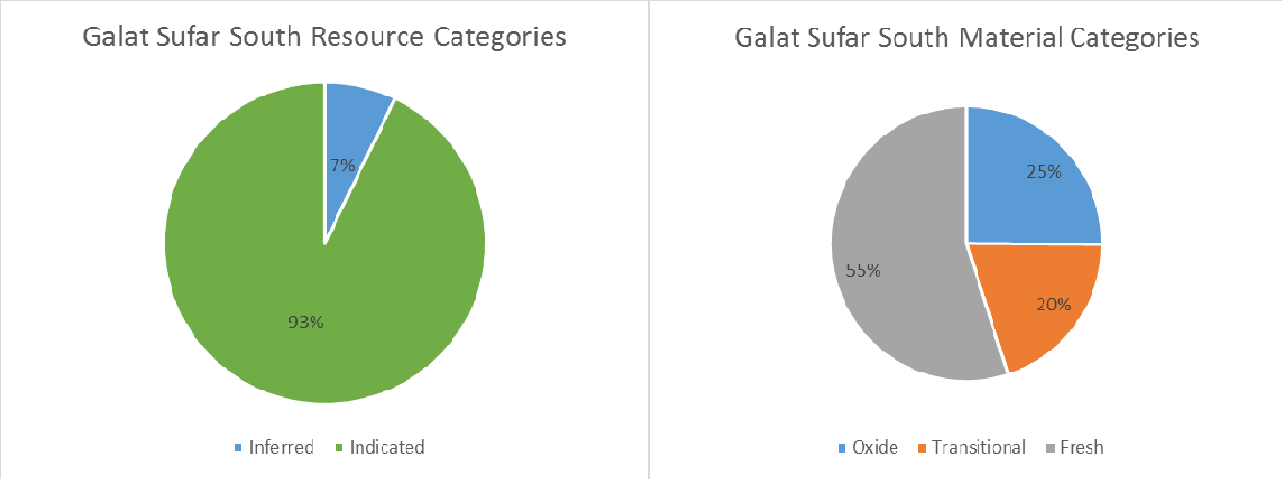


Table 16.8 GSS Pits Inventory

	Oxide		Transitional		Fresh		Total	
	'000 t	Grade	'000 t	Grade	'000 t	Grade	'000 t	Grade
Inferred	818	1.18	315	1.26	1,854	1.63	2,987	1.47
Indicated	9,651	1.36	7,998	1.35	21,045	1.43	38,694	1.40
Waste	22,436	-	29,024	-	39,124	-	92,457	-
Total Material	**33,369**	**-**	**37,936**	**-**	**62,832**	**-**	**134,138**	**-**

Figure 16.2 GSS Resource and Material Categories for Crusher Feed



WD

Although much smaller than the GSS pits to the west, the WD deposit contains mineralization at over 1.8 times the grade. The deposit is exploited through a three stage 150 m deep pit (Figure 16.3). The overall strip ratio for the WD pit is 4.3:1.

The total insitu economic material for Wadi Doum is 2.78 Mt at 2.52 g/t with 11.97 Mt of waste.

Table 16.9 shows the material contained within the GSS pits while Figure 16.4 shows the relative proportions of resource and material categories for the crusher feed.

Figure 16.3 3D View of the WD Designed Pit

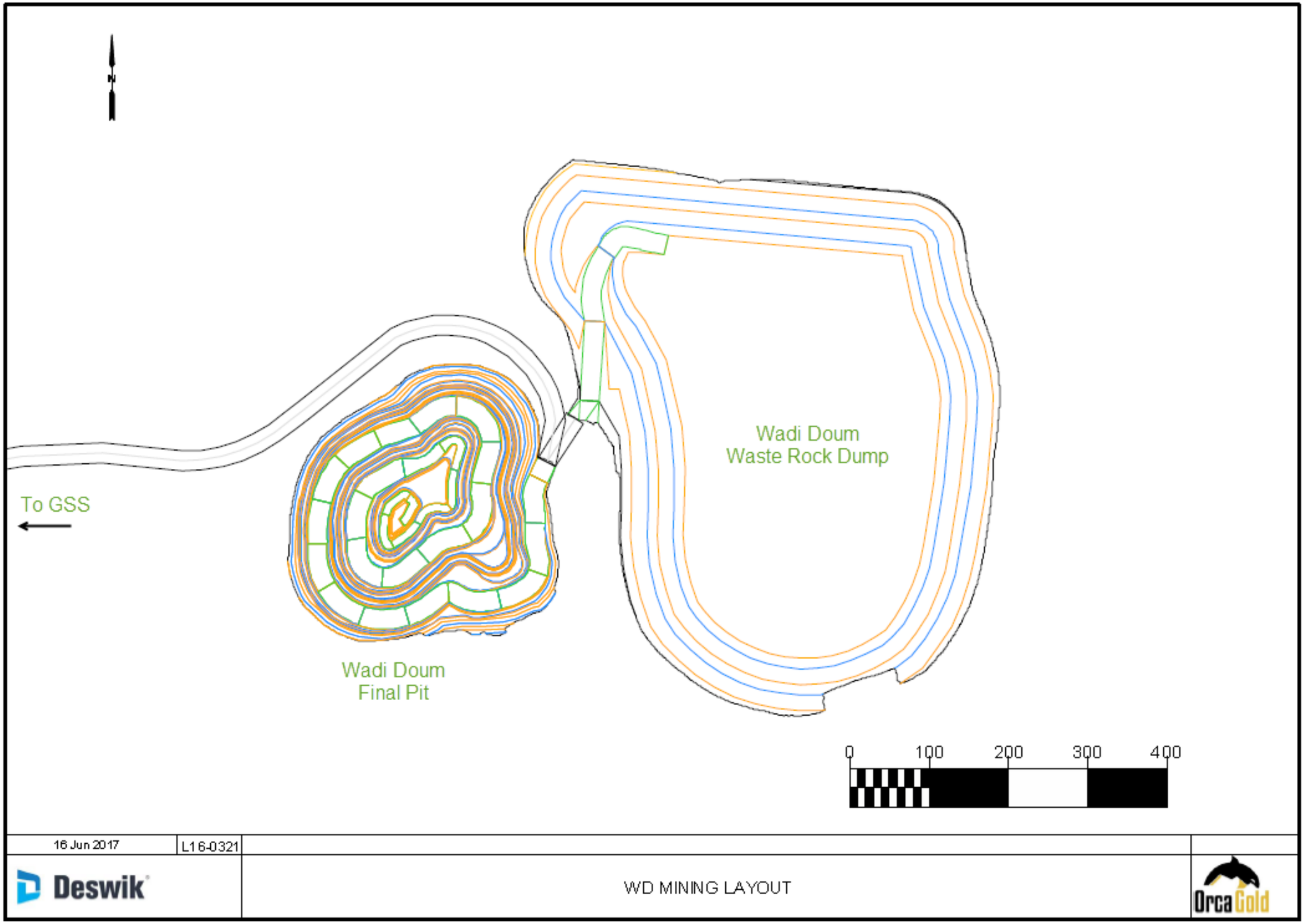
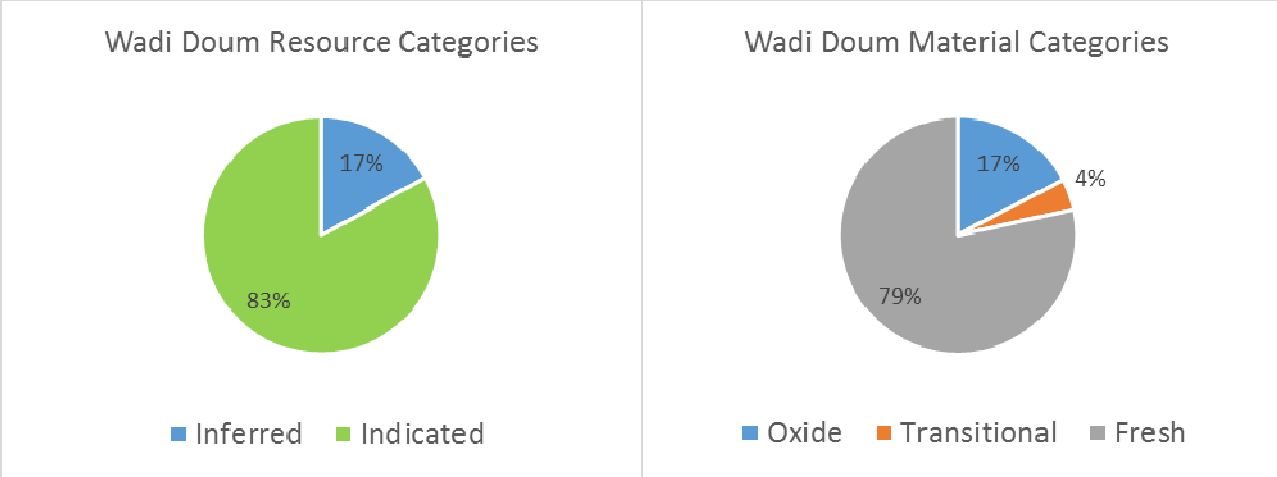


Table 16.9 WD Pit Inventory

	Oxide		Transitional		Fresh		Total	
	'000 t	Grade	'000 t	Grade	'000 t	Grade	'000 t	Grade
Inferred	132	1.70	13	1.37	316	2.41	461	2.18
Indicated	336	2.42	106	2.69	1,816	2.62	2,257	2.59
Waste	3,048	-	945	-	7,980	-	11,973	-
Total Material	**3,516**	**-**	**1,064**	**-**	**10,112**	**-**	**14,691**	**-**

Figure 16.4 WD Resource and Material Categories for Crusher Feed



Combined Summary

The two deposits produce a total of 44.4 Mt @ 1.47 g/t Au of treatable material. The tonnage of treatable material in the Indicated category is 92% of the total treatable material. Total contained gold is 2.10 Moz.

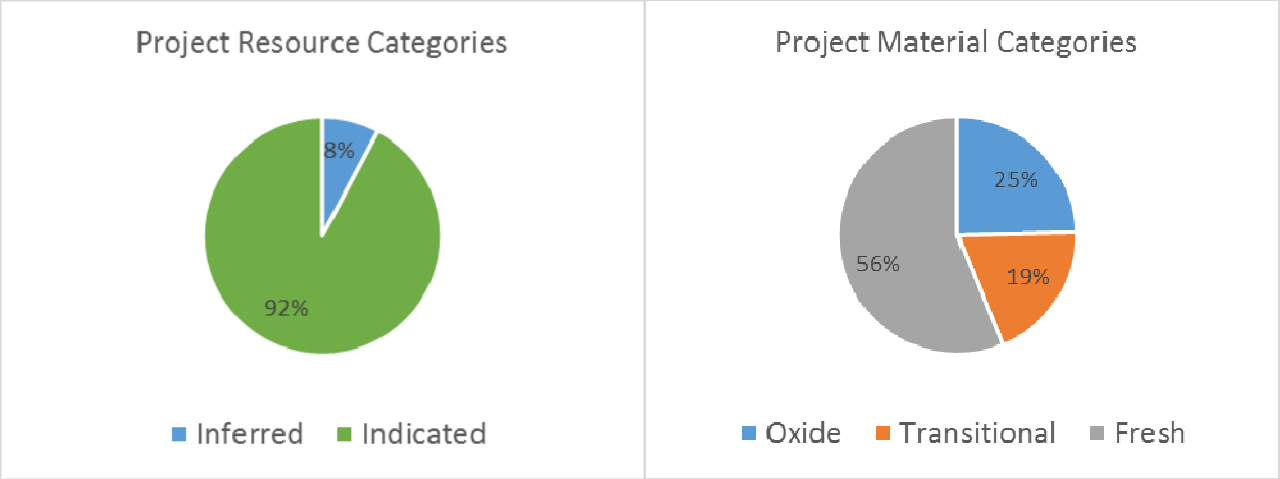
Waste material is 104.4 Mt with a strip ratio of 2.35:1. Total material movement is 148.8 Mt.

These results are summarized in Table 16.10 and Figure 16.5.

Table 16.10 Combined Project Inventory

	Oxide		Transitional		Fresh		Total	
	'000 t	Grade	'000 t	Grade	'000 t	Grade	'000 t	Grade
Indicated	9,986	1.40	8,103	1.37	22,862	1.53	40,951	1.46
Inferred	950	1.26	328	1.26	2,170	1.74	3,448	1.56
Waste	25,948	-	30,570	-	47,912	-	104,430	-
Total Material	**36,885**	**-**	**39,001**	**-**	**72,944**	**-**	**148,829**	**-**

Figure 16.5 Project Resource and Material Categories for Crusher Feed



16.6.4 Waste Rock Dumps, Haul Roads and ROM Pads

Waste dumps were designed for both GSS and WD. Given the terrain and lack of other land use, there is very little restriction on waste dump capacity so dump location was chosen to best suit the extraction requirements.

A ROM pad for GSS was also designed and can cater for six separate ROM fingers, which will facilitate blending of the crusher feed. A skyway has been designed at the back of the ROM pad which facilitates the construction of the fingers at a height of 10 m, allowing approximately 20,000 t of rock to be stockpiled on each finger. It is envisioned that the crusher will be fed with two front end loaders working in unison. A rockbreaker will be required on a periodic basis to break up any oversize from the pits, although the crusher has been designed to accommodate single boulders up to 800 mm which should manage most ROM feed.

While space has been allocated for a ROM pad at WD, no specific design has been completed. It is unlikely that ROM material will accumulate at WD to such quantities as to require multiple lifts being constructed.

Surface haul roads have been designed in a preliminary form although detailed cut and fill analysis has not yet been completed and is not required for this level of study. The topography and climate will mean that relatively simple haul road construction will be sufficient. Surface haulage distances were estimated for the various deposits to allow for the calculation of mining costs.

16.6.5 Mining Schedule

The WD pit was divided into three stages to assist with waste stripping and crusher feed exposure. The two larger GSS pits were also divided into several cutbacks while the rest of the pits were treated as single pits for the purposes of scheduling.

A ramp up period of two years was assumed at the start of the schedule following pre-stripping and minor rock stockpiling in the six months prior to commencement of processing. The target for Year 1 was 85% of the ultimate 3.4 Mtpa crusher feed production rate, and the second year 90%. From Year 3, the full processing rate of 3.4 Mtpa was achieved until the final year. Mining dilution and recovery were not included in the schedule, which are accounted for in the Resource Estimate.

A combined mining schedule was produced which preferentially treated the higher grade material from WD, while meeting the annual production targets with additional material from GSS. The mining schedule shows a 13.2 year mine life, with WD being completed in Year 7.

The mine plan is presented in Table 16.11. Figure 16.6 shows the scheduled crusher feed and average gold grade while Figure 16.7 shows the pit extraction sequence.

Table 16.11 Mine Plan

	Units	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
GSS Oxide Feed Tonnes	'000 t	10,468.7	209.9	2,450.0	1,973.4	601.7	1,233.3	1,711.3	827.5	130.8	13.9	276.7	237.8	254.7	365.2	108.9	73.6
GSS Oxide Feed Grade	g/t	1.35	1.23	1.72	1.47	1.25	1.23	1.11	1.03	1.38	0.81	1.45	0.92	1.13	1.11	1.31	1.06
GSS Transition Feed Tonnes	'000 t	8,312.4	0.1	56.6	692.2	1,347.2	622.0	1,091.5	1,270.6	1,129.1	401.0	657.7	217.7	76.6	353.5	345.4	51.1
GSS Transition Feed Grade	g/t	1.35	1.81	1.48	1.56	1.69	1.23	1.13	1.21	1.36	1.41	1.38	1.07	1.38	1.24	1.18	1.11
GSS Fresh Feed Tonnes	'000 t	22,899.7	-	0.9	4.6	1,005.1	1,104.5	148.6	852.4	2,074.1	2,985.0	2,465.6	2,944.5	3,068.7	2,681.4	2,945.6	618.7
GSS Fresh Feed Grade	g/t	1.45	-	1.22	1.12	1.40	1.81	1.10	1.19	1.32	1.28	1.33	1.42	1.55	1.56	1.54	1.76
GSS Total Feed Tonnes	'000 t	41,680.8	210.0	2,507.5	2,670.3	2,954.0	2,959.8	2,951.4	2,950.4	3,334.0	3,400.0	3,400.0	3,400.0	3,400.0	3,400.0	3,400.0	743.4
GSS Total Feed Grade	g/t	1.40	1.23	1.72	1.50	1.50	1.45	1.12	1.15	1.33	1.30	1.35	1.36	1.51	1.48	1.49	1.65
WD Oxide Feed Tonnes	'000 t	467.9	42.6	317.7	102.8	4.8	-	-	-	-	-	-	-	-	-	-	-
WD Oxide Feed Grade	g/t	2.21	1.88	2.28	2.18	1.87	-	-	-	-	-	-	-	-	-	-	-
WD Transition Feed Tonnes	'000 t	118.7	0.0	37.3	67.8	13.6	-	-	-	-	-	-	-	-	-	-	-
WD Transition Feed Grade	g/t	2.55	1.10	2.28	2.75	2.23	-	-	-	-	-	-	-	-	-	-	-
WD Fresh Feed Tonnes	'000 t	2,131.7	-	25.2	255.5	432.5	450.0	450.0	450.0	68.5	-	-	-	-	-	-	-
WD Fresh Feed Grade	g/t	2.59	-	2.33	2.73	2.44	2.58	2.62	2.64	2.58	-	-	-	-	-	-	-
WD Total Feed Tonnes	'000 t	2,718.2	42.6	380.2	426.0	450.9	450.0	450.0	450.0	68.5	-	-	-	-	-	-	-
WD Total Feed Grade	g/t	2.52	1.88	2.28	2.60	2.43	2.58	2.62	2.64	2.58	-	-	-	-	-	-	-
Total Oxide Feed Tonnes	'000 t	10,936.6	252.5	2,767.7	2,076.3	606.4	1,233.3	1,711.3	827.5	130.8	13.9	276.7	237.8	254.7	365.2	108.9	73.6
Total Oxide Feed Grade	g/t	1.39	1.34	1.79	1.51	1.25	1.23	1.11	1.03	1.38	0.81	1.45	0.92	1.13	1.11	1.31	1.06
Total Transition Feed Tonnes	'000 t	8,431.1	0.1	93.9	760.0	1,360.8	622.0	1,091.5	1,270.6	1,129.1	401.0	657.7	217.7	76.6	353.5	345.4	51.1
Total Transition Feed Grade	g/t	1.37	1.78	1.80	1.67	1.69	1.23	1.13	1.21	1.36	1.41	1.38	1.07	1.38	1.24	1.18	1.11
Total Fresh Feed Tonnes	'000 t	25,031.4	-	26.1	260.1	1,437.6	1,554.5	598.6	1,302.4	2,142.6	2,985.0	2,465.6	2,944.5	3,068.7	2,681.4	2,945.6	618.7
Total Fresh Feed Grade	g/t	1.55	-	2.30	2.70	1.72	2.03	2.25	1.69	1.36	1.28	1.33	1.42	1.55	1.56	1.54	1.76
Total Feed Tonnes	'000 t	44,399.1	252.6	2,887.7	3,096.3	3,404.9	3,409.8	3,401.4	3,400.4	3,402.5	3,400.0	3,400.0	3,400.0	3,400.0	3,400.0	3,400.0	743.4
Total Feed Grade	g/t	1.47	1.34	1.79	1.65	1.62	1.60	1.32	1.35	1.36	1.30	1.35	1.36	1.51	1.48	1.49	1.65
GSS Total Waste Tonnes	'000 t	92,457.0	1,316.3	4,259.5	4,495.2	5,855.1	6,936.3	10,570.9	10,610.5	9,091.4	7,926.3	9,363.2	8,254.4	5,262.5	3,734.6	4,279.3	501.5
GSS Total Mined Tonnes	'000 t	134,137.8	1,526.3	6,767.0	7,165.5	8,809.1	9,896.1	13,522.3	13,560.9	12,425.4	11,326.3	12,763.2	11,654.4	8,662.5	7,134.6	7,679.3	1,245.0
GSS Strip Ratio		2.22	6.27	1.70	1.68	1.98	2.34	3.58	3.60	2.73	2.33	2.75	2.43	1.55	1.10	1.26	0.67
WD Total Waste Tonnes	'000 t	11,972.9	421.2	2,042.0	3,225.3	2,698.5	1,707.0	1,003.0	815.7	60.1	-	-	-	-	-	-	-
WD Total Mined Tonnes	'000 t	14,691.1	463.8	2,422.2	3,651.4	3,149.4	2,157.0	1,453.0	1,265.7	128.6	-	-	-	-	-	-	-
WD Strip Ratio		4.40	9.89	5.37	7.57	5.98	3.79	2.23	1.81	0.88	-	-	-	-	-	-	-
Total Waste Tonnes	'000 t	104,429.8	1,737.5	6,301.5	7,720.5	8,553.6	8,643.3	11,574.0	11,426.1	9,151.6	7,926.3	9,363.2	8,254.4	5,262.5	3,734.6	4,279.3	501.5
Total Mined Tonnes	'000 t	148,828.9	1,990.1	9,189.2	10,816.8	11,958.5	12,053.1	14,975.4	14,826.6	12,554.1	11,326.3	12,763.2	11,654.4	8,662.5	7,134.6	7,679.3	1,245.0
Overall Strip Ratio		2.35	6.88	2.18	2.49	2.51	2.53	3.40	3.36	2.69	2.33	2.75	2.43	1.55	1.10	1.26	0.67

Figure 16.6 Scheduled Crusher Feed

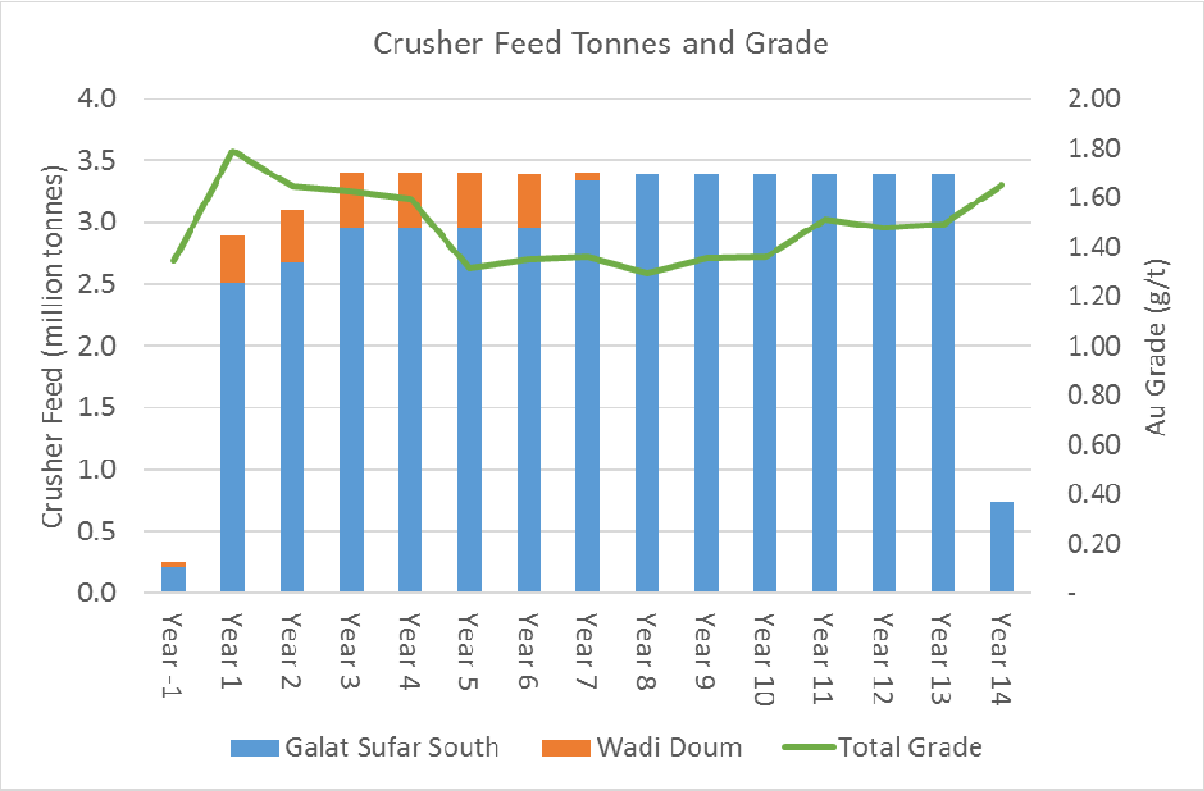
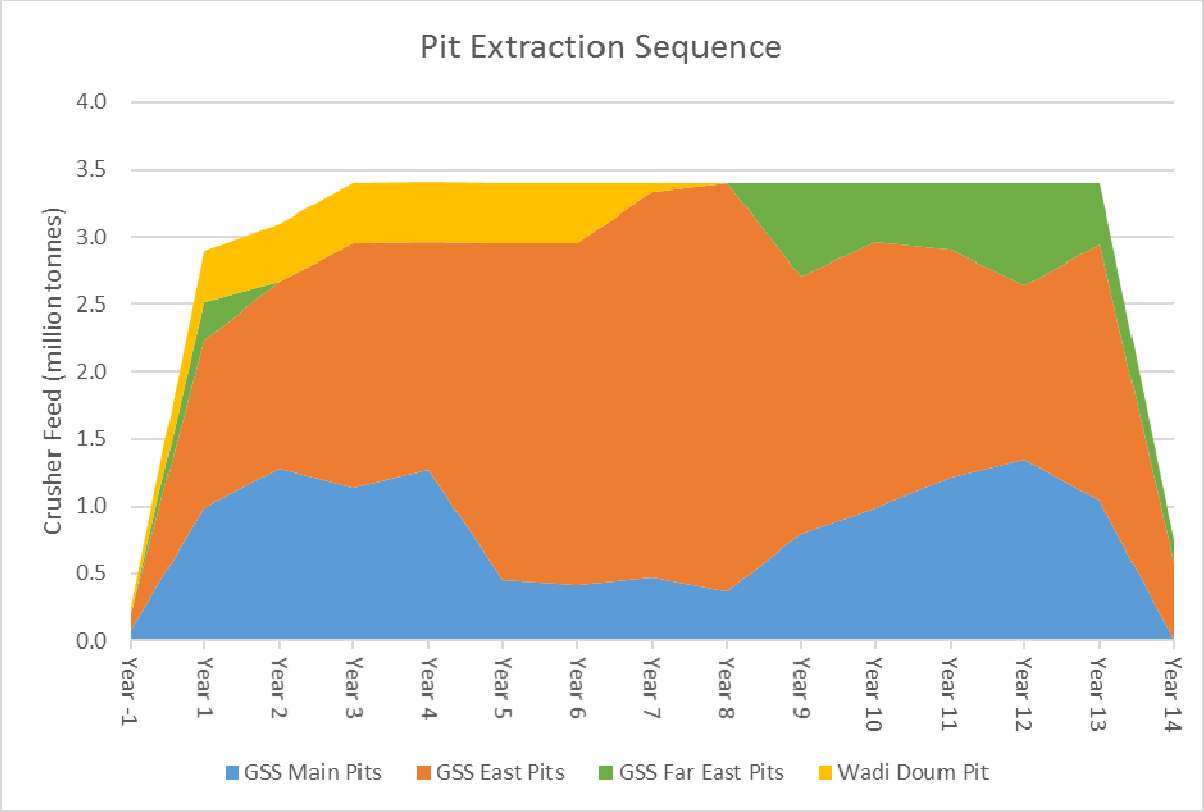


Figure 16.7 Pit Extraction Sequence



16.7 Operating Strategy

The project has been developed based on use of a mining contractor, on a cost per tonne basis. Explosives and fuel are included in this rate, although supply of one or both of these may be through Orca.

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

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

Orca will not undertake any mining activities directly. Therefore, all mobile maintenance (and waste management from such maintenance) will be the responsibility of the contractor.

A second contract may be required for the haulage from WD to GSS; however, this could be incorporated into a single mining and haulage contract. This will be investigated further during the Feasibility Study.

It is also likely that the mining contractor will be required to use two different excavator sizes during the course of the contract. This will add efficiencies around items such as dilution and feed recovery, as well as providing flexibility for optimal extraction.

17. RECOVERY METHODS

17.1 Overview

The metallurgical testwork conducted to date, specifically the 2016 and 2017 work completed by SGS South Africa and Vancouver, confirmed that the GSS and WD material is amenable to gold and silver recovery via conventional cyanidation techniques. Furthermore, flash flotation and regrind of flotation concentrate prior to leaching did not offer any additional economic benefit, and has not been included in the flowsheet.

A trade-off study comparing a three-stage crush ball mill circuit versus a SAG and ball mill (SAB) circuit concluded that there was little material difference between the options in terms of economics. However, the SAB circuit was selected on the basis of lower potential for dust generation, ease of maintenance and potential for expansion.

The design of the process plant has been based on a nominal capacity of 3.4 Mtpa.

The process plant design incorporates the following unit process operations:

- Single stage primary crushing with a jaw crusher to produce a crushed product size P_{80} of between 45 to 100 mm depending on feed type.
- A crushed rock surge bin to provide surge capacity between the crushing and grinding circuits. Surge bin overflow will be conveyed to an emergency stockpile. Feed from the emergency stockpile will be reclaimed by front end loader (FEL) to feed the mill during periods when primary crushing is off-line.
- A grinding circuit configured as a two stage circuit with a SAG mill and ball mill, both with the ability to operate in closed circuit. The circuit will produce a P_{80} of 53 µm for dominant oxide, approximately 75 µm for transition dominant and between 80 and 106 µm for fresh dominant feed.
- Pre-leach thickening to increase the slurry density feeding the leach circuit to minimise tankage, improve slurry mixing characteristics, and reduce overall reagent consumption.
- Leach and CIL circuit incorporating three dedicated leach tanks ahead of six stages of CIL for gold adsorption.
- Split AARL elution circuit, electrowinning, mercury retorting and gold smelting to recover gold and silver from the loaded carbon to produce doré, and safely remove mercury.
- Tailings thickening to recover and recycle process water from the CIL tailings.
- Tailings pumping to the tailings storage facility (TSF).

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

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

Figure 17.1 Overall Process Flow Diagram

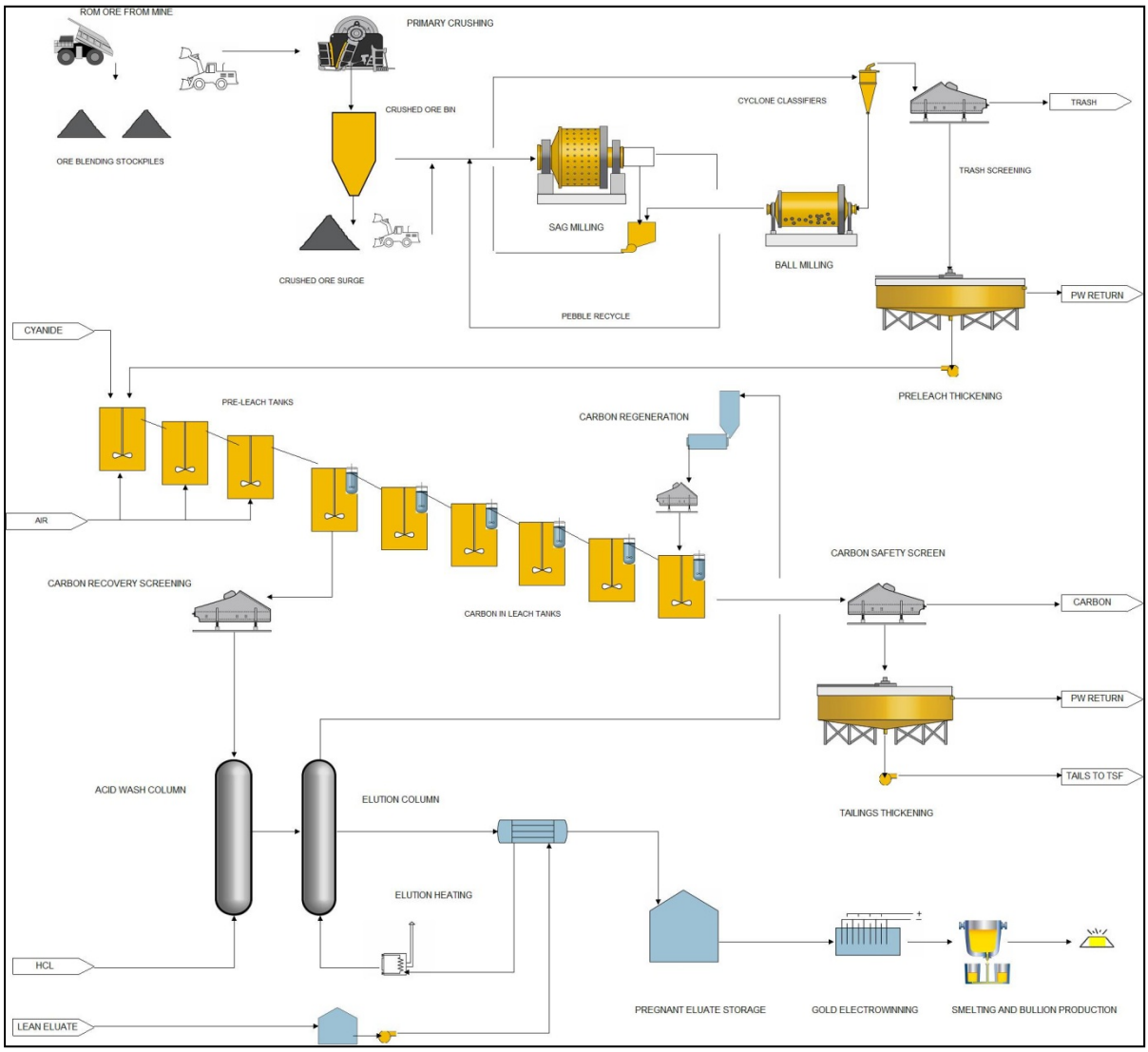


Figure 17.2 Plant Layout

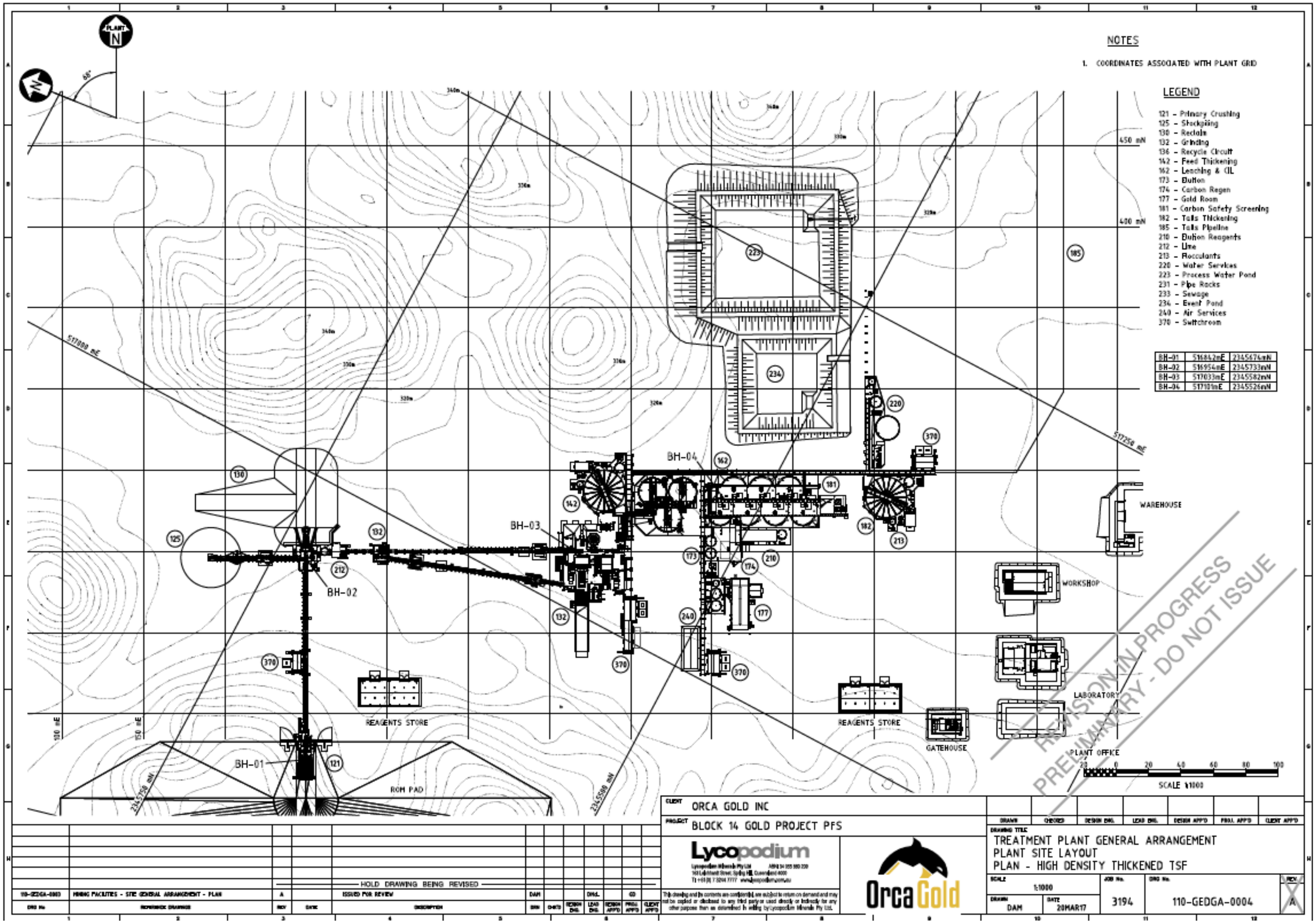
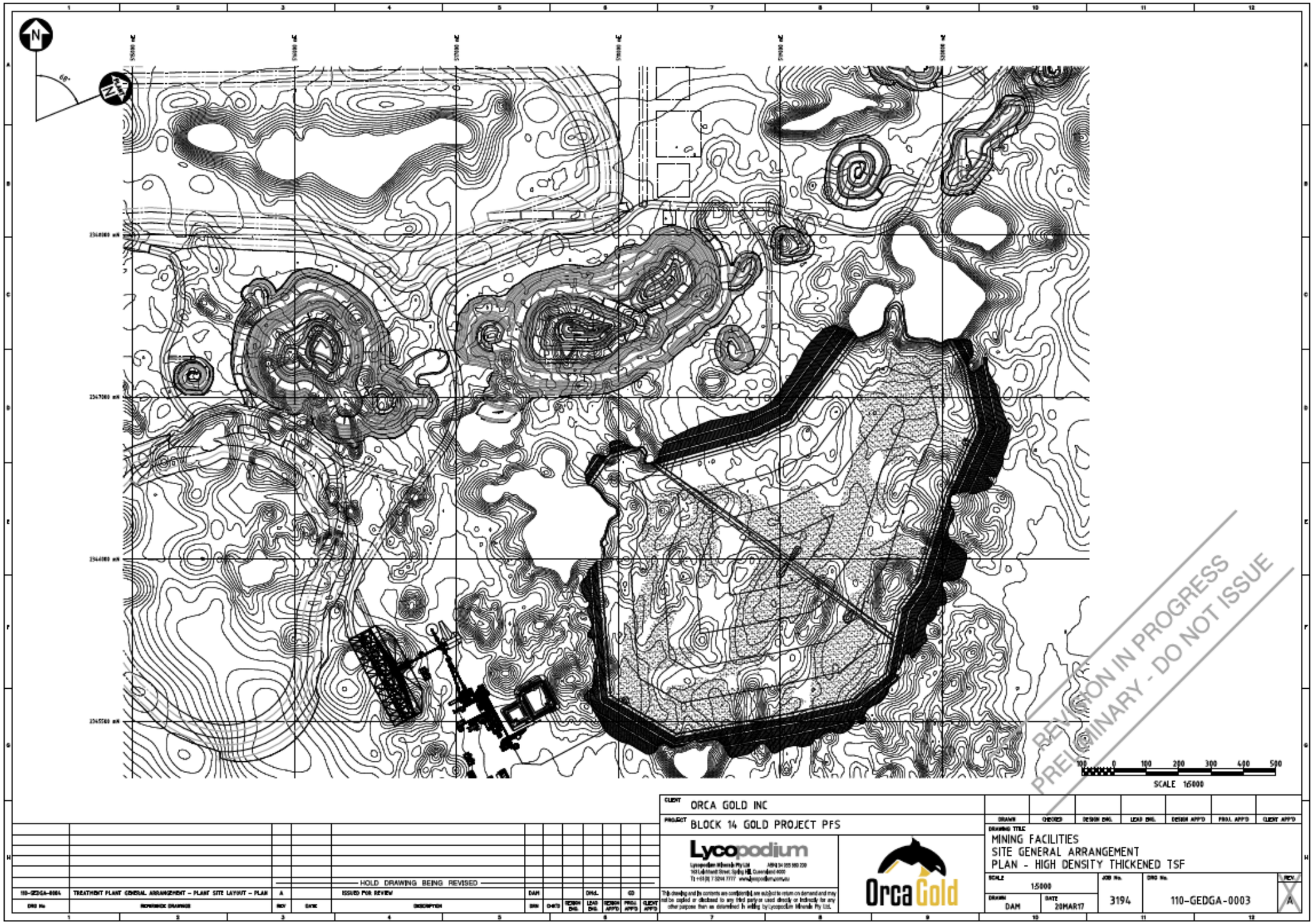


Figure 17.3 GSS Site Plan



17.2 Process Design Basis

The process plant design is based on a robust metallurgical flowsheet designed for optimum recovery while minimising plant capital costs. A number of trade-off studies have been carried out to support the flowsheet selection. The flowsheet chosen is based on well proven unit operations in the industry.

Key process design criteria for the plant are listed in Table 17.1. Inputs into the design criteria include metallurgical testwork conducted by SGS, Orca advice, comminution modelling by Orway Mineral Consultants (OMC) and Lycopodium calculations and modelling.

Table 17.1 Key Process Design Criteria

Parameter	Units	Oxide	Transition	Fresh	Source
Plant Capacity	tpa	3,400,000	3,400,000	3,400,000	Orca
Gold Head Grade	g Ag/t	1.41	1.35	1.75	Orca
Silver Head Grade	g Ag/t	2.55	3.35	4.75	Orca
Design Gold Recovery	%	91	83	83	Testwork
Design Silver Recovery	%	39	56	61	Testwork
Crushing Plant Utilization	%	85.0	85.0	85.0	OMC
Plant Availability	%	91.3	91.3	91.3	Lyco
Crushing Work Index (CWi)	kWh/t	10	10	10	OMC
A*b (SMC Test)		87 - 105	45.2	36.4 - 65.3	Testwork
Bond Ball Mill Work Index (BWi)	kWh/t	7.9 - 12.2	11.9 - 12.3	12.5 - 13.2	Testwork
Abrasion Index (Ai)		0.10	0.16 - 0.35	0.16 - 0.36	Testwork
Grind Size (P_{80})	µm	53	75	80 - 106	Orca / OMC
Pre-Leach Thickener Flux Rate	$t/m^2.h$	0.83	1.04	1.04	Testwork
Leach Circuit Residence Time	hrs	48	48	48	Orca
Leach Slurry Density	% w/w	40	40	40	Testwork
Number of Leach Tanks		3	3	3	Lyco
Number of Adsorption Tanks		6	6	6	Lyco
Cyanide Consumption – Plant	kg/t	1.19	1.52	1.41	Testwork / Lyco
Quicklime Consumption – Plant	kg/t	0.63	0.56	0.43	Testwork / Lyco
Elution Circuit Type		Split AARL	Split AARL	Split AARL	Lyco
Frequency of Elution	strips / wk	7	7	7	Lyco
Tailings Thickener Flux Rate	$t/m^2.h$	0.83	1.04	1.04	Testwork

Note that reagent consumption in the plant includes an allowance for residual cyanide, quicklime supply at 90% active CaO and the impact of leaching using site water compared to tap water used in the metallurgical testwork.

17.3 Process Description

17.3.1 Run-of-Mine (ROM) Pad

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

Feed will be blended under the guidance of mine geologists and process personnel to maintain a relatively constant feed grade to the process plant. Feed blending will also take cognisance of the consistency of the feed, with hard and soft material being mixed to assist with management of the mill feed in terms of hardness and size distribution.

17.3.2 Crushing Circuit

ROM feed will be transferred from the finger stockpiles into the ROM bin by FEL. A fixed grizzly will be installed over the ROM bin to protect the jaw crusher and downstream chutes from blockages. A mobile rock breaker will be used to break any oversize rocks that may be retained on the grizzly.

ROM feed will be withdrawn from the ROM bin at a controlled rate by a variable speed apron feeder which will feed a vibrating grizzly. Vibrating grizzly oversize will feed the primary jaw crusher. Vibrating grizzly oversize and crusher product will discharge onto the primary crusher sacrificial conveyor which then transfers feed onto the surge bin feed conveyor. The surge bin feed conveyor will be fitted with a weightometer to indicate the instantaneous and totalised crusher product rate.

Under normal operating conditions the crushing rate into the surge bin will exceed the rate of withdrawal of feed to the milling circuit. Crushed feed will overflow the surge bin and be directed on to the conveyor feeding the emergency stockpile. When required, feed from the emergency stockpile will be loaded by FEL into the surge bin to maintain mill feed when the crushing circuit is off line.

Crushed mill feed will be withdrawn from the surge bin at a controlled rate by a variable speed apron feeder and fed via the mill feed conveyor directly to the SAG mill. A weightometer will indicate the instantaneous and totalised mill feed tonnage.

Quicklime, used for pH control in the leach circuit, will be added directly onto the SAG mill feed conveyor.

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

Two dust collectors will be provided around the primary crushing and surge bin areas to provide dust removal at transfer points. Water sprays will also be provided at the grizzly and over the emergency stockpile.

The crushing circuit will be controlled from a dedicated crusher control room. The FEL driver will ensure feed is maintained to the crushing circuit and will communicate with the crusher control room using a two way radio to supply information on crusher feed operation.

17.3.3 Grinding and Classification

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

The SAG mill will discharge via a trommel with trommel oversize, consisting of pebbles and worn steel grinding media, reporting to the pebble conveyors. A belt magnet will be installed to remove worn and broken media from the system. Pebbles will be returned to the SAG mill feed conveyor.

Combined undersize product from the SAG mill and ball mill will gravitate to the cyclone feed hopper where it will be diluted with process water and pumped to hydrocyclone cluster for classification.

The combined cyclone overflow stream, with a nominal pulp density of 30% w/w solids will gravitate to the trash screen. Cyclone overflow will be screened on a vibrating trash screen to remove any misreporting coarse particles and trash that would otherwise report with the circuit carbon or block the intertank screens. Trash material will report to the trash bunker.

The underflow from each hydrocyclone will combine in the underflow launder. A portion of the underflow will be returned to the SAG mill when required, with the remainder reporting to the ball mill feed chute.

Flowmeters will be provided on the mill feed and cyclone feed hopper water addition lines to allow control of water addition rate using control valves. A nucleonic density gauge located on the cyclone feed line will indicate the cyclone feed density.

The grinding area will be serviced by two dedicated vertical spindle sump pumps which will allow spillage and clean up to be returned to the circuit via the cyclone feed hopper. A drive in sump will be provided to allow coarse material to be removed via FEL.

17.3.4 Pre-Leach Thickening

Trash screen underflow will be thickened in a high rate thickener. The feed slurry will be de-aerated in the thickener feed box prior to entry into the thickener. Flocculant will be added in the feed launder and feed well. Flocculant will be diluted with water in a static mixer to ensure good dispersion throughout the feed stream.

Thickener underflow will be pumped to a two stage leach feed sampler located above the leach feed distribution box using the duty / standby leach feed pumps. This leach feed sample will be used for metallurgical accounting purposes.

Thickener overflow will gravitate to the pre-leach thickener overflow tank from where it will be pumped to the process water pond for re-use around the plant.

The pre-leach thickener area will be serviced by the pre-leach area sump pump which will allow spillage to be directed to the pre-leach thickener feed or directly to the leach feed distribution box.

17.3.5 Leach and CIL Circuit

The leach and CIL circuit will consist of three leach tanks followed by six CIL tanks, with a total residence time of 48 hr. The tanks will be interconnected by launders, with slurry flowing by gravity from tank to tank. Each tank will be fitted with a dual impeller mechanical agitator to ensure uniform mixing and particle suspension. The CIL tanks will also be installed with mechanically swept woven wire intertank screens to retain the carbon. All tanks will be fitted with bypass facilities to allow any tank to be removed from service for agitator or screen maintenance. Low pressure air will be supplied down the central shaft of the tank agitators to provide the oxygen required for gold and silver leaching.

Slurry from the leach feed distribution box will normally be directed to the first of the three dedicated leach tanks. There will also be the option of directing slurry to the second tank if the first tank is off-line. Sodium cyanide solution will be delivered via a ring main and can be distributed to

any of the leach tanks using actuated control valves. Slurry from leach tank No. 3 will gravitate to the first of the CIL tanks.

Fresh or regenerated carbon will be returned to the circuit at CIL tank No. 6 and will be advanced counter current to the slurry flow by pumping slurry and carbon from CIL tank No. 6 to CIL tank No. 5 and so on. The intertank screen in CIL tank No. 5 will retain the carbon while allowing the slurry to return by gravity to CIL tank No. 6. This counter-current process will be repeated until the carbon eventually reaches CIL tank No. 1, the first adsorption tank. A recessed impeller pump will be used to transfer slurry to the loaded carbon recovery screen mounted above the acid wash column in the elution circuit. The carbon will be washed and dewatered on the recovery screen prior to reporting to the acid wash column. The associated slurry and wash water will return to CIL tank No. 1.

Tailings slurry from the last CIL tank will gravitate to the vibrating carbon safety screen to recover any carbon escaping from worn screens or overflowing tanks. Screen undersize will gravitate to the tailings thickener. Screen oversize containing carbon will be collected in the fine carbon bin for potential return to the circuit.

Barren carbon returning to the adsorption circuit from the carbon regeneration kiln will be screened on the carbon sizing screen to remove fine carbon. The sized and regenerated carbon will report directly to CIL tank No. 6, with undersize fine carbon gravitating to the carbon sizing screen.

A cyanide analyser and HCN monitor will be provided for monitoring purposes. A CIL area gantry crane will also be provided to facilitate removal of intertank screens for maintenance.

Four vertical spindle sump pumps will be provided in the leach and CIL areas to return spillage and clean up to the appropriate tanks.

17.3.6 Tailings Thickening and Disposal

Carbon safety screen undersize (CIL tailings) will gravitate via a two stage CIL tails sampler to the thickener feed box. Other miscellaneous waste streams, such as scrubber bleed streams and acid waste will also report to this feed box. Flocculant will be diluted using a static mixer prior to being added to the tailings thickener to enhance solids settling rates. Tailings thickener overflow will gravitate to the process water pond. Thickener underflow will be pumped to the TSF using two stages of centrifugal pumping.

Water from the surface of the TSF will be recovered from the decant system and pumped directly to the process water pond. Underdrainage and seepage from around the TSF drainage system will be pumped back into the TSF.

The tailings thickener area will be serviced by one vertical spindle sump pump. Any spillage collected within this area will be directed to the tailings thickener feed box.

A cyanide destruction circuit will not be provided as the TSF will be suitably lined and there is no evidence of significant bird activity in the area. Bird deterrents and monitoring will be used during operations to assess if further action is required.

17.3.7 Elution and Gold Room Operations

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

- Acid washing of carbon.
- Optional cold cyanide wash to remove copper from loaded carbon.

- Stripping of gold and silver from loaded carbon using the split AARL method.
- Electrowinning of gold and silver from pregnant solution.
- Filtration of electrowinning sludge.
- Removal of mercury using a mercury retort.
- Smelting of retorted products to produce a silver and gold doré.

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

Loaded carbon will be recovered on the loaded carbon recovery screen and directed to the rubber lined acid wash column. The acid wash column fill operation will be controlled manually. All other aspects of the acid wash and the carbon transfer sequence to elution will be automated. Acid washing of the carbon will commence after carbon transfer is complete.

Dilute hydrochloric acid, 3% w/w HCl in treated water, will be prepared prior to use and stored in the dilute acid make-up tank. During acid washing, the dilute solution of hydrochloric acid will be pumped into the column in an up-flow direction to remove contaminants, predominantly carbonates, from the loaded carbon. After the soak period has elapsed, the loaded carbon will be rinsed with treated water. This rinse water will displace any residual acid from the loaded carbon. Dilute acid and rinse water will be disposed of in the tailings thickener. Acid-washed carbon will be hydraulically transferred to the elution column for stripping.

A vertical spindle sump pump will be provided in the acid wash area to direct spillage to the tailings thickener.

The elution sequence will be fully automated, with actuated valves used to direct solution to and from the appropriate destinations once certain set-points or time periods are met.

The elution circuit will incorporate an optional cold cyanide wash step if copper loading on carbon becomes an issue. In this case, the operator can select a cold cyanide wash step prior to the solution re-circulating and heating step. Prior to a cold cyanide wash, the stripping water tank should be filled with treated water rather than barren electrowinning solution.

Treated water from the stripping water tank, along with cyanide injected into the pump suction, will be pumped into the column to soak for approximately half an hour. Following this soak time, treated water will be used to rinse the carbon, with the cold cyanide wash liquor reporting to the copper bleed tank. This solution will be directed to the evaporation pond where it will mix with brine reject from the water treatment plant and be evaporated.

The split AARL elution sequence will begin with the fill of the elution column and pre-heat of lean eluate solution with simultaneous injection of caustic and cyanide into the lean eluate pump suction. The solution will be re-circulated through the heat recovery and primary heat exchangers, through the elution column, through the hot side of the heat recovery heat exchangers and back into the lean eluate tank until a temperature of 95°C is achieved. The sequence will then automatically shift to the elution phase, with the temperature set point raised to 125°C, and five bed volumes (BV) of solution pumped from the lean eluate tank, through the heat exchangers and elution column to the pregnant solution tank. Caustic will be added to the pregnant solution tank during this step to ensure that a high enough solution pH is attained for electrowinning.

Following this step, five BV of treated water or barren electrowinning solution will be pumped from the stripping water tank, through the heat exchangers and elution column and into the lean eluate tank to provide lean solution for the next stripping cycle. The temperature set point will be maintained at 125°C for this step.

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

A vertical spindle sump pump will be provided in the elution column are to direct spillage to the leach feed distribution box.

Soluble gold and silver recovery from pregnant solution will be carried out by electrowinning onto stainless steel cathodes. A dedicated rectifier, per electrowinning cell, will supply the necessary current to electroplate the gold and silver onto the cathode.

Once sufficient pregnant solution is available within the pregnant solution tank, electrowinning will be initiated by starting the pregnant solution pump. The flow of pregnant solution to the cells will be evenly split across the electrowinning distribution box and manual control valves will assist the desired linear velocity to be achieved. During the electrowinning cycle the electrowinning cell discharge will be continuously returned to the pregnant solution tank via gravity.

Once the target barren solution grades have been achieved, the electrowinning cycle is complete and barren solution will discharge to the pregnant solution tank. Barren solution from the pregnant solution tank will be returned to the leach circuit via the barren solution pump. To conserve water the barren solution will be re-used as strip solution, with the option to direct this solution to leach feed when required.

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

Upon completion of electrowinning, precious metal sludge will be washed off the cathodes with a high pressure cathode washer. The gold and silver bearing sludge will gravitate to a sludge hopper, from where it will be pumped to a pressure filter.

The filter cake will be thermally dried and retorted using an electric mercury retort, supplied complete with water chiller, condenser unit, mercury trap, and carbon adsorption column on the off-gases. Mercury will be collected in a mercury flask.

Retort product solids will be mixed with a prescribed flux mixture (silica, nitre and borax), prior to being charged into the diesel fired gold furnace. The fluxes added will react with base metal oxides to form a slag, whilst the gold and silver remains as a molten metal. The molten metal will be poured into moulds to form doré ingots, which will be cleaned, assayed, stamped, and stored in a secure vault ready for dispatch. The slag produced will periodically be returned to the grinding circuit, via the SAG mill.

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

17.3.8 Carbon Regeneration

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

The carbon will be heated to 650 - 750°C and held at this temperature for 15 mins to allow regeneration to occur. Regenerated carbon from the kiln will be quenched and pumped to the carbon sizing screen. The screen oversize (regenerated, sized carbon) will return to the CIL circuit while the quench water and fine carbon will report to the carbon safety screen.

The carbon regeneration kiln off-gases will be scrubbed for mercury removal.

New carbon will be added to the carbon quench vessel to ensure that the carbon is sized over the carbon sizing screen prior to entering the CIL tanks.

A vertical spindle sump pump will be provided in the carbon regeneration area to direct any spillage back to the CIL train via the carbon sizing.

17.3.9 Reagents

Lime

Quicklime will be stored in a lime silo and will be metered onto the SAG mill feed conveyor using a rotary valve. Quicklime will be delivered to site in bulk bags and transferred to the lime silo using a hoist and lifting frame to a bag breaker mounted at the top of the silo. The silo will be fitted with a dust collector.

Hydrochloric Acid

Concentrated hydrochloric acid (32% w/w) will be delivered to site in 1,000 L bulk boxes. The concentrated hydrochloric acid will be transferred into the dilute acid make-up tank by a positive displacement, hose type pump. Treated water will be added to the dilute acid tank to achieve a solution concentration of 3% w/w. The solution will be mixed by using the acid wash pumps. Following completion of the mixing cycle, the dilute acid will be pumped to the acid wash column during the acid wash sequence.

The hydrochloric acid storage area will be serviced by an air operated dedicated floor sump pump which will pump to the tailings thickener.

Cyanide

Cyanide will be delivered as dry briquettes in one tonne bulk bags. Cyanide will be added to the mixing tank via a bag breaker and be dissolved in treated water to achieve the required 20% w/v reagent strength. The facility to dose caustic into the cyanide mixing tank to maintain a suitable solution pH will also been provided. The cyanide solution will be transferred to the cyanide storage tank on completion of a mix.

Cyanide will be dosed to the leach circuit via a ring main and control valves, while a dedicated pump will provide cyanide for cold cyanide wash and elution pre-soak as required.

A vertical spindle sump pump will be provided to service the cyanide and caustic mixing areas. This pump will report to the leach feed distribution box.

Caustic

Caustic (sodium hydroxide) will be delivered to site in one tonne bulk bags of 'pearl' pellets. Caustic will be added to the mixing tank via a bag breaker and be dissolved in treated water to achieve the required 20% w/v concentration. Caustic solution will be pumped to elution and electrowinning as required. The facility to dose caustic into the cyanide mixing tank will also be provided.

Flocculant

Flocculant for use in the pre-leach and tailings thickeners will be delivered to site in 700 kg bulk bags. Flocculant bags will be lifted by hoist to a bag breaker on the flocculant feed hopper. The vendor supplied flocculant mixing plant will automatically mix batches of flocculant with treated water and transfer the mixed flocculant to the flocculant storage tank after each mixing cycle is complete.

Flocculant will be distributed to the pre-leach and tailings thickener using dedicated variable speed positive displacement dosing pumps.

A vertical spindle sump pump will be provided to transfer any spillage to the tailings thickener.

17.3.10 Plant Utilities

Raw Water

Raw water for the project will be pumped to the plant from the bore field, approximately 80 km west of the plant site. A number of production bores will discharge into the raw water transfer tank, which will serve as a buffer between the bore pumps and the plant. A series of raw water transfer pumps will transfer raw water into the raw water tank.

The raw water tank in the process plant will have sufficient capacity to minimise the impact of short term supply interruptions. Duty / stand-by water pumps will be provided for the raw water distribution to the plant.

Treated Water

Treated water for the process plant will be produced by treating raw water in the water treatment plant. The treatment plant will be a containerised system consisting of pre-filtration (auto backwashing multimedia filters and cartridge filters), iron removal, anti-scalant dosing to prevent membrane scaling, reverse osmosis desalination, pH adjustment, and a clean-in-place system for membrane cleaning.

Treated water will report to the treated water storage tank and will be pumped to distribution points around the plant as required, including reagent mixing, elution area water and the closed circuit chillers for the SAG mill, ball mill and the mercury retort. Treated raw water from the treated water storage tank will be pumped using dedicated pumps for use as gland service water.

Brine reject from the water treatment plant will report to the brine tank. Brine water will be pumped to the mine for use as haul road dust suppression water. Any excess brine will report to the evaporation pond.

Fire Water

Fire water for the process plant will be drawn from the base of the raw water tank. Suctions for other water services fed from the raw water tank will be at an elevated level to ensure a fire water reserve always remains in the raw water tank.

The fire water pumping system will consist of an electric jockey pump to maintain fire ring main pressure, an electric fire water delivery pump to supply fire water at the required pressure and flowrate, and a diesel driven fire water pump that will automatically start in the event that power is not available.

Fire hydrants and hose reels will be placed throughout the process plant, fuel storage, and plant offices at intervals that ensure complete coverage in areas where flammable materials are present.

Potable Water

Treated water will be supplied to the plant potable water treatment plant. The water treatment facility will include micro filtration, ultra-violet sterilisation and chlorination. Potable water will be stored in the plant potable water tank and will be reticulated to the site ablutions, and other potable water outlets. A dedicated safety shower water tank will allow the safety shower and drinking fountain water to be reticulated on a ring main system to assist in keeping the potable water at a suitable temperature for use.

Process Water

The plant process water will consist of pre-leach and tailings thickener overflow and TSF decant return water, with raw water make-up as required. The process water pond will be situated adjacent to the raw water tank such that the raw water tank overflows to process water. With this arrangement the raw water tank can be kept full at all times.

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

Air Services

High pressure air at 700 kPa(g) will be provided by two high pressure air compressors operating in a lead-lag configuration. The entire high pressure air supply will be dried and can be used to satisfy both plant air and instrument air demand. Dried air will be distributed via the plant air receiver, crushing area receiver and the mill area instrument receiver.

Low pressure air will be supplied by several duty low pressure air blowers. A standby blower will also be provided. Low pressure air will be reticulated to the leach and CIL tanks and delivered down the agitator shaft.

Diesel

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

18. PROJECT INFRASTRUCTURE

18.1 Water Supply

The estimated water demand is in the order of 7,000 m^3/day (81 L/sec) and 34.0 Mm^3 over the projected life of mine (Table 18.1).

Table 18.1 Proposed Water Demand for the 3.4 Mtpa Option

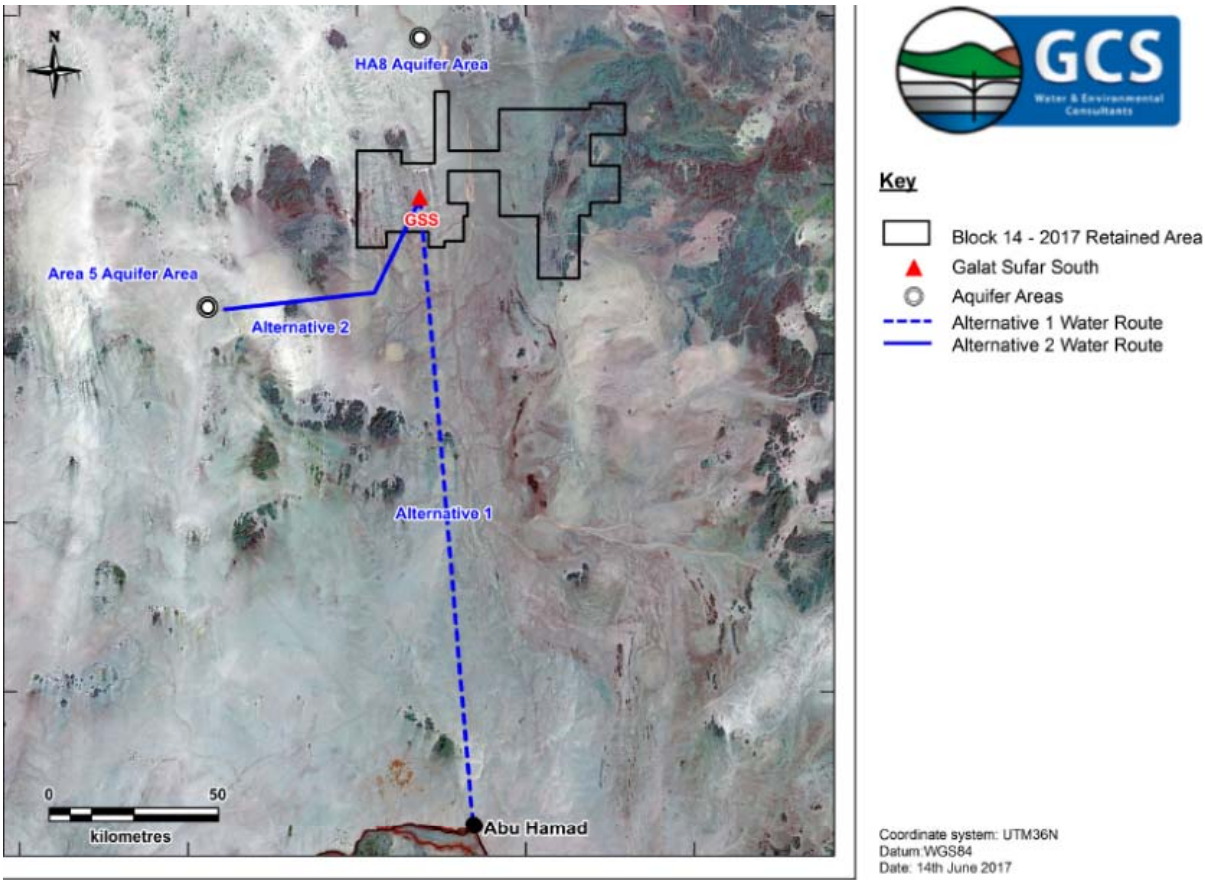
m^3/day	LOM (years)	Total (m^3)	Plant Throughput (Mtpa)
7,000	13.2	33,981,500	**3.4**

A water supply infrastructure concept study identified two potential water sources, the Nile and a local aquifer. Reconnaissance of the aquifer gathered information for the study.

The two alternatives sources of water, as shown in Figure 18.1 were identified:

- **Alternative 1:** Establishing a pump station and pipeline from the Nile River in Abu Hamed, located to south of the project area.
- **Alternative 2:** Establishing a pump station and pipeline from boreholes, located 80 km to the south west of the project area.

Figure 18.1 Water Supply Options



18.1.1 Alternative 1: Pipeline from the Nile River

To supply water to the project, extraction of water from the Nile River at Abu Hamad, located approximately 197 km south of GSS was considered due to the practicality of obtaining water and

access to existing infrastructure. Along the route there are several sectors where illegal artisanal mining is carried out, and there may be the risk of damage to the pipeline due to the movement of people and equipment from these sites.

The pipeline would be 197 km long and 630 mm diameter constructed from high density polyethylene (HDPE). The pumping system would consist of a water extraction barge pump inside the Nile, plus a main pump station with two pumps (1 operating + 1 stand by), which will have capacity to supply, continuously, 100% flow demanded by the process plant. The equipment would be supplied with power from the local grid with installed power of 220 kW.

18.1.2 Alternative 2: Borehole Well Field

As an alternative to the water supply from the Nile, a concept design of a groundwater extraction and pumping system from Area 5 (Figure 18.1) to the GSS area was completed. The proposed area for well field development is located approximately 80 km south west of the GSS project area.

In order to verify the viability of Alternative 2, regional and local scale groundwater resource investigations have been carried out in Area 5, approximately 80 km south west of the proposed mining area was identified (Figure 18.2). SkyTEM Surveys of Denmark, an airborne geophysical contractor specialising in water exploration were contracted to fly an electromagnetic survey where two historic production wells are located, “Station 6 Well” and “Bir Mufta Well” (see Figure 18.3). The survey in Area 5 returned positive results over a large area which was followed up with drilling.

Figure 18.2 Block 14 Groundwater Exploration Drilling Areas with Regional Topographical Setting

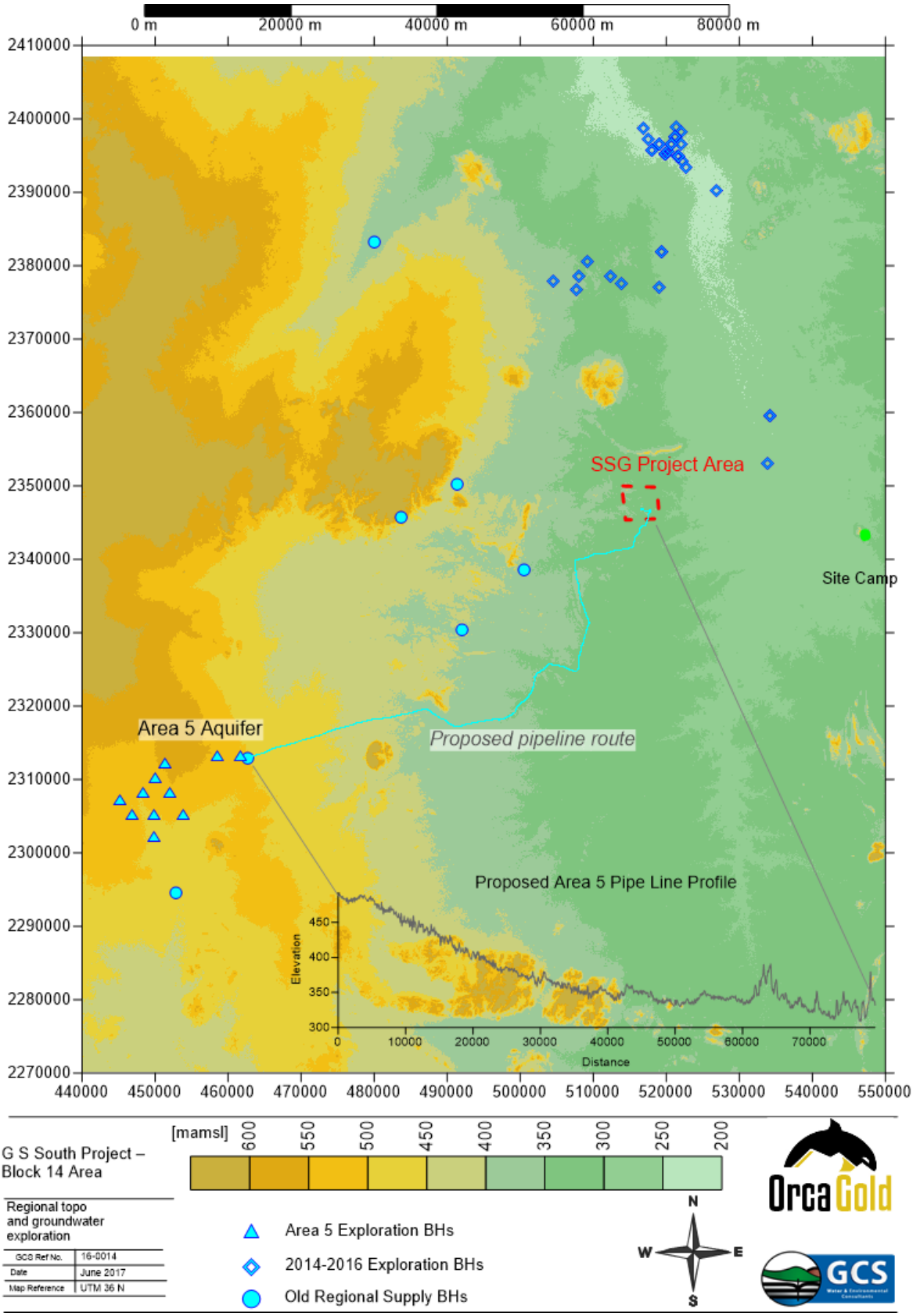
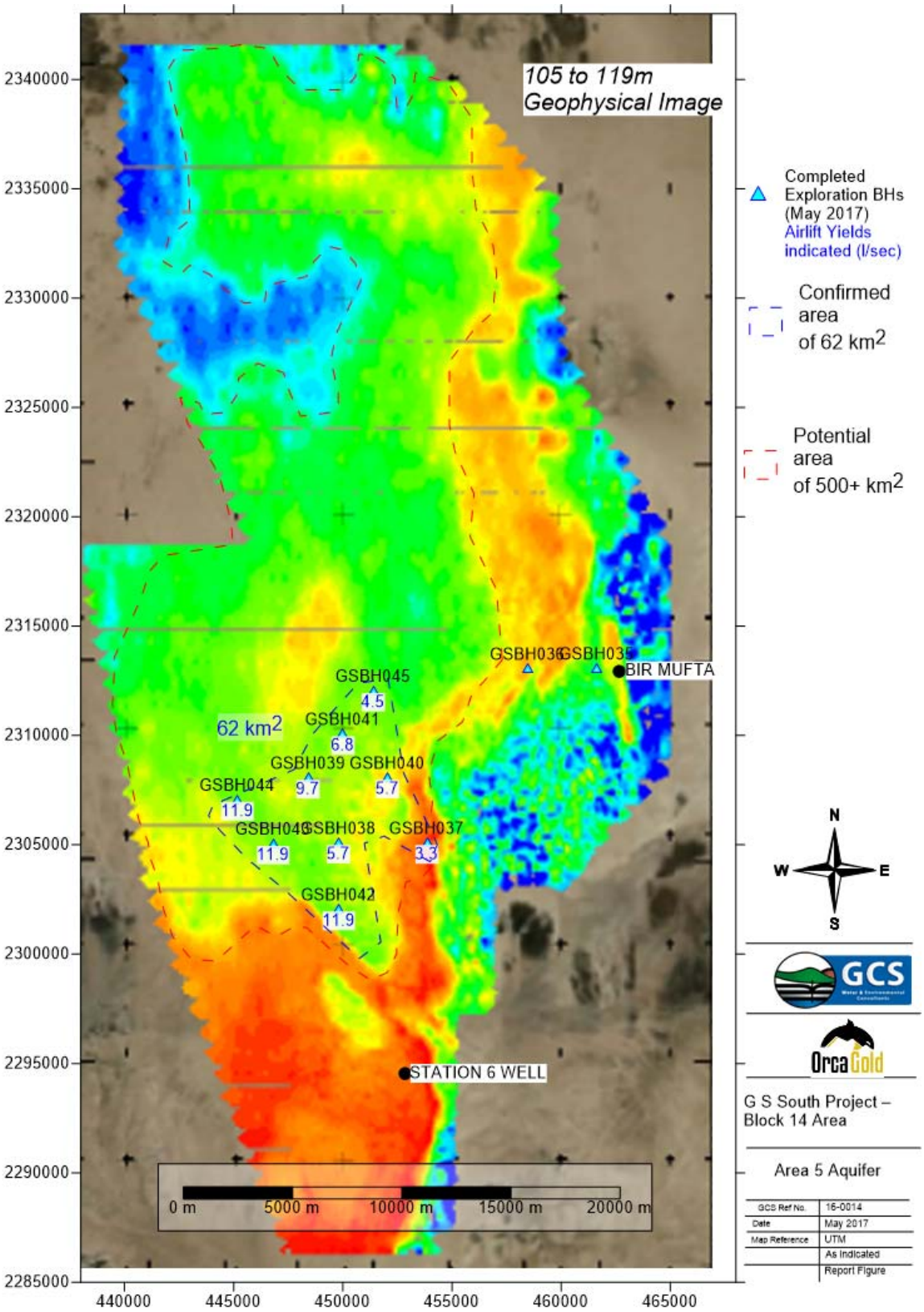


Figure 18.3 Area 5 Aquifer



18.1.3 Preliminary Groundwater Resource Indicators

Indications for Area 5

Eight exploration boreholes have been drilled within the Area 5 aquifer, which indicated high and constant airlift yields with the occurrence of groundwater within an inter-layered basin type sequence of course to medium grained sandstone (Table 18.2). Figure 18.4 illustrates the groundwater interception elevation depths within the layered basin sequence.

The results support the well field development as designed and that the identified aquifer is able to sustain the proposed mine water demand over the LOM period (Table 18.3). There is potential to extend the current identified aquifer extent in term of lateral and vertical dimensions. Further drilling and testing will aid aquifer delineation and characteristics.

Preliminary pump test has been completed on four of the boreholes and the testwork will be continued. Pump yields were limited due to the narrow diameter of the exploration boreholes, constant rate tests over 48 and 24 hr periods achieved between 3.1 and 18 L/s (Table 18.2). It can be seen that aquifer transmissivity[1] values range between 100 and 1,000 m^2/day (Table 18.2).

Table 18.2 Summary of Exploration Borehole Airlift and Preliminary Pump Tests

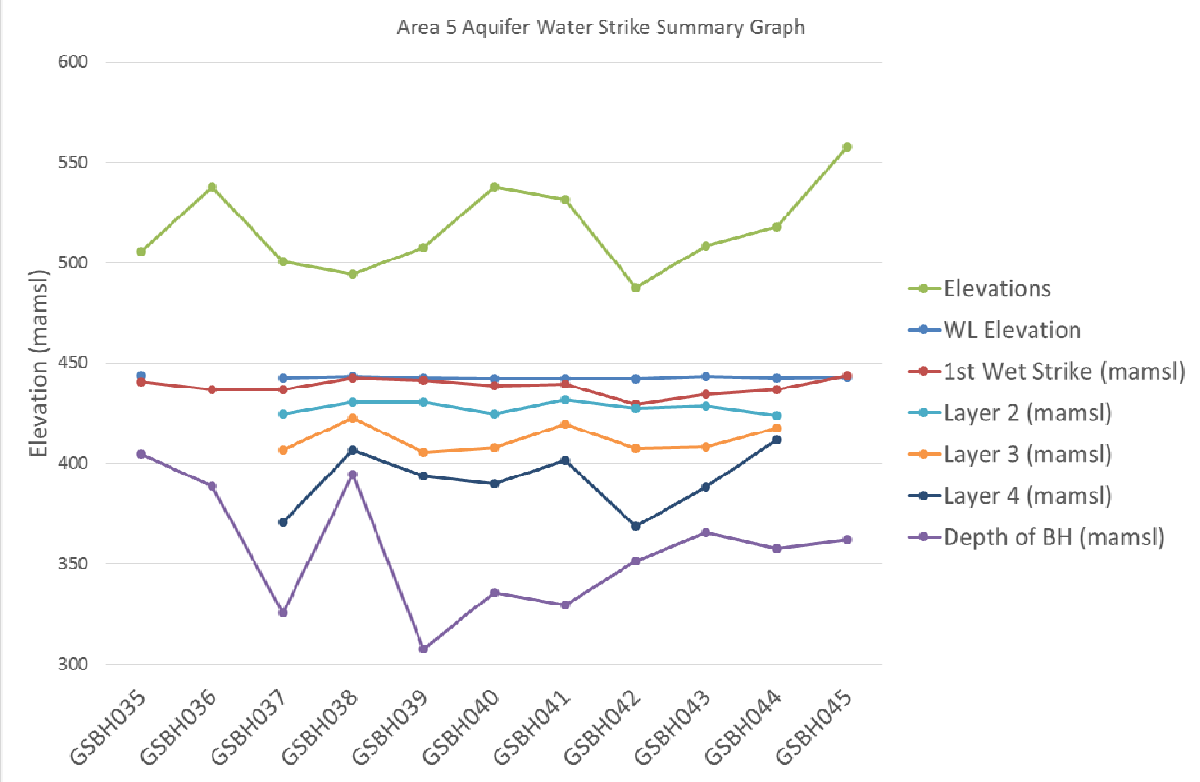
No.	Borehole ID	East	North	Completion Date	Depth (m)	Final Airlift Yield (L/sec)	Pump Depth (m)	Pump Rate (L/sec)	T High (m^2/day)	T Low (m^2/day)
1	GSBH035	461638.41	2312999.76	25-Apr-17	101	-	-	-	-	-
2	GSBH036	458470.69	2312975.27	27-Apr-17	149	-	-	-	-	-
3	GSBH037	453853.68	2305018.22	02-May-17	175	3.3	75	3.1	235	100
4	GSBH038	449801.62	2305022.18	06-May-17	100	5.7	-	-	-	-
5	GSBH039	448409.93	2307996.14	10-May-17	200	9.7	70	3.58	345	230
6	GSBH040	452049.77	2307999.49	16-May-17	202	5.7	-	-	-	-
7	GSBH041	450003.46	230995.71	21-May-17	202	6.8	-	-	-	-
8	GSBH042	449817.92	2301979.32	24-May-17	136	11.9	55	3.8	1,000	620
9	GSBH043	446807.00	2304980.00	28-May-17	143	11.9	104	18	1,000	720
10	GSBH044	445173.31	2306990.07	07-Jun-17	160	11.9	-	-	-	-
11	GSBH045	451392.96	2311970.78	11-Jun-17	196	4.5	-	-	-	-

Table 18.3 Theoretical Resource Available in the Area 5 Aquifer under Unconfined Conditions

Aquifer Zone	Area (km^2)	Main Aquifer Thickness (m)	Spec Yield	Indicative Volume (Unconfined) m^3
Area 5	62.0	10	10%	62,000,000

[1] *Transmissivity is the rate of flow under a unit hydraulic gradient through a unit width of aquifer of given saturated thickness. The transmissivity of an aquifer is related to its hydraulic conductivity*

Figure 18.4 Area 5 Groundwater Interception Elevations Summary Graph



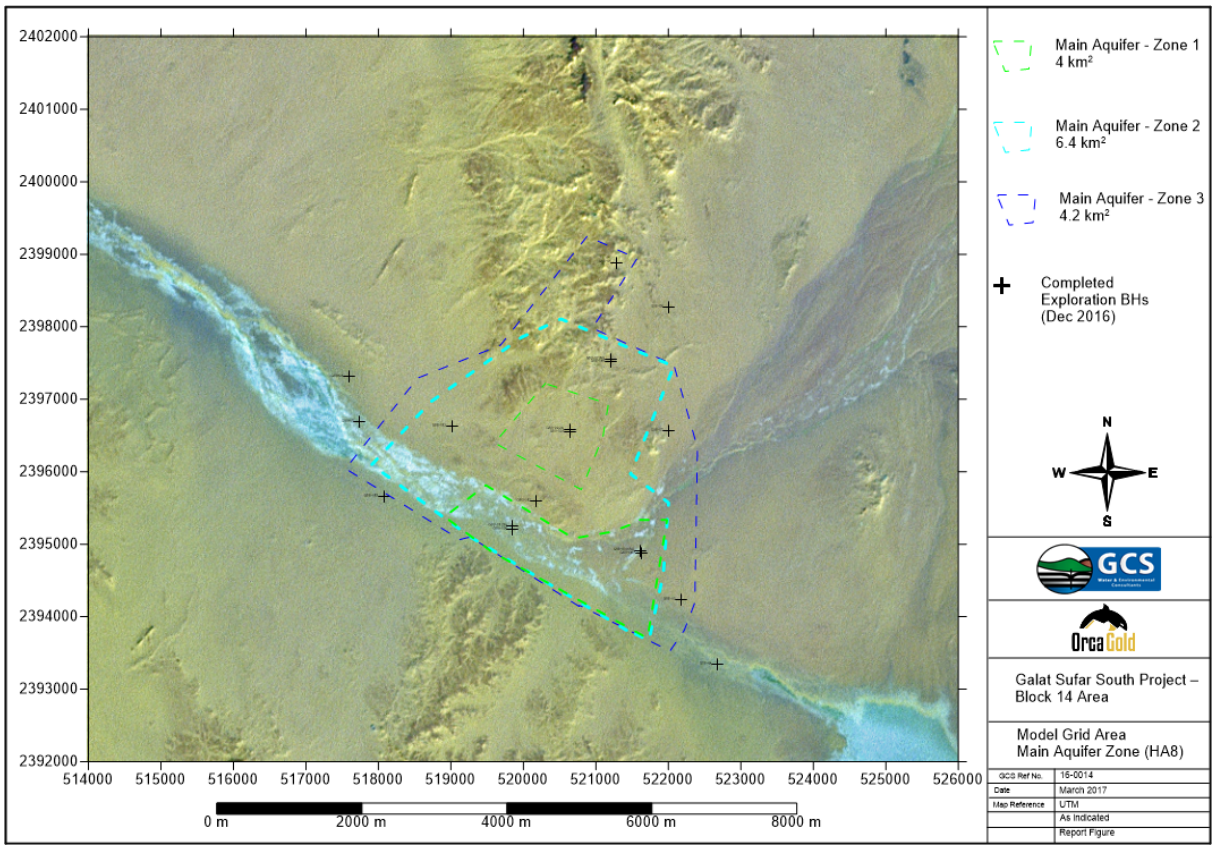
Indications for HA8

A second groundwater extraction area, located 50 km north of the project area, had been identified in 2016. After completion of the groundwater exploration, it was concluded that the aquifer potential for long term groundwater abstraction is limited and only viable for lower mine through put options. This might be considered for back-up supply purposes.

Based on the available information from the boreholes completed in HA8 and observations made during the drilling phase:

- The main groundwater strikes occurred within an interbedded sequence of course grained sandstone and sandy clays between 70 and 120 m below ground level (mbgl).
- Figure 18.4 supplies an overview of the main area where groundwater was intersected. This area is approximately 4 km^2 in size and demarcated around the successful boreholes. The total aquifer zone is approximately 14 km^2 in size.
- If one considers the available pump test data, it can be conservatively assumed that aquifer storage in this identified zone is between 10 Mm^3 and 20 Mm^3.

Figure 18.5 HA8 Aquifer Zones Delineated after Pump Tests Completed



HA8 Aquifer zones delineated after pump tests completed in December 2016.

Proposed Mining Areas - GSS and WD

0.7% of samples from the exploration programme intersected groundwater at the GSS and WD areas.

18.1.4 Typical Well Field Design Indicators

The well field would be developed and constructed for the abstraction of groundwater to a central distribution point:

- Depth of the boreholes will be in the order of 150 m.
- Drilling diameter will be in the order of 12” or 305 mm.
- Casing to line the boreholes will be 8” or 203 mm inner diameter and it is recommended that a combination of thick-wall uPVC casing and 304 stainless Steel V Wire Screens be used.
- For the water demand of about 7,000 m^3/day, between eight and 11 production boreholes will be required. The estimated cost per production borehole is US$85,000.
- To battery of deep wells and a main pumping station, which will pump the water to GSS through a 500 mm diameter HDPE pipeline. A diesel generator will be located at the main pump station.

The proposed follow-up testwork will provide further information for the well field design.

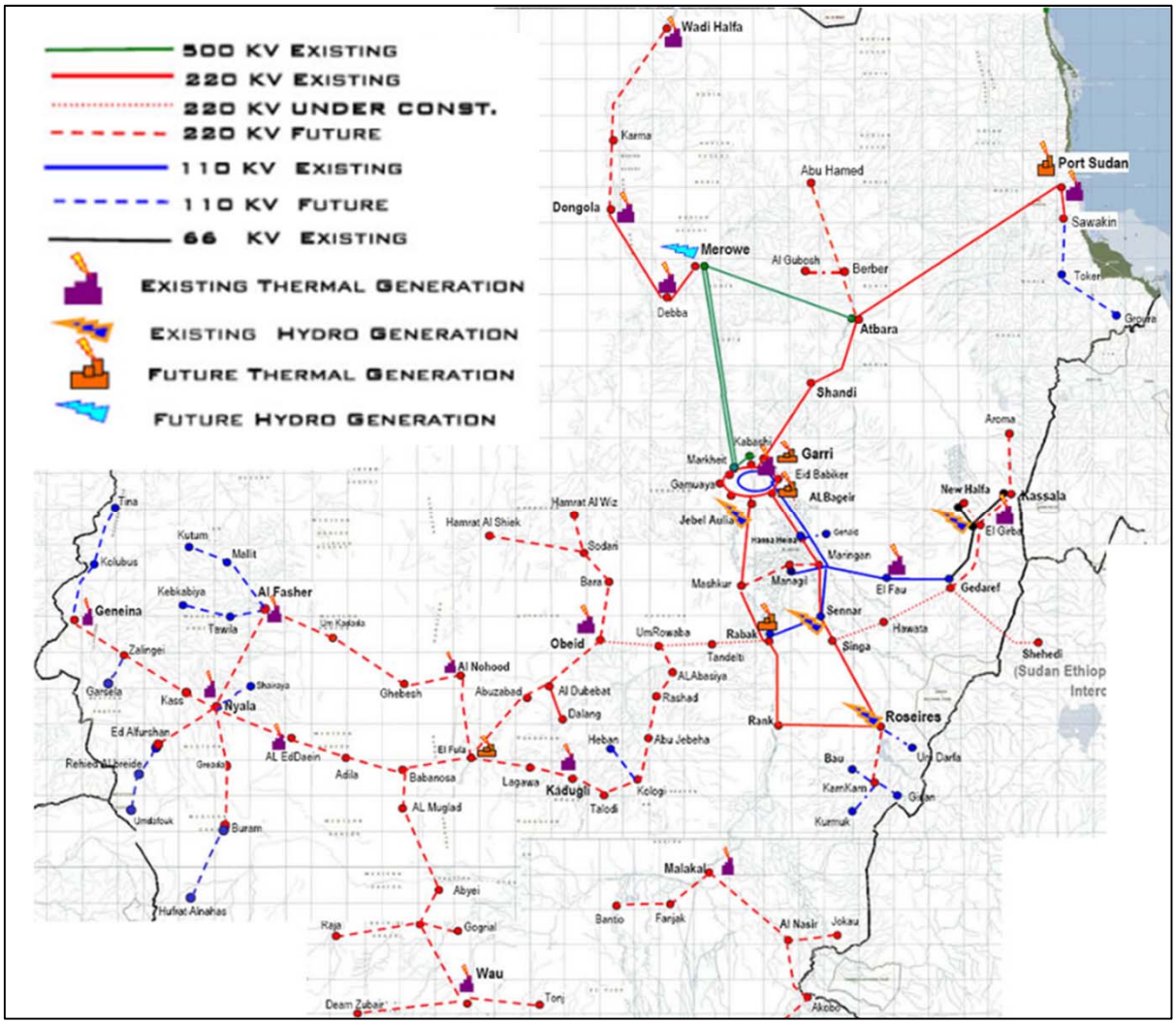
18.2 Electric Power

In Sudan, the Ministry of Water Resources and Electricity has divided the National Electricity Co-operation (NEC) into two different companies:

- Sudanese Electrical Transmission Company Ltd (SETCO) - SETCO is responsible for the electrical power grid and power supply.
- Sudanese Electrical Distribution Company Ltd (SEDC) - SEDC is responsible for operation and maintenance of the distribution infrastructure and networks.

Electrical power in the Sudan is produced from both hydro and thermal generation stations, with the transmission network depicted in Figure 18.7. The total output of the generation is approximately 1,500 MW. The actual consumption of domestic and bulk users in Sudan is approximately 1,750 MW, so there is a shortage in the power production of approximately 250 MW. The shortfall in power generation is supplemented from the Ethiopian grid.

Figure 18.6 Sudanese Power Generation & Distribution Network



Due to the shortfall in supply, every factory or an industrial project connected to the local power grid utilizes standby diesel generators in case of sudden power cuts, and this would be a necessary addition to the Block 14 project to utilize grid connection.

The current cost per kWh, including a 50% government subsidy is:

- Domestic users: US$0.02 /kWh.
- Bulk consumers: US$0.04 /kWh.

18.2.1 Project Power Supply Demand

The project plant and remote infrastructure power supply requirements have been evaluated based on the study mechanical equipment list and typical load / operating factors, and a plant availability of 8,000 hr/yr and factored for the design plant throughput. On this basis, the expected power demand for the process plant is estimated to be approximately 18.0 MW.

18.2.2 Project Power Supply Options

Two possible supply options were investigated:

- **Option 1**: Supply from the local power grid.
- **Option 2**: Self generation.

Option 1: Supply from the Local Power Grid

The closet take-off point substation or electrical line to the project is in the town of Abu Hamad, approximately 200 km south from the site. The connection of the project would involve the extension of this existing substation with an additional 220 kV feeder bay, 200 km of new 220 kV transmission line and a 220/33 kV receiving substation at the mine. The capital costs for this connection were estimated during the 2016 PEA at a total implementation cost of US$66.2M, based on a transmission line construction cost of US$221k /km and US$22M in substation construction costs and project management. A further allowance of US$5M has been allowed in the cost model to cover the cost of emergency generation to sustain site and remote infrastructure during outages.

Although the company would have to pay the total cost of the construction of this power line, the ownership of the power line would cede to the NEC, who would have the right to provide power to any other consumer from this line.

Option 2: Self Generation

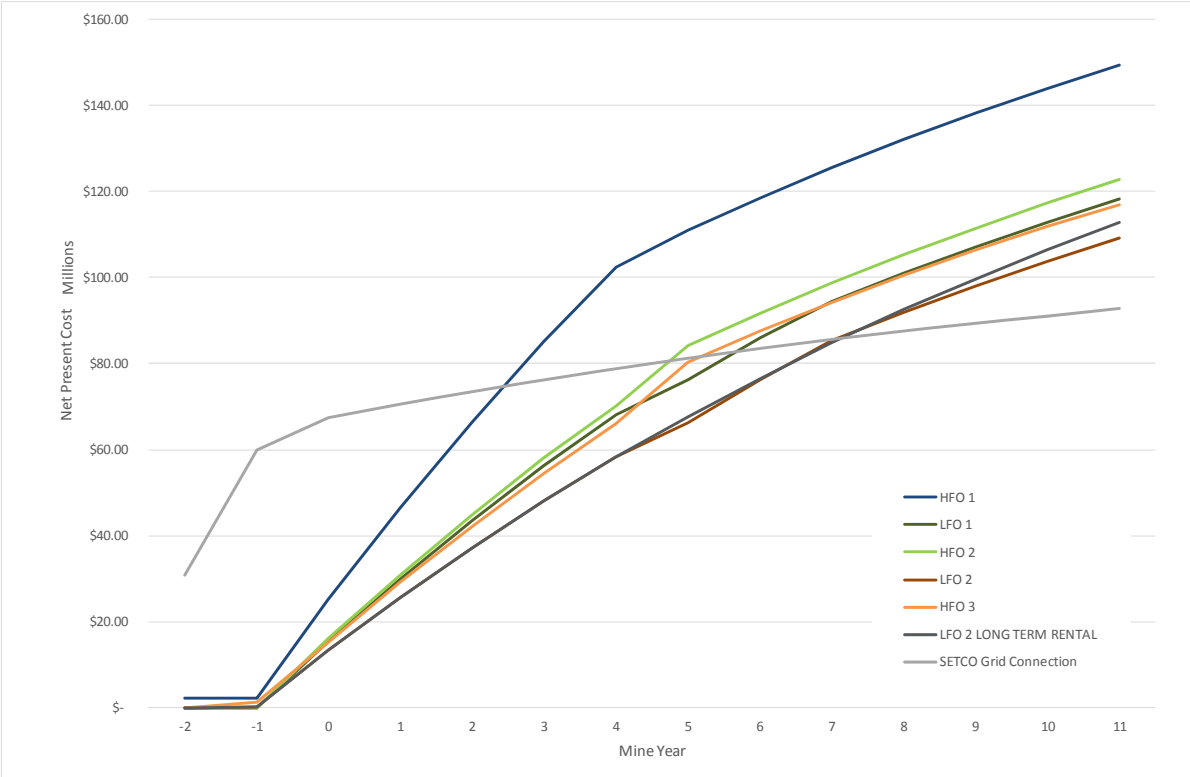
Budget tenders were solicited from the market for the provision of onsite power generation facilities. Both LFO and HFO fuel supply options were explored and collated into the evaluation.

On the basis of the offers tendered the preferred self generation option would utilize multiple diesel engine driven turbine generators to provide the total generation capacity. Based on a BOOT style contract, (build own operate transfer after five years), a power cost of US$0.168 /kWh was used in the operational cost analysis.

Selected Option

In order to select the most viable power supply option a Net Present Cost comparison was developed. Figure 18.7 below shows the calculated Net Present Cost of each key power option explored within this review. While grid connection provides the overall lowest NPC to the project over the life of mine, the payback period on the significant upfront capital doesn't occur until Year 7 of the mining operation which has been deemed unviable. The high delivered to site HFO price renders the HFO offerings uncompetitive against the less efficient LFO units. On the basis of this analysis Option 2 (self generation) based on LFO fuel has been selected as the preferred option at this stage.

Figure 18.7 Power Supply Options Comparison



18.3 Tailings Storage Facility

18.3.1 General

The principal design components of the tailings storage facility (TSF) consist of a paddock facility for storage of thickened tailings, including basin preparation, underdrainage works, decant water recovery system, embankment, monitoring instrumentation and closure. Refer to Knight Piesold report "*BR401-00126 TDR M17002 Geochemical Assessment of Tailings*" for details of the TSF design.

The key objectives on which the design is based are as follows:

- To produce a safe and economical design for tailings disposal at the Block 14 Project which complies with the relevant local, national, and international regulations, guidelines, and standards.
- To eliminate, manage or control environmental, health, and safety risks with zero harm aspiration.
- To produce a design which complies with relevant local and international regulations, guideline and standards using best available technology.
- Permanent, secure, and total containment of all solid waste material within an engineered disposal facility.
- Achieve a high density, consolidated tailings mass by employing controlled sub-aerial deposition of tailings.
- Control, collect, and remove free draining liquids from the tailings during operation for recycling as process water to the maximum possible extent.
- To reduce seepage from the facility during operation and on closure.
- Cost-efficient utilization of available material for embankment construction.

- Provide monitoring features for all aspects of the facility and associated works to ensure adopted performance standard can be measured and achieved.
- Effectively divert surface runoff from the upstream catchments around the facility.

A two cell paddock facility has been proposed to store high density thickened tailings. It is envisaged that two cells will be required to allow alternate deposition, and to improve drying and settling of the tailings to enable upstream raising.

18.3.2 Dam Failure and Environmental Consequence Category

An assessment was undertaken in accordance with ANCOLD guidelines. There are two consequence categories which need to be assessed in accordance with ANCOLD guidelines; the Dam Failure Consequence Category and the Environmental Spill Consequence Category. These were determined to be Medium High C and Minor High C respectively.

18.3.3 Embankment Configuration and Staging

The TSF is located within the valley located to the east of the process plant. The TSF has been designed as two cell paddock facility to allow alternate deposition and subsequent upstream raising. The facility will have a starter embankment, one downstream raise and then upstream raises up to LOM with a rate of rise less than 2 m. The starter embankment and the downstream raises will predominantly be constructed utilizing waste rock sourced from the pit whilst the subsequent upstream raises will be constructed with tailings with a 4 m wide outer select rockfill cover. A divider wall would be required to separate the two cells. The divider wall will be constructed of waste rock and will be raised by centreline technique over the LOM.

Construction of the embankment will be staged over the LOM with the aim of deferring the capital and operating costs involved with the embankment construction.

18.3.4 Basal Liner and Underdrainage

A full basal liner has been proposed for the tailings storage facility. In order to avoid damage to the liner, a 300 mm sand blanket has been proposed above the rock areas to form suitable surface to deploy the basal liner. The extent of the proposed basal liner may be reviewed once the nature of the founding conditions is well defined on completion of a geotechnical investigation.

The design incorporates a basin underdrainage system which will be installed above the basal liner. Any seepage from the tailings mass will be intercepted by these drains and discharged to the underdrainage sumps located at the geographically lowest points within the basin. The water collected within these sumps located at the bottom of the underdrainage towers will be pumped back to the supernatant pond and recycled back to the plant via the decant system. The design also incorporates a toe drain running along the upstream toe of the embankment to drain the tailings adjacent to the embankment.

18.3.5 Decant System and Operational Spillway

Supernatant water will be pumped back to the plant for reuse via two decant towers (one per cell) constructed and raised during operation. The decant towers will comprise a slotted concrete tower constructed of 1.8 m diameter pre-cast concrete pipes surrounded by a clean competent rockfill, which will limit the inflow of tailings solids into the tower whilst allowing water to flow into the towers.

Calculations indicate that each cell has sufficient storm storage capacity to store inflows generated as a result of the design storm event. Since the likelihood of TSF embankment overtopping during operation is considered low an operational spillway for the TSF is not considered necessary.

18.3.6 Monitoring and Instrumentation

Allowance has been made for the following:

- Survey prisms will be installed at regular intervals within the TSF along the crest of embankments to monitor any movements.
- A series of vibrating wire piezometer will be installed under the stack to monitor the saturation level in the tailings.
- To allow monitoring of groundwater levels and water quality, five groundwater monitoring stations will be installed around the perimeter of the TSF and two ground water monitoring stations installed downstream of the process pond.

18.3.7 Closure Capping

As rainfall at the project site is very low only a thin store and release cover is proposed to close the facility. The layer will further reduce the potential for erosion through either run-off of water or wind.

18.4 Surface Water Management

The process plant site and majority of the related infrastructure have been located within a valley having a significant upstream catchment (~83 ha). Hence, a clean water diversion channel has been proposed to be constructed along the west extremity of the process plant site to prevent flooding of the process plant during extreme rainfall events.

18.5 Fuel Supply and Storage

The two principal consumers of diesel will be the mining contractor and the power station. Allowance has been made in the capital estimate for site diesel storage, distribution to the mine services diesel day tank, distribution to the power station diesel day tank and heavy and light vehicle re-fueling facilities. In addition, a plant diesel day tank and reticulation via ring main has also been included as described in Section 17.

Allowance has been made for two 2,000 m^3 above ground vertical storage tanks for the bulk site diesel storage. Fuel storage requirements will be reviewed during the next phase as part of reviewing the BOOT power station tenders. If diesel power generation continues to be the preferred option, there is deemed to be adequate fuel supply available in Sudan and local transport contractors in the area in order to supply the Project with the required fuel.

18.6 General Infrastructure

18.6.1 Plant Buildings

The following plant buildings have been included in the capital estimate:

- Plant warehouse and office.
- Reagent storage facility.
- Plant workshop and office.
- Main office building.
- Plant office and control room.

- Clinic and emergency response building.
- Plant security gatehouse.
- Changerooms.

18.6.2 Mine Buildings

The following mine buildings have been included in the capital estimate:

- Mine warehouse and office.
- Reagent storage facility.
- Mine workshop and office.
- Heavy vehicle workshop
- Vehicle washdown facility
- Mine ablutions building.
- Changerooms.

18.6.3 Camp / Accommodation Village

An accommodation village, with all necessary facilities, has been included in the capital estimate. The number of accommodation units has been based on the number of employees and visitors anticipated on site at peak manning. The following buildings have been allowed:

- Camp gatehouse.
- Kitchen and mess hall.
- Office and storage.
- Laundry.
- Recreation facility.
- Accommodation units to suit both national and expatriate workers.

18.6.4 Communications

The site Information, Communication and Telephony Management System requirements need to be addressed during the next phase of the study. Currently there is limited access available on the local cellular network.

18.6.5 Security

The project will be monitored 24 hr a day by on site security personnel. Access to the plant will only be via the main security gate. The accommodation village will also have a gatehouse to restrict entry to Orca personnel and approved visitors only.

18.6.6 Health and Safety

The entire project site will be fenced to restrict access to the public, and in particular off-road vehicles that may be travelling through the desert.

18.6.7 Laboratory

An allowance for an on-site modular laboratory has been made in the capital estimate, along with a sample preparation shed.

18.6.8 Sewage

Site sewage will be managed on site via a vendor supplied sewage treatment plant, with cleaned effluent disposed of via spray fields or soak wells. Treatment plant sludge will be suitable for direct landfill burial.

18.6.9 Roads & Transport

The main supply route to the site will be either via Khartoum or Port Sudan. Roads are tarred between these cities and Abu Hamad (the closest main town to the project area). Bridges along these roads have been washed out by heavy rains during 2013. Access to the Project area from Abu Hamad is via a site access road, comprising a track through the desert with no speed restrictions or maintenance regime.

This track is also used by artisanal and small scale miners, water delivery vehicles and nomadic travellers. The routes are not clearly defined but are heavily used.

The GSS site is located 100 km east of the No. 6 railway station, North Sudan and is accessible from site via desert tracks.

18.6.10 Airstrip

Allowance has been made for a 1,000 m x 60 m unsealed air strip suitable to accommodate a Cessna 206H.

19. MARKET STUDIES AND CONTRACTS

No Market Studies were carried out for this study. The final product of the Block 14 project will be gold doré bars. These can be sold in the current market at prevailing global gold prices. Gold bullion sells on several international markets, the most well-known being the London Metals Exchange or LME.

No material contracts have been entered into as of the date of this report. Construction and mining contracts will be investigated in the next stage of the study.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

Mineesia Ltd, a UK based consultancy, was engaged in 2014 to undertake an assessment of the environmental and social conditions relating to the Block 14 project, and to oversee the planning and collection of environmental and social baseline data. This study has led to the preparation of a draft Environmental and Impact Assessment (EIA) to accompany the Preliminary Feasibility Study for the Project. The study is based on a site visit between November and December 2014, and ongoing baseline data collection with analysis based on the proposed plan to mine and process material at GSS with additional material being trucked in from a satellite mining operation at WD.

20.2 Sudan Legal Setting

The EIA for the Project has been undertaken in accordance with the requirements of the relevant Sudanese mining and environmental legislation. A summary of the key legislation considered is provided in Table 20.1.

Table 20.1 Key Sudanese Legislation

	Name	Year	Description
Mining	Mineral Wealth and Mining Resources Development Act	2015	Requirements for reporting mineral discoveries and control of licensing and mining contracts
Environmental	Environmental Conservation Act	2001	Protection of natural resources and requirement for EIA
	Environmental Protection Act	2001	Role of Federal & State authorities in environmental protection
	Water Resources Act	1996	Protection of freshwater resources

Exploration activities have been undertaken under the auspices of the Concession Agreement, which gives MSMCL the right to search for and win minerals, through the establishment of a mine operation. The Concession Agreement provides a framework of guidelines for compiling environmental management programmes and promoting management strategies to cope with mining-related effects on the environment. The concession agreement is based on the Mineral Development Act 2015 and Regulations, and covers the technical, financial, environmental and social aspects.

20.3 International Requirements and Guideline

The Project is classified as a Category A development in accordance with International Finance Corporation (IFC) Guidelines. As such, the Project is subject to an EIA process.

The primary requirement for any International Project is to comply with local and national regulations of the host country. These regulations are often supplemented by standards and guidelines from international financial institutions, particularly with regard to environmental and social components of the Project. The IFC Sustainability Framework, with associated Performance Standards on Environmental and Social Sustainability provide the basis for most EIA assessments.

Additional guidance is provided by the Equator Principles, which provide an approach to determine, assess, and manage environmental and social risk in project financing. The IFC Performance Standards (PSs) were revised in 2012 and apply to all projects that are not yet operational that require funding from International Financial Institutions.

The Project will adopt Orca Gold policies and implement its Environmental Protection and Management Programme (EPMP) to guide environmental and social management, and stakeholder and community relations. The Project will aim to conform to the environmental and social requirements of the IFC International Finance Corporation Performance Standards, its associated Environmental Health and Safety guidelines, ICMM and Equator Principles where they are relevant to the Project.

20.4 Project Permitting

Sudan has developed an integrated permitting infrastructure for formal mining operations. The development of the Project will be subject to receiving environmental approval of its design, environmental management programme and appropriate mitigation measures where required. Based on the provisions of the various legal requirements and sectoral laws as well as policies of different departments, the impacts of the proposed project will be assessed and appropriate mitigation measures recommended where appropriate.

The institutions at National Level responsible for the implementation and monitoring compliance to both national and international agreements include:

- The Higher Council for Environment and Natural Resources (HCENR).
- Ministry of Environment, Natural Resources and Physical Development (MENRPD).
- Federal Ministries of Health, Industry and Agriculture.
- Ministries and Councils at State Level.

The environmental acts and laws provide standards to be applied in assessing the probable environmental impacts of the project. It is important to note here that State Organs and Local laws deal with issues at State or Local levels, while the Federal Acts are more concerned with general directives and set limits and standards to certain environmental concerns without going into details of problems of local nature (Table 20.1).

20.5 Project Design and EIA

A Terms of Reference (TOR) for the EIA for the Project was submitted to the Sudan Mineral Resources Company (SMRC) for approval in May 2015. The TOR identified potential key environmental and socially sensitive areas and was based on the Project design below:

- Open pit mines – to be developed sequentially using standard open cast mining techniques.
- Processing Plant – material treated with Carbon-in-Leach (CIL) extraction methods, with a throughput of 3.4 Mtpa.
- Waste facilities – comprising waste rock dumps for stockpiling overburden and waste material from the open pit, potentially a tailings storage facility for storage of slimes from the CIL processing plant, and a domestic waste facility for non-mineral wastes.

- Water supply – including a reverse osmosis water treatment plant to treat raw water and potential construction of evaporation ponds for disposal of the brine produced by the reverse osmosis plants. Initial water requirements are expected to be up to 1.5 Mm^3/annum (4,000 m^3/day).
- Power plant – fired by diesel or heavy fuel oil.
- Associated infrastructure – including haul roads, offices, workshops, ablutions, and sewage treatment systems, explosives storage and a laboratory.
- Accommodation camp – to house employees.

20.6 Baseline Environmental Setting

20.6.1 Physiography

See Section 5.5.

20.6.2 Climate

As discussed in Section 5.2, the climate is arid with a hot season from June to September, when the maximum temperatures range from 45°C to 49°C and minimum of 23°C to 35°C. The coolest months of the year are January and February, with daytime high temperatures of 28°C and cool nights of approximately 5°C. The project area is hot and dry, with maximum summer temperatures of 52°C, and subject to strong winds in summer, predominantly from the north. The highest wind speeds are from the NNW direction. Dry northerly winds prevail, and in winter, strong cold winds and sandstorms are frequent (Countryside Studies, 2013). Data collected from site to date indicates that average wind speed on site is 11.6 ms-1, with a maximum wind speed of 28.6 ms-1. The winds result in dusty conditions, and there are no days which are completely windless.

20.6.3 Air Quality

Due to the remoteness of the Project area and absence of major industrial activities in the region, air quality at the site is not considered to be affected by any other industries. There are gaseous emissions from vehicles and mining equipment operated by artisanal miners in the locality. Their works are extensive covering large areas, both in terms of surface (colluvial) mining and hard rock pits. Their activities are transient; moving from one prospect to the next and the land surface is scoured and pockmarked by these workings. There is extensive use of mobile equipment, such as excavators, tractors and trucks together with individuals using hand held metal detectors. These sources of emissions are considered limited and widely dispersed. This dispersion combined with the lack of major industry in the vicinity of the Project region suggests that ambient air quality can be regarded as free from anthropogenic impact. As such, no further data has been collected regarding gaseous constituents.

As a result of the desert landscape around the Project site, the main impact on air quality in the region is naturally occurring suspended particulates (windblown dust) from the desert soils. The relatively flat topography and wind patterns create frequent sand storms (sometimes termed dust storms) which result in extreme concentrations that create low visibility across the Project area.

In 2014, the Project established a dust monitoring programme using passive samplers installed at the main camp, the drill camp and at the GSS prospect, to provide adequate coverage of the Project area. The drill camp pad was moved to the Wadi Doum prospect in April 2016 to monitor conditions in that area. The results provide a good baseline for the risk of dust impact naturally occurring in the area.

20.6.4 Noise and Vibration

The Project is located in a remote region with no permanent settlements in the vicinity. Potential receptors (excluding persons working on mine project) of noise and vibration in any future mining development are limited to artisanal and illegal small scale miners in the area, which are themselves already existing sources of noise and vibration. Given the small size, remoteness and impermanent nature of those operations no noise assessment of them has been undertaken. It is not known if these operations have measures to control and protect their workers from noise and vibration impacts.

20.6.5 Surface Water

As a result of the high rate of evaporation and low rainfall there are no permanent watercourses on-site or within the local vicinity. However storms can produce ephemeral floods in wadis and across open ground. Water quality data for wadis are not available due to the intermittent nature of flow. When the wadis do flow, erosion of the soils results in high levels of suspended solids. The nearest open water of significance is the Nile River, near Abu Hamed, approximately 200 km south of the Project site.

20.6.6 Groundwater Status and Supply

As described in Section 18, the Project area is within an arid zone, with groundwater emerging from a well at Talat Abda. The project area is close to the Nubian Sandstone formation which forms part of a regional aquifer system covering Sudan, Egypt, Libya, and Chad. Drilling activities have provided evidence of limited groundwater, although groundwater investigation and development are in their infancy in Sudan.

20.6.7 Groundwater Quality

Water quality results indicate that groundwater through the area is not continuous, which is supported by evidence that some monitoring wells in the area are dry. Analytical results show that the groundwater samples exceed aesthetic guidelines for a number of parameters. With the exception of Talat Abda, the water is not considered suitable for livestock or irrigation, and the presence of high levels of sulphate, chloride and sodium further support this assessment. The high levels of manganese present issues with water use for domestic (non-drinking) purposes. Scaling and staining are likely to be prevalent if this water is used without pre-treatment. Under the criteria available for irrigation use, water from these wells would be classified as 'unsuitable' for irrigation purposes, particularly for sensitive plants such as food crops.

20.6.8 Soils, Pre- and Existing Land Use

The Harmonised World Soil Database (FAO et al, 2009) indicates a mosaic of soil groups within the Project region, namely:

- Arensol is a sandy soil featuring very weak or no soil development.
- Fluvisol is a young soil located in alluvial deposits.
- Leptosol is a very shallow soil over hard rock or unconsolidated very gravelly material.
- Regosol comprises unconsolidated material derived from freshly deposited alluvium or sands with very limited soil development.
- Luvisol is a soil with subsurface accumulation of high activity clays and high base saturation.

The nature of the soils in the Project area and surrounding region in combination with hyper-aridity and scarce water resources has meant that potential soil productivity has been of low to nil capacity.

Nomadism in the Project area and its surrounding region is considered very limited too none due to the absence of oases and wells.

The predominant land use of the region has been artisanal mining. Mineral wealth has represented an inherent land capacity in the Nile River basin and some adjacent lands throughout history. The development of gold mining over time, from Predynastic (ca. 3000 BC) until the end of Arab gold production times (about 1350 AD), including the Pharaonic period in the remote deserts of Egypt, is well known. Around 250 gold production sites were mined during different periods in the remote region of the Eastern Desert of Egypt, which is now upper northern and north-eastern Sudan. Gold mining (and to a lesser extent copper and iron) in the above regions and around the greater Project area has a long and important history in the evolvement of various civilisations and cultures in the Nile basin region.

The existing land use is restricted to mining activities, due to the arid conditions. The major land use in the area is artisanal gold mining, primarily illegal (not licenced) interspersed with licenced operations that are being encouraged, and the issuance of exploration permits to registered companies, by the national Government.

20.6.9 Flora and Fauna

The Project area occurs within the desert ecoregion, one of sixteen terrestrial regions within the Nile River Basin. The desert ecoregion contains few visible plants, with most areas covered by bare soil or rock. Some shrubs of *Acacia tortilis* can be found in the true desert, but most plants are only visible in the rainy season and survive the dry season as seeds (these are annual plants).

There is considerable variation in vegetation described north of the Project, particularly along wadis. Scanty and stunted vegetation dominated by *Acacia*, *Tamarix*, and *Calotropis* species has been found along some wadis. Where water is available from the crystalline basement, doum palms (*Hyphaene thebaica*) and sparse thorny shrubs exist. None of the plant communities or single plants identified are rare or unusual in a national or international context. None of the species recorded are included in the IUCN Red List.

The desert habitats in the Project area provide a harsh environment with extreme temperatures, little water and little food for wildlife. They are capable of supporting only a limited suite of specialist species, although the habitats they inhabit are common and widespread. Wildlife presence is being assessed on an ongoing basis through the use of both passive infra-red (PIR) cameras, as well as opportunistic photographs of other wildlife. A database of the wildlife present in the area is being developed, based on lists of fauna that potentially occur there. PIR cameras have been installed at the main camp burn pit and at GSS: these are downloaded monthly.

To date, Golden Jackals (or African golden wolf, *Canis anthus*, as they are now known) have been captured by the PIR cameras (Figure 20.1). Other species recorded by Project personnel in the area, but not captured by the cameras, include the Dorcas gazelle (*Gazella dorcas*), wildcats (the Hausa wildcat, *Felis silvestris hausa*, and the East African wildcat, *Felis silvestris rubida*) and Ruppell's foxes (*Vulpes rueppelli*i). No protected species such as gerbils have been identified in the exploration area. Several snakes (unidentified) and scorpions have been captured and killed in the camp. There are no known endangered species present in the Project area.

Figure 20.1 Golden Jackal Captured by Remote Camera

Source: Orca Gold.

20.7 Baseline Social Setting

20.7.1 Population

The Project is situated in the remote and largely uninhabited north-western corner of Sudan, and therefore does not have a sustained population in place.

20.7.2 Regional and Local Economic Structure

The low rainfall in the region, combined with thin and poorly developed soils, and rocky plateau topography, limit the width of the Nile's floodplain, and thus the extent of fertile land, to approximately 1 km east and west. The agricultural zone along the Nile is therefore unsuitable for large-scale irrigation projects, and farmers are restricted to small, intensively cultivated fields nourished by water raised from the Nile.

The main source of employment in the region arises from illegal artisanal mining and associated support services. Revenue for communities is also generated by fees imposed on the illegal miners. A small amount of formal employment is provided by mineral exploration companies, including MSMCL. These workers migrate from time to time, once they have completed exploration activities in their project areas.

Abu Hamad is the only significant local population centre. People from the Ababda tribe have settled in small towns and villages close to the Nile and, wherever possible, Orca employs some members of this community as staff or as casual labourers.

There are a large number of illegal artisanal miners working in the region. Their works are extensive covering large areas, both in terms of surface mining and hard rock pits. Their activities are transient, moving from one prospect to the next, and the land surface is scoured and pockmarked by these workings. There is extensive use of heavy equipment, such as excavators, tractors and trucks

together with individuals using hand held metal detectors. It is understood that the artisanal miners are not local and come from all regions of Sudan. All processing of the material produced by the artisanal miners takes place in Abu Hamad.

20.7.3 Landscape and Visual

No established communities are present in the area that would be potential receptors for landscape and visual impacts. The terrain is general flat, with some rocky outcrops. Artisanal and small scale miners utilize the area and have created visual impacts of the area. No permanent sealed access roads will be created in the region, as access will be restricted to a graded earth road. As such, development of the Project is not anticipated to detract from the existing landscape.

20.7.4 Infrastructure

Further to the description in Section 5.4, there is no existing infrastructure such as electricity or water supply present. Supply routes would be through Khartoum and Port Sudan, with tarred roads between Abu Hamed and these cities. Access to the exploration camp and Project area is via desert tracks from Abu Hamad that are used by the artisanal mining community. Routes through the desert are not clearly defined but are heavily used. Two cafeterias are located along the route, which are predominantly fuel stations with telephone communications. Within the project area roads are unpaved, and in some areas sand has become compact due to continual usage of road systems. These compacted roads are seen more around the main camp, as heavy water trucks and lorries use these roads daily.

To the extent relevant to the Project, the infrastructure requirements for the current and planned exploration activities as well as development of the project are minimal and operational requirements can be satisfied using local and regionally available materials and services, mainly from Abu Hamad, Atbara (where the company operates a small logistics office) and Khartoum (administration office).

20.7.5 Land Acquisition and Resettlement

Since there are no formal local communities in the vicinity of the project, no resettlement is anticipated. Under Sudanese legislation, the land is owned by the State, and the operation Concession Agreement gives the company the right to utilize the land. As such, no land acquisition will be required.

20.7.6 Archaeology and Cultural Heritage

The region has been worked historically, with workings dating to circa 4,000 BC. Sites of interest have been observed during the exploration phase, but none have been found within the target resource areas of GSS or WD. However, illegal artisanal mining operations have used these workings as an indicator of prospective zones and as such these areas are often disturbed. Some areas of Pharaonic interest have been identified in the exploration block, although they had already been disturbed when observed by MSMCL (Figure 20.2). The Project has implemented a chance finds procedure to mitigate against any impacts from their exploration activities. The Chance Finds Procedure defines a series of steps to minimize physical impacts to cultural heritage by providing a process for conducting an archaeological look ahead-survey, monitoring of ground disturbing activities, and responding to any tangible cultural heritage encountered unexpectedly during exploration.

Figure 20.2 Historic Artefacts

Left: Pharaonic tomb. Right: Grinding stone (Source: Orca Gold).

20.8 Potential Environmental Impacts

The EIA includes an assessment of the environmental and social impacts of the Project's planned development. The assessment was based on the site visit and undertaken according to internationally accepted procedures for such assessments. The assessment of impacts included:

- An examination of existing baseline conditions taking into account existing artisanal mining operations and their corresponding impacts, as requested by the local authorities.
- Consultation with local and national stakeholders to identify impacts of concern.
- An estimation of the nature and scale of impacts caused by the construction, operation and eventual closure of projected / anticipated / planned development.
- A description of mitigation measures to be adopted to eliminate, avoid, reduce, or compensate for adverse environmental impacts.
- An evaluation of the residual impacts present after the mitigation measures have been implemented.

The Project is likely to give rise to a range of low level environmental and social impacts. However, assuming the implementation of the proposed mitigation measures, these impacts are considered manageable and controllable and therefore would enable effective environmental and social development, operation and closure of the Project.

20.9 Environmental Protection and Management

A provisional Environmental Protection and Management Plan (EPMP) has been prepared for the Project. MSMCL is committed to managing the impacts of its operations, in conformance with recognised international best practice. The purpose of the EPMP is to ensure that appropriate control and monitoring measures are in place to deal with all significant impacts of the Project. The EPMP has been designed so that it can be reviewed and updated on a systematic basis in line with company policies. It is also designed to be developed throughout the life of the Project. A basic EPMP has been developed for Exploration activities, which will be expanded to include Construction, then Operational Phases of the Project.

The EPMP includes details of the area of impact, objectives to reduce negative or enhance positive impacts, specific targets adopted to achieve those objectives, and definition of responsibilities for implementing the programme. It is a live document that can be reviewed and updated on a systematic basis, in-line with the principles of continual improvement.

20.10 Monitoring

MSMCL has implemented an environmental and social monitoring during the exploration phase of the Project. The objective of monitoring is to characterise environmental conditions, including groundwater, air quality (specifically airborne dust) and ecology, and will continue to observe any changes in the social environment of the Project area. This information is, and will continue to be analysed to inform the environmental management of the Project and support the development of action levels and response plans for future monitoring of construction, operation and closure phase impacts.

MSMCL has already developed and implemented appropriate sampling procedures for water and dust sampling. Passive infrared cameras will continue to be used to detect the presence of wildlife.

Groundwater levels are recorded on a monthly basis, and water quality samples have been collected and sent for analysis on a six monthly basis. Interactions with artisanal and small scale miners are recorded in a daily diary, along with wildlife observations and any other items of environmental interest.

20.11 Public Consultation

As part of the environmental assessment, public consultation and disclosure is required. In order to ensure that the Project is developed and operated in an appropriate manner, MSMCL endorses the concept that effective engagement with its stakeholders is an essential component of the assessment process and its on-going 'license to operate'. MSMCL is therefore committed to a pro-active programme of communications with all relevant stakeholders. A Public Consultation and Disclosure Plan (PCDP) has been developed by MSMCL.

The Project has few stakeholders due to the remoteness of the location. The closest people to the site are illegal artisanal miners. The Project has engaged with these illegal miners previously, and they may perceive that they are adversely impacted or consider themselves to be representatives of impacted people. However, including these artisanal miners in the EIA and Public Consultation process and documenting their concerns could legitimise their activities. As such, any engagement with these miners is considered informative only, and no formal consultation has been undertaken.

Given the nature and location of the Project, a series of small gatherings have been held with distant communities (Geibat peoples, as well as people in Abu Hamed and Atabara). These form the basis used to conduct the Public Consultation. This targets the consultation process on those who most likely will be affected by the Project. A log of these gatherings has been kept, summarizing the numbers of people engaged with, their activities and any issues or concerns they may have with the project.

21. CAPITAL AND OPERATING COSTS

21.1 Introduction

The overall study capital cost estimate was compiled by Lycopodium and is presented here in summary format. The various elements of the Project estimate have been subject to internal peer review by Lycopodium and have been reviewed with Orca for scope and accuracy.

The capital cost estimate was developed to an accuracy level range of ±30% to cover engineering, procurement, construction, and start-up of the mine and processing facilities, as well as the ongoing sustaining capital costs. The capital cost estimates were developed for a conventional open pit mine, CIL process plant and supporting infrastructure for an operation capable of treating 3.4 Mtpa of material. For the purpose of this PEA, power supply via a third-party Build Own Operate Transfer, and a contract mining scenario have been assumed.

The estimate covers the direct costs of purchasing and constructing the CIL facility and infrastructure components of the project and an allowance for mining related infrastructure.

Indirect costs associated with the design, construction and commissioning of the new facilities, owner's costs, and contingencies have also been estimated, based on percentages of the direct capital cost estimate. Risk amounts are specifically excluded from this estimate. A breakdown of the capital cost estimates is shown in Table 21.1.

All costs are estimated in United States dollars (US$) as at 1Q17.

21.2 Capital Cost Summary

The capital estimate is summarized in Table 21.1 and 21.2. The initial project capital cost is estimated at US$211M, including a contingency allowance of US$41M.

Table 21.1 Capital Estimate Summary (1Q17, ±30%)

Main Area	US$'000s
Mine Costs	$8,332
Treatment Plant Costs	$122,392
TSF Initial cost	$7,902
Engineering	$15,810
Owners Costs	$15,077
Subtotal	**$169,514**
Contingency	$41,113
Grand Total	**$210,627**

The total LOM cost is estimated at US$303M including sustaining capital costs of US$92M, as shown in Table 21.2.

Table 21.2 Sustaining Capital Estimate Summary (1Q17, ±30%)

Main Area	US$'000s
TSF	$54,709
TSF closure	$4,418
Generator	$1,535
Other	$31,385
Grand Total	**$92,046**

21.3 Direct Capital Costs - Mining

Total pre-production mine development costs have been estimated by Deswik Europe to be US$8.3M, primarily associated with waste pre-stripping.

21.4 Capital Cost Estimate – Process Plant and Infrastructure

To develop the process plant and infrastructure cost estimate a mechanical equipment list was compiled, based on the process requirements, and major equipment pricings were obtained from technology / equipment suppliers or from a database of similar size projects and factored as required for the project capacity. The estimated cost was benchmarked against similar sized projects and was found to be within the cost range of the various projects.

Project infrastructure includes mine infrastructure as itemised in Section 18.6.2.

The EPCM estimate was factored based upon Lycopodium's recent experience with similar type and size of project. Expenses such as catering and accommodation for the Engineer's site personnel, as well as site telecommunications costs, are included in the estimate.

A contingency allowance is included to make specific provision for uncertain elements of cost within the project scope. Contingencies do not include allowances for scope changes, escalation, or exchange rate fluctuations. Contingency has been applied to all parts of the process plant estimate.

21.5 Operating Costs – Mining

Contract open pit mining costs were derived by Deswik Europe from first principles based on equipment required and include pit and dump operations, road maintenance, mine supervision and technical services cost. In addition Wadi Doum mining costs include the haulage of material to the process plant. The average open pit operating cost (US$/t mined) is shown below:

Table 21.3 Mining Costs

Mine Area	Mineralized Rock (US$ /t)	Waste Rock (US$ /t)
Main	2.79	2.55
East	2.83	2.59
NEZ	2.57	2.34
Wadi Doum	2.71	2.57
Total	**2.79**	**2.55**

In addition, a transfer cost of US$7.74 /t will be incurred on material from Wadi Doum.

21.6 Operating Cost – Plant and Infrastructure

The Operating Cost Estimate (OPEX) for the plant and infrastructure has been divided into multiple cost centres, with Fixed and Variable costs calculated for each cost centre for each different material types. The operating cost estimate is presented in Table 21.4 and is deemed to have an accuracy of ±30% based on pricing as at 2Q17. The process operating cost includes all direct costs to produce gold bullion for the Project.

In general, costs have been built up from first principle estimates, with quotations obtained for major reagents and consumables and consumption rates based on metallurgical testwork, calculations or modelling. Minor reagents, laboratory, expatriate labour rates and a number of G&A costs have been sourced from the Lycopodium database. Power consumption has been calculated from the gross power required to achieve the desired grind size on each material type, based on OMC comminution modelling, plus the remaining installed power from the mechanical equipment list, with suitable drive efficiency and utilization applied and factored for the design throughput. The total power draw was used to calculate power costs based on a diesel fuel price of US$0.50 /L.

Table 21.4 Operating Cost per Material Type

Cost Centre	Fixed US$'000/y	Main Fresh	Main Trans	Main Oxide	East Fresh	East Trans	East Oxide	Wadi Fresh	NE Oxide
		Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t
Power (excluding grinding)	3,127	1.60	1.60	1.60	1.60	1.60	1.60	1.60	1.60
Grinding Power	-	2.72	2.97	3.05	3.47	3.35	3.15	3.92	2.43
Operating Consumables	-	6.03	5.46	4.13	5.19	6.29	5.10	6.27	4.21
Maintenance Materials	2,921	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Laboratory	581	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06
Process & Maintenance Labour	2,633	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Administration Labour	2,647	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
General & Administration	6,106	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
TOTAL	**18,015**	**10.65**	**10.33**	**9.09**	**10.56**	**11.55**	**10.15**	**12.10**	**8.54**

21.7 Exclusions

The following items have been excluded from the operating cost estimate:

- All sunk costs.
- Government monitoring and compliance costs.
- All head office costs and corporate overheads.
- Withholding taxes and other taxes such as GST or VAT.
- Escalation.
- Financing costs.
- Foreign exchange fluctuations.
- Interest charges.
- Political risk insurance.
- All costs associated with areas beyond the battery limit of the study.
- Plant rehabilitation costs.
- Land compensations costs.
- Subsidies to local communities.

- Licence fees.
- Royalties.
- Contingency
- All mining and exploration costs, except power costs for mining services within Lycopodium scope of work.
- Maintenance costs of all mine, haul and plant access roads.
- Gold refining costs.
- Bullion transport costs.
- Bullion marketing costs.
- Bullion insurance in transit costs.
- Tailings storage costs, including future lifts and rehabilitation which are included in sustaining capital.
- Tailings dust suppression costs.
- External Government TSF monitoring and compliance costs.
- Any rehabilitation or closure costs.

22. ECONOMIC ANALYSIS

22.1 Introduction

The economic analysis is based on Indicated and Inferred Mineral Resources and mine schedule as per Table 22.4.

A preliminary economic analysis has been carried out for the project using a cash flow model. The model is constructed using annual cash flows by taking into account annual processed tonnages and grades for the CIL feed, process recoveries, metal prices, operating costs and refining charges, royalties and capital expenditures (both initial and sustaining).

The financial assessment of the project is carried out on a "100% equity" basis and the debt and equity sources of capital funds are ignored. No provision is made for the effects of inflation. Current Sudan tax regulations are applied to assess the tax liabilities. All amounts in this section are presented in US$. Discounting has been applied from the first year of operation.

The model reflects the base case and technical assumptions as described in the foregoing sections of this report.

22.2 Model Inputs and Assumptions

The model inputs and assumptions used in the economic analysis are summarized in Table 22.1 and, unless otherwise stated, is used in the model.

Table 22.1 Model Inputs and Assumptions

Model Inputs	Unit / Value
Base Currency	US$
Base Date	2nd Quarter 2017
Money Terms	July 2017
Sudan Royalty (charged against Revenue)	7%
Sudan Tax Rate	15%
NPV Discount Rate	7%
Metal Price Scenario – Fixed	US$1,200 / oz
Refinery Charges & Shipping	US$4 /oz
Assumptions	
Capex excludes Finance Charges & Fees	
Capex excludes Pre-production Investigations	
Capex Amortisation/Depreciation based on no salvage value	
Capex excludes Escalation	
Tax paid on an Annual basis	

22.2.1 Capital Cost Expenditures

Pre-production capital expenditures are defined in Table 22.2. Sustaining capital for the Plant, Mining, and TSF expansion costs have been phased over the life of the project; see Table 22.3.

22.2.2 Royalties

Royalties at 7% have been included for the LOM, and will be charged against the revenue.

22.2.3 Cost of Sales

Cost of Sales includes freight and refining costs. A value of US$4.00 /oz gold recovered has been allowed for in the model.

22.2.4 Depreciation

Depreciation is calculated using the units of production method starting with first year of production and can be summarized as follows:

- Initial pre-production capex depreciated over the total LOM, based on units of production.
- Capitalised pre-production costs (i.e. cumulative exploration and PEA costs) to date depreciated over the total LOM, using estimated total capitalised pre-production costs of US$47.9M up to date (31 Mar 2017).
- The annual sustaining capital is assumed to be largely:
  - repairs and maintenance; or
  - short life items, which would not be capitalised, then depreciated over several years.

Remaining sustaining capital items (i.e. TSF) each depreciated separately based on respective remaining LOM.

22.2.5 Inflation

Inflation was not included in the cash flow analysis.

22.2.6 Operating Costs

Annual fixed and variable costs, as per Sections 21.5 and 21.6, are included in the cash flow.

22.3 Financial Model

The operating costs, capital costs, mining and production schedule, and other technical considerations, defined elsewhere in this report are reflected in the post-tax project cash flow model in Table 22.5.

The pre-tax and post-tax financial results of the project are summaries in Table 22.4. On a pre-tax basis, the project has a Net Present Value (NPV) of US$278.2M at a discount rate of 7%, an Internal Rate of Return (IRR) of 26.5%, and undiscounted payback of 2.6 years following commencement of production; on a post-tax basis the NPV is US$227.7M at a discount rate of 7%, the IRR is 23.1% and the discounted payback period is three years following commencement of production.

Figure 22.1 shows the pre-tax and post –tax cumulative cash flow for the project over the LOM; the payback period corresponds to when the cumulative cash becomes positive during Year 5 for the pre-tax and the post-tax model. Figure 22.2 shows the annual and cumulative post-tax cash flow.

Table 22.2 Pre-production Capital Expenditure

Item	Unit	Total	Year	
			-2	-1
Mine	US$'000	8,332	1,961	6,371
Process Plant	US$'000	122,392	73,435	48,957
TSF	US$'000	7,902	4,741	3,161
EPCM	US$'000	15,810	9,486	6,324
Owner	US$'000	15,078	9,047	6,031
Construction Sub Total	**US$'000**	**169,541**	**98,671**	**70,844**
Contingency	US$'000	41,113	24,668	16,445
Construction Total	**US$'000**	**210,627**	**123,338**	**87,289**

Table 22.3 Sustaining Capital Expenditure

Item	Unit	LOM
TSF	US$'000	54,709
TSF Closure	US$'000	4,418
Generator	US$'000	1,535
Other	US$'000	31,385
Sustaining Total	US$'000	**92,046**

Figure 22.1 Cumulative Cash Flow

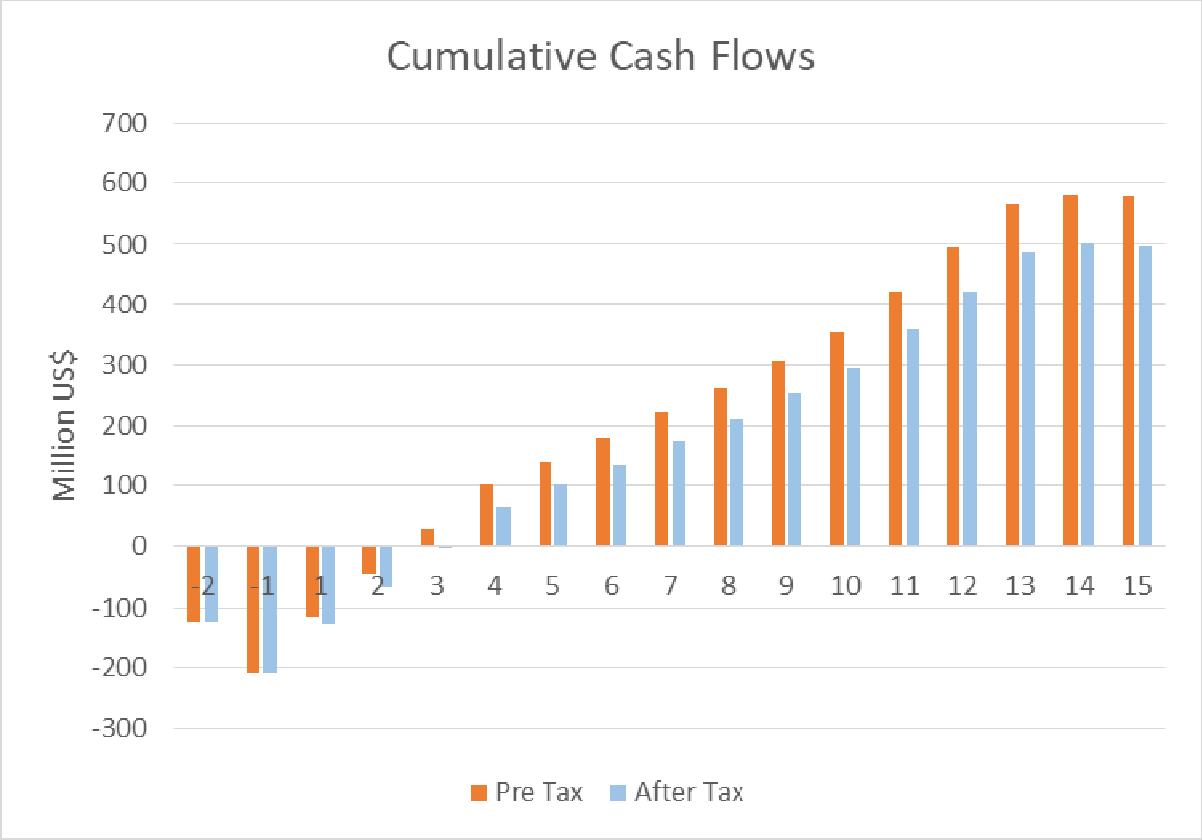


Table 22.4 Financial Model (Operations)

Block 14 - PEA May '17 vs1j b.xlsx			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Mining																		
East Total Tonnes	'000 t	26,561.3		126.6	1,239.9	1,393.4	1,821.2	1,686.7	2,505.0	2,542.6	2,862.5	3,035.5	1,908.3	1,975.8	1,690.2	1,292.6	1,906.7	574.1
East Total Au Grade	g/t	1.28		1.13	1.36	1.23	1.26	1.20	1.08	1.12	1.28	1.27	1.29	1.36	1.38	1.37	1.43	1.82
Main Total Tonnes	'000 t	11,826.5		83.4	992.1	1,276.9	1,132.8	1,273.1	446.4	407.9	471.5	364.5	794.7	985.8	1,216.1	1,345.8	1,035.7	-
Main Total Au Grade	g/t	1.73		1.39	2.12	1.78	1.89	1.78	1.32	1.34	1.64	1.51	1.50	1.53	1.82	1.74	1.73	-
NEZ Total Tonnes	'000 t	3,293.1		-	275.5	-	-	-	-	-	-	-	697.0	438.4	493.7	761.5	457.6	169.3
NEZ Total Au Grade	g/t	1.26		-	1.85	-	-	-	-	-	-	-	1.35	1.00	1.22	1.19	1.21	1.08
WD Total Tonnes	'000 t	2,718.2		42.6	380.2	426.0	450.9	450.0	450.0	450.0	68.5	-	-	-	-	-	-	-
WD Total Au Grade	g/t	2.52		1.88	2.28	2.60	2.43	2.58	2.62	2.64	2.58	-	-	-	-	-	-	-
Total Mined Tonnes	'000 t	44,399.1		252.6	2,887.7	3,096.3	3,404.9	3,409.8	3,401.4	3,400.4	3,402.5	3,400.0	3,400.0	3,400.0	3,400.0	3,400.0	3,400.0	743.4
Total Mined Au Grade	g/t	1.47		1.34	1.79	1.65	1.62	1.60	1.32	1.35	1.36	1.30	1.35	1.36	1.51	1.48	1.49	1.65
Total Mined Indicated Tonnes	%	92%		78%	92%	98%	98%	96%	94%	94%	97%	96%	91%	93%	89%	90%	83%	50%
Total Mined Inferred Tonnes	%	8%		22%	8%	2%	2%	4%	6%	6%	3%	4%	9%	7%	11%	10%	17%	50%
Total Waste Tonnes	'000 t	104,429.8		1,737.5	6,301.5	7,720.5	8,553.6	8,643.3	11,574.0	11,426.1	9,151.6	7,926.3	9,363.2	8,254.4	5,262.5	3,734.6	4,279.3	501.5
Global Strip Ratio		2.4		6.9	2.2	2.5	2.5	2.5	3.4	3.4	2.7	2.3	2.8	2.4	1.5	1.1	1.3	0.7
Processing																		
Oxide Tonnes Processed	'000 t	10,936.6			3,020.2	2,076.3	606.4	1,233.3	1,711.3	827.5	130.8	13.9	276.7	237.8	254.7	365.2	108.9	73.6
Oxide Au Grade Processed	g/t	1.39			1.75	1.51	1.25	1.23	1.11	1.03	1.38	0.81	1.45	0.92	1.13	1.11	1.31	1.06
Oxide Ag Grade Processed	g/t	1.82			2.60	2.45	1.15	1.29	1.16	1.34	3.17	2.10	1.07	0.68	0.83	0.82	0.97	0.78
Oxide Recovered Au Metal	'000 ozs	443			155	92	22	44	55	25	5	0	12	6	8	12	4	2
Oxide Recovered Ag Metal	'000 ozs	218			87	56	7	17	21	12	5	0	3	2	2	3	1	1
Transition Tonnes Processed	'000 t	8,431.1			94.0	760.0	1,360.8	622.0	1,091.5	1,270.6	1,129.1	401.0	657.7	217.7	76.6	353.5	345.4	51.1
Transition Au Grade Processed	g/t	1.37			1.80	1.67	1.69	1.23	1.13	1.21	1.36	1.41	1.38	1.07	1.38	1.24	1.18	1.11
Transition Ag Grade Processed	g/t	2.77			3.39	3.75	4.25	2.39	1.99	2.09	2.70	3.46	2.71	1.64	2.06	1.86	1.77	1.67
Transition Au Recovered Metal	'000 ozs	308			4	34	61	21	33	42	41	15	24	6	3	12	11	2
Transition Ag Recovered Metal	'000 ozs	420			6	48	91	28	43	53	56	22	33	8	3	14	13	2
Fresh Tonnes Processed	'000 t	25,031.4			26.1	260.1	1,437.6	1,554.5	598.6	1,302.4	2,142.6	2,985.0	2,465.6	2,944.5	3,068.7	2,681.4	2,945.6	618.7
Fresh Au Grade Processed	g/t	1.55			2.30	2.70	1.72	2.03	2.25	1.69	1.36	1.28	1.33	1.42	1.55	1.56	1.54	1.76
Fresh Ag Grade Processed	g/t	2.90			12.17	14.49	5.40	5.14	11.05	6.12	2.43	1.99	1.81	1.80	1.78	1.71	1.91	2.70
Fresh Au Recovered Metal	'000 ozs	1,026			2	19	66	86	37	59	76	100	86	110	126	111	119	28
Fresh Ag Recovered Metal	'000 ozs	1,461			6	69	149	148	122	151	111	130	96	113	115	95	120	37
Total Tonnes Processed	'000 t	44,399.1			3,140.3	3,096.3	3,404.9	3,409.8	3,401.4	3,400.4	3,402.5	3,400.0	3,400.0	3,400.0	3,400.0	3,400.0	3,400.0	743.4
Total Au Process Grade	g/t	1.47			1.75	1.65	1.62	1.60	1.32	1.35	1.36	1.30	1.35	1.36	1.51	1.48	1.49	1.65
Total Ag Process Grade	g/t	2.61			2.70	3.78	4.18	3.24	3.17	3.45	2.55	2.16	1.93	1.71	1.72	1.63	1.87	2.44
Total Au Recovered Metal	'000 ozs	1,776			161	145	149	150	125	125	123	115	122	123	137	135	134	32
Total Ag Recovered Metal	'000 ozs	2,099			99	173	247	192	186	216	171	153	132	123	120	113	134	39

Block 14 - PEA May '17 vs1j b.xlsx			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Pre-Production CapEx																			
Mine	US$ '000	8,332	1,961	6,371															
Process Plant	US$ '000	122,392	73,435	48,957															
TSF	US$ '000	7,902	4,741	3,161															
EPCM	US$ '000	15,810	9,486	6,324															
Owner	US$ '000	15,078	9,047	6,031															
Construction Sub Total	US$ '000	169,514	98,671	70,844															
Contingency	US$ '000	41,113	24,668	16,445															
Construction Total	US$ '000	210,627	123,338	87,289															
Sustaining CapEx																			
TSF	US$ '000	54,709			7,603	10,825	2,582	3,073	3,051	3,588	3,212	3,029	2,612	2,881	3,063	3,063	3,063	3,063	-
TSF Closure	US$ '000	4,418																	4,418
Generator	US$ '000	1,535			-	-	-	-	1,535	-	-	-	-	-	-	-	-	-	-
Other	US$ '000	31,385			2,615	2,615	2,615	2,615	2,615	2,615	2,615	2,615	2,615	2,615	2,615	2,615	-	-	-
Sustaining Total	US$ '000	92,046	-	-	10,219	13,441	5,198	5,689	7,201	6,204	5,827	5,645	5,227	5,496	5,678	5,678	3,063	3,063	4,418
Revenue																			
Gold Revenue	US$ '000	2,120,596	-	-	191,991	172,539	177,513	179,201	149,569	149,774	146,362	137,602	146,148	146,550	163,542	160,734	160,556	38,516	-
Silver Revenue	US$ '000	35,626	-	-	1,673	2,937	4,197	3,267	3,154	3,674	2,909	2,590	2,243	2,082	2,043	1,916	2,277	664	-
Revenue		2,156,222	-	-	193,664	175,476	181,710	182,467	152,723	153,448	149,271	140,192	148,391	148,631	165,585	162,650	162,833	39,180	-
Selling Costs	US$ '000	(15,501)	-	-	(1,037)	(1,270)	(1,584)	(1,370)	(1,244)	(1,368)	(1,176)	(1,071)	(1,018)	(982)	(1,029)	(990)	(1,075)	(286)	-
Royalties	US$ '000	(150,936)	-	-	(13,556)	(12,283)	(12,720)	(12,773)	(10,691)	(10,741)	(10,449)	(9,813)	(10,387)	(10,404)	(11,591)	(11,385)	(11,398)	(2,743)	-
Op Costs - Mining	US$ '000	(410,294)		(5,063)	(24,821)	(30,186)	(34,038)	(34,063)	(40,375)	(41,454)	(33,502)	(30,572)	(33,850)	(31,609)	(24,372)	(20,236)	(22,357)	(3,795)	-
Op Costs - Process	US$ '000	(704,139)			(47,894)	(49,382)	(54,479)	(54,466)	(54,507)	(55,073)	(54,577)	(53,971)	(53,549)	(53,650)	(53,556)	(53,443)	(53,929)	(11,664)	-
Operating Profit	US$ '000	**880,416**			**106,355**	**82,355**	**78,889**	**79,795**	**45,906**	**44,812**	**49,567**	**44,764**	**49,586**	**51,987**	**75,038**	**76,595**	**74,074**	**20,693**	**-**
Net Cash Flows, before tax	US$ '000	**577,743**	**(123,338)**	**(87,289)**	**96,137**	**68,914**	**73,691**	**74,106**	**38,705**	**38,609**	**43,740**	**39,119**	**44,359**	**46,490**	**69,359**	**70,916**	**71,011**	**17,630**	**(4,418)**
NPV Pre Tax	US$ '000	**278,226**																	
IRR Pre Tax		**26.5%**																	
Depreciation		**(313,273)**	**-**	**-**	**(24,098)**	**(22,625)**	**(23,538)**	**(24,111)**	**(20,450)**	**(20,908)**	**(20,859)**	**(20,048)**	**(21,761)**	**(22,451)**	**(26,011)**	**(26,932)**	**(29,373)**	**(10,109)**	**-**
Other Sustaining		**(35,803)**	**-**	**-**	**(2,615)**	**(2,615)**	**(2,615)**	**(2,615)**	**(2,615)**	**(2,615)**	**(2,615)**	**(2,615)**	**(2,615)**	**(2,615)**	**(2,615)**	**(2,615)**	**-**	**-**	**(4,418)**
Tax	US$ '000	**(80,364)**	**-**	**-**	**(11,946)**	**(8,567)**	**(7,910)**	**(7,960)**	**(3,426)**	**(3,193)**	**(3,914)**	**(3,315)**	**(3,782)**	**(4,038)**	**(6,962)**	**(7,057)**	**(6,705)**	**(1,588)**	**-**
Net Cash Flows, after tax	US$ '000	**497,379**	**(123,338)**	**(87,289)**	**84,190**	**60,347**	**65,781**	**66,146**	**35,279**	**35,415**	**39,826**	**35,804**	**40,577**	**42,452**	**62,398**	**63,859**	**64,306**	**16,043**	**(4,418)**
NPV After Tax	US$ '000	**227,699**																	
IRR After Tax		**23.1%**																	
Cash cost per ounce Au	US$	**701**	**0**	**0**	**533**	**624**	**663**	**662**	**828**	**837**	**790**	**806**	**789**	**770**	**646**	**625**	**643**	**553**	**0**
All-in sustaining cost per ounce Au		**752**	**0**	**0**	**598**	**719**	**700**	**702**	**875**	**888**	**839**	**857**	**834**	**818**	**690**	**670**	**669**	**657**	**0**
															-				
Payback period - Pre Tax	Years	**2.6**																	
Payback period - After Tax	Years	**3.0**																	

22.4 Financial Summary

The results of the financial model are summarized in Table 22.5.

Revenue generated per lithology is shown in Figure 22.2; where total oxide provides 25%, total transition provides 17% and total fresh provides 58% of the total revenue.

A breakdown of the total cash costs is shown in Figure 22.3. The combined variable and fixed plant operating costs account for 63% of the total cash cost; of the plant operating cost, fixed costs account for 34%, and variable costs account for 66% of the plant operating costs.

Table 22.5 Financial Model Summary

Description	Units	LOM
Tonnage Feed	Mt	44.4
Feed Grade Processed (average)	g/t	1.47
Gold Recovery (average)	%	84.5
Production Period	Years	13.2
Waste Rock	Mt	104.4
Total Mined	Mt	148.8
Stripping Ratio	W:O	2.35:1
Gold Production	'000 oz	1,776
Annual Gold Production (average)	'000 oz/y	135
Silver Production	'000 oz	2,099
Gross Revenue	US$M	2,156.2
Operating Profit	US$M	880.4
Total Operating Costs	US$M	1,114.4
Pre-production Capital Cost	US$M	210.6
Sustaining Capital Cost	US$M	92.0
Total Capital Cost	US$M	302.7
Mining Cost / t Mined	US$ /t	28.93
Process Cost / t Processed	US$ /t	15.86
Cash Cost / oz Au recovered	US$ /oz	701
AISC / oz Au recovered	US$ /oz	752

Figure 22.2 Revenue Generated per Material Type

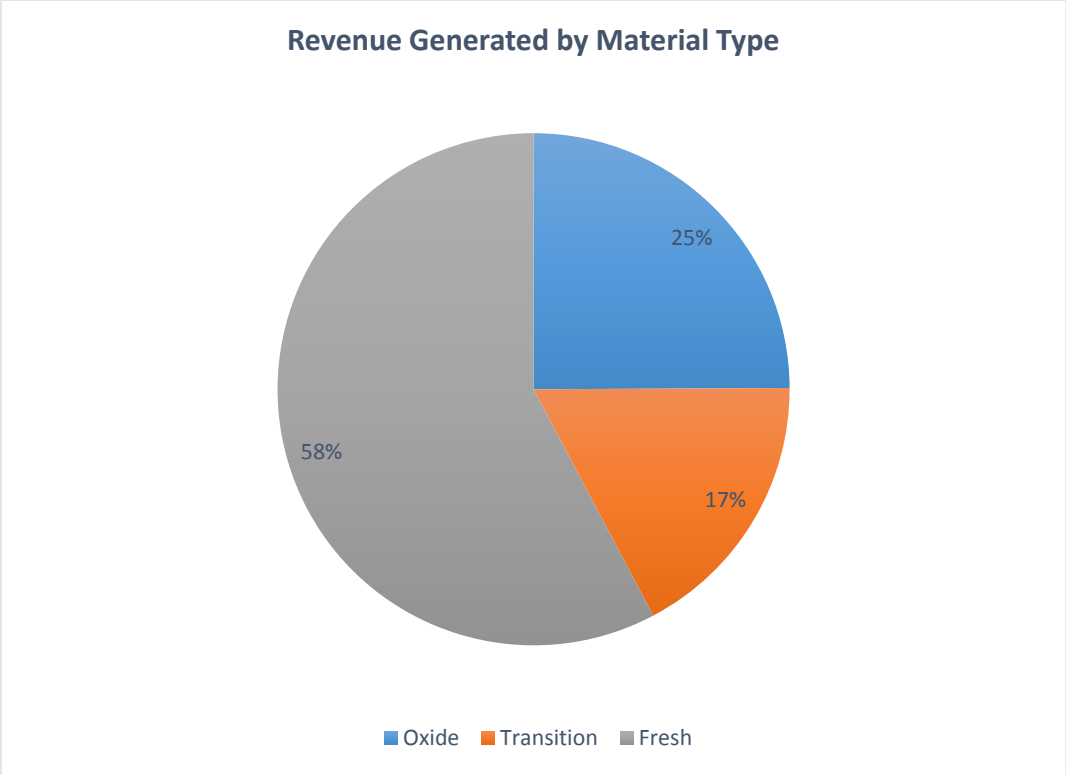
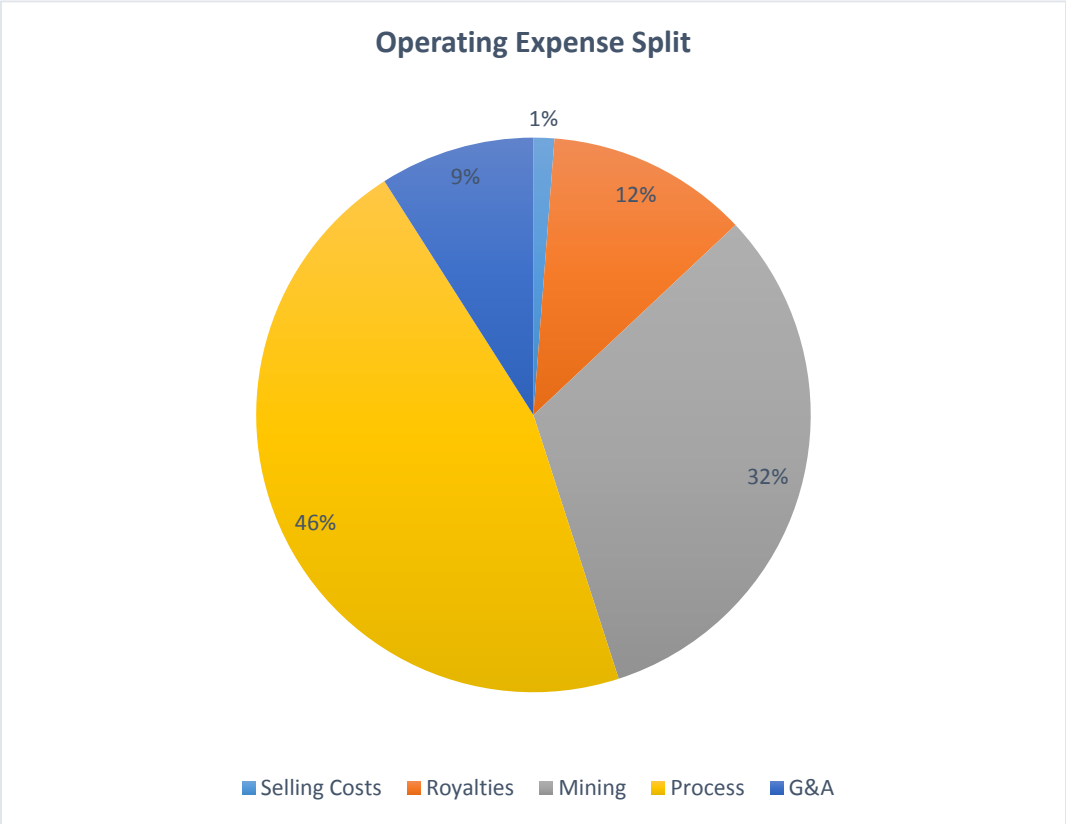


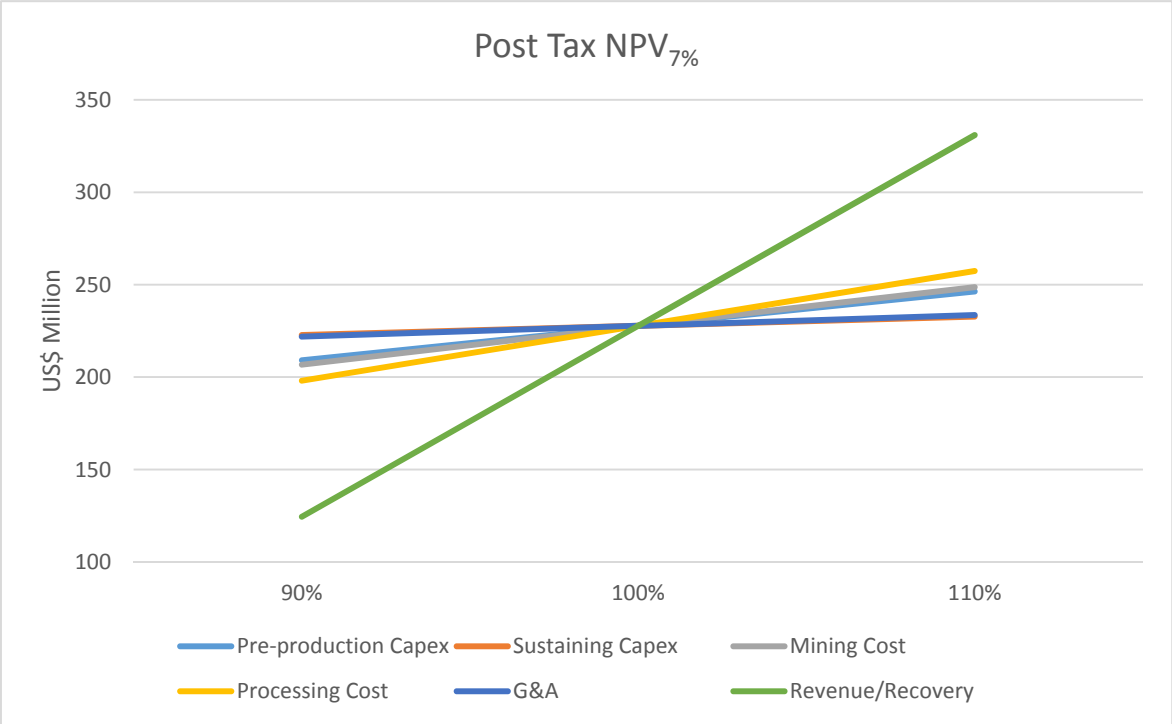
Figure 22.3 Operating Expense Split

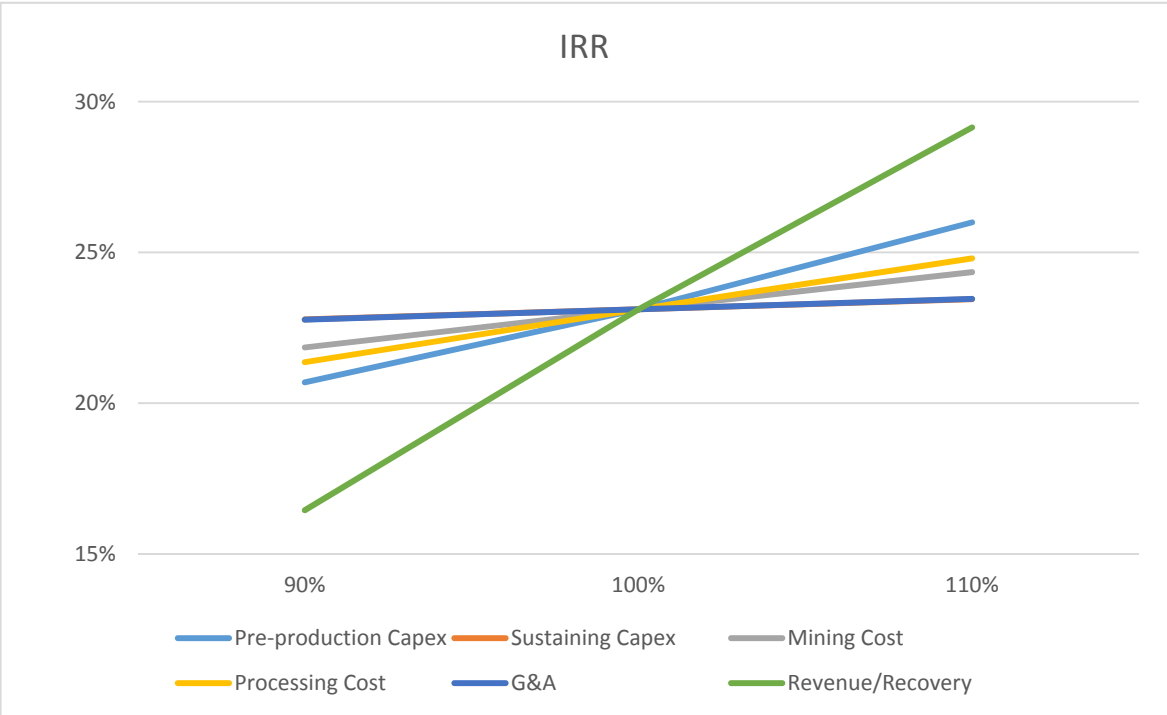


22.5 Single Parameter Sensitivities

Figure 22.4 shows the changing post-tax $NPV_{7\%}$ and IRR for varying single parameter sensitivities for revenue, pre-production and sustaining capital costs, mining, plant and G&A operating costs and revenue / gold recovery. Figure 22.4 also shows the post-tax IRR sensitivity to parameters that the NPV is most sensitive revenue / recovery.

Figure 22.4 NPV and IRR Sensitivity





23. ADJACENT PROPERTIES

Managem International operate a pilot plant scale operation 100 km south of Block 14 at Gabgaba and are known to be exploring for gold mineralization in the Block 15 exploration licence.

Tahe Minerals are operating a small gold mine 200 km to the west of Block 14 in the Wadi Halfa area. There is no publicly available information on the project.

24. OTHER RELEVANT DATA AND INFORMATION

There is no additional information or explanation required in order to make this report understandable and not misleading; all relevant information has been summarized in the Revised PEA Report and Appendices.

25. INTERPRETATION AND CONCLUSIONS

25.1 Geology

Eight percent of the in-pit resources are currently in the inferred category. A short RC drill programme will be carried out to upgrade these into Indicated Resources and to provide coverage in several areas where the pits show the potential to go deeper.

25.2 Mining

The mining of the GSS and WD deposits has been shown to be technically feasible through conventional open pit mining methods. Pit optimizations show that both deposits have economically viable material under the assumed economic and physical parameters. Work completed since the 2016 PEA has improved the project due to:

- Improved geotechnical wall parameters.
- Lower processing costs due to higher throughput and change to water source.
- Lower fuel price.

A total of 44.4 Mt of crusher feed at a gold grade of 1.47 g/t is contained within eleven pits across the two deposits, along with some 104.4 Mt of waste, resulting in a strip ratio of 2.35:1. Over 90% (92%) of this crusher feed is contained within the Indicated category.

A combined mining schedule demonstrates that a processing rate of 3.4 Mt can be sustained for 13.2 years, including an initial ramp up period in Years 1 and Year 2. A pre-production period of six months is utilized to strip waste from both deposits and stockpile minor amounts of crusher feed, ready for the commissioning of the processing plant. The pit at WD is mined for the first seven years and blended with material from GSS. This maximises early returns by blending the significantly higher grade material from the WD deposit with the lower grade material from GSS.

Opportunities exist to increase the mining rate should an expansion of processing capacity be available. This would have the effect of lowering the impact of fixed operating costs, thereby lowering the cut-off grade and potentially producing larger pits.

25.3 Mineral Processing and Recovery Methods

The most economically effective process scheme identified as a comminution circuit followed by the adsorption of gold onto activated carbon, through the carbon-in-leach (CIL) process.

The design of the comminution circuit and the metal recovery plant is based on a nominal capacity of 3.4 Mtpa.

The proposed process route is a proven and robust concept with very little associated risks, apart from operational hazards stemming from the reagents that are used in the process. The plant design will take cognisance of this fact and good engineering practise and industry standards will be applied to design a safe operating plant.

25.4 Major Infrastructure

25.4.1 Selected Water Supply Option

Eight exploration boreholes have been drilled within the Area 5 aquifer, which indicated high and constant airlift yields with the occurrence of groundwater within an inter-layered basin type sequence of course to medium grained sandstone. The results support the well field development as designed and that the identified aquifer is able to sustain the proposed mine water demand over the LOM period. There is potential to extend the current identified aquifer extent in terms of lateral and vertical dimensions. Further drilling and testing will aid aquifer delineation and characteristics.

25.4.2 Selected Power Supply Option

Based on the trade-off carried out, self-generation is the most likely option with the least risk associated with it.

25.4.3 Roads

Surface haul roads have not been designed although the topography and climate will mean that relatively simple haul road construction will be sufficient. Surface haulage distances were estimated for the various deposits to allow for the calculation of mining costs.

No additional site access road networks or off-site infrastructure will be developed for this project from Abu Hamad (the closest main town to the project area) due to the use of desert tracks that are capable of handling the loads from heavy vehicles.

25.5 Environmental

There are currently no objections to the development of the Project. The current Exploration Project has been mentioned as an example of good practice by the SMRC, as the National authority.

There are few receptors in the area, with no human settlements in proximity. Even so, the Project has commenced a number of environmental studies, with a view to developing a detailed database covering at least 12 months. These baseline studies for an EIA have commenced at an early stage of the Project. The data being collected requires a long lead time to ensure some reliability in the results. The remoteness and arid conditions mean that it is hard for wildlife and those animals present tend to avoid human activity. The use of remote cameras provides the opportunity to record these fauna. Other wildlife records are captured through daily observations. Climate data and weather data has been collected and compared, to provide reliable data for the EIA and design teams. Water data from the existing boreholes and Talat Abda well are being collected, even though there are no known sources of potable water and few potential water users in the vicinity. Social data is also being collected during the Exploration stage, with continuous engagement with artisanal and small scale miners resulting in the collection of information that will be used in the EIA.

From a legal perspective, the Project is authorized under the Concession Agreement, which gives MSMCL the right to establish a mining operation in a responsible manner. MSMCL has the responsibility to manage the effects of the mining activity in such a way as to mitigate the negative impacts. MSMCL has initiated environmental measures through their Exploration Statement, which is specific to exploration work and includes an EPMP to mitigate impacts associated with the work. In accordance with the Concession Agreement, the EPMP is a dynamic document that will be revised and updated as the Project progresses.

The development of the Project is likely to give rise to a range of environmental and social impacts. However, assuming the implementation of mitigation measures proposed in the EIA, these impacts are considered manageable and controllable. Therefore the development, operation and closure of the Project could be undertaken in an effective environmental and social manner.

25.6 Economic Analysis

Based on a gold price of US$1,200 /oz, the Project has a pre-tax IRR of 26.5% and pre-tax $NPV_{7\%}$ of US$278.2M. The Project has a post-tax IRR of 23.1% and a post-tax $NPV_{7\%}$ of US$227.7M.

The Pre-tax undiscounted payback period is 2.6 years and Post-tax, undiscounted payback period is 3.0 years from start of production.

The NPV and IRR are most sensitive to the metal price / recovery, plant and mine operating cost.

25.7 Risks and Opportunities

25.7.1 Risks

The following risks were identified; a formal Risk Response Plan will be developed during the Definitive Feasibility Study to mitigate these risks:

- Exploration may not enable the Inferred Resource within the open pits to be classified as Indicated Resources.
- Mining contract rates may be higher than expected given the remote location.
- Skilled labour will be difficult to source in the local area, meaning that initially expatriate operators and trainers will be required, although experienced operators can be found in Egypt. This has the possibility of increasing contract mining rates and plant operating personnel for the first several years.
- Due to the location of the plant, transport and logistics of fuel and reagents might be problematic and sufficient buffer capacity and storage will be designed in the event of logistical problems and non-delivery of items to the project.
- The current Direct Capital Cost estimate has been shown to be within a level of accuracy range of ±30 %. The risk is that the direct cost might increase by 30% once more detail engineering and development is completed. Based on Figure 22.4 it can be seen that the project will be able to absorb this increase at the current (July 2107) metal spot price.

25.7.2 Opportunities

Mining

Both feed grade and operating costs can be refined by further optimizing the extraction sequence of the various GSS pits.

Material haulage from Wadi Doum could be improved if long distance haulage units which can be loaded at the face are utilized. This would negate the need for ROM stockpiles and re-handling of the crusher feed, potentially saving US$0.40 – 0.50 /t of crusher feed from Wadi Doum.

A review of the hydrogeology will lead to a greater understanding of the effects of groundwater on the pit walls (if any), and could allow for further steepening.

Mineral Processing

The increase in spatial variability data and linking with dominant lithologies will identify the level of variability of physical properties (hardness) and gold recovery and lime / cyanide consumptions, which will enable optimization of mine scheduling and therefore financial modelling accuracy.

Increasing the Geochemistry data base will assist in developing relationships with gold recovery and/or cyanide consumption.

Carbon modelling data will provide data to define and optimize the design of the overall leaching, adsorption and elution circuits.

Power Supply

Alternative energy sources can be investigated in future project phases including waste-to-liquid plants and solar panel arrays. A trade-off between capital cost and operating cost savings should be carried out. Currently power costs account for 9% of the plant operating costs.

26. RECOMMENDATIONS

26.1 General

The Revised PEA has demonstrated a strong project with several opportunities for improvement. Accordingly, the Company has approved the decision to commence a Definitive Feasibility Study (DFS) of the Block 14 project focused on optimizing the Project towards a development decision in early 2018.

The estimated cost of the DFS inclusive of resource and geotechnical drilling, testwork, trade off studies, engineering and DFS engineering to be US$6.34M. A breakdown is provided in Table 26.1.

Table 26.1 DFS Cost Estimate Summary

Description	Cost ($'000)
Consultants	461
Testwork	343
Mining & Geotechnical	392
EIA	109
Hydrology	37
Resource Drilling	5,000
Total	**6,342**

26.2 Additional Recommendations

26.2.1 Environmental

The final Project design still has to be finalised, but by initiating the EIA process early, results can be used to improve the design, as well as maximising the benefits of the EIA without incurring excessive costs. There are a few improvements that the Project should undertake in the near future, namely:

- Examine the potential for renewable power supplies.
- Maintain a grievance procedure to identify and pre-empt potential tensions with artisanal and small scale mining operations, as well as distant communities that may feel that the Project affects them.

26.2.2 Mining

The following follow up work is required:

- Extend geotechnical investigations to cover the full depth of the open pits.
- Complete waste dump and haul road design to allow for more accurate estimate of haulage requirements.
- Consider using a blending algorithm to improve the scheduling based on material blending to provide a more consistent grade and material composition through the processing plant.

26.2.3 Metallurgical Testwork

The next stage of work will focus on:

- Complete comminution testing to determine the variability within each dominant zone, domain and lithology.
- Completion of testing of the effect of primary grind on gold recovery.

- Performing carbon modelling in order to optimize the retention time and carbon advancement rate.
- Performing testwork to determine effect of site water from post leach thickener overflow to optimize the process water circuit.

26.2.4 Water Supply from Borehole Well Field

The following follow up work is required:

- Drilling of at least five larger diameter test boreholes within 50 m from the existing observation / exploration boreholes. These boreholes will be constructed with 8"casing.
- Detailed pump tests at an optimal constant discharge rate for at least 24 to 48 hrs per borehole.
- It is also recommended to drill an additional four exploration boreholes to determine the full extents of the aquifer to the north and south.
- Detailed assessment based on the data, to include a preliminary numerical groundwater model to illustrate the zone of depression and aquifer boundaries.

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QUALIFIED PERSONS CERTIFICATE

Certificate of Qualified Person – Dr Geoffrey Duckworth, B.Eng (Chem), M.Eng.Sc, PhD, FIChemE, MIEAust, FAusIMM, RPEQ 2702

I, **Geoffrey Alexander Duckworth**, do hereby certify that:

1. I am employed as a Senior Consultant – Process, with the firm Lycopodium Minerals Pty Ltd, 153 - 163 Liechardt Street, Spring Hill, Queensland 4000 Australia, specialising in minerals processing engineering consulting, contracting for a wide variety of clients, and have been so employed since 2008.
2. This certificate applies to the technical report with an effective date of 30 May 2017, and titled *"Revised Preliminary Economic Assessment, NI 43-101 Technical Report - Block 14 Project, Republic of Sudan"*.
3. I am a practising Metallurgical Engineer and registered Fellow of the Australian Institute of Mining and Metallurgy.
4. I am a graduate of the Royal Melbourne Institute of Technology with a Bachelor of Engineering (Chemical) 1974 and post graduate of the University of Queensland Australia with a M.Eng.Sc degree (1979) and PhD (1982), both in Mining and Metallurgy. I have practiced my profession continuously since 1974.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I visited the Galat Sufar South Project between 3 October and 9 October 2016. The purpose of the visit was to review site location, access, infrastructure requirements, and allocation of layout areas for the project.
7. I am responsible for Sections 17, 18, 21, 22, 25, 26, and 27.
8. I am independent of Orca pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Orca or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed the 6th day of July, 2017 at Brisbane

Geoff Duckworth, B.Eng (Chem), M.Eng.Sc, PhD, FIChemE, FAusIMM, RPEQ 2702
Study Manager

I **Nicholas James Johnson** hereby state:-

1. I am a consulting Geologist, with the firm of MPR Geological Consultants Pty Ltd, 19/123A Colin Street, West Perth, WA 6005, Australia.
2. This certificate applies to the technical report with an effective date of 6th July 2017, and titled "Revised Preliminary Economic Assessment, NI 43-101 Technical Report - Block 14 Project, Republic of Sudan".
3. I am a practising a practising Geologist and registered Member of the Australian Institute of Geoscientists.
4. I am a graduate of the Latrobe University, Melbourne, Australia with a Bachelor of Science (Honours) degree in Geology (1988). I have practiced my profession continuously since 1988.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I visited the Galat Sufar South Project between 17th January and 21st January 2014. The purpose of the visit was to review the exploration practices and project geology.
7. I am responsible for sections 1.5 and 14.
8. I am independent of Orca pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Orca or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 6th day of July 2017 at Perth Australia.

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Nicolas James Johnson, B.Sc (Hons) (Geol), MAIG, Consulting Geologist

I **Pieter Ferdinandus Labuschagne** hereby state:-

1. I am a consulting Hydrogeologist, with the firm of GCS (Pty) Ltd and situated in the Durban office in South Africa (4A Old Main Road, Kloof, 3610).
2. This certificate applies to the technical report with an effective date of 6th July 2017, and titled "Revised Preliminary Economic Assessment, NI 43-101 Technical Report - Block 14 Project, Republic of Sudan".
3. I am a practising Hydrogeologist and registered Member of the South African Council for Natural Scientific Professions – SACNASP (Pr.Sci.Nat.400386/11).
4. I am a graduate of the University of the Free State, Bloemfontein, South Africa with a Master's of Science degree in Hydrogeology (2004). I have practiced my profession continuously since 1998.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I visited the Galat Sufar South Project between 23 November and 2 December 2016. The purpose of the visit was to review the groundwater exploration drilling and aquifer testing.
7. I am responsible for sections 1.9.1, 18.1 and 26.2.4.
8. I am independent of Orca pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Orca or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 6th day of July 2017 at Kloof, South Africa.

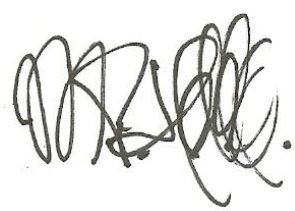
_____

Pieter Labuschagne, M.Sc Hydrogeology (Pr.Sci.Nat 400386/11)

I **Michael Peter Hallewell** hereby state:-

1. I am a consulting Metallurgist, with the firm of MPH Minerals Consultancy Ltd, Office S015, Tremough Innovation Centre, Penryn, Cornwall, TR10 9TA.
2. This certificate applies to the technical report with an effective date of 6th July 2017, and titled "Revised Preliminary Economic Assessment, NI 43-101 Technical Report - Block 14 Project, Republic of Sudan".
3. I am a graduate with a B.Sc (Engineering) degree in Minerals Engineering from the University of Birmingham, UK. I have 35 years practical experience in Minerals Processing as Plant Manager, Consulting or Senior Metallurgist in precious metals, base metals and ferrous metals industry.
4. I am a Fellow of the South African Institute of Mining & Metallurgy (RSA), a Fellow of the Institute of Materials, Minerals and Mining (London, UK), Fellow of the Minerals Engineering Society (UK) and a Chartered Engineer.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I am responsible for Sections 1.6, 13, 25.7.2 and 26.2.3 of the report.
7. I am independent of the issuer as described in section 1.5 of NI 43-101.
8. I do not beneficially own, directly or indirectly, any securities of Orca or any associate or affiliate of such company.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 6th day of July 2017



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Michael Peter Hallewell, B.Sc (Hons), F.I.M.M.M., F.S.A.I.M.M., F.M.E.S., C.Eng, Consulting Metallurgist

I **Carl Steven Nicholas** hereby state:-

1. I am a Chartered Environmental Consultant, with the company of Mineesia Limited, 4 Mace Farm, Cudham, Kent, TN14 7QN, UK.
2. This certificate applies to the technical report with an effective date of 6th July 2017, and titled "Revised Preliminary Economic Assessment, NI 43-101 Technical Report - Block 14 Project, Republic of Sudan".
3. I am a practising Environmental Consultant and registered Member of the Institute of Materials, Minerals and Mining.
4. I am a graduate of Imperial College, London, UK with a Masters in Environmental Diagnosis, with a Bachelor of Science (Honours) degree in Biodiversity Conservation and Environmental Management. I have practiced my profession continuously since 2005, and have 12 years practical experience in Environmental Impact Assessments for mining projects.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I visited the Galat Sufar South Project between 24th November and 3rd December 2014. The purpose of the visit was to review the baseline conditions and establish priorities for environmental management for the project.
7. I am responsible for sections 1.10, 20, 25.5 and 26.2.1.
8. I am independent of Orca pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Orca or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI 43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 6th day of July 2017

Carl Steven Nicholas, M.Sc, B.Sc (Hons), DIC, CEnv, MIMMM

I **Chris Reardon** hereby state:-

1. I am a Principal Mining Engineer, with the firm of Deswik Europe Ltd and situated in the Amersham office in England (Sycamore House, 1 Woodside Road, Amersham, Buckinghamshire HP6 6AA).
2. This certificate applies to the technical report with an effective date of 6th July 2017, and titled "Revised Preliminary Economic Assessment, NI 43-101 Technical Report - Block 14 Project, Republic of Sudan".
3. I am a practising Mining Engineer with over 20 years of relevant experience in open pit mining operations, 14 of which have been in open pit gold mines.
4. I am a graduate of the University of Queensland, Australia with a Bachelor's of Science degree in Geology (2004). I have practised my profession continuously since 2005.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I visited the Galat Sufar South Project between 3rd and 8th October 2016. The purpose of the visit was to review the site layout, geological setting and logistics associated with the development of an open pit mining operation.
7. I am responsible for sections 1.7, 16 and 21.5, 25.2, 25.7.1, 25.7.2 and 26.2.2.
8. I am independent of Orca pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Orca or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 6th day of July 2017 at Amersham, England.

Chris Reardon