PRELIMINARY ECONOMIC ASSESSMENT

NI 43-101 TECHNICAL REPORT

BLOCK 14 GOLD PROJECT

REPUBLIC OF THE SUDAN

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Date and Signature Page

The independent Qualified Person (within the meaning of NI 43-101) for the purposes of this report is Keith Bright, representing SGS Time Mining (Pty) Ltd. Further review and authoring has been undertaken by Nic Johnson and Mike Hallewell.

The SGS Time Mining 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.

This report entitled Preliminary Economic Assessment Technical Report, Block 14 Gold Project Republic of Sudan, was prepared and signed by the following authors:

Signed, this 30th day of August 2016

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CONTENTS

List of Figures

List of Tables

1 EXECUTIVE SUMMARY

1.1 Introduction

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

The PEA has been prepared by SGS Time Mining (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 3,747 km^2. It is located in the Nubian Desert in the northern region 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 May 19, 2010. A purchase agreement dated 1st 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 USD $9.5 million in three instalments, in exchange for an increasing ownership interest.

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 4 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 below:

Start Date	Expiry Date	Area (km²)	Terms
19th May 2010	19th May 2014	7,046	Initial exploration period.
19th May 2014	19th May 2015	7,046	Additional year compensation period granted, extending the expiry date of the initial exploration period.
19th May 2015	19th May 2017	3,747	Formal notification of Mineral Discovery Areas, relinquishment of 50% of the Initial Exploration area.
19th May 2017	19th May 2018		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. Air travel to other destinations in the country is only possible with local carriers, using a network of minor airfields of varying capacity and quality. The airport at Port Sudan also has international flight handling capability although services are limited to neighbouring countries.

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°C to 49°C and minimum of 23°C to 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.

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 5 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.

To the extent relevant to the mineral 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).

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 metres 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 Mineralisation

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 subordinate 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.

Gold mineralisation is being exploited by artisanal miners throughout Block 14. The geological setting is varied but four main types or 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; and
- 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.

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 mineralisation. 1:25,000 scale mapping 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 mineralisation 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.

6 Domains of mineralisation 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 mineralisation 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 mineralisation being present to the north and south with a similar, although not fully defined, trend. Shear fabrics in oriented core are dominated by a 110º orientation dipping steeply to the west. Sub-ordinate 010° trending shear is also present although these north going areas of mineralisation 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 mineralisation is hosted within a sheared sericitised microdiorite. Steep to vertical mineralised 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 mineralisation 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 mineralisation. The first phase of mineralisation 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 mineralisation, but generally the k-feldspar alteration contains weak, variable mineralisation. 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 mineralised 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 a millimetre scale to a maximum of 1.5 m.

1.4.2 Wadi Doum (WD)

The main, high grade mineralisation 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-mineralised felsic and mafic dykes. In contrast to the volcaniclastics the rocks on the hill dip 75° to the east. Mineralisation on the hill is associated with stringer zones within the syenite and in places smaller shears.

The high grade mineralisation 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 mineralisation often appears un-affected by structure whereas the mineralisation hosted by the syenite on and around the summit of the hill does appear structurally controlled.

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

The mineralisation types being targeted within the Block 14 project are broadly categorised into 3 groups, namely, Orogenic Gold, Volcanogenic Massive Sulphide, and Rift Associated Epithermal.

Evidence of orogenic gold mineralisation 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 mineralisation 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 mineralisation.

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

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 mineralisation 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 mineralisation.

1.5 Mineral Resource Estimate

In February 2016, 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 utilised RC and diamond drilling data supplied by Orca in February 2016. Details of this sampling and assay are described in later 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 more than 50 degrees in 5 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 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 metres depth with around 90% of the Indicated Resources, and 70% of the Inferred resources occurring at depths of less than 160 metres. The WD estimates extend to around 210 metres depth with around 90% of the Indicated Resources, and 70% of the Inferred resources occurring at depths of less than 100 metres.

For GSS the combined oxidised and transitional material hosts around 38% and 20% of the Indicated and Inferred resources respectively with the remainder lying in fresh rock. For WD the combined oxidised and transitional material hosts around 15% and 21% of the Indicated and Inferred resources respectively.

GSS						
Material	**Indicated**			**Inferred**		
	Mt	**Au g/t**	**Au koz**	**Mt**	**Au g/t**	**Au koz**
Oxide	5.3	1.79	305	1.1	1.6	57
Transition	4.7	1.76	266	0.8	1.6	41
Fresh	16.3	1.72	901	8.1	1.7	443
Total	**26.3**	**1.74**	**1,471**	**10**	**1.7**	**547**

WD						
Material	**Indicated**			**Inferred**		
	Mt	**Au g/t**	**Au koz**	**Mt**	**Au g/t**	**Au koz**
Oxide	0.12	3.26	13	0.2	2.7	17
Transition	0.08	3.11	8	0.1	2.3	7
Fresh	1.16	2.86	107	1.2	2.0	77
Total	**1.36**	**2.91**	**127**	**1.4**	**2.1**	**95**

Combined						
Material	**Indicated**			**Inferred**		
	Mt	**Au g/t**	**Au koz**	**Mt**	**Au g/t**	**Au koz**
Oxide	5.4	1.82	318	1.3	1.8	74
Transition	4.8	1.78	274	0.9	1.7	49
Fresh	17.5	1.80	1,008	9.3	1.7	520
Total	**27.7**	**1.80**	**1,599**	**11.4**	**1.7**	**641**

1.6 Metallurgy

The extensive metallurgical test work conducted on various different composite samples from the deposit clearly indicates that the two main material types, GSS (Main & East) and WD, are amenable to direct cyanidation.

The diagnostic leach and gold deportment studies conducted by SGS RSA show that gold deportment is mainly associated with pyrite and at an extremely fine grain size. The gold dissolution in cyanide is sensitive to grind size. The following gold recoveries have been determined through test work.

Pit	Material	% Recoveries
GSS	Oxide	92
	Transitional	87
	Fresh	80
Wadi Doum	All	83

The next stage of work will focus upon the optimisation of:

- Flash Flotation Reagents and Flotation time;
- Regrinding of Optimised Flash Flotation Concentrate;
- Cyanide Leach parameters on optimised reground flash flotation concentrates; and
- Variability of following parameters by material zones and lithologies:
  - Bond Ball mill work index
  - Head Assay and Clay content
  - Flash Flotation Response using optimised conditions from aforementioned program
  - CIL Response

1.7 Mining

A review of existing Block 14 gold concession geotechnical data was conducted by SRK Consulting (UK) Limited. The report from this review has been used as a basis for subsequent optimisation and pit design.

Basic geotechnical data from 18 cored boreholes were provided for the review. These consisted of lithology, rock strength, RQD, weathering state and rock fabric. A structural database was also provided, gathering mainly data from the vein and foliation orientation. At this stage, no systematic logging of natural open joint orientations and conditions has been recorded.

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 130 metres deep. The GSS deposit will be exploited by seven separate pits, five of which are shallow oxide/transitional pits only. The other two are deeper than the WD pit at 150 and 205 metres.

While the mineralisation 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.

It is assumed that mining is conducted by a mining contractor, utilising a mining fleet comprised of Caterpillar 777 rigid body haul trucks (90t) with suitably sized loading unit. 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.

There were two main scenarios examined for the GSS deposit, based around the treatment of the oxide. The base case was to treat oxide through a CIL plant, along with the transitional and fresh material. An alternative case was proposed where the oxide would be treated using a heap leach plant prior to the commencement of the CIL processing of transitional and fresh material.

Despite the fact that the Heap Leach option has a lower processing cost for oxide, and generates more crusher feed tonnes, the relative processing recoveries for heap leach and CIL means that more ounces are generated from the CIL option. For this reason, the Revenue Factor = 0.95 shell from the CIL optimisation was chosen as the basis for design.

The WD optimisation considered the use of a local or remote processing plant as two options. The decision was made to use the GSS processing plant for WD material. The additional operating margin from a local processing plant is insufficient to cover the additional capital costs. In addition, benefits can be gained by treating the WD feed at a higher throughput in the GSS plant providing a grade boost to GSS feed while the GSS pits are developed. The Revenue Factor = 0.95 shell for the GSS plant scenario has been chosen as the basis of design for WD

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

Wall angles, batter and berm 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 metres was assumed, 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 over 90% of the total crusher feed and 87% of the contained ounces. The deposit is exploited through 7 individual pits extending along strike for nearly 3 kilometres. The overall strip ratio for the pits is 1.97:1.

Although much smaller than the GSS pits to the west, the WD deposit contains mineralisation at nearly double the grade. The deposit is exploited through a two stage 130m deep pit. The overall strip ratio for the WD pit is 3.91:1.

While no waste dump technical designs have been completed, areas have been set aside for waste dumps at both deposits, and a ROM pad at GSS. Given the topography and lack of surface water, construction of these is unlikely to create major issues. A second ROM pad will be required at WD to act as a transfer pad and space is available for this.

The WD pit was divided into two stages to assist with waste stripping and crusher feed exposure. The GSS pits were treated as single pits for the purposes of scheduling. Further refinement of the schedule would include the development of cutbacks for the West 1 and East 2 pits. The other pits are too small to warrant phased extraction.

A ramp up period of 12 months was assumed at the start of the schedule. The target for Year 1 was 1.5 Mt of crusher feed, with all subsequent years targeting 1.8 Mt. Mining dilution and recovery were not included in the schedule.

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 16-year mine life, with WD being completed in Year 6.

The Owner 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 will be required for the haulage from WD to GSS.

1.8 Recovery Methods

Metallurgical test work conducted to date, with specific reference to Gold Deportment and Diagnostic Leach Test Work, validate that the GSS and WD material types are amenable to gold recovery via cyanidation.The most economically effective process scheme identified is the adsorption of gold onto activated carbon, through the carbon-in-leach (CIL) process preceded by a comminution circuit.

The design of the comminution circuit and the recovery plant is based on a nominal capacity of 1.8 Mtpa.

Oxide, transitional and fresh Run-of-Mine (ROM) material from GSS and WD is fed into primary crusher and secondary & tertiary crusher circuit. The material is reduced in size from the top size of 600mm to 80% passing 8mm before it is fed into the ball mill. The material is then ground to 80% passing 75 µm and fed to the CIL circuit for leach and adsorption onto carbon. The cyclone underflow stream returns to the ball mill for further grinding.

The CIL tails are treated in a detoxification circuit where reagents sodium metabisulphite (SMBS) and copper sulphate are added in high dissolved oxygen slurry to dissociate free cyanide to a level of less than 50 ppm.

After detoxification the slurry is pumped to the tailings thickener. The thickener overflow is distributed back into the process circuit and the underflow is pumped to the tailings storage facility. No TSF design has been completed at this stage of the project, and an existing design for conventional surface thickened tailings storage facility with an approximate footprint of 57 hectares was used to estimate costs.

There is upward potential in gold recovery using flash flotation followed by ultra-fine grinding and cyanide leaching of the flash flotation concentrates to recover some of the more refractory gold associated with pyrite. This has option has not been included in the current design and will be addressed in the Pre-Feasibility Study.

1.9 Project Infrastructure

1.9.1 Water Supply

The estimated water demand is in the order of 4,000 m^3/day (46 l/s).

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; and
- **Alternative 2:** Establishing a pump station and pipeline from boreholes, located approximately 50 km north of the project area.

In order to verify the hydrological viability of Alternative 2, regional and local scale groundwater resource related investigations, involving remote sensing, ground geophysics and drilling, have been carried out in selected areas of the Project area since 2012. Based on geophysical surveys completed in late 2015, a follow up drill programme, testing two low resistivity anomalies hosted within the Nubian Sandstone basin was completed in early July 2016.

Based on available drilling observation data in area HA8, it can be conservatively assumed that aquifer storage in HA8 is between 22 million and 42 million cubic metre. This range is based on 16 to 20 m of aquifer thickness and 10 to 15% net specific yield for the observed strata. The proposed daily 4000 m^3/day water demand can be supplied to the mine for the entire LOM form the HA8 area.

After confirmation of the Alternative 2 hydrological capacity and comparing the capital and operating cost of the two alternatives, Alternative 2 was selected as the preferred alternative for the purposes of the PEA design and economic assessment.

1.9.2 Electric Power

Two possible supply options were investigated:

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

The total estimated cost to connect to the grid was USD 66.2 million.

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.

For the self-generating option, the project will require 3 x 3MW and 3 x 1MW containerised diesel engine driven turbine generators. The estimated investment is USD 8.1 million, excluding ancillary electrical equipment and installation of the system.

The current cost of diesel fuel, including transportation to GSS site is USD 0.65/l. Based on the consumption figures of 0.23 l/kWh, a power cost of USD 0.15/kWh was used in the operational cost analysis.

In order to select the most viable power supply option, a capital and process power cost trade-off was carried out for the two identified options over the Life-of-Mine production schedule. For the purpose of the trade-off the grid supply option includes the capital cost for back-up generation and excludes maintenance and replacement costs for capital equipment.

The results show that there is no break-even point, and if one considers the fact that it might be possible to move or sell the generator sets, while ownership of the distribution network would be transferred to Sudan's National Electricity Cooperation, Option 2 (Self generation) was selected.

1.9.3 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 were 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.

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 major cities like Khartoum, Port Sudan, or 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.

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, as 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 covering at least 12 months. 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 impact, which they have begun doing through the implementation of an Exploration Statement. This statement 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.

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 of -20% to +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 1.8 million tonnes of material per annum. For the purpose of this PEA, a contract mining scenario has 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 reported as EPCM/Home Office Cost and Field Costs, and contingencies have also been estimated, based on percentages of the Direct Capital Cost Estimate. Risk amounts are specifically excluded from this estimate.

The Life of Mine capital cost (CAPEX) is estimated at US$138 million, comprising US$101 million direct costs, US$17 million indirect costs and US$20 million contingency. The direct cost estimate is inclusive of the total estimated TSF cost; this cost will however be phased over the life of the project, see Table 22-3. The total pre-production capital cost is US$122 million.

Due to the use of mining contractors, who will provide the mining fleet, the mining capital costs comprise US$5 million in Year 1 to cover items such as:

- Principal mine office, including fittings, furniture and computer systems;
- Fuel Storage and Dispensing Facility; and
- Light vehicles for principal mining team.

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 3 years.

Unit costs were determined for the following items:

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

Unit mining costs averaged $2.88/t.

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 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 $0.034/t/10m 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 $8.48/t for crusher feed contributed by WD.

1.11.3 Operating Cost – Process Plant & Infrastructure

Processing operating costs from similar operations have been used; with the exception ofvariable power costs. The theoretical power draw was calculated from each of the processed material type Work Indices values, and installed power from the mechanical equipment list. The power draw was used calculate the power cost / tonne based on the generator consumption and a diesel fuel price of $0.65/l.

Operating Cost was summarised into the following cost centres:

- Power;
- Operating Consumables;
- Maintenance Materials;
- Laboratory;
- Process & Maintenance Labour;
- Administration Labour; and
- General and Administration Costs.

Fixed and Variable costs were applied individually to the cost centres for each of the different processed material types.

A fixed cost of $ 14.3 million was applied per annum and a variable cost averaged $10.84/t.

1.12 Economic Analysis

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

On a pre-tax basis, the project has a Net Present Value (NPV) of US$156 million at a discount rate of 7 %, an Internal Rate of Return (IRR) of 25%, with a payback after 3 years of production; on a post-tax basis, the NPV is US$128 million at a discount rate of 7 %, the IRR is 22%, and the and the payback is 4 years after start of production.

1.13 Recommendations

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

1.13.1 Environmental

MSMCL is still to finalize the final Project design, 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:

- Conduct social baseline surveys of Abu Hamad and record primary economic activities of people there and those working in the vicinity of the Project; and
- Develop a grievance procedure to identify and pre-empt potential tensions with artisanal and small scale mining operations.

1.13.2 Mining and Geology

The following follow up work is required:

- Review geotechnical information and complete additional geotechnical investigations in order to determine if wall steepening is possible;
- Complete waste dump and haul road design to allow for more accurate estimate of haulage requirements; and
- 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

Froth Flotation provides the potential for Orca to coarsen the primary grind size since the flotation testwork response of East and Main was extremely encouraging.

Accordingly, the Company has approved the decision to commence a Pre-Feasibility Study (PFS) of the Block 14 project, focused on optimizing the Projectand declaring a maiden Reserve in late 2017.

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 northern region 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 as currently envisaged to comprise two open pit 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

SGS Time Mining has reviewed all other technical work completed on the Project by the Company and its other contractors and consultants to a sufficient level to enable SGS Time Mining to present its own opinions on the Project and to derive an audited NPV for the Project.

This report comprises a preliminary economic assessment (PEA) of the Block 14 Project and has been prepared by SGS Time Mining (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).

Work undertaken by SGS Time Mining in compiling this report has been managed by Mr Anton van Tonder, and overseen and reviewed by Mr Keith Bright. Mr Keith Bright is a Qualified Person (QP) as defined in the National Instrument 43-101 of the Canadian Securities Administrators (NI 43-101).

2.2 Property Inspections by the Author

Mr Anton van Tonder and Mr Keith Bright visited Sudan and the Block 14 project between the 10th and 15th January 2016.

Nic Johnson of MPR Geological consultants, who is responsible for sections 1.4 and 14 of the report detailing project resources, visited the Block 14 Project between the 17th and 21st January 2014.

2.3 Declaration

SGS Time Mining's opinion contained herein from the effective date is based on information collected by SGS Time Mining throughout the course of the PEA investigations, which in turn reflect various technical and economic parameters at the time of writing the report. Due to the nature of the mining business, these conditions can change significantly over relatively short periods of time, and actual results may be considerably more or less favourable.

This report may include technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages; calculations inherently involve a degree of rounding and accordingly introduce a margin of error. Where these occur, SGS Time Mining does not consider them to be substantial.

SGS Time Mining is not an insider, associate or an affiliate of Orca Gold Inc. or any affiliate that has acted as advisor to the Company, its subsidiaries or its affiliates in connection with this project, except for SGS Mineral Services (UK. Canada, South Africa), that were appointed by the Company to conduct the mineralogy and metallurgical test work.

The results of the technical review by SGS Time Mining are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

2.4 Abbreviations

As	Arsenic
Au	Gold
AAS	Atomic Absorption Spectrometry
B14WC	Block 14 Water Concession
CA	Concession Agreement
CAE Fusion	Geological Data Management System
cm	centimetre
CIL	Carbon-in-Leach
CRM	Certified Reference Material
°C	Degree Celcius
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/t	grams per tonne
HA	Hydrological Area
HQ	Exploration drill size (96mm OD / 63.5mm 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
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	metre
m^3	cubic metre
mm	milimetre
min	minute
NPV	Net Present Value
NQ	Exploration drill size (75.5mm OD / 47.6mm ID)
masl	Metres above sea level
MDA	Mineral Discovery Area
Meyas Nub	Previously known as Emdehan Multi-activities Company Ltd
MIK	Multiple Indicator Kriging
MoM	Ministry of Minerals

Mtpa	Million tonne per annum
MSMCL	Meyas Sand Mineral Company Ltd
NPV	Net Present Value
PDS	Particle Size Distribution
PFS	Pre-Feasibility Study
ppm	Parts per million
PQ	Exploration drill core size (122.6mm OD / 85mm ID)
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
s	second
SMCL	Sand Metals Company Ltd
SMRC	Sudan Mineral Resource Company
SD	Standard Deviation
SG	Specific Gravity
SiB	Stay-in-Business / Sustaining Capital
TDS	Total Dissolved Solids
TSF	Tailings Storage Facility
PVR	Present Value Ratio (NPV/PV of Net Negative Cash Flow)
VMS	Volcanogenic Massive Sulphide
WD	Wadi Doum

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, environmental issues and the legal status of the exploration permits included in the Project. The author has relied on the information provided in sections 1.1 and 4.3.The Author has reviewed the Concession Agreement provided by Orca (Concession Agreement for The Exploration and Exploitation of Gold and Associated Metals and Minerals in Lower Gabgaba Area (Block 14), dated 19th May 2010) which details the terms and conditions of the tenement holding including environmental responsibilities. Metallurgical test work, as per section 13, undertaken on mineralisation from GSS and WD was conducted by ALS Metallurgy in Perth Western Australia, SGS Minerals Services UK Limited, SGS Mineral Services South Africa (Pty) Ltd and SGS Mineral Services Canada Inc.

SGS Time Mining has not independently investigated the representability or accuracy of this test work and has relied on the information provided by ALS Metallurgy, SGS Minerals Services UK Limited, SGS Mineral Services South Africa (Pty) Ltd and SGS Mineral Services Canada Inc. in their reports in the preparation of Sections 1.6 and 13 of the report:

- Metallurgical test work conducted upon Selected Gold Ore Samples from the Galat Sufar South Gold Project, Sudan, report A15503, dated February 2014. ALS Metallurgy, Perth, Australia.
- Metallurgical test report for the Galat Sufar South and Wadi Doum Deposits, Project Number 10866-583, dated 21st October 2015. SGS Mineral Services UK Ltd.
- Comminution Test Work Report on Six Samples from Orca Gold Inc., Proposal 15/835 rev 1. Dated 6th April 2016. SGS Mineral Services South Africa.
- Mineralogical and Metallurgical Report 15/640: Gold Deportment Studies on Three Samples from the Galat Sufar South and Wadi Doum Deposits, Sudan. Dated 18th May 2016. SGS Mineral Services South Africa.
- Heap Leach Amenability Report 16/074. Dated 26th July 2016, SGS Mineral Services South Africa.
- Gravity Recoverable Gold and Flash Flotation Test work 16/235, Dated 22nd July 2016, SGS Mineral Services South Africa.
- Cyanidation Test work on Gold and Silver bearing samples 16/379. Dated 22nd July 2016, SGS Mineral Services South Africa.
- Grinding circuit designs based on the small-scale data for the Orca Gold Project Report CAQC-15688-001. Dated 2 August 2016, SGS Canada Inc.

The author of this report has relied upon the following other expert reports which provided information regarding the mineral rights, geology, mining methods, hydrogeological, and environmental sections of the Report as noted below:

Mineral Rights: the Author has relied on the Concession Agreement provided by Orca (Concession Agreement for The Exploration and Exploitation of Gold and Associated Metals and Minerals in Lower Gabgaba Area (Block 14), dated 19th May 2010) which details the terms and conditions of the mineral tenement holding including environmental responsibilities and the which has been used to inform Sections 1.2 and 4 of the Report.

Geology: the Author has relied on information provided by Orca Gold Inc for sections 1.4, 7, 8, 9, 10 and 11 specifically the NI 43-101 Independent Technical Report for Orca Gold Inc., Effective Date March 11 and The Discovery and Geology of the Galat Sufar South Deposit, Republic of the Sudan. 2015, Newgengold Conference Proceedings.

Mining: the Author has relied on information provided by Deswik Europe Limited for sections 1.7 and 16 as detailed in the report Mining Contribution for Orca Gold PEA dated 19th July 2016.

Hydrogeology: the Author has relied on information provided by GCS Water and Environmental Consultants for sections 1.9.1 and 18.1 as detailed in the report Groundwater Exploration and Groundwater Resource Assessment for the Galat Sufar South Project, Sudan – Block 14 Area, Summary Report for PEA Supplement. Dated July 2016. GCS Water & Environmental Consultants.

Environment and Social: the Author has relied on information provided by Mineesia Limited forsections 1.10 and 20 as detailed in the report Block 14 Environmental and Social Summary for Orca Gold PEA, Dated 31st March 2016, Mineesia Ltd.

Pipeline Studies: the Author has relied on information provided by Propipe for sections 1.9 and 18.1 as detailed in the report Galat Sufar South Project Water Supply: Conceptual Study Final Report P1050-G-RP-001 Rev B. Dated March 2016. ProPipe Process & Pipeline Projects.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Block 14 mineral exploration project covers an area of 3,747 km^2. It is located in the Nubian Desert in the northern region of the Republic of the Sudan, close to the international border with Egypt (Table 4-4 and 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.

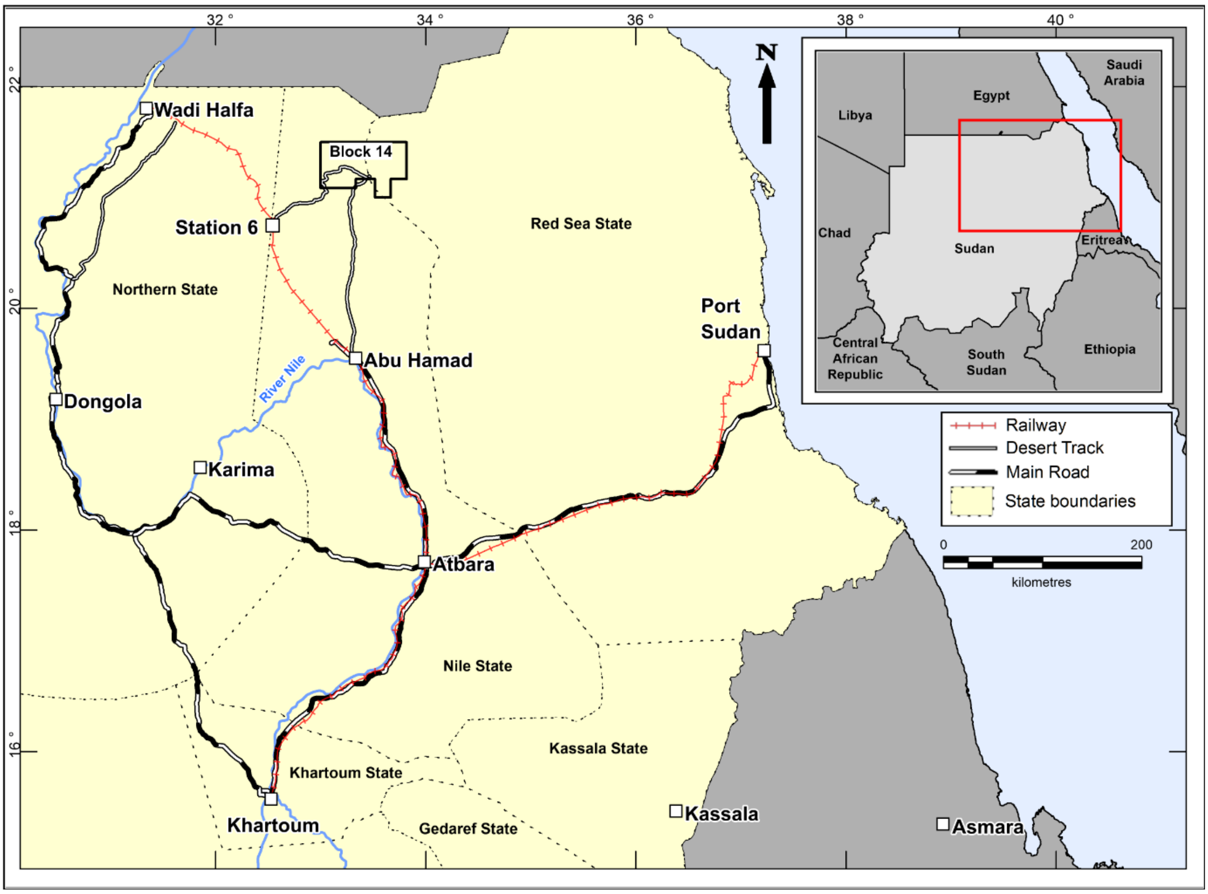


Figure 4-1: Block 14 current tenement boundary, showing regional location (inset) local state boundaries (approximate) and major towns. Source Orca Gold Inc.

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 and as such is main Government body to which MSMCL reports its activities.

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 EPL is issued for an initial period of 3 or 4 years (4 years in the case of Block 14) renewable for two subsequent extension periods of 2 years and 1 year for Block 14 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 procure the incorporation of a new company for the purpose of holding the newly 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 also 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 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 Exclusive Prospecting License (EPL) for Block 14 was originally granted to Meyas Nub under a Concession Agreement dated May 19, 2010.

A purchase agreement dated 1st 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 USD $9.5 million in three instalments, in exchange for an increasing ownership interest. Payments have been made in line with Table 4-1 and Meyas Nub retain the remaining 30% interest in MSMCL.

Table 4-1: Purchase agreement schedule of payments and ownership interest.

Date	Payment	Total ownership interest
01/03/2012	USD $3.5 million	35.0%
01/09/2013	USD $3.0 million	52.5%
01/09/2014	USD $3.0 million	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	3,747 km²
Concession Type	Gold and associated minerals
Annual Surface Rental ($/km²)	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 (requires 50% reduction in area)	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 4 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 (km²)	Terms
19th May 2010	19th May 2014	7,046	Initial exploration period.
19th May 2014	19th May 2015	7,046	Additional year compensation period granted, extending the expiry date of the initial exploration period.
19th May 2015	19th May 2017	3,747	Formal notification of Mineral Discovery Areas, relinquishment of 50% of the Initial Exploration area.
19th May 2017	19th May 2018		Relinquishment of 50% of remaining area (excluding mineral discovery area)

On 27th February 2015, MSMCL submitted a declaration of Mineral Discovery Areas (Table 4-5 and Figure 4-2) and requested the first extension period in line with the concession agreement. The Mineral Discovery Areas (MDAs) define areas within which either mineral resources are located or areas where work to date has shown potential for mineral resources to be developed. The MDAs are excluded from the calculation of the area to be relinquished. The permit extension has been granted based on the coordinates shown in Figure 4-2and Table 4-4and the licence is currently valid until 19th May 2017.

Table 4-4: 2015 Block 14 pillar coordinates (WGS-84).

	Block 14	
Pillar	**Longitude**	**Latitude**
1	33° 00' 00"	21° 30' 00"
2	33° 49' 00"	21° 30' 00"
3	33° 49' 00"	21° 10' 00"
4	33° 40' 00"	21° 10' 00"
5	33° 40' 00"	21° 00' 00"
6	33° 31' 00"	21° 00' 00"
7	33° 31' 00"	21° 10' 00"
8	33° 20' 00"	21° 10' 00"
9	33° 20' 00"	21° 05' 00"
10	33° 00' 00"	21° 05' 00"

Table 4-5: Block 14 Mineral Discovery Area Coordinates.

MDA	Pillar	Longitude	Latitude
GSS	M1	21° 17' 00”	33° 04' 00”
GSS	M2	21° 09' 00”	33° 04' 00”
GSS	M3	21° 09' 00”	33° 15' 00”
GSS	M4	21° 17' 00”	33° 15' 00”
WD	M5	21° 18' 00”	33° 40' 00”
WD	M6	21° 13' 00”	33° 40' 00”
WD	M7	21° 13' 00”	33° 45' 00”
WD	M8	21° 18' 00”	33° 45' 00”
Liseiwi	M9	21° 28' 00”	33° 40' 00”
Liseiwi	M10	21° 23' 00”	33° 40' 00”
Liseiwi	M11	21° 23' 00”	33° 46' 00”
Liseiwi	M12	21° 28' 00”	33° 46' 00”

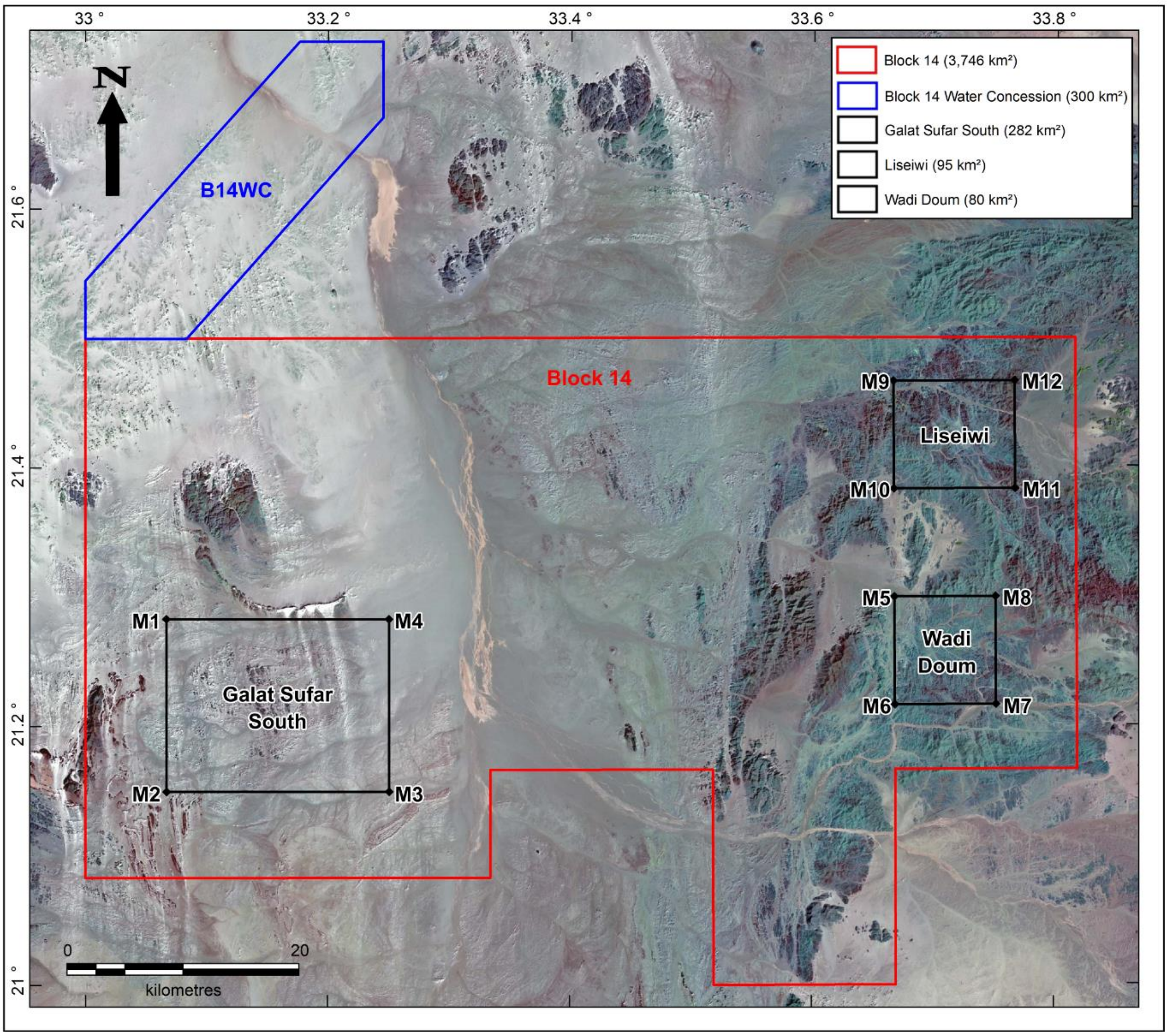


Figure 4-2: Map showing Block 14 and Block 14 Water Concession, together with Mineral Discovery Areas and pillar coordinates, overlain on Landsat imagery acquired on behalf of Orca Gold Inc.

In addition to the mineral exploration tenement, effective from 19th May 2015, MSMCL secured an addendum to the EPL which grants additional 300 km^2 for the purpose of exploring for, and exploiting water (Figure 4-2and Table 4-6). The Block 14 Water Concession (B14WC) provides a two-year initial exploration period, corresponding with the current year extension period of Block 14. The B14WC expires on 19th May 2017 if no water discovery is made during the period. If a water discovery is made (as is the case) the Company has the right to complete a feasibility study on the extraction of the water resource in support of a mining operation.

Table 4-6: Block 14 Water Concession, pillar coordinates (WGS-84).

	Block 14 Water Concession Block 14 (300 km^2)	
Pillar	**Longitude**	**Latitude**
1	33° 00' 00.0"	21° 32' 41.9"
2	33° 10' 38.1"	21° 43' 46.8"
3	33° 14' 47.2"	21° 43' 46.8"
4	33° 14' 47.2"	21° 40' 16.4"
5	33° 05' 00.0"	21° 30' 00.0"
6	33° 00' 00.0"	21° 30' 00.0"

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 cooperative manner.

The hot season which extends from May to September makes exploration more difficult but it does not impact significantly upon the working programme, although reconnaissance in remote areas away from Orca's permanent camp is minimised during this period, much like many other exploration projects in this region.

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. Air travel to other destinations in the country is only possible with local carriers, using a network of minor airfields of varying capacity and quality. The airport at Port Sudan also has international flight handling capability although services are limited to neighbouring countries.

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: 20t reverse circulation drill rig being transported to Block 14 on desert tracks. 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°C to 49°C and minimum of 23 to 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 5 events of rain >1 mm and one event producing 17.5 mm since June 2015 (Figure 5-4).

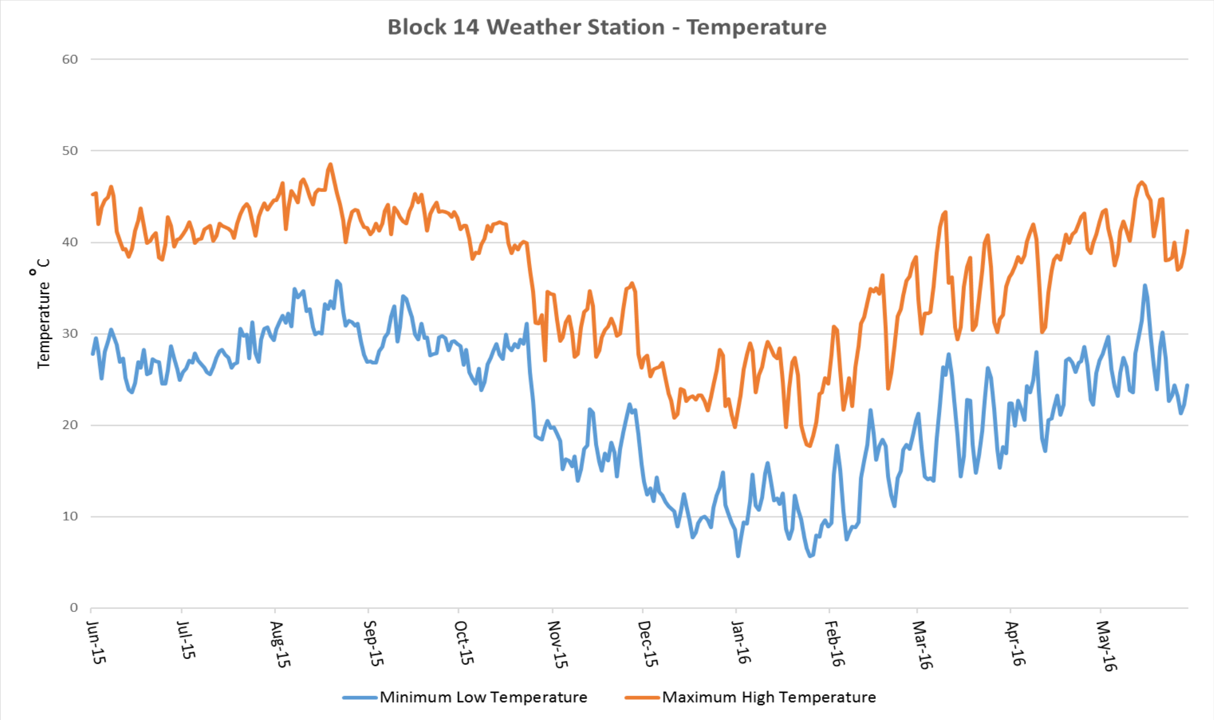


Figure 5-2: Site temperature records from Block 14 between 3 June 2015 and 4th June 2016. Source: Orca Gold Inc.

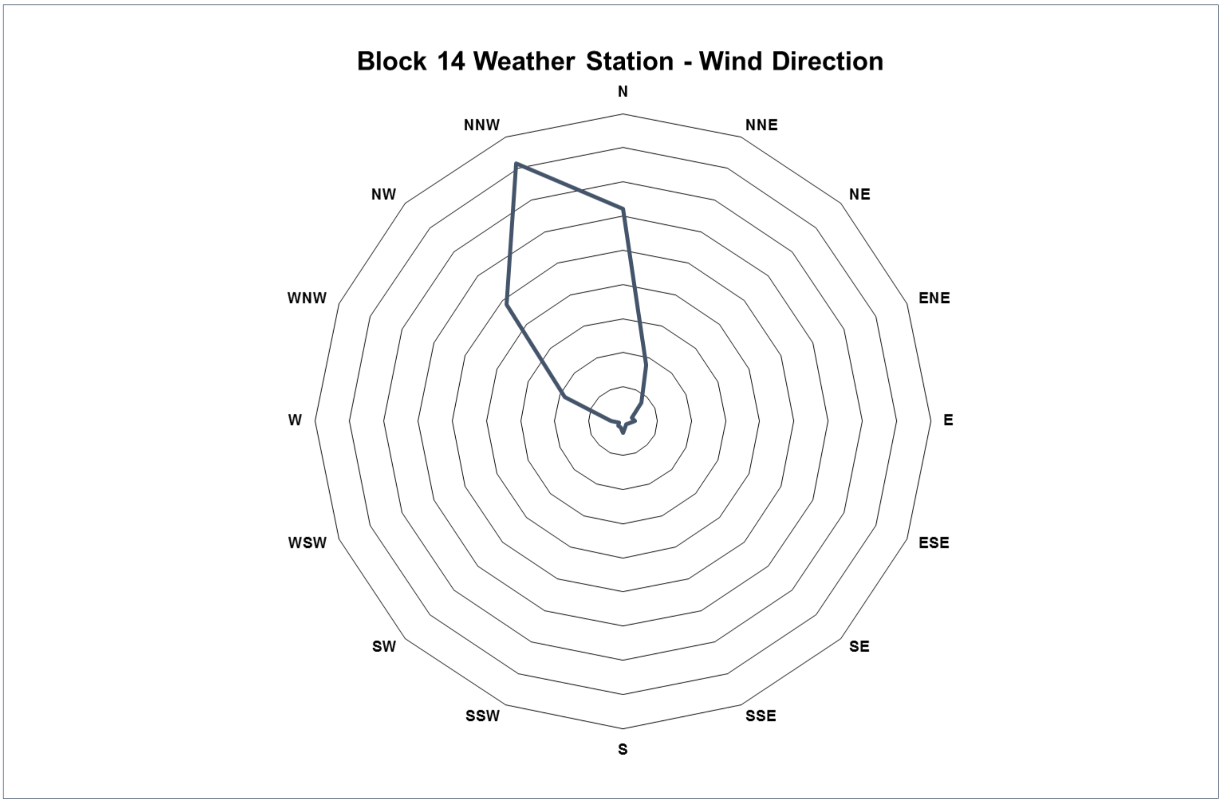


Figure 5-3: Wind direction (origin) recorded at Block 14 between 3 June 2015 and 4 June 2016. Source: Orca Gold Inc.

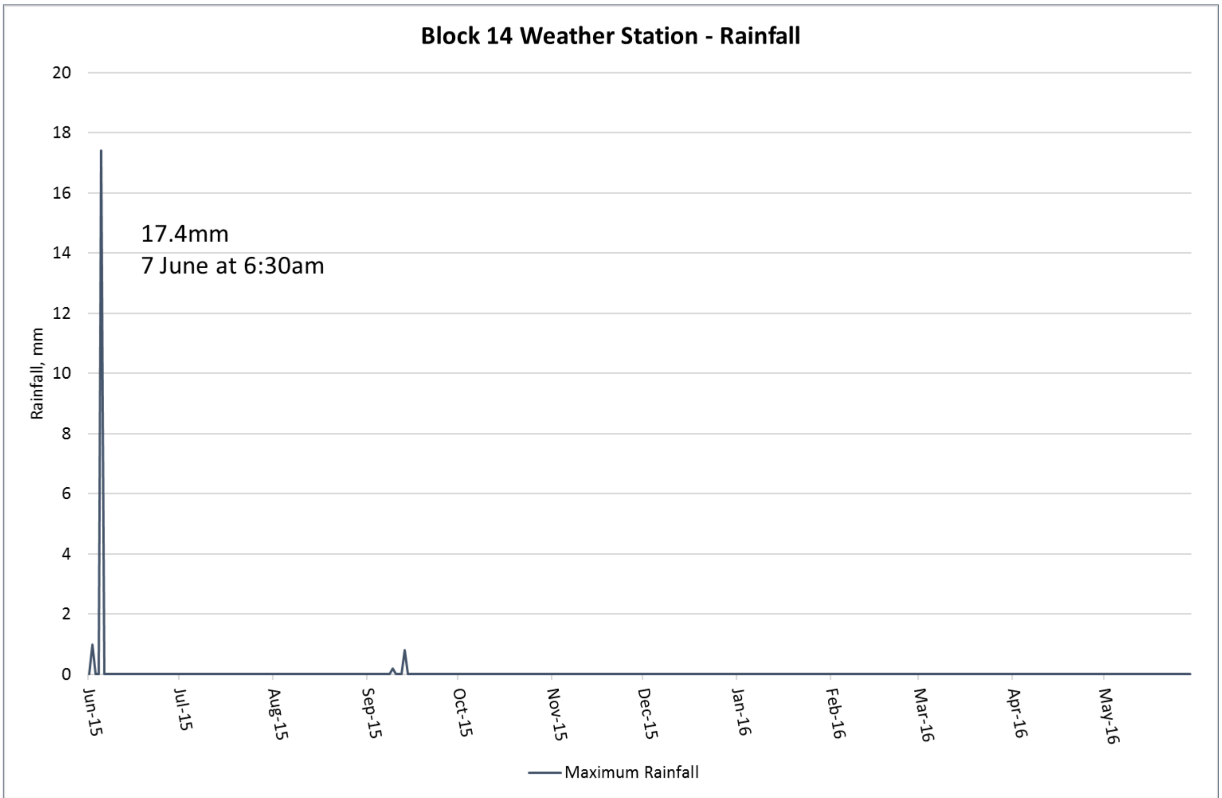


Figure 5-4: Rainfall recorded at Block 14 between 3 June 2015 and 4 June 2016. Source: Orca Gold Inc.

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 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).

5.4 Infrastructure

The Company operates a 100 man exploration camp in Block 14 comprising converted sea containers and brick built buildings that house exploration and support staff. Power is self-generated (Figure 5-5) from fuel delivered as needed in 25,000 litre tankers. Fresh water is supplied by tanker from the River Nile.

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 (Figure 5-1). Two fuel stations with telephone communications, known locally as Cafeterias are located along the route.

Figure 5-5: MSMCL exploration camp, showing containerised accommodation and brick built buildings. 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.

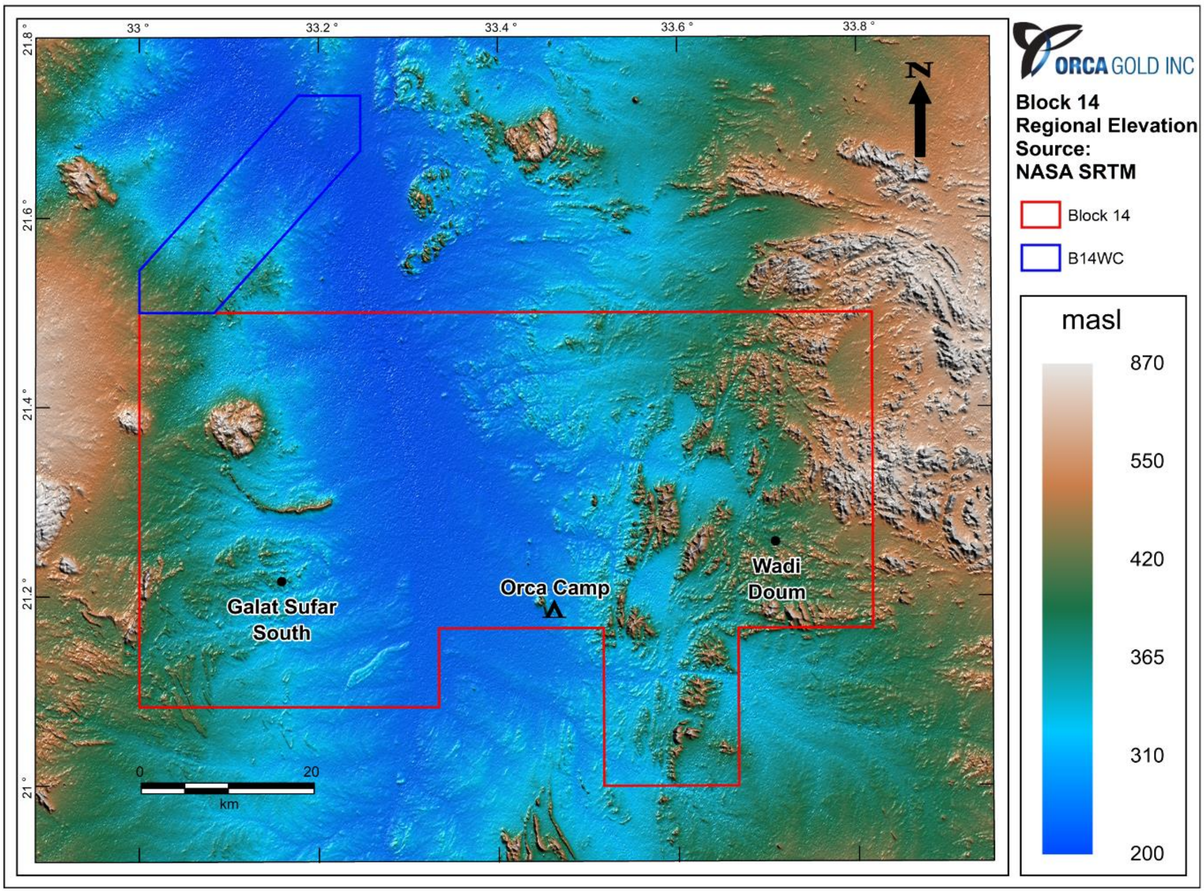


Figure 5-6: Physiography of the Block 14 tenement. Source: Orca Gold Inc.

Figure 5-7: Photograph of Wadi Gabgaba looking east showing the physiography and sparse vegetation. 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.

5.6 Authors Comments

In the opinion of the Author:

There is sufficient suitable land available within the exploration concession for the planned tailings disposal, mine waste disposal, and mining-related infrastructure such as the open pit, process plant, workshops and offices.

The mining license provides the license holder with exclusive access and use of the Project area. This does not give the license holder ownership of the land, but does made the land available for construction, operational and infrastructure needs. No surface rights are currently held; however, the process for obtaining such rights is well understood.

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 cooperation 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 – 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. The programme was initiated with a large scale drainage survey based on 63 µm silt fraction at an average density of 1 sample per 9 km^2. The samples were analysed by fire assay and aqua regia digest, ICP-MS multi-element analysis.

In 1998 RRI completed a 1:1,000,000 scale geological map of the Red Sea Hills. The map draws from the prospecting work carried out by RRI for Minex with field support and collaboration from GRAS. Large areas of the mapping coverage are in extremely remote areas and have never 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 drill holes 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.

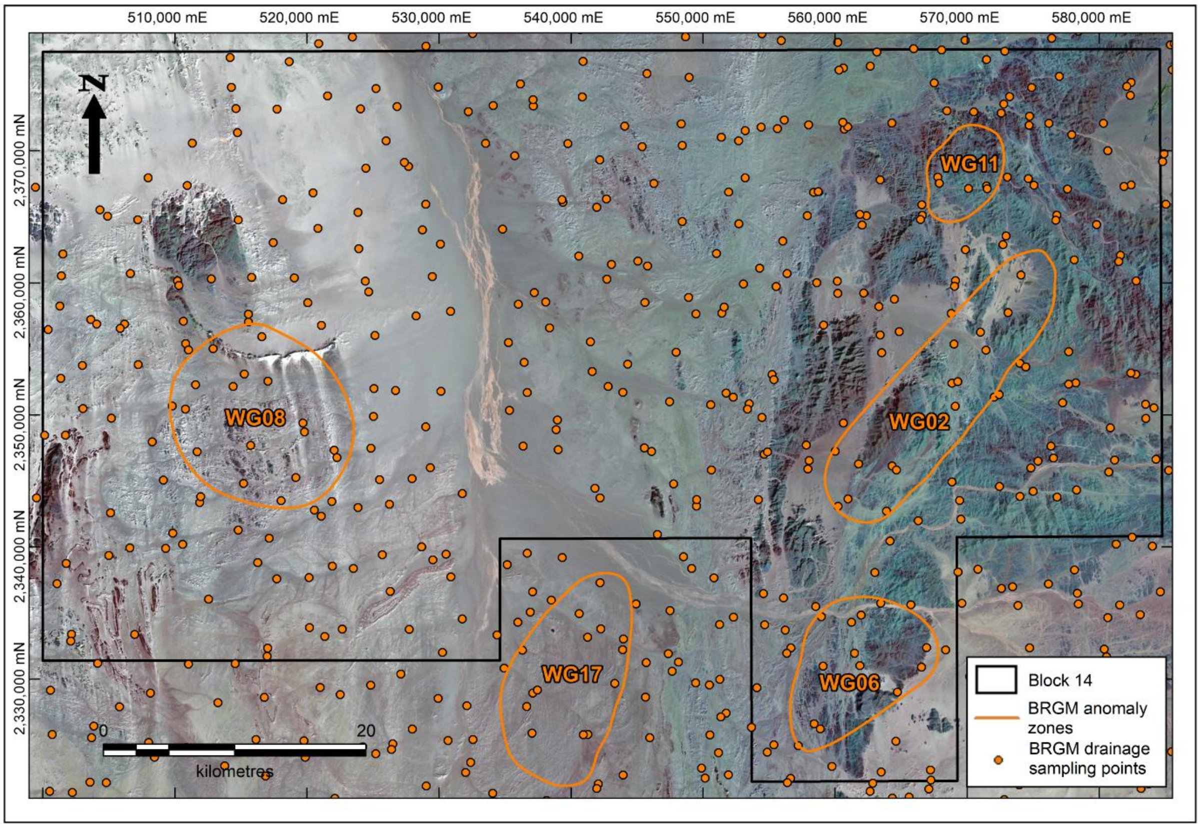


Figure 6-1: BRGM Nubian Gold Project drainage anomalies and sampling points taken from BRGM compilation, 2004. Overlain on Landsat 741 Image acquired by Orca in 2012. Source: Orca Gold.

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 4 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 82t in 2015 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 (700-1,500tpd) mines although further details and forecast production figures are not available. In addition, over the last 3 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).

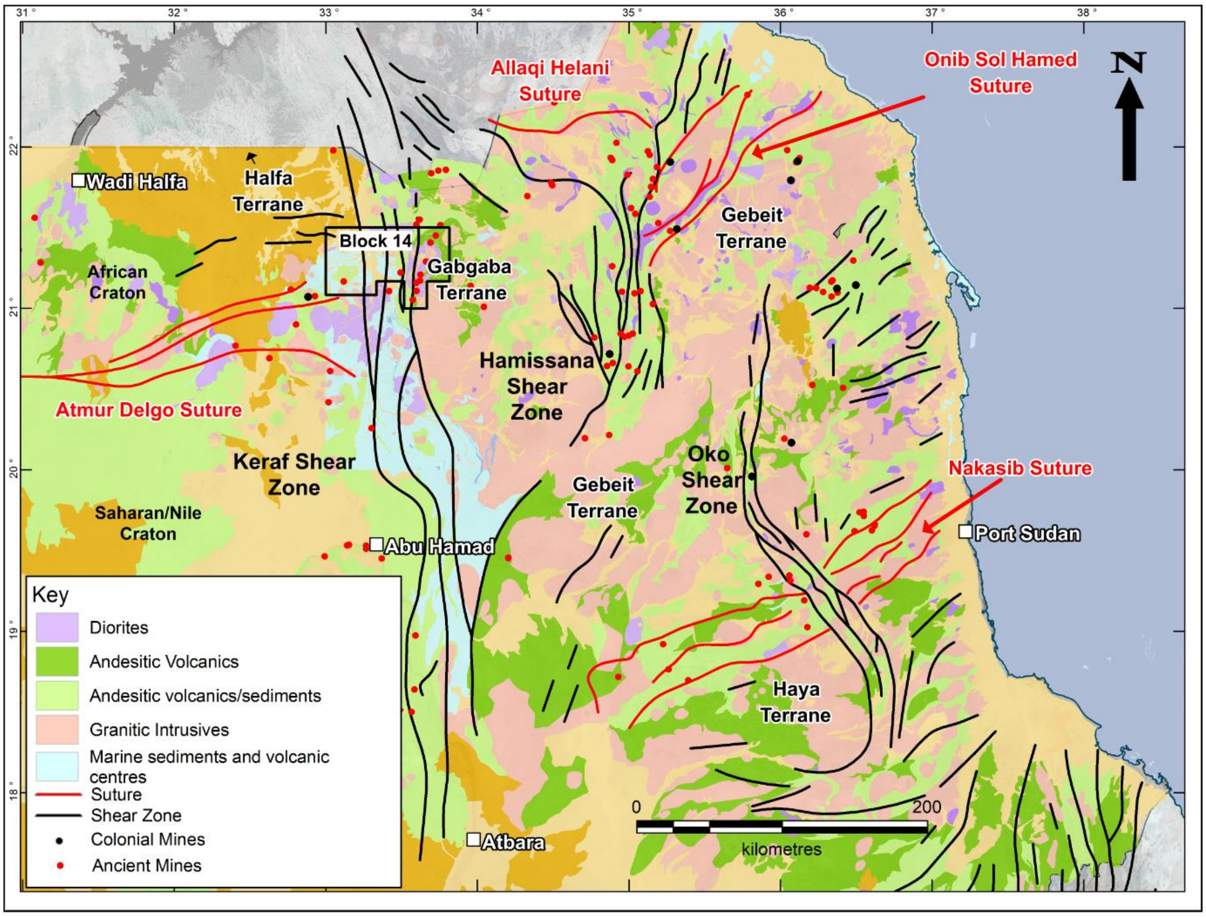


Figure 7-1: Regional schematic geology showing the assemblage of arc-accretionary belts of Neoproterozoic age accreted to the African craton to the west. Points indicate the locations of ancient and colonial mining activities. Source: Orca Gold Inc.

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).

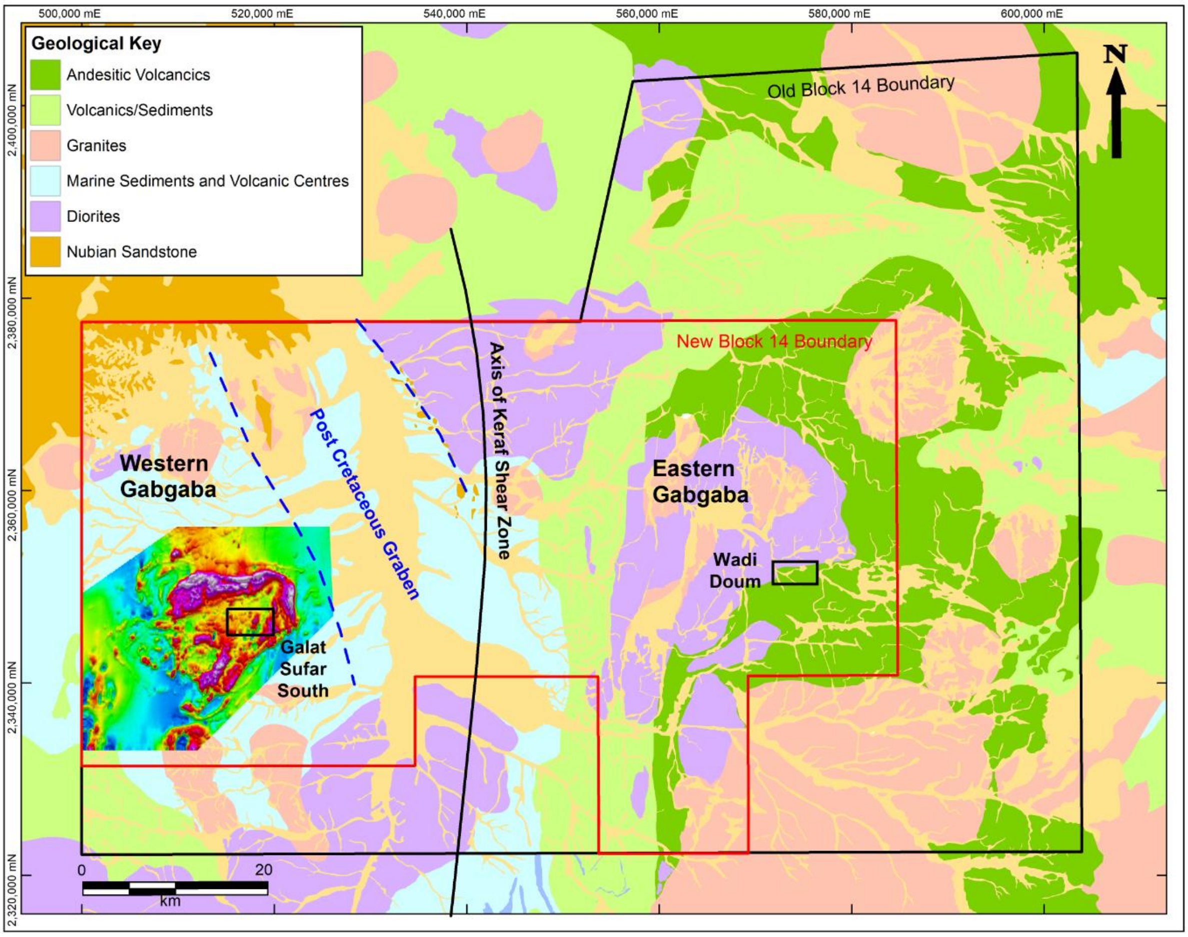


Figure 7-2: Regional geology of Block 14 EPL showing the axis of the Keraf Shear Zone and the position of a Post Cretaceous graben in Wadi Gabgaba. High Resolution Airborne Magnetic (HiRAM) Total Magnetic Intensity Image is overlain showing the annular high magnetic (HiRAM) Total Magnetic Intensity Image is overlain showing the annular high magnetic Western Gabgaba 'donut' shape which is related to mafic volcanics cored by an intermediate to sub-alkalic volcano-sedimentary assemblage

Eastern Gabgaba (Figure 7-2) is dominated by thick sequences of andesitic volcanics with subordinate 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 (<500Ma).

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 Mineralisation

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 subordinate 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 Mineralisation

Gold mineralisation is being exploited by artisanal miners throughout Block 14. The geological setting is varied but four main types or artisanal mining are apparent (Figure 7-3):

1. Colluvial mining using metal detectors that have a reliable penetration of 20 – 30cm. 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 >30m below surface.

4. Mechanised, open pit, hard rock mining to depths of up to 30m. 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 subordinate quartz veining for example at GSS and the surrounding area.

Figure 7-3: 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

7.2.3.1 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 volcanics (Figure 7-4).

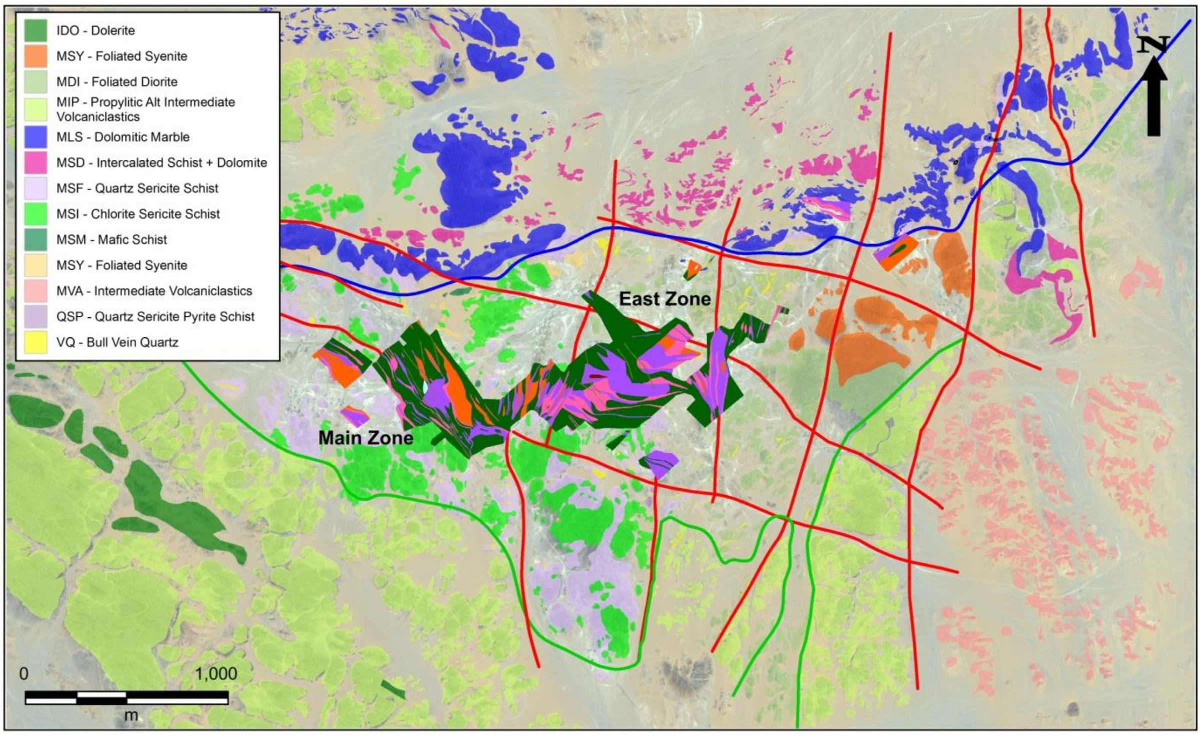


Figure 7-4: GSS geology - Blue line contains package of interleaved carbonates and volcanics, Green line marks boundary of proximal quartz - sericite altered series of intermediate volcanics, diorite and syenite intrusives that host the GSS deposit with *distal chlorite edpidote altered andesitic volcanics. Red line marks principal shears oriented 010º and 110º that control mineralisation. Source: Orca Gold.*

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 mineralisation (Figure 7-4). 1:25,000 scale mapping 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 mineralisation 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 (Figure 7-5). 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.

7.2.3.2 *Mineralisation*

6 Domains of mineralisation have been differentiated and resource modelled (Figure 7-5):

- 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 150m. 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 mineralisation 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 mineralisation 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 mineralisation 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 mineralisation is hosted within a sheared sericitised microdiorite. Steep to vertical mineralised 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 mineralisation hosted within a sericite/carbonate altered diorite/microdiorite.

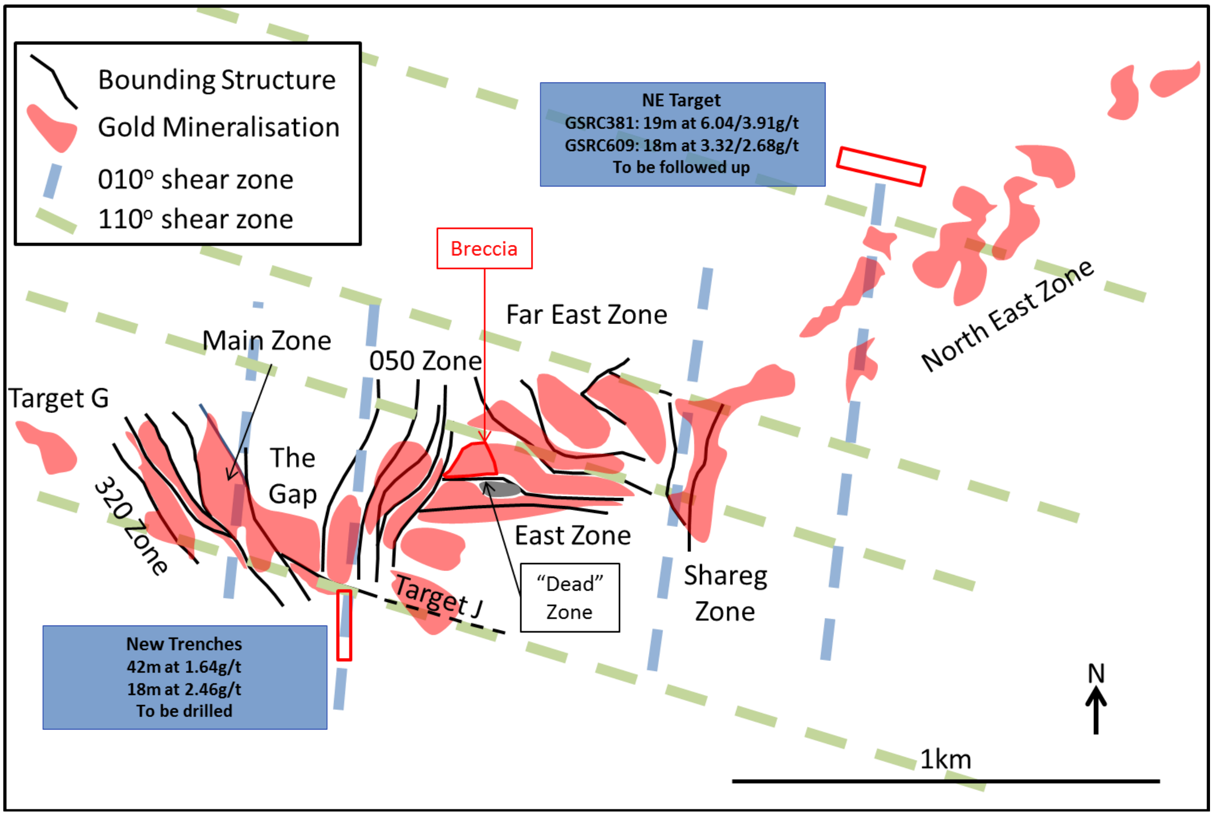


Figure 7-5: GSS deposit model showing surface expression of bounding structures and stylised position of gold mineralisation in relation to large-scale regional shears defined by high resolution satellite imagery and ground magnetic surveys. Source: Orca Gold Inc.

7.2.3.3 Alteration

Alteration is pervasive and deposit-scale units are defined by their alteration assemblages which are variably zoned outward from the gold mineralisation (Table 7-1). The first phase of mineralisation 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	Mineralisation
↑	QSP	Quartz sericite pyrite schist	strong
	QSS	Quartz sericite schist	weak
	MDI	Foliated diorite	weak, variable
	KDI	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 mineralisation, but generally the k-feldspar alteration contains weak, variable mineralisation. 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 mineralised 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 mineralisation 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 (<0.75cm), late, un-mineralised felsic and mafic dykes. In contrast to the volcaniclastics the rocks on the hill dip 75° to the east. Mineralisation on the hill is associated with stringer zones within the syenite and in places smaller shears.

The high grade mineralisation 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 mineralisation often appears un-affected by structure whereas the mineralisation hosted by the syenite on and around the summit of the hill does appear structurally controlled.

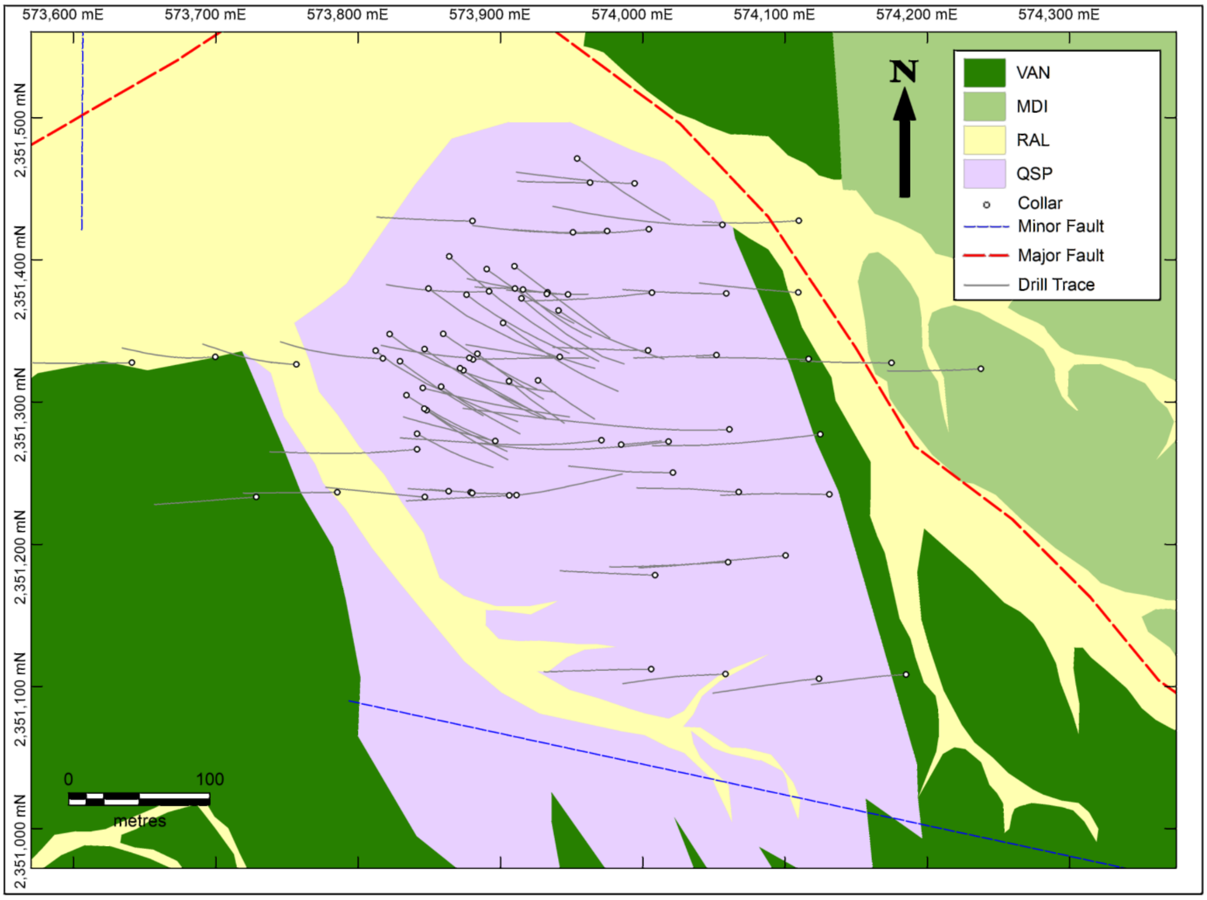


Figure 7-6: The Geology of the WD Deposit. Source: Orca Gold

8 DEPOSIT TYPES

The mineralisation types being targeted within the Block 14 Project are broadly categorised into 3 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 (2.4Moz gold, Saudi Arabia Mining Co., 2015) and the Zara Mine in Eritrea (0.9Moz 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 3.0Moz gold Gabgaba Project

Evidence of orogenic gold mineralisation 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 mineralisation 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 mineralisation.

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

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

The Author has been unable to verify the project information above and this information is not necessarily indicative of the mineralisation on the property that is the subject of this technical report.

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 March 11th 2014.

Table 9-1 below shows the work completed since the commencement of exploration on the Block 14 Project.

Table 9-1: Summary of work completed on Block 14, 2012 - 2016.

Surface Sampling	
Rock Chip samples	4,690
Soil Samples	2,682
Chip Channels (m)	56,486
Trenching (m)	40,923
Airborne Geophysics	
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)	79,458
Diamond core drilling (m)	5,310
Water Borehole Drilling (m)	2,263
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 2 diamond core holes at WD.

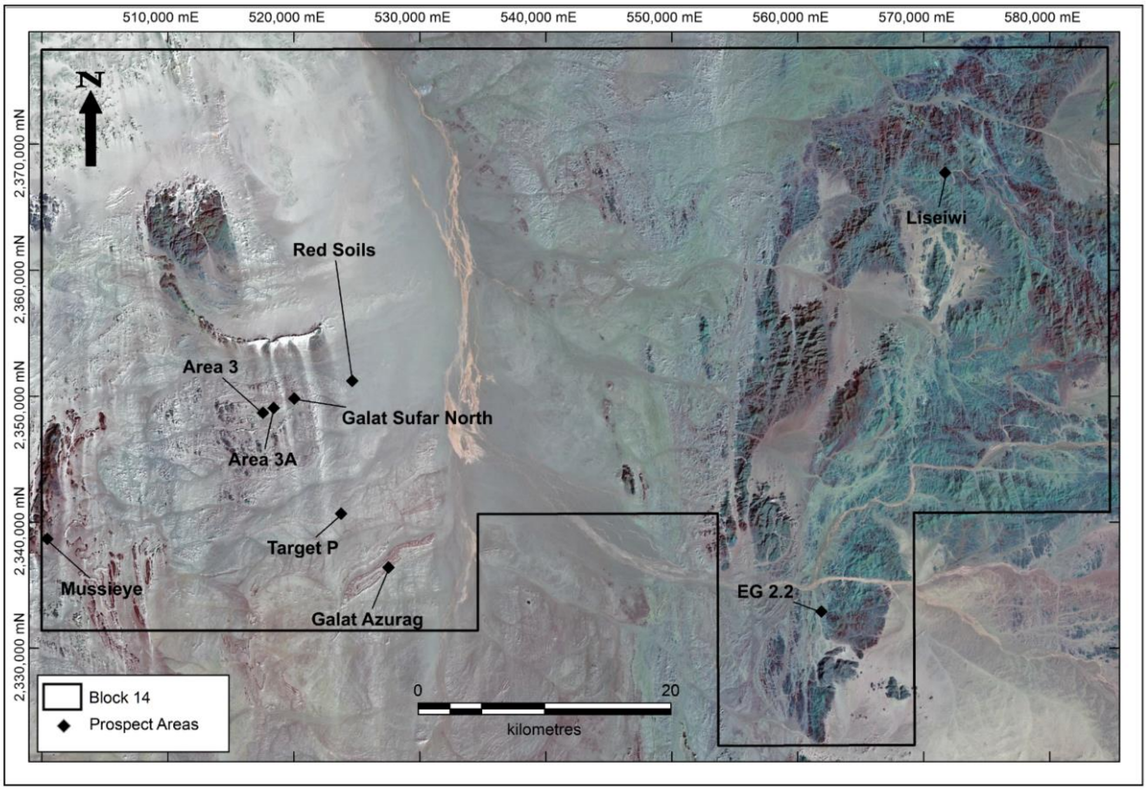


Figure 10-1: Prospect areas which have been drilled. Source: Orca Gold Inc.

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	17	5,060	449	58,207
WD	2	250	77	10,062
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 P			3	114
Total	**19**	**5,310**	**631**	**79,458**

Figure 10-2: RC Drilling at Main Zone, GSS. Source: Orca Gold.

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 rigs. (Figure 10-2).

10.1.1 Drilling and Sampling Method

RC samples are collected at 1m 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 1m (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. Samples for all drilling since July 2013 were weighed routinely.

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 93%. Statistics relating to poor recoveries are shown in Table 10-2.

Table 10-2: Samples with poor RC recoveries

Oxidation State	Samples	Recovery <60%	%
Oxide	10,333	375	3.6
Transition	15,743	142	0.9
Fresh	34,346	345	1.0
Total	**60,422**	**720**	**1.4**

Water is recorded in isolated intervals. 575 samples (Table 10-3) were logged as having contained some water present, often at rod changes, and these were not saturated with water.

114 samples logged as wet returned grades >0.50 g/t out of a total of 12,776 samples which returned >0.50 g/t – 0.89%.

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	14,430	35	0.24
Transition	19,783	20	0.10
Fresh	43,118	520	1.21
Total	**77,331**	**575**	**0.74**

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 97%.

Core recovery of <80% accounted for 2.9% of the core drilled. Recoveries of <60% were recorded from 0.80% of the core runs.

5 core samples had recoveries of <60% and returned grades >0.50 g/t out of a total of 1,928 samples which returned >0.50 g/t – 0.26%.

52 core samples had recoveries of <80% and returned grades >0.50 g/t out of a total of 1,928 samples which returned >0.50 g/t – 2.70%.

10.1.3 Drill Hole Surveying

10.1.3.1 Collar Location Surveying

Drill hole 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 drill hole name, was placed at the collar (Figure 10-3). After drilling all collars were surveyed using differential GPS (DGPS) equipment.

Figure 10-3: Drill hole collar with details as reference after completing the hole. Source: Orca Gold.

10.1.3.2 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 6m 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.

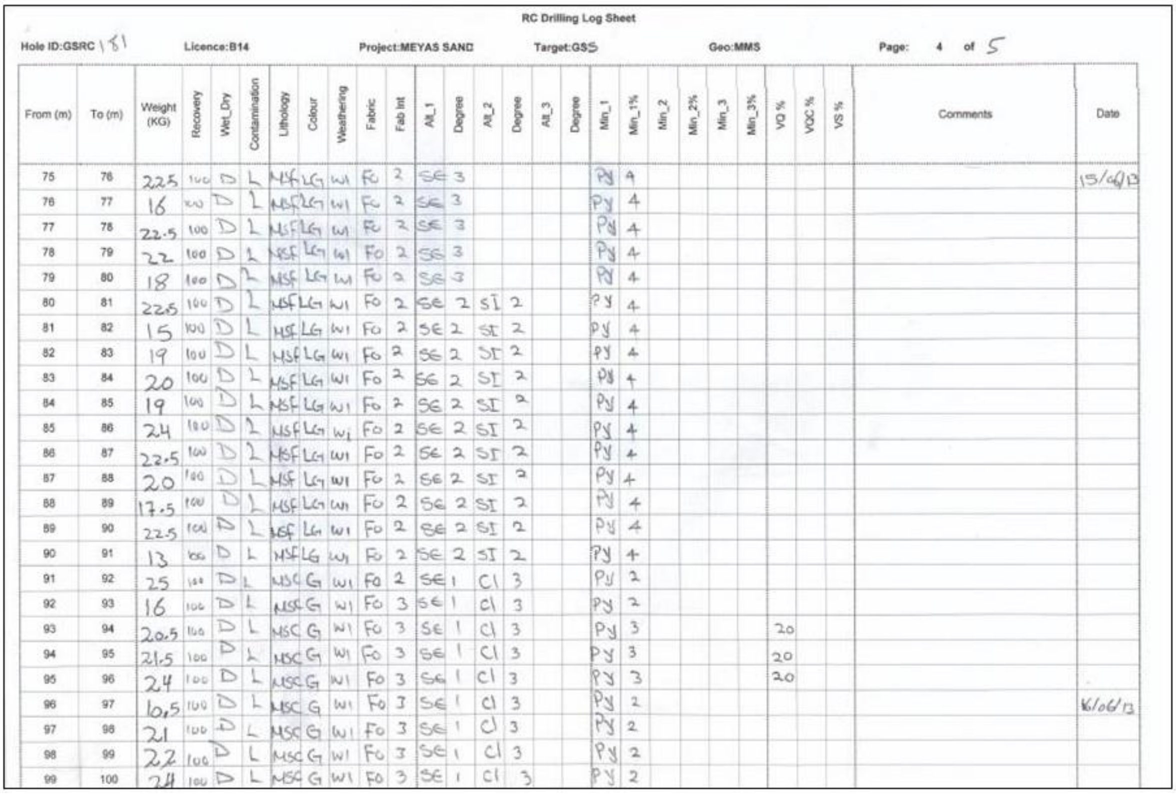
A Reflex Ez-Trac single-shot survey tool (Reflex) was used for all drilling, surveying at 50m intervals during the drilling of the hole. Due to significant azimuth variation seen in the RC drilling up to the end of June 2013, a Reflex Gyro multi-shot survey tool (Gyro) was used to carry out surveys on completion of all new holes (in addition to the reflex used during drilling). It was also used to re-survey all mineralised 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

10.1.4.1 RC Drilling

RC drill chips were geologically logged at 1m 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).



RC Drilling Log Sheet

Hole ID:GSRC 181 Licence:B14 Project:MEYAS SAND Target:GSS 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	225	100	D	L	MSF	LG	W1	Fo	2	SE	3					Py	4									15/06/13
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									

Figure 10-4: Example of an RC geological Log. Source: Orca Gold.

RC drill chip samples were sieved at 1m 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: Picture showing a 1m interval sieved RC chip sample in a chip tray. Source: Orca Gold.

10.1.4.2 Diamond Drilling

To facilitate geotechnical and structural logging diamond drill holes 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 mineralisation. Rock type, stratigraphic subdivisions, alteration, oxidation and mineralisation are routinely recorded.

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

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 4m 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

A total of 19 core holes were systematically sampled for the purpose of density measurements, 17 holes from GSS and 2 holes from WD.

A total of 825 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 5m intervals. The lithology, depth and oxidation condition are recorded. The samples are dried in an oven for 24 hours (at 100^{0}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 10-4 below shows a summary of the density measurements made.

Table 10-4: Density Sampling.

Zone	Lithology	GSS		WD	
		g/cm3	Samples	g/cm3	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: View over GSS Resource Area) has identified a significant mineralised system and Mineral Resource over an area of 2 x 1 km (Figure 10-7). Mineralisation is hosted within variably sheared intermediate intrusives and is associated with intense quartz-sericite-carbonate alteration and pyrite. Drilling at GSS has identified significant mineralisation, with true widths in excess of 80 m intersected (Figure 10-8 and Figure 10-9).

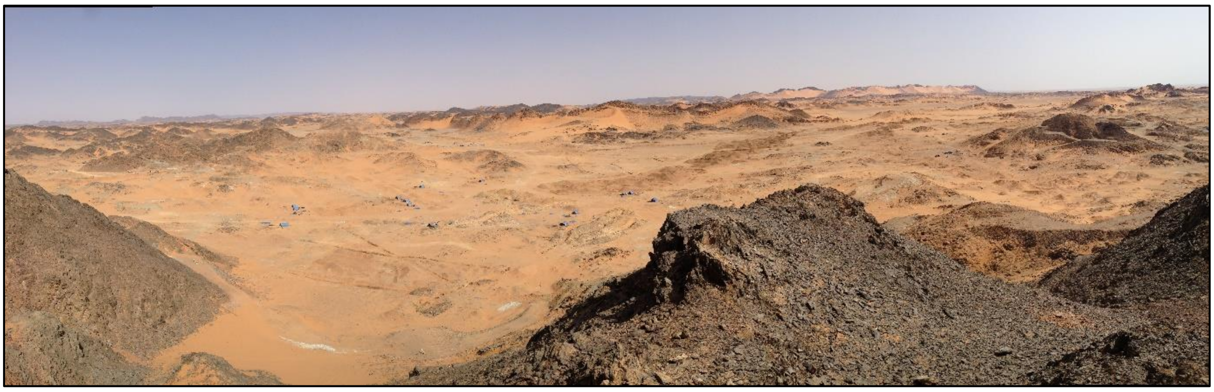


Figure 10-6: View over GSS Resource Area

A selection of drill intercepts is shown in Table 10-5 below:

Table 10-5: Selected Intercepts from GSS. True widths are 65-75% of intercept widths.

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

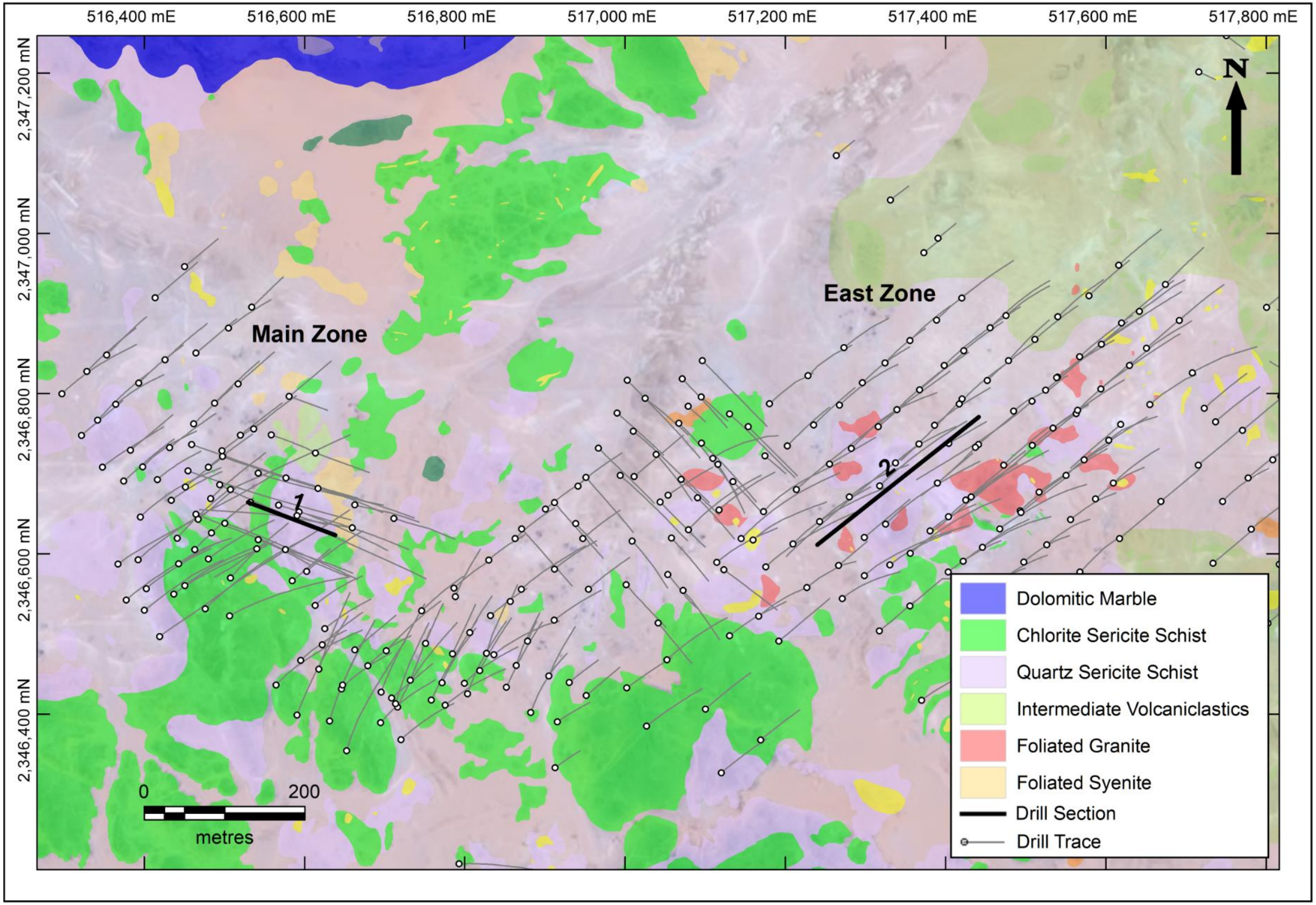


Figure 10-7: GSS: Geology, mineralisation and drill traces. Source: Orca Gold.

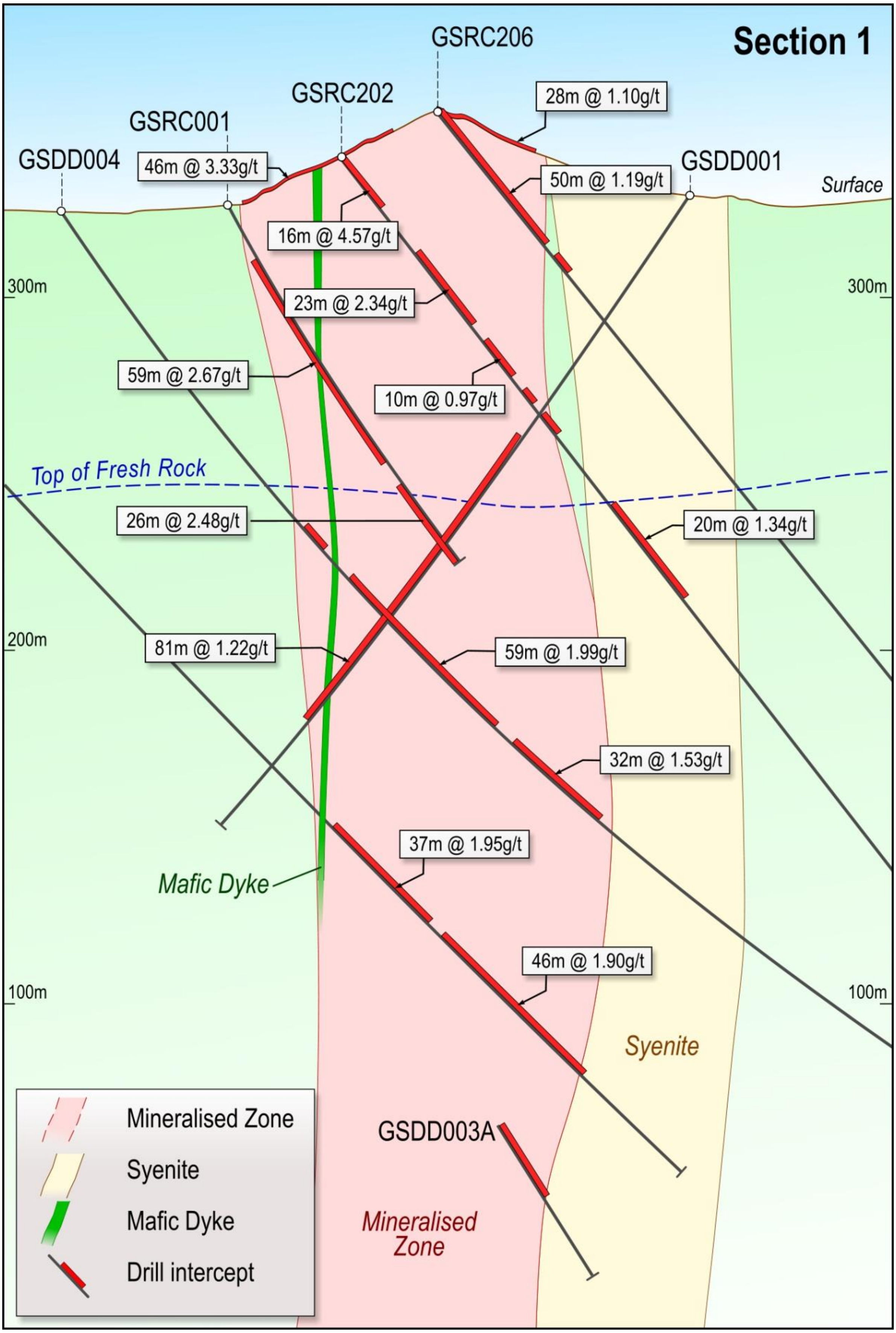


Figure 10-8: Main Zone Drill Section 1. Source: Orca Gold.

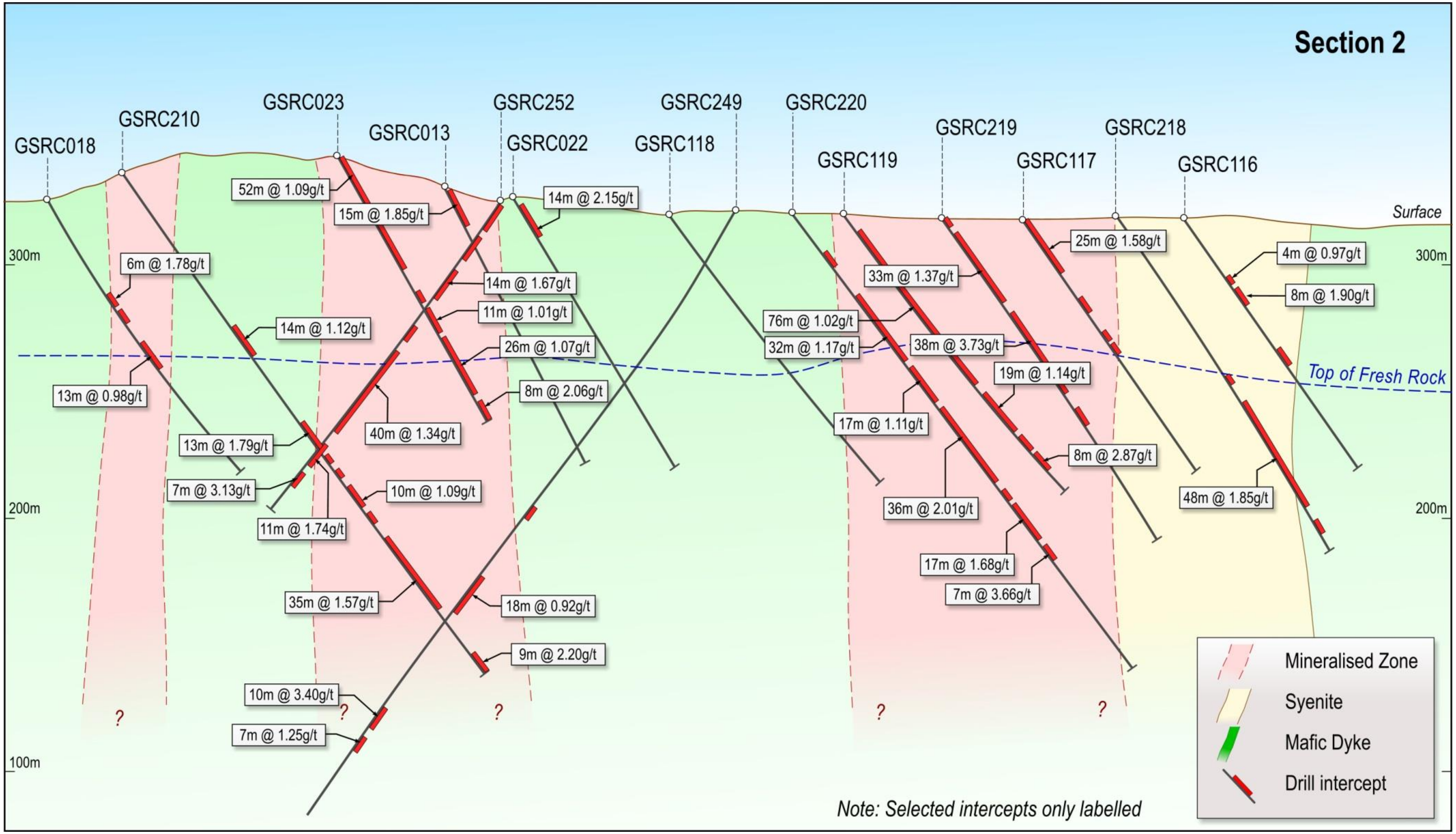


Figure 10-9: East Zone Drill Section 2. Source: Orca Gold.

10.2.2 WD

Drilling at WD has identified significant mineralisation and a Mineral Resource over an area of 300m by 300m and centred on a small hill as shown in Figure 10-10 and Figure 10-11 below.

Figure 10-10: View over WD Resource Area

High grade mineralisation 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 mineralisation 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. True widths are 60-75% of intercept.

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

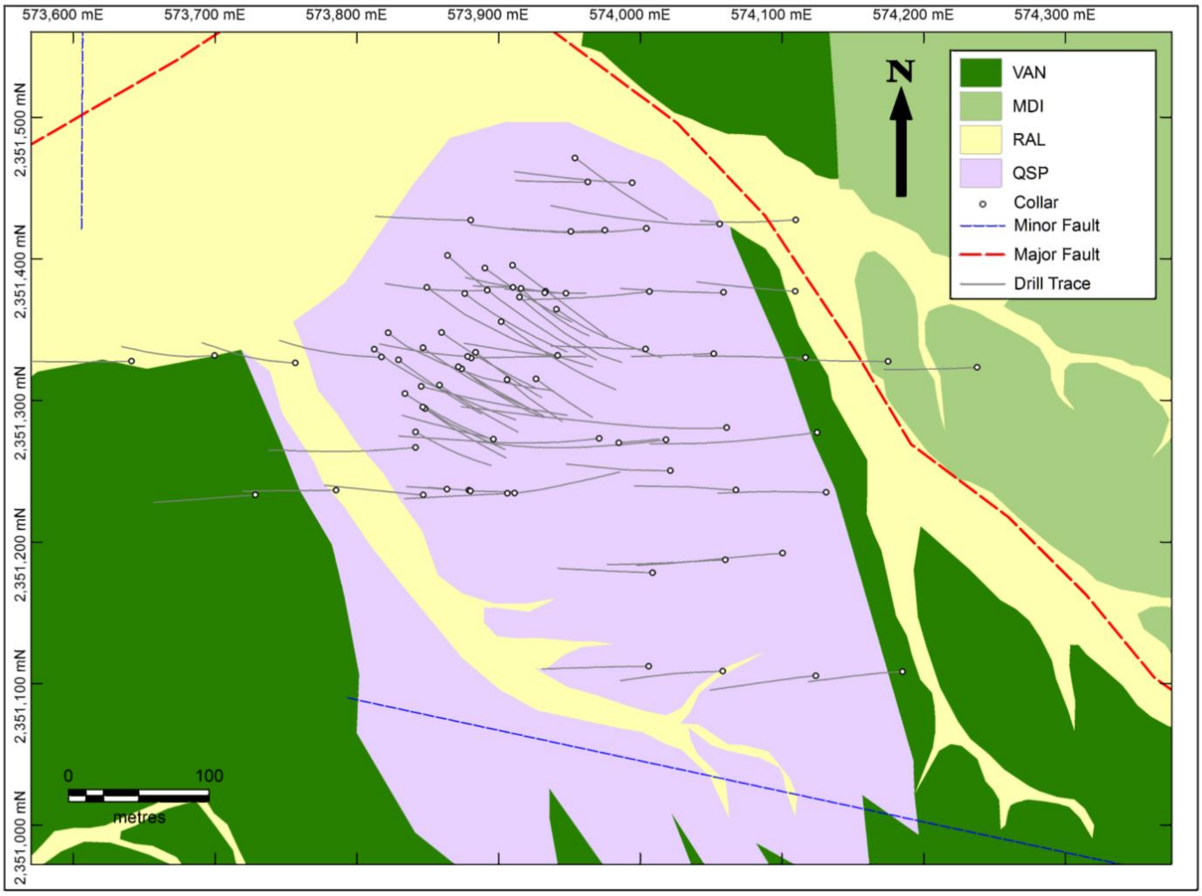


Figure 10-11: Location of targets drilling within the Block 14 Project. Source: Orca Gold.

10.2.3 Other Prospects Drilled

11,189m of RC drilling (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.

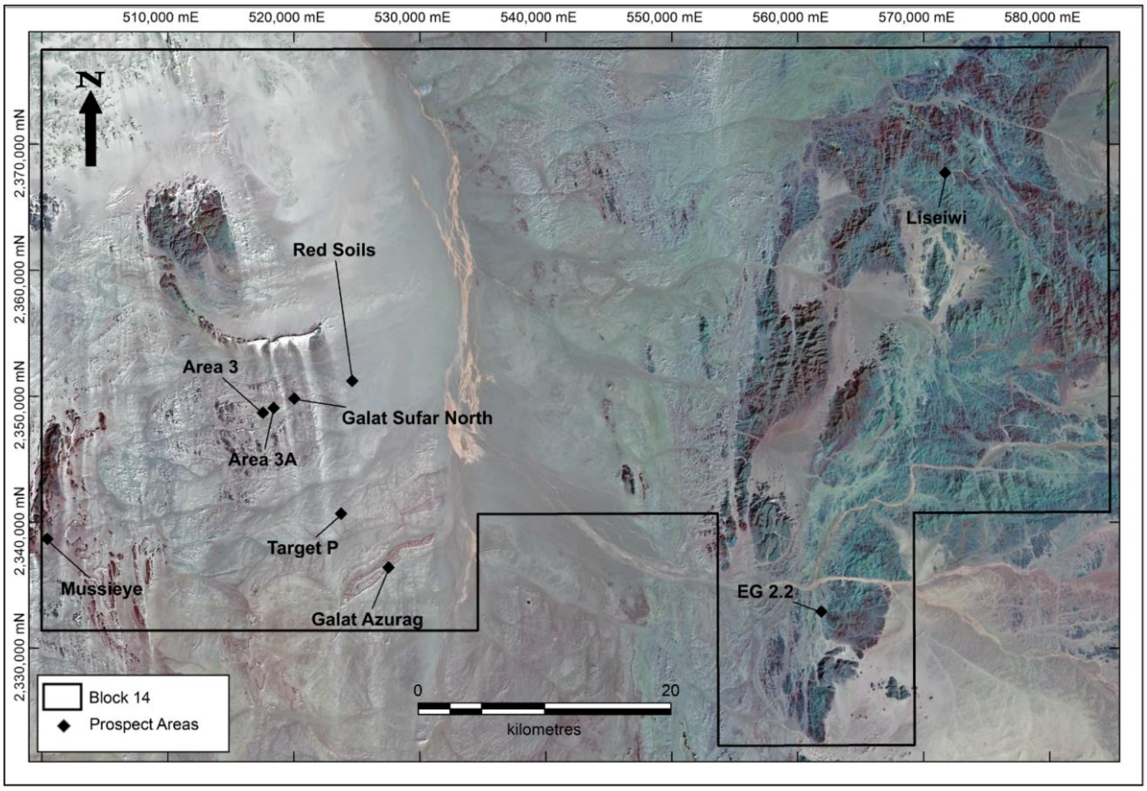


Figure 10-12: Location of targets drilling within the Block 14 Project. Source: Orca Gold, Landsat 741 Image acquired by Orca in 2012.

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. True widths are 60-75% of intercept

Prospect/Target	Hole ID	Depth From	Depth To	Interval	No Top Cut	Top Cut 10 g/t
					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

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 jaw crusher at another 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.
- QAQC samples comprising standards (~60g 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, 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 75 µm, 5% sieve tests.
- A split of 250g pulp is shipped to the Rosia Montana laboratory and a second 500g 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 50g 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.

Figure 11-2: Sealed drums of samples after arrival at sample preparation facility. Source: Orca Gold.

Figure 11-3: Jaw Crusher. Source: Orca Gold.

Figure 11-4: Riffle Splitter. Source: Orca Gold.

Figure 11-5: Storage of archived pulp samples. Source: Orca Gold.

11.4 Quality Control and Quality Assurance

Orca has instigated external QAQC processes to monitor the reproducibility of geochemical, trenching and drilling data. The QAQC 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	QAQC Sample Type	QAQC 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, have their own internal quality performance processes. These follow best practice guidelines required for qualification under International Organisation for Standardisation ("ISO") standards. The standard QAQC 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 mineralisation being targeted. CRMs are supplied by Geostats Pty Ltd of Perth, Australia in sealed 80g 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

11.4.5.1 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 50g fire assay with AAS finish (with multi element suite). These pulp samples represent 5 complete drill holes (1 x diamond, 4 x RC) and consisted of 125g pulp splits taken from the reference samples stored at the Atbara preparation facility.

11.4.5.2 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 7 drill holes (All RC) and consisted of 500g 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 QAQC 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 QAQC sample data for the sampling completed on the project by Orca. Overall, the QAQC 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 summarised 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 VALUE	RESULT AVERAGE	STDEV of Results	% Fail
DRILLING (DD, RC)	BLANK	4,196	0.05	0.006	0.002	0%
SURFACE (RCHIP, DRAINAGE)	BLANK	89	0.05	0.005	0.005	0%
TRENCH (TR, HC)	BLANK	928	0.05	0.006	0.004	0%

Note, all samples assaying below detection limit of 0.01 ppm (DL) arerounded to ½ DL or 0.005ppm.

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.

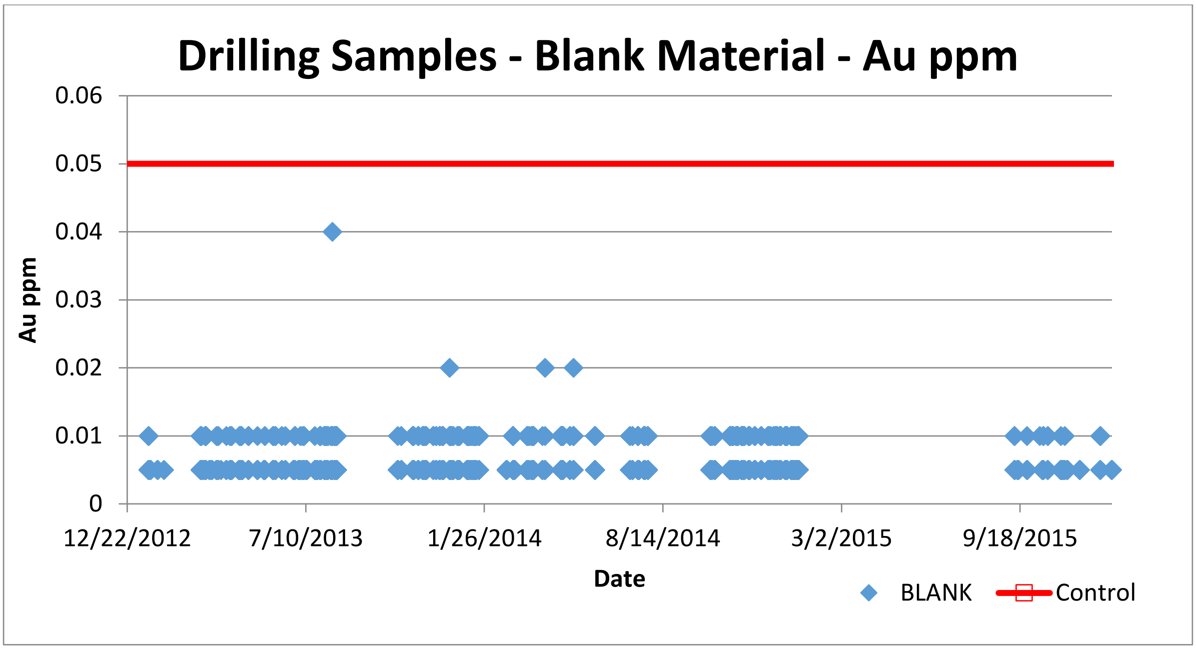


Figure 11-6: Blanks Samples for Drilling Samples, (50g Fire Assay).

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 are detailed in Table 11-3.

Table 11-3: Quantity of duplicate pairs collected by different sampling methods.

Type	Quantity of pairs
Diamond Drill Core	284
Reverse Circulation Drilling	3,788
Trench	1,108

11.5.2.1 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 1ppm, 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.

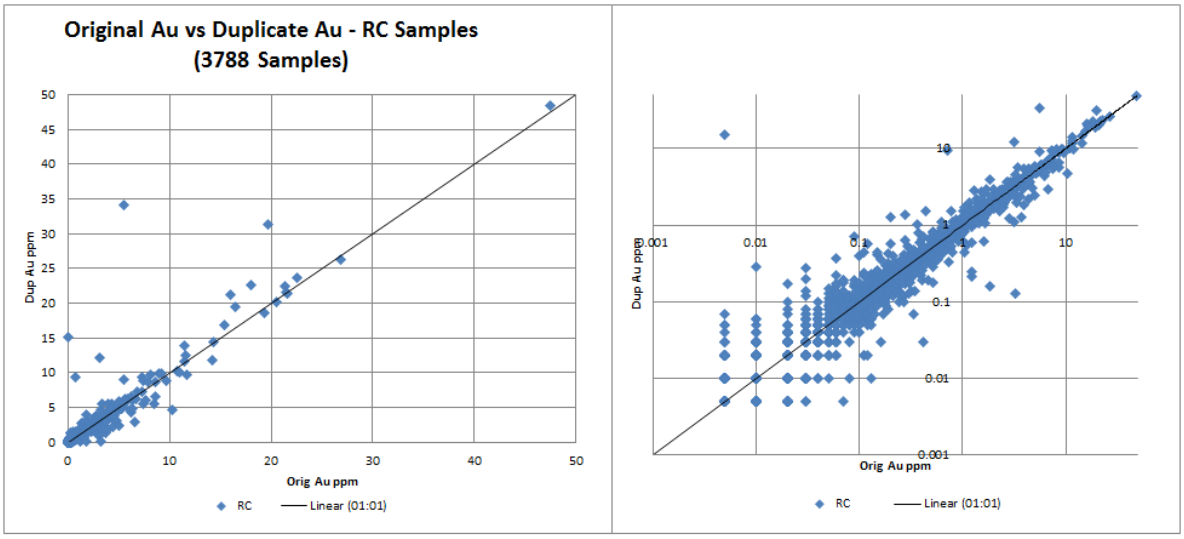


Figure 11-7: Normal and Log scatter plots of RC field duplicates.

Ranked 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 33.73 at 94.8%) compared to the DD duplicates (41.6% 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.

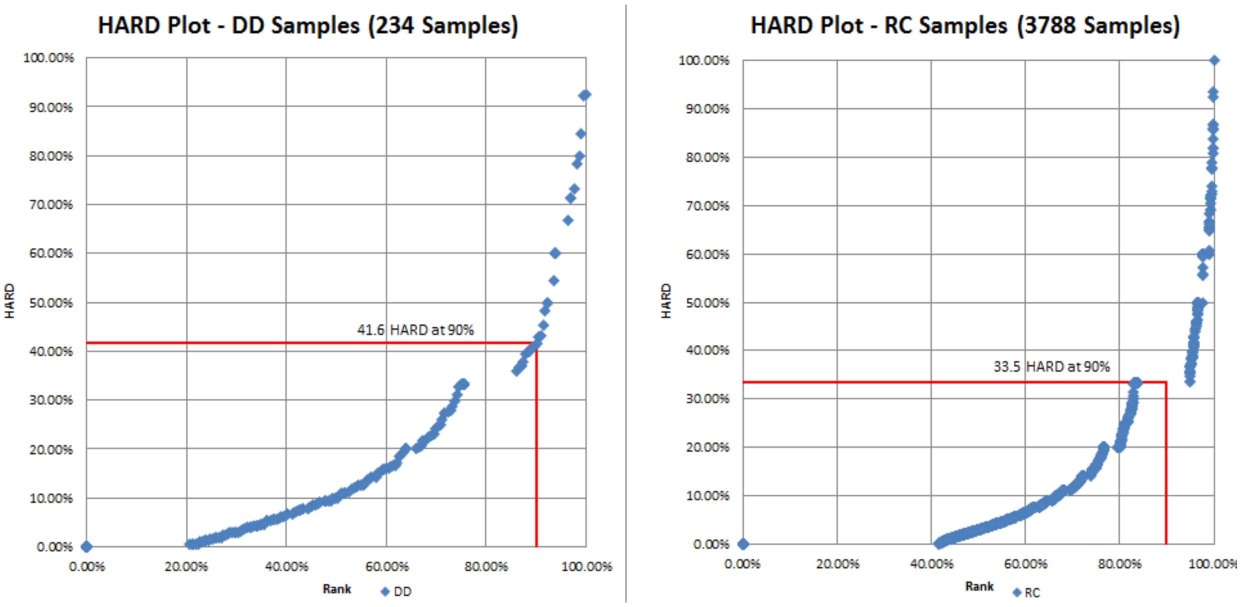


Figure 11-8: Ranked Hard Plots for DD duplicate samples and RC duplicate samples.

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 893 duplicate pairs compared to the total of 4,022 for the previous two sets. Therefore ¾ of the previous data set were <0.3 ppm, this is in part due to the utilisation 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.

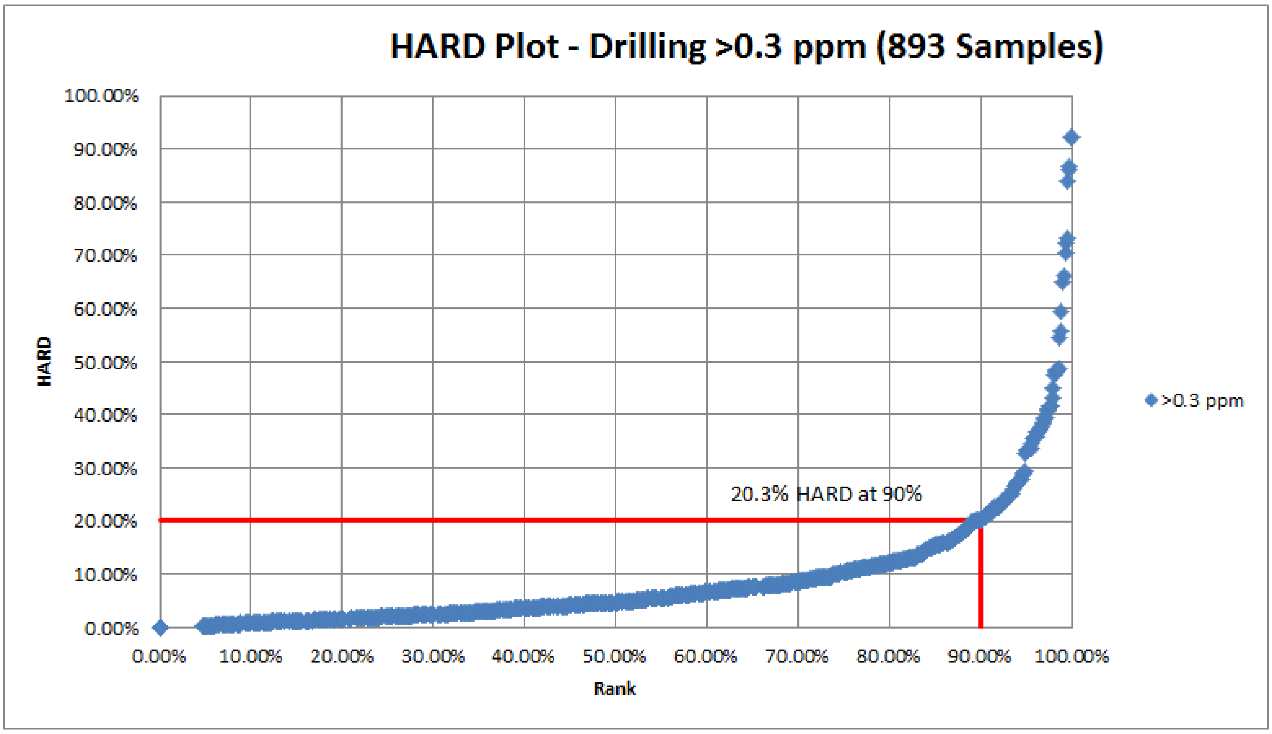


Figure 11-9: Ranked Hard Plot for all drilling (DD and RC) duplicates where original sample grade is >0.3 ppm.

11.5.2.2 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.

RC %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 5ppm 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.

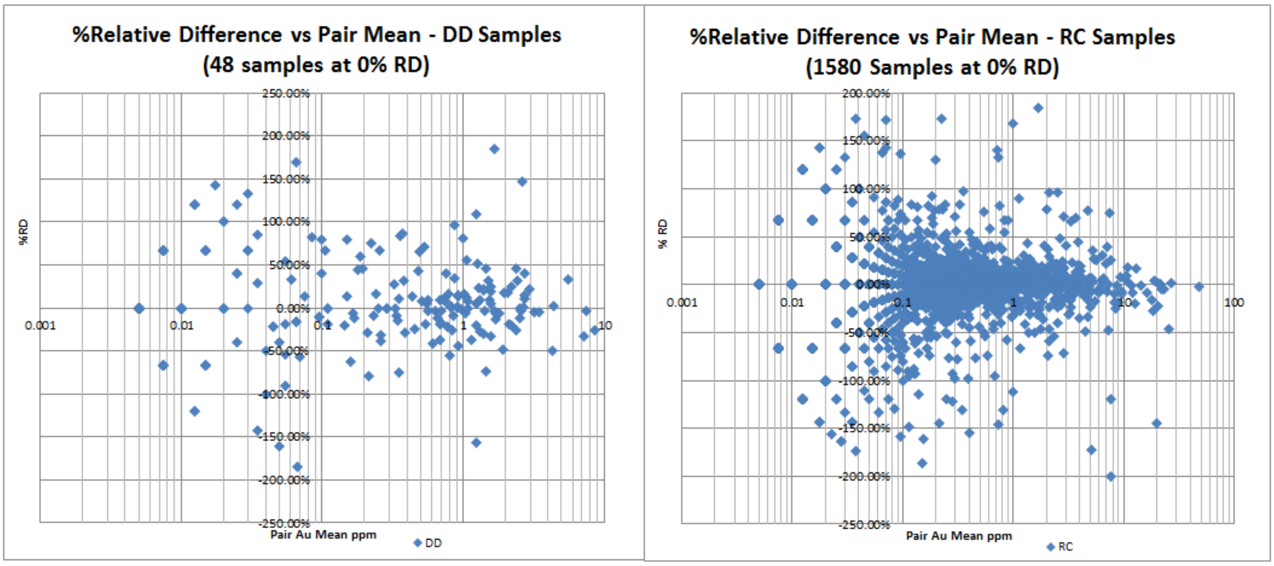


Figure 11-10: %Relative Difference plot for DD duplicates and RC duplicates.

11.5.3 Pulp Duplicates (Umpire Assay)

973 pulp duplicates 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 QAQC 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 5 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 5 outliers identified only account for 0.5% of the data set and may be the result of nugget gold or poor subsampling of the pulp either from the pulp reject or from the submitted pulp.

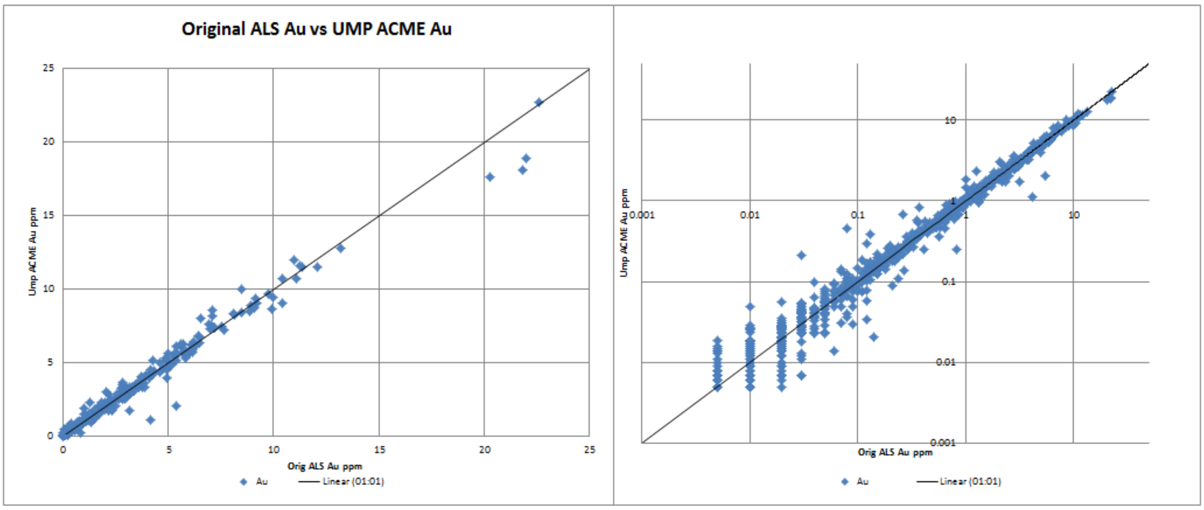


Figure 11-11: Normal and Log X-Y Scatter plots for Pulp Duplicate Umpire Assays.

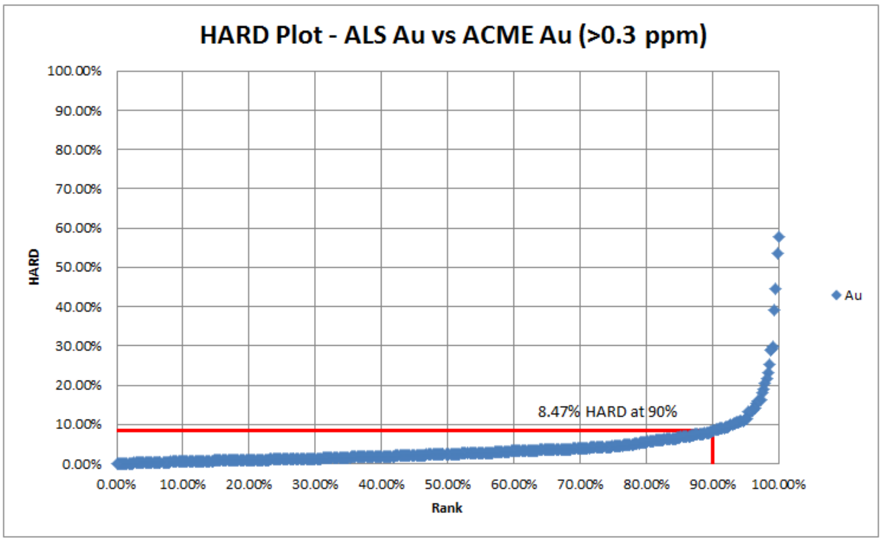


Figure 11-12: Ranked HARD plot of all duplicates with original result >0.3 ppm (483 Samples).

11.5.4 Bottle Roll Pulp Duplicate

481 samples of 500g were assayed by bottle roll. For the QAQC 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 6 points demonstrating positive relative difference above 50% and secondly 6 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 1112). 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.

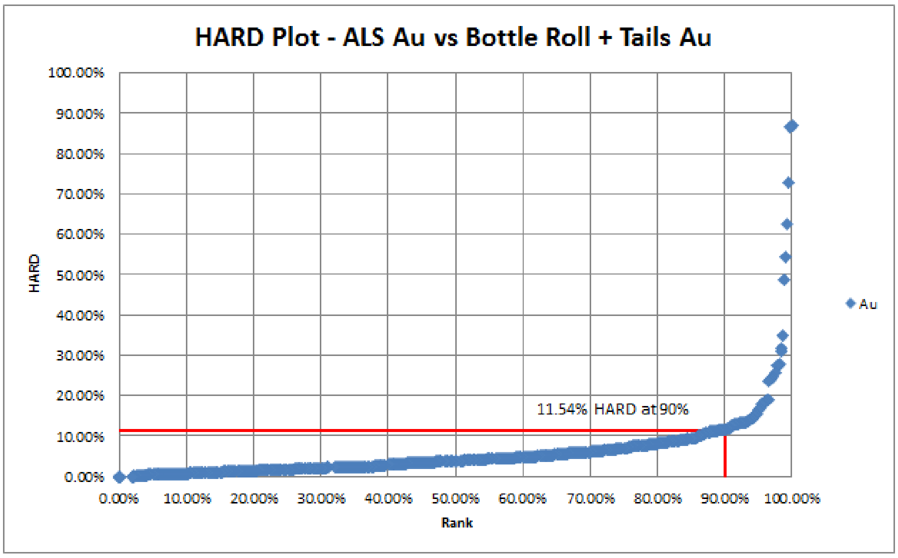


Figure 11-12: Ranked HARD plot of Bottle Roll duplicate pairs.

11.5.5 Certified Reference Material (CRM)

11.5.5.1 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 QAQC 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 QAQC, ±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 summarised for drilling below (Table 11-4).

11.5.5.2 Drilling CRM

A total of 2,139 CRM samples have been included in drilling sample sequences as summarised 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 AVERAGE	STDEV of Results	% 3SD Fail (#)	% 2SD Fail (#)	% 1SD Fail (#)
G307-7	370	7.87	7.90	0.167	0 %	0 %	10.8 % (40)
G303-2	440	4.11	4.24	0.107	0 %	0 %	5.9 % (26)
G900-7	359	3.22	3.23	0.081	0 %	0 %	2.8 % (10)
G308-3	255	2.47	2.50	0.056	0 %	0 %	2 % (5)
GBMS304-5	4	1.62	1.52	0.022	0 %	0 %	75 % (3)
G300-9	559	1.53	1.51	0.028	0 %	0 %	1.1 % (6)
G901-7	22	1.53	1.48	0.030	0 %	0 %	0 %
G908-3	180	1.03	1.03	0.024	0 %	0 %	2.8 % (5)
G907-1	573	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	1003	0.4	0.39	0.024	0.1 % (1)	0 %	0.2 % (2)
GLG303-1	273	0.164	0.16	0.008	0 %	0 %	4.4 % (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	4108			Number	2	5	138
				%	**0.0%**	**0.1%**	**3.4%**

11.6 Author's Statement

Mr Nic Johnson, the author responsible for the project Mineral Resource Estimates witnessed in 2014 (17th – 21st January) all aspects of the collection, preparation and dispatch of samples carried out by Orca personnel. The sample collection and preparation, analytical techniques and security protocols implemented by Orca for the Project are consistent with standard industry practice and are suitable for the reporting of exploration results. QAQC 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 QAQC conducted by Orca are adequate for and consistent with the author's understanding of the style of mineralisation targeted by Orca.

12 DATA VERIFICATION

In accordance with National Instrument 43-101 guidelines, Mr Nic Johnson, the author responsible for the project Mineral Resource Estimates visited the Block 14 project area between the 17th and 21st January 2014.

The purpose of the site visit was to inspect selected areas within the permits to ascertain the geological setting, witness the extent of the exploration work and visit the principal targets within the Block 14 project.

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;
- QAQC procedures and control data;
- Data and database management systems;
- Sample handling and storage;
- The sample preparation facility in Atbara;
- Future work programmes and budgets for the next 12 months; and
- The author has reviewed procedures, drill data and QAQC 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 QAQC 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; and
- 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 Author's Statement

Mr Nic Johnson, the author responsible for the project Mineral Resource Estimates has assessed the geological work undertaken, the surface geochemical and drilling data for the Block 14 project and concluded that all logging, sampling and QAQC procedures implemented by Orca to date were undertaken to a high standard when compared to industry practice.

The Author has independently reviewed available QAQC 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 test work have been completed since 2014. Table 13-1 below shows a summary of all test work undertaken.

Table 13-1: Metallurgical Sample Selection.

<table>
<tr><th>Metallurgical Laboratory</th><th>Year</th><th>Sample</th><th>Test Work</th></tr>
<tr><td rowspan="14">ALS Metallurgy (Perth)</td><td rowspan="14">2014</td><td>Main Zone Fresh 1</td><td rowspan="4">Comminution</td></tr>
<tr><td>Main Zone Fresh 2</td></tr>
<tr><td>East Zone Fresh 1</td></tr>
<tr><td>East Zone Fresh 2</td></tr>
<tr><td>Main Zone Fresh 1</td><td rowspan="7">Gravity/Direct Cyanidation</td></tr>
<tr><td>Main Zone Fresh 2</td></tr>
<tr><td>East Zone Fresh 1</td></tr>
<tr><td>East Zone Fresh 2</td></tr>
<tr><td>East Zone Oxide 1</td></tr>
<tr><td>East Zone Transition 1</td></tr>
<tr><td>GSS High Grade 1</td></tr>
<tr><td>Main Zone Fresh 1</td><td rowspan="2">Diagnostic Leach</td></tr>
<tr><td>Main Zone Fresh 2</td></tr>
<tr><td>Main Zone Fresh 1</td><td>Qemscan Mineralogical Analysis</td></tr>
<tr><td rowspan="20">SGS Mineral Services (UK)</td><td rowspan="20">2015
2016</td><td>East Zone Oxide 2</td><td>Oxide Coarse Ore Leach</td></tr>
<tr><td>East Zone Fresh 3</td><td rowspan="5">Direct Cyanide Leach</td></tr>
<tr><td>Main Zone Fresh 3</td></tr>
<tr><td>WD Fresh 1</td></tr>
<tr><td>WD Fresh 2</td></tr>
<tr><td>WD Fresh 3</td></tr>
<tr><td>East Zone Fresh 3</td><td rowspan="2">Carbon in Leach</td></tr>
<tr><td>Main Zone Fresh 3</td></tr>
<tr><td>East Zone Fresh 3</td><td rowspan="2">Gravity/Direct Cyanidation</td></tr>
<tr><td>Main Zone Fresh 3</td></tr>
<tr><td>East Zone Fresh 4</td><td rowspan="5">Flotation</td></tr>
<tr><td>Main Zone Fresh 4</td></tr>
<tr><td>WD Fresh 1</td></tr>
<tr><td>WD Fresh 2</td></tr>
<tr><td>WD Fresh 3</td></tr>
<tr><td>East Zone Fresh 4</td><td rowspan="5">Fine Grinding/Intensive Leach of Float Concentrate</td></tr>
<tr><td>Main Zone Fresh 4</td></tr>
<tr><td>WD Fresh 1</td></tr>
<tr><td>WD Fresh 2</td></tr>
<tr><td>WD Fresh 3</td></tr>
<tr><td rowspan="7">SGS Mineral Services (RSA)</td><td rowspan="7">2016</td><td>Main Zone Transition 1</td><td rowspan="6">Comminution</td></tr>
<tr><td>Main Zone Fresh 5</td></tr>
<tr><td>East Zone Oxide 3</td></tr>
<tr><td>East Zone Transition 2</td></tr>
<tr><td>East Zone Fresh 5</td></tr>
<tr><td>WD Fresh 4</td></tr>
<tr><td>East Zone Fresh 4</td><td>Diagnostic Leach</td></tr>
</table>

Metallurgical Laboratory	Year	Sample	Test Work
		Main Zone Fresh 4	
		WD Fresh 1	
		WD Fresh 2	
		WD Fresh 3	
		East Zone Oxide 4	Heap Leach
		East Zone Oxide 5	
		Main Zone Oxide 1	
		Main Zone Fresh 4	Knelson CVD Gravity/Flash Flotation Amenability

13.1 ALS Metallurgy Pty Ltd 2014

13.1.1 Sample Selection and Head Assays

The samples from GSS detailed in Table 13-2 were submitted for metallurgical test work:

Table 13-2. ALS 2014 Metallurgical Sample Selection.

Sample	Type	Drill Holes	Total Interval (m)
Main Zone Fresh 1	Diamond Core	GSDD001/004	9
Main Zone Fresh 2	Diamond Core	GSDD002/005	9
East Zone Fresh 1	Diamond Core	GSDD007A	7
East Zone Fresh 2	Diamond Core	GSDD008	12
East Zone Oxide 1	RC Drill Chips	GSRC088/175/177	20
East Zone Transition 1	RC Drill Chips	GSRC108/175/176/177/178	9
GSS High Grade Zone 1	RC Drill Chips	GSRC081/082/161/183	14

Composites were prepared by Orca by combining RC drill chips and half drill core to provide samples of between 20 and 36 kg. The seven samples were subjected to complete head assay as shown in Table 13-3.

Table 13-3: Head Assays.

Sample	Orca Au g/t	ALS Head Assays							
		Au g/t	Ag g/t	As ppm	Cu ppm	Ctotal	Corganic	Stotal	Ssulphide
Main Zone Fresh 1	1.75	1.78	4.0	190	85	2.58	<0.03	2.84	2.56
Main Zone Fresh 2	1.74	1.65	4.0	150	105	2.07	<0.04	3.58	3.48
East Zone Fresh 1	1.78	1.38	2.0	40	80	2.37	<0.05	1.98	1.8
East Zone Fresh 2	1.70	1.54	2.1	30	85	1.56	<0.06	2.18	2.04
East Zone Oxide 1	1.68	1.30	1.2	60	110	0.51	<0.07	0.12	0.08
East Zone Transition 1	1.78	1.54	2.0	50	105	0.96	0.06	1.48	1.24
GSS High Grade Zone 1	27.31	21.30	24.0	650	355	0.87	0.06	2.26	2.04

13.1.2 Comminution results

The four fresh samples were subjected to preliminary comminution test work. The results are shown in Table 13-4 below:

Table 13-4: Bond Ball Mill Work Index test work results

Sample	Micrometres		Gbp (g/rev)	Bwi (kWh/t)	Bond Abrasion index
	F80	P80			
Main Zone Fresh 1	2,797	76	1.315	13.90	0.1061
Main Zone Fresh 2	2,796	75	1.770	10.90	0.0695
East Zone Fresh 1	2,848	80	1.363	14.00	0.1518
East Zone Fresh 2	2,824	73	0.913	18.50	0.2528

13.1.3 Gravity/Direct Cyanidation Test work

Gravity/direct cyanidation gold recovery test work was conducted on all seven composite samples in order to determine gold extraction characteristics. For all seven drill core and RC chip composite samples, the agitated vessel 48-hour direct cyanide leach tests were conducted at a P80: 75 µm grind size and at 40% solids using Perth tap water. Results are shown below in Table 13-5.

Table 13-5: Gravity/Direct Cyanidation test work results.

Sample	% Extraction @ hours						Au Grade (g/t)		Consumption (kg/t)	
	Gravity	2	4	8	24	48	Calc Head	Res	NaCN	Lime
Main Zone Fresh 1	11.77	73.79	75.90	77.55	78.75	78.75	1.74	0.37	1.02	0.32
Main Zone Fresh 2	14.81	73.59	74.87	76.55	77.78	77.78	1.71	0.38	0.81	0.33
East Zone Fresh 1	16.25	82.45	83.42	85.31	85.77	88.47	1.52	0.18	0.92	0.36
East Zone Fresh 2	10.18	77.46	81.57	86.48	86.92	86.92	1.61	0.21	0.89	0.31
East Zone Oxide 1	16.52	81.31	82.29	84.69	85.63	89.29	1.49	0.16	0.85	0.46
East Zone Trans 1	15.12	78.75	81.49	82.26	83.38	86.66	1.87	0.25	0.89	0.43
GSS High Grade 1	23.10	86.76	89.55	94.10	94.10	95.33	22.10	1.03	1.50	0.38

For all the samples tested the gold leaching kinetics appears to be relatively fast with the majority of the gold leaching in the first 8 hours. Similarly for all samples the lime and sodium cyanide reagent consumption rates are relatively low. Head values derived from the leach data compare well with the head assay values. For the Main Zone lithologies, the lower gold recoveries were thought to have been due to:

- presence of very fine gold that was not being liberated at a P80: 75 µm grind size;
- a portion of the gold may be refractory in nature and locked up in sulphide or other minerals; and
- solid solution gold present in the sulphide minerals.

Multistage diagnostic leach test work was recommended to determine the cause of the lower recoveries in the Main Zone Fresh samples

13.1.4 Multi-stage Diagnostic Leach

Sub-samples of the two Main Zone composite samples were submitted for gravity separation followed by amalgamation of the gravity concentrate and multi-stage, sequential diagnostic gold leach test work as detailed below:

- The sample was submitted for a direct leach to determine the distribution of free cyanidable gold.
- The leach residue was then subjected to 3M HCl digestion to destroy the carbonate minerals. The residue from hydrochloric acid digestion was subsequently cyanide leached to determine the quantity of gold released from the carbonate minerals.
- The residue from the direct leach was subjected to dilute nitric acid digestion to destroy the arsenical minerals such as arsenopyrite and arsenical pyrite. The residue from the nitric acid digestion stage was subsequently cyanide leached to determine the released gold content.
- The residue from the HNO3/cyanidation stage was subjected to aqua regia digestion to destroy all remaining sulphide minerals (mostly pyrite) and simultaneously release the contained gold into solution.
- The residue from the aqua regia digestion was fire assay smelted to determine the silicate (gangue) encapsulated gold content.

Results of this test work are shown in Table 13-6 below:

Table 13-6: Multistage Diagnostic Leach test work results.

Diagnostic Stage	Description	Main Zone Fresh 1		Main Zone Fresh 2	
		g/t	%	g/t	%
Amalgamation on Gravity	Free gravity Gold Content	0.22	13.03	0.43	21.65
Direct Leach	Cyanidable Gold Content	1.10	64.85	1.16	58.15
Dil. HCL Digest /Direct	Carbonate Locked Gold Content	0.03	1.80	0.06	2.94
Dil. HNO3 Digest/Direct	Arsenical Mineral locked Gold	0.32	18.89	0.32	16.08
Aqua Regia Digest	Pyritic Sulphide Mineral Locked	0.014	0.85	0.02	1.02
Total Fire Assay Smelt	Silicate (gangue) encapsulated gold	0.01	0.59	0.003	0.15
Total Gold Content		1.689	100	1.986	100
Head Assay		1.775		1.645	

For the Main Zone Fresh 1 Composite the average head assay and the diagnostic leach calculated head assay match well. A total of 77.88% of the gold was recovered at the selected P80 75 µm grind size, of which 13.03% was recoverable by gravity.

The diagnostic gold analysis also shows that 1.80% of the remaining gold is locked up in carbonate or reactive sulphide minerals, 18.89% of the gold is locked in arsenopyrite, 0.85% of the gold is locked in pyrite and 0.59% of the gold is encapsulated in silicates.

For the Main Zone Fresh 2 Composite the average head assay and the diagnostic leach calculated head assay match well. A total of 79.80% of the gold was recovered at the selected P80 75 µm grind size, of which 21.65% was recoverable by gravity. The diagnostic gold analysis also shows that 2.94% of the remaining gold is locked up in carbonate or reactive sulphide mineral, 16.08% of the gold is locked in arsenopyrite, 1.02% of the gold is locked in pyrite and 0.15% of the gold is encapsulated in silicates.

As the head assay data indicated a relatively low level of arsenic in both Main Zone Fresh 1 and 2 composite samples (190 ppm and 150 ppm As, respectively), the elevated percentage of locked gold in arsenopyrite falls into question. It is possible that the majority of the locked gold may not be in arsenopyrite mineral, but may be in a pyrite mineral. If the pyrite mineral present is in the form of a more reactive or less stable form of pyrite, then it can undergo digestion in the dilute nitric digest stage rather than the aqua regia digest stage. This would result in the elevated locked gold content being reported as "arsenopyrite locked gold", rather than "pyrite locked gold".

13.1.5 Qemscan Mineralogical Analysis

A 1,000 g sub-sample of the Main Zone Fresh 1 Composite was ground to P80: 75 µm prior to gravity separation via a Knelson separator. The resultant gravity concentrate and tail products were submitted for detailed QEMSCAN mineralogical analysis

Thirty-two gold grains hosted by 23 particles were detected during the Trace Mineral Search of one polished block of the gravity concentrate.

All of the gold grains are hosted by the pyrite; most are enclosed within the pyrite and the remainder are exposed at the surface of pyrite particles.

The maximum diameter of the gold grains typically ranges from 1 µm to 10 µm with the two largest grains detected measuring 23 x 5 µm and 12 x 1 µm.

The Ag content of the gold appears to be variable with the lowest measured Ag content 3% and the highest measured Ag content 29%. Gold with Ag less than 20% has been classified as argentian gold while the gold with Ag higher than 20% has been classified as electrum.

The resolution of the QEMSCAN method is about 1 µm. The pyrite may contain some "invisible" gold (i.e. <1 µm and solid solution type gold) and the deportment of such gold is unaccounted for.

The mineralogy report indicates that the majority of the locked gold is not in arsenopyrite mineral, but in a pyrite mineral. There are also indications that the pyrite mineral present is in the form of a more reactive or less stable form of pyrite, being more porous and containing between 1-2% Zn in solid solution.

If this is the case, then the pyrite may have undergone digestion in the dilute nitric digest stage rather than the aqua regia digest stage during the diagnostic analysis test work. This would result in the elevated locked gold content being reported as "arsenopyrite locked gold", rather than "pyrite locked gold".

For the fresh material, as most of the locked gold is relatively fine in size and the pyrite is well liberated at a P80: 75 µm grind size, it was recommended to examine a sulphide flotation (at P80: 75 µm) and subsequent pyrite concentrate ultra-fine grind/intensive leach process route with standard cyanidation on the flotation tail to improve overall gold recoveries.

13.2 SGS Mineral Services UK

13.2.1 Sample Selection and Head Assays

SGS Minerals Services UK conducted three phases of work for Orca Gold Inc. These projects were conducted under the numbers SGS510, SGS583, and SGS585.

Samples for GSS comprised quarter HQ and PQ core, samples from WD were made up of RC chips. Individual intervals were composited by SGS to form the following samples:

Table 13-7: SGS UK Samples for test work

SGS Project	Sample	Weight kg	Head Au g/t	Head Ag g/t
510	East Zone Fresh 3	45	1.76	3.20
	Main Zone Fresh 3	39	2.24	6.77
	East Zone Oxide 2	24	1.16	0.97
583	East Zone Fresh 4	63	1.59	3.62
	Main Zone Fresh 4	68	2.14	2.68
	East Zone Oxide 2	24	1.16	0.97
585	WD Fresh 1	23	1.53	5.29
	WD Fresh 2	26	1.99	17.09
	WD Fresh 3	30	7.63	23.44

13.2.2 Oxide Coarse Ore Leach Tests

Coarse material bottle roll tests were conducted on splits of the East Zone Oxide 2 sample at crush sizes of 100% passing of 25 12, 6, 3.5 and 1.18 mm. The tests were conducted in duplicate on 2 kg material samples to mitigate the errors introduced from sample splitting at the coarse crush sizes tested. The individual 2 kg samples were placed in to glass bottles and pulped to 40% solids by mass using tap water and the pH was adjusted to between 10.5 and 11 using lime. Cyanide was added and maintained at 0.5 g/l CN or around 1 g/l NaCN. The bottles were placed on to the roller deck and rolled for 1min for every hour of leach time. The leach time for the tests was 14 days. To benchmark the results against what should be achievable using a tank leach approach a bottle roll test was conducted at a grind size of 100% passing 75 µm. Results are shown in Table 13-8: SGS UK Coarse Ore Bottle Roll Results below and in Figure 13.1.

Table 13-8: SGS UK Coarse Ore Bottle Roll Results

Leach Time	25 mm	12 mm	6 mm	3.35 mm	1.18 mm	75 µm
	Au	Au	Au	Au	Au	Au
2	10.58	12.70	12.77	14.74	22.73	84.14
4	21.40	28.60	33.58	40.17	49.90	85.19
6	25.99	33.54	37.70	44.58	53.01	86.29
24	39.15	49.69	51.10	56.45	64.22	90.62
48	45.44	55.66	56.91	61.10	67.69	**91.68**
96	51.83	62.52	61.91	65.75	71.21	
168	57.58	67.98	69.47	69.75	74.78	
240	58.49	69.05	70.56	70.83	73.22	
336	**62.91**	**73.01**	**73.05**	**72.63**	**79.65**	
Residue	37.09	26.99	26.95	27.37	20.35	8.32
Back Calc Head g/t	1.06	1.01	1.06	1.02	1.02	0.96
Reagent Addition (kg/t)	1.68	1.77	1.56	1.62	1.73	1.68
Reagent Consumption (kg/t)	0.43	0.52	0.15	0.28	0.53	0.19

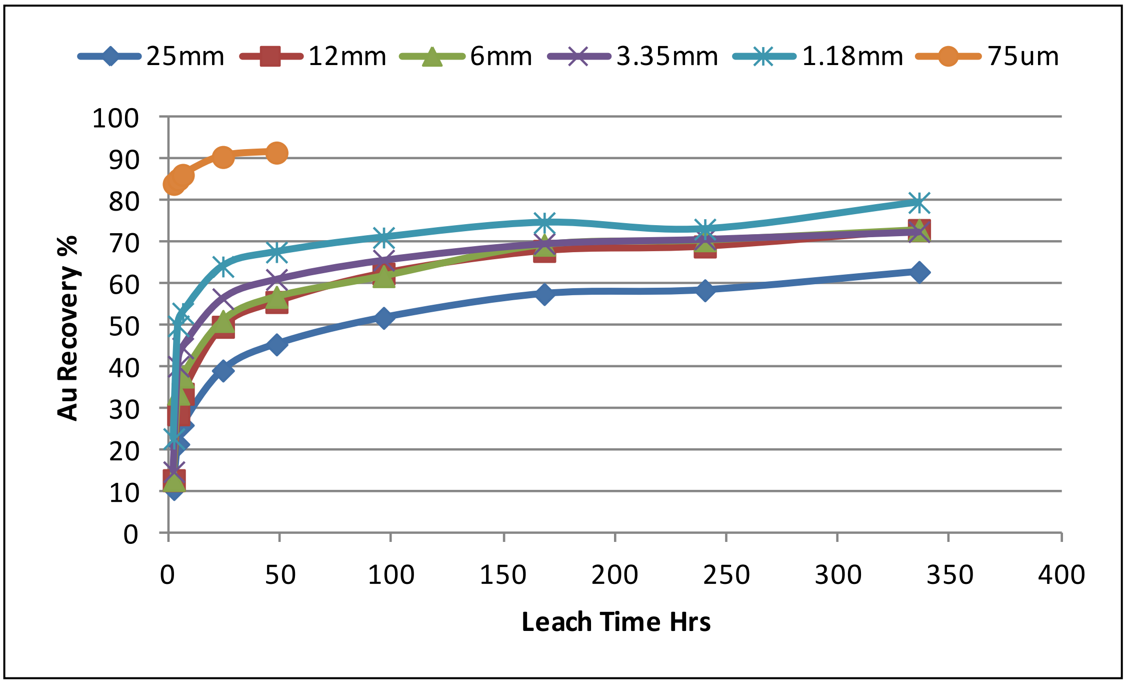


Figure 13-1: Kinetics of Cyanide Extraction of Gold for Coarse and Fine Ore Tests Performed on East Zone Oxide Sample.

13.2.3 Direct Cyanide Leaching

The East Zone Fresh 3 and Main Zone Fresh 3 samples were submitted to a series of fine material cyanide leach tests conducted at a range of grind sizes. The grind sizes tested were -150, -106, -75, and -53 µm. The sizes indicate the 100% passing size.

Tests were conducted on 1 kg samples stage ground to the target sizes. The samples were placed in to 5 litre glass bottles and pulped to 45% solids and adjusted to pH 10.5 to 11. Cyanide was added to reach a strength of 0.5 g/l CN which is around 1 g/l NaCN. This cyanide content was maintained through the leach with the levels monitored using silver nitrate titration. The leach tests ran for 48 hours. Results are shown in Table 13-9, Figure 13-2 and Figure 13-3.

Table 13-9: SGS UK Direct Cyanidation Tests

Hours	East Zone Fresh 3				Main Zone Fresh 3			
	150µm	**106µm**	**73µm**	**53µm**	**150µm**	**106µm**	**73µm**	**53µm**
2	58.09	62.93	57.20	61.66	52.69	59.30	63.49	62.24
4	67.52	73.01	69.80	73.49	60.24	66.79	72.33	69.39
6	68.50	75.32	73.76	76.58	62.89	68.23	74.68	70.90
24	71.58	76.80	77.74	80.00	66.99	71.09	77.06	75.32
48	**72.82**	**77.36**	**78.46**	**82.68**	**69.61**	**73.20**	**79.36**	**77.52**
Residue	27.18	22.64	21.54	17.32	30.39	26.80	20.64	22.48
Back Calc Head g/t	1.66	1.72	1.81	1.56	2.19	2.24	2.20	2.09
NaCN Addition (kg/t)	1.71	1.65	1.71	1.63	1.50	1.50	1.48	1.63
NaCN Consumption (kg/t)	0.26	0.37	0.24	0.21	0.13	0.14	0.28	0.25
CaO Addition (kg/t)	0.45	0.37	0.44	0.43	0.37	0.39	0.48	0.75
CaO Consumption (kg/t)	0.45	0.37	0.44	0.43	0.37	0.39	0.48	0.75

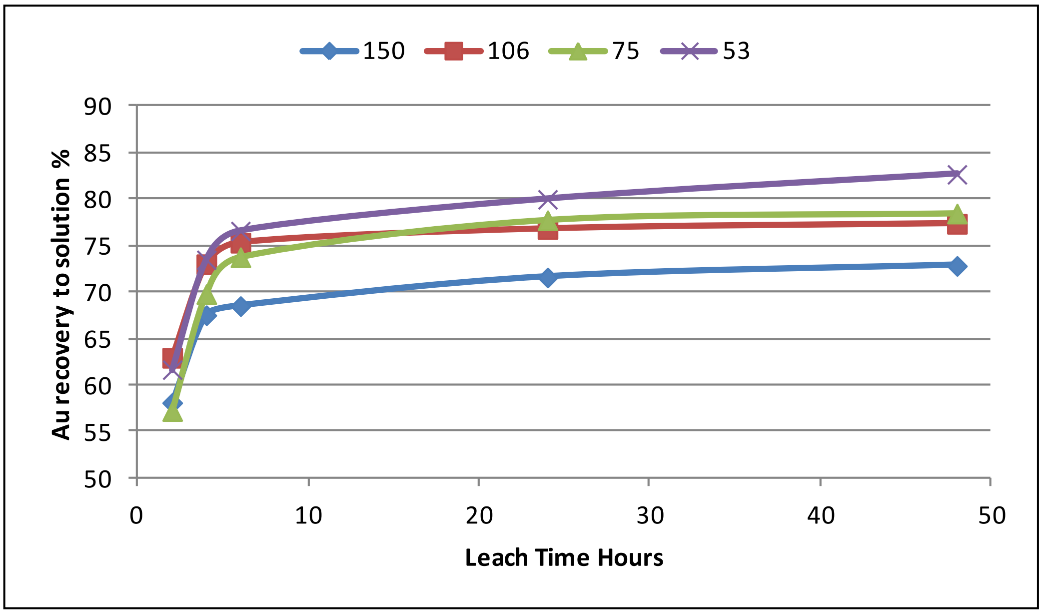


Figure 13-2: Kinetics of Cyanide Extraction of Gold for East Zone Fresh 3 Sample.

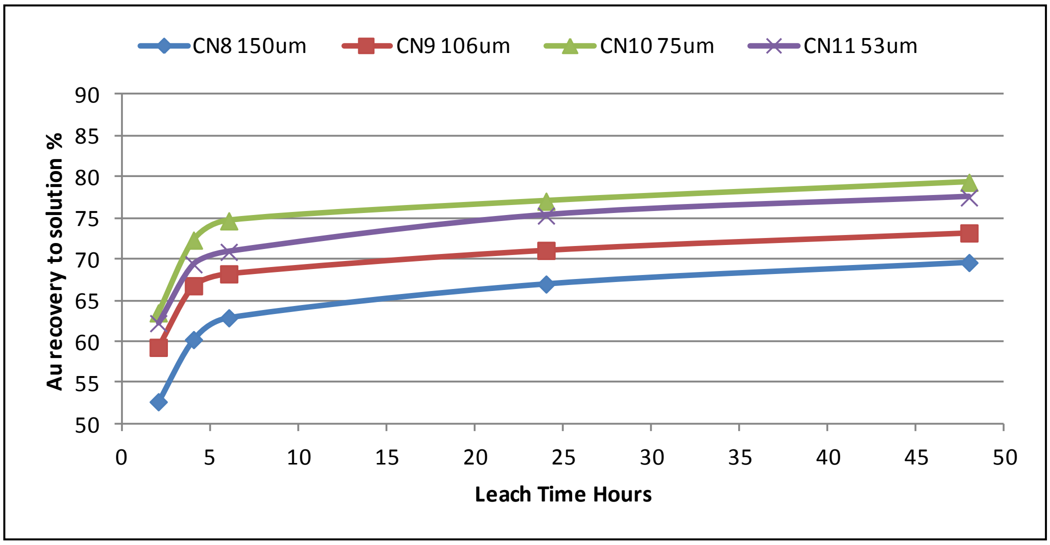


Figure 13-3: Kinetics of Cyanide Extraction of Gold for Main Zone Fresh 3 Sample.

Fine material leach tests were also conducted on the three samples from the WD deposit. The samples were stage ground to pass 75 µm for the tests. Results are shown in Table 13-10 below.

Table 13-10: SGS UK WD Direct Cyanidation Tests

Hours	WD Fresh 1	WD Fresh 2	WD Fresh 3
2	45.71	61.32	78.52
4	59.22	68.38	83.87
6	64.87	72.21	90.49
24	73.41	77.36	94.69
48	78.09	77.16	93.02
Residue	21.91	22.84	6.98
Back Calc Head g/t	1.53	1.99	7.63
NaCN Addition (kg/t)	2.09	1.84	1.83
NaCN Consumption (kg/t)	0.97	0.66	0.77
CaO Addition (kg/t)	0.31	0.23	0.29
CaO Consumption (kg/t)	0.31	0.23	0.29

13.2.4 Carbon in Leach tests

Carbon in leach bottle roll tests were completed on the East Zone Fresh 3 and the Main Zone Fresh 3 samples. The object of the tests was to check for any pregnant leach of gold or silver back into the sample from the solution caused by naturally occurring carbon. The results of the tests are shown in Table 13-11 below.

The total gold extraction is calculated from the gold absorbed onto the carbon plus any gold remaining in solution at the end of the test. The total gold recoveries for each test are comparable to those seen in the direct cyanide leach tests. This indicates that there is no pregnant leach robbing components in the material causing significant impact to the gold metallurgy.

Table 13-11: SGS UK Carbon in Leach tests

Hours	East Zone Fresh 3			Main Zone Fresh 3		
	106µm	75µm	53µm	106µm	75µm	53µm
48	1.72	2.45	1.70	0.70	0.69	0.67
Carbon	76.94	80.17	77.10	73.19	79.52	79.77
Total	78.65	72.62	78.80	73.88	80.21	80.44
Residue	21.35	27.38	21.20	26.12	19.79	19.56
Back Calc Head g/t	1.78	1.86	1.79	2.20	2.22	2.27
NaCN Addition (kg/t)	1.88	1.84	1.81	1.60	1.70	1.57
NaCN Consumption (kg/t)	0.69	0.66	0.63	0.43	0.54	0.56
CaO Addition (kg/t)	0.26	0.22	0.32	0.47	0.53	0.52
CaO Consumption (kg/t)	0.27	0.22	0.33	0.45	0.53	0.52

The results showed that similar gold and silver recoveries were obtained when compared to the direct leach results. This indicates that there is no pregnant leach robbing components in this material type.

13.2.5 Gravity Gold Recovery and Tailings Leach

1 kg sub-samples from the Main Zone Fresh 1 and East Zone Fresh 1 composites, ground to 100% -75 µm were passed through a Falcon SB40 unit. The concentrate was then cleaned over a Mozley table with the concentrate from this submitted for analysis. The Mozley table tailings and the Falcon tailings were then recombined and submitted for cyanide leach testing.

Table 13-12: SGS UK Carbon in Leach tests

Product	East Zone Fresh 3	Main Zone Fresh 3
Gravity Concentrate	15.12%	27.87%
Tailings Leach Hours		
2	63.69	50.14
4	66.67	52.98
6	67.25	52.36
24	69.32	55.37
48	68.32	53.97
Residue	16.56	18.16
Back Calc Head g/t	1.52	1.98

During the Mozley panning of the Falcon concentrates no free gold was observed on the table and the concentrate consisted of predominantly sulphide material likely pyrite.

Overall gold recovery was 84.4% for the East Zone Fresh 1 sample and 81.8% for the Main Zone Fresh 1 sample. Although the recovery of gold overall is higher than achieved by the direct leach approach it should be considered that the gravity concentrate was not leached. It is likely that, when accounting for the efficiency of leaching this highly sulphidic product, that the overall gold recovery would be very similar to that seen from the direct leach approach.

The main point noted from the testing was that gold recovery appeared linked to sulphide recovery. Bulk sulphide flotation was therefore recommended for investigation as a means to produce a low mass gold concentrate for leaching.

13.2.6 Flotation Test Work

Initially a series of mesh of grind flotation tests were performed on the East Zone Fresh 4 and Main Zone Fresh 4 samples to select a primary grind size to use for further testing.

The reagents used and the dosages along with the final mass pull and metal recoveries to the flotation concentrate are shown in Table 13-14.

The results show that all tests gave high gold recoveries to concentrate at around 93% to 94% at grind sizes below 125 µm for both samples. The sulphide recovery is similarly high at over 97% for the East Zone and over 99% for the Main Zone Fresh samples. Targeting a high sulphur recovery was a key objective due to the intimate association of the gold and silver with the sulphide minerals.

A primary grind size of 125 µm was chosen for further test work aimed at studying reagent regime using a float time of 15 min.

Table 13-13: SGS UK mesh of grind flotation tests

Sample	Grind Size µm	Float Time Mins	Aero 208 g/t	Pax g/t	MIBC g/t	pH	Sodium Carbonate g/t	Mass Pull %	S Recovery %	Au Recovery %
East Zone Fresh 4	180	15	50	100	32	10	480	14.79	97.79	94.09
	150	15	50	100	32	10	600	13.95	97.60	93.68
	125	15	50	100	32	10	580	12.72	97.71	94.06
	106	15	50	100	32	10	640	12.04	97.73	92.81
	75	15	50	100	32	10	680	12.65	98.01	94.48
Main Zone Fresh 4	180	15	50	100	32	10	480	14.79	98.05	90.09
	150	15	50	100	32	10	600	13.95	98.99	91.65
	125	15	50	100	32	10	580	12.72	99.26	93.00
	106	15	50	100	32	10	640	12.04	99.24	92.70
	75	15	50	100	32	10	680	12.65	99.48	94.32

Table 13-14: SGS UK Flotation Rougher Reagent test results

Sample	CuSO4 g/t	Aero 407 g/t	Aero 208 g/t	Pax g/t	MIBC g/t	pH	Sodium Carbonate g/t	Mass Pull %	S Recovery %	Au Recovery %
East Zone Fresh 4			20	50	32	9.5	As Required	13.73	98.12	95.61
	50		20	50	32	natural	As Required	15.87	97.49	92.07
	50	20	20	50	32	natural	As Required	13.38	97.93	92.74
	50	20	20	50	32	natural	As Required	13.45	97.36	92.78
	50	20	20	50	32	natural	As Required	14.63	98.14	94.39
			20	100	32	natural	As Required	15.81	98.17	94.85
			20	200	32	natural	As Required	16.04	97.35	93.33
			20	100	32	natural	As Required	15.51	97.72	94.32
			20	50	32	natural	As Required	15.17	97.49	93.59
			20	25	32	natural	As Required	14.21	95.65	91.20
				50	32	natural	As Required	14.99	97.42	93.25
Main Zone Fresh 4			20	50	32	9.5	As Required	15.06	98.04	90.49
	50		20	50	32	natural	As Required	15.45	97.35	92.50
	50	20	20	50	32	natural	As Required	14.48	98.03	90.42
	50	20	20	50	32	natural	As Required	14.84	97.70	91.49
	50	20	20	50	32	natural	As Required	15.61	95.66	88.12
			20	100	32	natural	As Required	18.30	98.08	89.60
			20	200	32	natural	As Required	16.30	98.56	92.27
			20	100	32	natural	As Required	17.51	97.61	89.37
			20	50	32	natural	As Required	17.15	96.80	90.48
			20	25	32	natural	As Required	16.33	98.31	88.88
				50	32	natural	As Required	16.10	98.54	89.07

The results for the tests were largely similar when considering the possible variances between the tests introduced from minor mineralogical variance from sample splitting, variance in concentrate pulling rate by the test operator, and assay variance.

The results showed that flotation could be performed at natural pH which would save cost on pH regulation in the final plant. Results also showed that there was no significant effect of using specialised collectors for the recovery of gold (MX900) or was any benefit seen using a collector useful for collecting tarnished gold (Aero 407). The addition of copper sulphate did not assist in the collection of sulphides which may have assisted co-collection of gold and silver through mineralogical association. Testing Showed that satisfactory results could be obtained using a simple flotation regime of Aero 208 and PAX and hinted that PAX by itself could be sufficient.

Cleaner flotation tests were then run using a primary grind of -125µm and a rougher flotation regime of natural pH, 20 g/t Aero 208, and 50 g/t of PAX. Two tests were run on each of the composite samples with one of each following re-grinding of the rougher concentrate to -25um. The conditions for the tests and results are shown below in Table 13-15.

Table 13-15: SGS UK Cleaner Flotation test results

Sample	Regrind time mins	Cleaner Float time Mins	PAX g/t	Mass Pull % Ro/Cl		S Recovery % Ro/Cl		Au Recovery % Ro/Cl	
East Zone Fresh 4		10.5		15.02	9.22	97.23	95.90	94.23	88.81
	25	10.5		13.85	7.19	97.84	73.39	94.63	86.28
		10.5	20	14.48	9.47	98.11	97.05	94.69	91.03
	25	10.5	20	13.59	7.97	97.37	85.14	93.32	88.14
	60	45.5	110	13.92	10.09	97.41	96.92	93.18	92.37
Main Zone Fresh 4		10.5		15.75	9.59	93.60	92.49	90.13	97.18
	25	10.5		15.24	8.84	98.79	82.47	92.38	88.93
		10.5	20	14.75	9.76	98.79	98.03	91.39	89.79
	25	10.5	20	14.37	8.28	98.29	82.02	91.86	88.57
	60	45.5	110	14.92	10.19	97.73	97.56	89.86	89.54

Results show that re-dosing with PAX prior to the cleaning stage assisted in gold recovery to the cleaner concentrate increasing recovery by around 1.5 to 2%. The results show that regrinding the concentrate decreased the gold recovery to the concentrate but also decreased the mass recovery to the concentrate indicating that the kinetics of gold recovery in respect to float time was decreased or slowed.

The East Zone Oxide 2 composite from GSS was also submitted for flotation test work with poor results as shown in table 13-16.

Additional tests were conducted to try to recover the oxidised sulphide minerals present. The results show that gold recovery to the flotation concentrate remained poor.

Table 13-16: SGS UK Cleaner Flotation test results

Grind Size µm	Float Time Mins	NaSH g/t	Aero 208 g/t	Pax g/t	MIBC g/t	pH	Sodium Carbonate g/t	Mass Pull %	S Recovery %	Au Recovery %
125	15	500	50	200	32	10	200	7.34	19.73	53.05
106	15	500	50	200	32	10	200	4.94	18.72	47.41
90	15	500	50	200	32	10	200	5.65	17.78	58.15
75	15	500	50	200	32	10	200	5.32	28.72	56.15
65	15	500	50	200	32	10	200	6.22	17.29	56.63

The three fresh samples from WD were also tested using flotation using a similar regime to that used on the GSS samples. Results are shown in Table 13-17 below:

Table 13-17: SGS UK WD Flotation test results

Sample	Grind Size µm	Float Time Mins	Aero 208 g/t	Pax g/t	pH	Mass Pull %	S Recovery %	Au Recovery %
WD Fresh 1	150	15	50	200	10	10.95	97.00	91.32
	106	15	50	200	10	12.80	99.10	93.75
	75	15	50	200	10	13.45	97.59	95.95
WD Fresh 2	150	15	50	200	10	31.79	98.26	97.37
	106	15	50	200	10	28.72	96.50	97.37
	75	15	50	200	10	31.90	98.11	97.47
WD Fresh 3	150	15	50	200	10	23.59	97.68	94.19
	106	15	50	200	10	24.47	98.66	95.37
	75	15	50	200	10	28.82	98.57	94.79

13.2.7 Fine Grinding of Flotation Concentrates and Intensive Leaching

A series of rougher/cleaner flotation tests was conducted on the Main Zone Fresh 4 and East Zone Fresh 4 samples to provide a concentrate for intensive cyanide leach testing. The flotation tests were conducted using a standard regime. The results show that a flotation cleaner concentrate provided a mass pull of between 6 and 7% for both samples and at best recovering 88.7% of the gold for the East Zone and 94.8% for the Main Zone Fresh samples.

The cleaner flotation concentrates were submitted for intensive cyanidation testing. The tests were conducted at 15% solids by weight and 10 g/l CN at pH 10.5 to 11. The cleaner flotation concentrate was stage ground to pass 20 µm prior to leach testing. To assist the leach air was sparged into the pulp during the test, recorded dissolved oxygen values shows that the levels were between 8 and 10 mg/l during the leach tests.

The results show that overall recovery of gold to the cyanide solution reached 78% for the East Zone Fresh 4 sample and 83% for the Main Zone Fresh 4 sample.

The Rougher concentrates from the flotation of the WD Fresh samples were also subjected to the same fine grind and intensive leach returning leach results of between 63% and 90%.

13.3 SGS Mineral Services South Africa

13.3.1 Comminution Test Work

Six representative composite samples were submitted to SGS South Africa for comminution test work. For each sample, the following comminution tests were carried out:

- Bond Ball Mill Work index (BBWi) test;
- Bond Rod Mill Work Index (BRWi) test;
- SAG Mill Comminution (SMC) test; and
- Bond Abrasion Index (Ai) test.

The BBWi, BRWi, Ai and SMC Tests were performed at SGS South Africa. The SMC data generated from the tests was sent to JKTech in Australia for analysis.

13.3.1.1 Bond Ball Mill Work Index

The Bond Ball Mill Work index results for the ORCA Gold samples show work indexes varying between 8.7 kWh/t to 12.0 kWh/t, and from soft to medium hardness material (Table 13-18). When studying the results based on the different zones it can be observed that there is a decrease in the work index between the fresh samples and the samples from more oxidized zones.

For the Main Zone samples, the work index decrease from 10.1 kWh/t (fresh) to 8.9 kWh/t (transition) while in East Zone samples the work index decrease from 12.0 kWh/t for the fresh material, 10.7 kWh/t for the transition material to 8.7 kWh/t for the oxide material.

Table 13-18: SGS RSA Bond Ball Mill Index Results

Sample ID	Limiting Screen Size µm	Net Production g/rev	F_{80} µm	P_{80} µm	Circulation Load %	Work Index kWh/t
Main - Fresh (MZ_FR5)	106	1.98	2 507	76	245	10.1
Main - Transition (MZ_TR1)	106	2.34	2 353	76	246	8.9
East - Fresh (EZ_FR5)	106	1.45	2 649	79	253	12.0
East - Transition (EZ_TR2)	106	1.88	2 454	78	252	10.7
East - Oxide (EZ_OX3)	106	2.38	2 141	74	252	8.7
WD - Fresh (WD_FR4)	106	1.79	2 601	72	246	10.6

13.3.1.2 Bond Rod Mill Work Index

The Bond Rod Mill Work index results for the ORCA Gold samples show work indexes varying between 8.0 and 13.1 kWh/t, and from soft to medium hardness material (Table 13-19). When studying the results based on the different zones it can be observed that, for the Main zone, the work index is similar between the fresh and the transitional samples, unlike the BBWi results. However, for the East Zone samples, a similar observation to the BBWi results can be made with a decrease in the work index from 13.1 kWh/t for the fresh material, to 10.1 kWh/t for the transitional material and 8.0 kWh/t for the Oxide material.

Table 13-19: SGS RSA Bond Rod Mill Index Results

Sample ID	Limiting Screen Size µm	Net Production g/rev	F_{80} µm	P_{80} µm	Circulation Load %	Work Index kWh/t
Main - Fresh (MZ_FR5)	1,180	13.07	11 329	837	98	10.7
Main - Transition (MZ_TR1)	1,180	13.25	10 681	858	102	10.9
East - Fresh (EZ_FR5)	1,180	9.84	11 160	862	99	13.1
East - Transition (EZ_TR2)	1,180	13.69	11 616	812	100	10.1
East - Oxide (EZ_OX3)	1,180	20.85	10 242	820	101	8.0
WD - Fresh (WD_FR4)	1,180	11.04	11 240	822	99	11.8

13.3.1.3 Bond Abrasion Index

Table 13-20 presents the Ai test results for the six composite samples. From the table values, 5 out of 6 samples are slightly abrasive and 1 sample (WD – Fresh) is considered as non-abrasive.

When analysing the results based on the different zones, it can be observed that, for the Main Zone samples, the abrasiveness does not vary significantly between the fresh and transition sample. For the East Zone samples, the abrasiveness is similar between the Fresh and transition sample (Ai around 0.38) but decreases to Ai = 0.1062 with the oxide sample.

Table 13-20: SGS RSA Bond Abrasion

SPECIMEN PARTICULARS		SPECIMEN TEST RESULTS				
Specimen No	Client	Test Paddle	Paddle Mass	Paddle Mass	Abrasion index	Life Factor
6501-	ID	No.	Before Test (g)	After Test (g)	Ai (g)	LF
PAT-01	Main - Fresh (MZ_FR5)	1	92.7570	92.5887	0.1683	2.41
			92.5887	92.4371	0.1516	2.58
PAT-02	Main - Transition (MZ_TR1)	2	92.9811	92.8196	0.1615	2.48
			92.8196	92.6573	0.1623	2.47
PAT-03	East - Fresh (EZ_FR5)	3	93.0111	92.6256	0.3855	1.28
			92.6256	92.2924	0.3332	1.45
PAT-04	East - Transition (EZ_TR2)	4	93.0091	92.6247	0.3844	1.28
			92.6247	92.3114	0.3133	1.52
PAT-05	East - Oxide (EZ_OX3)	5	93.0027	92.9078	0.0949	3.47
			92.9078	92.8016	0.1062	3.24
PAT-06	WD - Fresh (WD_FR4)	6	93.1399	93.0604	0.0795	3.85
			93.0604	92.9841	0.0763	3.94

13.3.1.4 SAG Mill Comminution

The samples tested, when compared to other material in JKTech's SMC database of over 35,000 test results, appears to be of medium hardness with the exception of the East Zone Oxide sample which can be classified as soft. A detailed elaboration and application on the parameters listed in Table 13-21 can be found in the SMC report, included in "Comminution Test Work Report on Six Samples from Orca Gold Inc, Proposal 15/835 rev 1. Dated 6 April 2016. SGS Mineral Services South Africa".

Table 13-21: SGS RSA SAG Mill Comminution

Sample ID	DWi kWh/m_3	DWi %	Mia kWh/t	Mih kWh/t	Mic kWh/t	A	b	SG	t_a	SCSE* kWh/t
Main - Fresh (MZ_FR5)	4.83	27	15.3	10.6	5.5	54.7	1.00	2.65	0.54	8.54
Main - Transition (MZ_TR1)	3.45	14	13.1	8.5	4.4	59.2	1.15	2.36	0.75	7.89
East - Fresh (EZ_FR5)	5.97	42	17.8	12.9	6.6	62.1	0.73	2.69	0.43	9.31
East - Transition (EZ_TR2)	3.69	16	13.2	8.7	4.5	63.9	1.04	2.46	0.70	7.88
East - Oxide (EZ_OX3)	2.04	5	8.4	4.9	2.5	60.2	1.98	2.43	1.27	6.55
WD - Fresh (WD_FR4)	5.38	34	16.2	11.5	5.9	56.9	0.89	2.71	0.48	8.90

13.3.1.5 Conclusions

Table 13-22 summarises the results of the various comminution test work:

Table 13-22: SGS RSA Comminution Summary

Sample ID			Work Index			
			SCSE kWh/t	BRWi kWh/t	BBWi kWh/t	Ai (average)
Main	Fresh	(MZ_FR5)	8.54	10.7	10.1	0.16
	Transition	(MZ_TR1)	7.89	10.9	8.90	0.16
East	Fresh	(EZ_FR5)	9.31	13.1	12.0	0.36
	Transition	(EZ_TR2)	7.88	10.1	10.7	0.35
	Oxide	(EZ_OX3)	6.55	8.0	8.70	0.10
WD	Fresh	(WD_FR4)	8.90	11.8	10.6	0.08

	Medium Hardness/Slightly abrasive
	Soft Hardness /Non abrasive

The results above indicate that the material from GSS and WD show generally below-average requirements in terms of comminution power (SMC, BBWi & BRWi) and media consumption (Ai).

These results will generally translate into higher throughput at lower operational costs on the comminution circuit. For the 3 zones studied (respectively Main, East and WD) the following observations were made:

- **Main Zone**

A decrease in the material's hardness can be observed for the fine grinding, with a lower BBWi observed for the transition material (8.9 kWh/t) compared to the fresh material (10.1 kWh/t).

- **East Zone**

A gradual decrease in the material hardness can be observed between the fresh, transitional and oxidized samples. This decrease is observed (Table 13-20) in coarse impact breakage and grinding (measured with the SCSE and BRWi) as well as fine grinding (BBWi). Additionally, a drop in the material abrasiveness is recorded for the East – Oxide sample.

- **WD**

It can be noted that, despite the material's hardness classification as medium, the WD sample is the least abrasive; with an Ai value classifying it as non-abrasive.

13.3.2 Interpretation of Comminution Results

It was requested that SGS design four grinding circuit configurations for the Project. The objective of the study was to develop grinding circuits capable of treating 2.0 million dry metric tons per annum (Mtpa).

A total of six samples were tested by SGS South Africa, see section 13.3.1, and the results were forwarded to SGS Canada Inc. for the completion of the grinding circuit designs.

Four grinding circuit options were identified, all following a primary stage of gyratory crushing:

- Semi-autogenous grinding (SAG mill) operated in closed circuit followed by a ball mill circuit (SAB configuration);
- Two-stage crushing followed by rod-ball milling,
- Two-stage crushing followed by two-stage ball milling; and
- Two-stage crushing followed by single-stage ball milling.

Each grinding circuit was simulated for two downstream process flowsheets, for a total of eight scenarios:

- Material leach (Straight CIL), and
- Flotation at a coarser grind followed by regrinding and cyanidation.

For the purpose of this PEA the leach results are summarised below, due to the fact that the suggested recovery method is based on a single stage ball mill straight CIL flowsheet, as per section 17. Reference can be made to "Grinding circuit designs based on the small-scale data for the Orca Gold Project Report CAQC-15688-001. Dated 2 August 2016. SGS Canada Inc.", for the full report and simulation results.

The starting point for all scenarios was the gyratory crusher product, and P80 was estimated at 125 mm, based on the A x b values, see

Figure 13-4. The circuit availabilities were assumed by SGS at 90% for all milling scenarios, and 66% for the crushing circuits.

The design criteria is summarised in Table 13-23.

Table 13-23: Grinding Simulation Design Criteria

Flow Sheet	Throughput		Mill Availability (%)	F_{80} (mm)	P_{80} (µm)	Mass Pull		Regrind (µm)
	(Mtpa)	(t/h)				(%)	(t/h)	
Straight CIL	2	254	90	125	75			
Flotation CIL	2	254	90	125	125	15	38	25

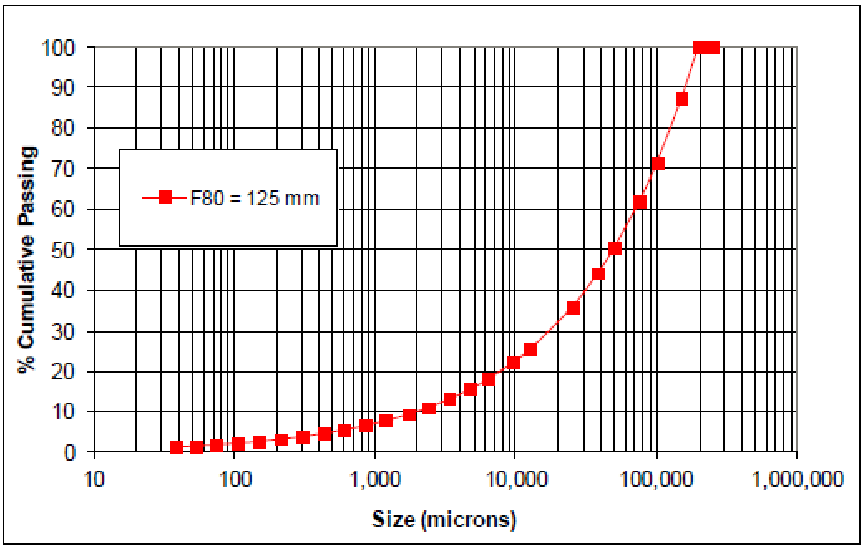


Figure 13-4: Estimated Primary Crusher Product Size Distribution

An upper limit on the maximum throughput rate of 25% was set by SGS, meaning that the maximum throughput rate for the two scenarios was 318 t/h. Similarly, the maximum final grind size was allowed to reach maximums of 85 µm and 135 µm for the Straight CIL and Flotation/CIL flowsheet options, respectively.

13.3.2.1 Single-stage Ball –Straight CIL Flowsheet Simulation

Eight simulations (SIM-71 to SIM-78) were performed for the scoping phase investigating the effect of feed size on ball mill dimension and overall power requirement. In each case, the ball mill power requirement was calculated for a P_{80} of 75 µm.

SIM-76 was selected as base-case scenario with a F_{80} of 8.0 mm. A single 16.0' x 28.5' low aspect ratio ball mill was required to achieve 2.0 Mtpa. A low aspect ratio ball mill was selected to allow more residence time in the mill and control recycle; this is desirable for single-stage ball milling operation that are required to grind coarse rocks to a fine grind. The ball mill should have internal dimensions of 4.72 x 8.53 m. The ball mill should be fitted with a 5,200 HP (3.9 MW) motor power.

Variability simulations were performed in simulations SIM-79 to SIM-84, using the selected mill. The ball mill power was fixed at the design power (3,156 kW) while the throughput rate and final grind were allowed to vary around the average. For SIM-83, the maximum throughput rate of 25% above the target (318 t/h) was met, and for SIM-81 the final P_{80} was maximized at 85 µm by reducing the tonnage.

The circuit P_{80} versus throughput rate data points are depicted in Figure 13-5. The recommended comminution equipment is summarized in Table 13-24.

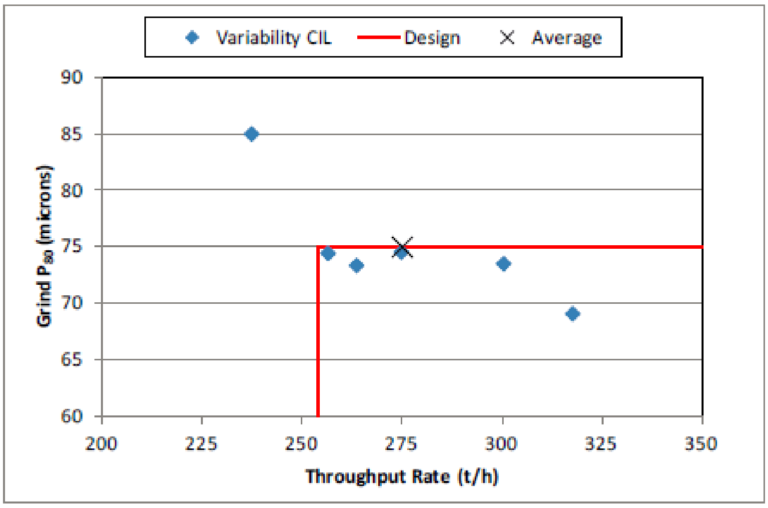


Figure 13-5: Throughput vs. Grind Relationship

Table 13-24: Single Stage BM - CIL Circuit Comminution Equipment Details

Description	Crusher[(a)]	Ball Mill	Total
Number in Parallel	2 stages	1	-
Mill Dimensions			
Nominal Dimension	-	16.0' x 28.5'	-
Inside Liner Dimension (metre)	-	4.72 x 8.53	-
% of Critical Speed (%)	-	75	-
Mill Speed (RPM)		7.9	-
Cone Angle (degree)	-	10 x 10	-
Grinding Steel			
Diameter Recommended (mm)	-	57	-
Design Ball Charge (%Vol.)	-	32	-
Maximum Ball Charge (% Vol.)	-	40	-
Motor			
Total Designed Power (kW)	359	3,156	3,515
Installed Power per mill (kW)	895	3,893	4,788
Installed Power per mill (HP)	1,200	5,200	6,400
Power Utilisation (%)	40	81	73
Classification			
Type	10-mm Screen	Hydro Cyclones	-
Circuit Performance			
F_{80} (µm)	125,000	8,000	125,000
P_{80} (µm)	8,000	75	75
Throughput Rate (t/h)	346	254	254
Availability (%)	66	90	
Specific Power Req. (kWh/t)	1.0	12.4	13.9

(a) Excludes Primary Crusher

13.3.3 Gold Deportment and Diagnostic Leach Test Work

Three feed samples from the GSS and WD deposits in Sudan were submitted for gold deportment test work. Feed samples were of fresh rock from the GSS Main Zone (MZ_FR4), one from the GSS East Zone (EZ_FR4) and the final from WD (WD_FR_COMP). In all three cases the samples provided were pulp splits from samples submitted for earlier metallurgical testing programs. The objective of the test work was to determine the leaching characteristics of the samples and to examine the nature of the refractory gold in the context of two processing options, firstly material leach and secondly rougher flotation, regrind and cyanidation.

13.3.3.1 Rapid Mineralogical Analysis

The results of the rapid mineralogical analysis indicated that pyrite exhibits a broad size range in all three samples. Consequently, the following three grind sizes were selected for the effect of grind tests: 80% passing 125 µm, 80% passing 75 µm and 80% passing 25 µm.

13.3.3.2 Effect of Grind Tests

Each of the three samples showed an improvement in gold recovery with finer grinding (16% for Main; 12% for East; 7% for WD, Table 13-25). At a coarse grind of 80% -125 µm, the WD sample yielded the highest dissolution at ~85% in comparison to ~74% and ~72% for the Main and East samples respectively. This suggests that the WD sample contains a higher proportion of liberated gold. At an intermediate grind size of 80% -75 µm, only marginal improvements in gold recovery were observed for the East and WD samples (<4%). This may signal a deficiency of medium-sized gold in these two samples. All three samples show relatively high gold dissolution at 80% -25 µm (~5 to 7%), which implies a significant proportion of fine-grained gold.

Acid digestion by nitric acid revealed that the majority of the refractory gold occurs in acid soluble minerals (80 to 91%) and that only a very small percentage of the total gold is locked within gangue/silicates (~2 to 3%). On this basis, there is upward potential to increase total recoveries beyond the ROM CIL dissolution benchmarks of 77 to 86%.

13.3.3.3 Head Chemical Analysis

The chemical compositions of the three feed samples were determined by a range of head assay methods. Head gold assays were reported as 2.1 g/t for Main, 1.3 to 1.8 g/t for East and 4.5 to 5.3 g/t for WD. Silver grades for the Main and East samples are comparable (~2 and 3 g/t, respectively), whereas a higher grade was observed for the WD sample (~13 g/t). Base metal concentrations are low for the Main and East samples, whereas moderate Zn and Pb concentrations were observed for the WD sample (0.2 to 0.4%). The results indicate the presence of silicate, oxide, carbonate and sulphur-bearing minerals in the three samples (e.g., quartz, mica, feldspar and sulphide minerals).

13.3.3.4 Heavy Liquid Separation

The objective of the HLS analyses was to produce a concentrate for mineralogical analyses. The analyses also draw attention to the amenability of the material to density-based methods of separation (e.g. GAT).

Table 13-25: Effect of Grind Dissolution tests

Target Grind		Mass Loss (%)	Au Grade (g/t)			Au Dissolution (%)		
			ROM	HNO_3 Res.	HNO_3 Res. (Calc)	Dissolution ROM CIL	Dissolution HNO_3 Res.	HNO_3 Ass. Gold
Main (MZ_FR4)	80% -25 µm	25.2	2.13	2.18	2.85	90.1	98.4	8.30
	80% -75 µm	24.9		2.54	2.84	83.5	97.7	14.2
	80% -125 µm	15.3		2.35	2.52	73.8	96.7	22.9
East (EZ_FR4)	80% -25µm	18.2	1.75	1.85	2.14	83.8	98.2	14.4
	80% -75µm	18.1		1.98	2.14	76.6	97.1	20.5
	80% -125µm	19.7		2.05	2.18	72.2	94.5	22.3
Wadi (WD_FR_COMP)	80% -25µm	23.6	4.53	6.27	5.93	91.5	98.7	7.20
	80% -75µm	22.9		5.55	5.87	86.2	98.7	12.5
	80% -125µm	20.8		5.61	5.71	84.7	97.1	12.4

Table 13-26: HLS Results for 80% -75 µm

Main (MZ_FR4)								
Fraction	**Mass**	**Mass %**	**Au Grade**	**Au Dist. %**	**U/D**	**S Grade**	**S Dist. %**	**U/D**
Sinks	31.33	6.24	18.57	54.41	771.8	37.76	67.34	978.9
Floats	273.32	54.45	0.78	19.94	-63.4	1.06	16.49	-69.7
Slimes	197.35	39.31	1.39	25.65	-34.7	1.44	16.17	-58.9
Total	502.00	100.00	2.13	100.00		**3.50**	100.00	
East (EZ_FR4)								
Fraction	**Mass**	**Mass %**	**Au Grade**	**Au Dist. %**	**U/D**	**S Grade**	**S Dist. %**	**U/D**
Sinks	26.61	5.31	15.84	48.03	805.1	41.96	59.52	1021.8
Floats	260.49	51.94	0.50	14.84	-71.4	1.26	17.50	-66.3
Slimes	214.41	42.75	1.52	37.13	-13.1	2.01	22.98	-46.3
Total	501.51	100.00	1.75	100.00		3.74	100.00	
Wadi (WD_FR_COMP)								
Fraction	**Mass**	**Mass %**	**Au Grade**	**Au Dist. %**	**U/D**	**S Grade**	**S Dist. %**	**U/D**
Sinks	36.46	7.07	35.47	55.37	683.0	32.51	51.43	627.4
Floats	205.86	39.92	1.36	11.99	-70.0	1.76	15.72	-60.6
Slimes	273.31	53.01	2.79	32.65	-38.4	2.77	32.85	-38.0
Total	515.63	100.00	4.53	100.00		4.47	100.00	

Red text indicates calculated values

The results of the HLS analyses (conducted at 80% -75 µm,

Table 13-26) indicate that approximately half of the gold was recovered into the concentrate fractions of the three samples ("sinks") with small to moderate mass pulls of ~5 to 7%.

Comparable sulphur and gold distributions in the HLS concentrate fractions suggest an association between sulphur and gold. Moderate distributions of gold were observed in the floats fractions of the Main and East samples (~20 and 15%), which indicates a small but notable association of gold with low density gangue minerals. The gold distribution in the slimes (-25 µm) fraction ranges from ~26 to 37% (typical for gold bearing material).

13.3.3.5 X-ray Diffraction Analysis

The XRD analyses showed relatively high concentrations of quartz, mica, dolomite, pyrite, chlorite and plagioclase, along with minor calcite, rutile and K-feldspar. The Main and WD samples are dominated by quartz and mica, whereas the East sample contains elevated concentration of quartz and plagioclase. These results correspond with the chemical analyses, which found high concentrations of SiO2, Al_2O_3, Fe_2O_3, CaO, K_2O and S.

13.3.3.6 QEMSCAN Bulk Modal Analysis

The QEMSCAN BMA results are consistent with the XRD analyses, which indicated high concentrations of quartz and mica in the Main and WD samples, and elevated plagioclase concentrations in the East sample (~21%). Pyrite concentrations are relatively high for all three samples (~6 to 10%). The HLS concentrate fractions are dominated by pyrite (~62 to 80%).

13.3.3.7 QEMSCAN Gold Deportment Assessment

- **Gold Speciation**

The QEMSCAN elemental deportment results (Table 13-27) indicate that gold occurs predominantly as native gold (>95%) with a small amount of petzite (<5%). Concentrations of petzite are markedly higher in the East sample. Petzite is known to be slow leaching during cyanidation.

Table 13-27: Gold Elemental Deportment (Mass %)

Sample	Fraction	Gold	Petzite
		Au-Ag	Ag_3AuTe_2
Main (MZ_FR4)	Feed	99.34	0.66
	HLS Conc	99.87	0.13
East (EZ_FR4)	Feed	96.92	3.08
	HLS Conc	95.61	4.39
Wadi (WD_FR_COMP)	Feed	99.70	0.30
	HLS Conc	99.97	0.03

Native gold grains from the Main and East samples are characterized by similar Ag concentrations (Avg. ~20%), whereas native gold from the WD sample displays significantly higher Ag concentrations (Avg. ~35%)

- **Gold-containing Particles**

Assessment of gold-containing particles indicates that the Main and East samples possess gold-containing particles with relatively low particle SGs (5.3 to 6.0). This suggests that a high proportion of gold is poorly liberated and locked within pyrite (SG of 5.01). Higher degrees of gold liberation can be inferred from the high particle SG in the WD sample.

- **Gold Grain Size Dimensions**

The gold grains in all three samples are typically less than 40 µm in size, however coarse-grained gold was observed in the WD sample (both feed and HLS concentrate fractions).

- **Gold Grain Exposure**

The gold grain exposure assessment indicates that most of the gold is exposed (~68 to 99% for the feed samples and ~65 to 90% for the HLS concentrate fractions). Strong pyrite-gold associations were evident in both the Main and East samples. The high percentage of exposed gold in the WD samples can be explained by the presence of coarse gold.

- **Exposure of -25 µm gold**

The exposure characteristics of gold grains smaller than 25 µm in size indicate that the degree of exposure diminishes with decreasing gold grain size and therefore improves with increasing gold grain size. The results also highlight the presence of fine-grained gold in the East sample and coarse-grained gold in the WD sample.

- **Gold Grain Liberation**

The gold grain liberation data indicates that a large proportion of gold occurs as locked gold in the Main and East samples (~53-64% for Main and ~52-95% for East). The large percentage of liberated gold in the WD sample can be explained by the presence of coarse gold.

The paucity of gold in the middlings category (where between 30% and 80% of the particle area is composed of gold) for all three samples suggests that gold occurs in two dominant forms: as liberated gold grains or as small grains attached to/occluded within larger particles, such as pyrite.

The results highlight a strong association between gold and pyrite in the Main and East samples, as well as gold and other sulphides in the WD sample.

- **Liberation of -25 µm gold**

The liberation characteristics of gold grains smaller than 25 µm highlight a number of key features: The dominance of locked and liberated gold over middlings gold, the occurrence of locked gold in fine size fractions and liberated gold in coarse size fractions, as well as the strong gold-pyrite associations in the fine size fractions.

- **Sulphur Deportment**

The results of the QEMSCAN sulphur deportment show that the majority of the sulphur occurs within pyrite (>93%). Small concentrations of sphalerite, arsenopyrite and galena were observed in the WD sample. Most of the gold-bearing pyrite grains display compact (porous-free) morphologies. Fine-grained inclusions of rutile, carbonates and base metal sulphides were commonly observed within pyrite. Pyrite is well liberated in both the feed samples (~94 to 95% liberated) and HLS concentrate fractions (~97 to 98% liberated). The grain size dimensions of pyrite illustrate an asymmetric distribution, with a mode around ~30 µm for the feed samples and ~50 µm for the de-slimed (+25 µm) HLS concentrate fractions. The liberation characteristics of pyrite are similar for each of the three samples. In almost all of the size fractions, pyrite is well liberated. Although locked pyrite accounts for approximately a third of the pyrite in the -5 µm fraction, this particular size fraction possesses very little pyrite (<1%). The mineral associations of pyrite for all three samples are comparable (quartz, mica, rutile and carbonate). However, pyrite in the WD sample also possesses an association with other sulphides (e.g., sphalerite, galena).

13.3.3.8 Diagnostic Leach Tests

The results of the diagnostic leach tests are consistent with the effect of grind tests conducted at the same grind size, 80% -75 µm. Gold dissolutions for Main, East and WD are ~83%, ~77% and ~85%, respectively, Table 13-28 (compare with ~84%, ~77% and ~86% from the effect of grind tests). The diagnostic leach tests also indicate that most of the refractory gold was released during nitric acid pre-treatment of the material (after HCl pre-treatment). This confirms the strong mineralogical gold-pyrite connection since sulphides are soluble in hot nitric acid. The small proportion of gold released during roasting of the East sample (~3%) may be linked to remnant gold-bearing sulphides after the nitric acid stage.

A comparison between the diagnostic leach results and QEMSCAN exposure values from the HLS concentrate show that the results are broadly comparable for the Main sample. However, the fine-grained nature of gold in the East sample and the coarse-grained nature of the gold in the WD sample are amplified in the HLS concentrate fractions.

Table 13-28: Diagnostic Leach Results (80% -75 µm)

Stage	Main (MZ_FR4)	East (EZ_FR4)	Wadi (WD_FR_COMP)
Cyanide Soluble (%)	82.88	77.35	85.29
Preg-robbed (%)	0.89	0.47	1.74
Hydrochloric acid leach (%)	2.16	0.78	0.83
Nitric acid leach (%)	12.70	17.33	11.40
Roast (%)	0.38	3.30	0.24
Silica/Gangue (%)	0.98	0.76	0.49
Total	**100.00**	**100.00**	**100.00**
Available by CIL Recovery	83.77	77.83	87.03

13.3.3.9 Conclusions

The gold deportment test work yielded a number of significant findings. One of the most salient findings was the connection between gold grain size and leaching behaviour. Contrasting trends were observed between the samples from the GSS (Main and East samples) and the WD deposits.

- **Main and East Zones**

The gold grades of the Main and East samples are relatively low (~1 to 2 g/t) and most of the gold is present in the form of native gold. Gold recoveries are moderate at coarse grind sizes (~75%) and improve with each successive stage of milling (up to a maximum of ~84% for East and ~90% for Main). These trends can be explained by the presence of fine-grained gold (<40 µm). Higher gold recoveries from the Main sample can be attributed to the presence of slightly larger gold grains. Regarding the refractory gold component, the diagnostic leach tests indicate that approximately 80 to 90% of the refractory gold occurs in sulphides – a feature confirmed by the mineralogical analysis. The major factor behind the moderate ROM gold recoveries from Main and East is probably due to the exceedingly fine-grained nature of particulate gold in pyrite. The possibility of a solid-solution gold component is small considering the scarcity of arsenic in the samples (e.g. arsenopyrite, arsenian pyrite).

- **WD**

The gold grade of the WD sample is relatively high in comparison to the Main and East samples (~5 g/t). The gold occurs as native gold but unlike the Main and East sample, elevated concentrations of alloyed silver were noted (Avg. ~35%). Gold recoveries are notably higher for the WD sample. Gold recoveries for the coarse, medium and fine grind sizes are ~85%, ~86% and ~91%, respectively. The higher recoveries are readily explained by the presence of coarse gold grains (>70 µm). Nevertheless, a fine-grained gold component is also evident as shown by the ~6% increase in recovery upon fine grinding to 80% -25 µm. This feature was observed in all three samples. The diagnostic leach results indicate that the bulk of the refractory gold occurs in sulphides (~88% at 80% -75 µm). Although pyrite is the dominant sulphide in the WD sample (>94%), other sulphides, such as sphalerite are galena, are also present.

13.3.4 Gravity Amenability Test Work /Flash Flotation Test Work

13.3.4.1 CVD Knelson Gravity Amenability Test Work

The use of continuous (CVD) centrifugal gravity concentrators has become common practice for recovering gold associated with sulphides where higher weight yields are required than the more traditional batch centrifugal concentrator. Three gravity amenability tests were carried out on 3 x 1 kg aliquots of sample at a grind of 80% -75 µm. A five pass gravity concentration test was allowed, the gravity concentrates from each pass, were further upgraded by heavy liquid separation. The sinks and floats were assayed to extinction. The final gravity tails were reserved for cyanidation tests.

Table 13-31, and Table 13-32, present the gravity concentration results for gold, silver and sulphur respectively. Figure 13-6presents the gravity recovery comparison per concentrate.

Due to poor gold metallurgical accountability, an additional gravity concentration test was conducted, to ascertain the gold accountability; the concentrates of this test were however not subjected to heavy liquid separation, but were assayed to totality, Table 13-29 presents the initial gold gravity test results.

The silver and gold recover display a similar trend in that, with each gravity pass, the recovery decreases. Maximum gold recovery of 77.0% was attained with a maximum silver recovery of 60.8%. With the exception of the first pass, the sulphur test results, also illustrate a decrease in the recovery with each gravity pass, this test yielded maximum sulphur recovery of 75.0%.

Table 13-29: Initial gravity concentration results; Gold

Average head assayed	2.14	g/t					
Calc'd head	1.43	g/t					
Accountability	66.7	%					
Fraction	Sink/Float	Mass	Mass	Gold Grade (g/t)		Recovery	Recovery (Cumulative)
		g	%	Discrete	Combined	%	%
P. Conc 1	Sink	10.9		9.50			
	Float	84.0	8.74	1.13	2.10	12.8	12.8
P. Conc 2	Sink	12.9		16.1			
	Float	104	10.7	1.00	2.67	20.1	32.9
P. Conc 3	Sink	10.5		11.7			
	Float	110	11.1	0.60	1.57	12.2	45.1
P. Conc 4	Sink	11.5		13.1			
	Float	107	10.9	0.60	1.81	13.9	59.0
P. Conc 5	Sink	5.65		13.1			
	Float	133	12.8	0.50	1.01	9.06	68.0
Tail		497	45.7	1.00			
Total		1086	100	1.43			
Primary Conc		589	54.3	1.79		32.0	100

Table 13-30: Repeat gravity concentration results; Gold

Average head assayed	2.14	g/t			
Calc'd head	2.45	g/t			
Accountability	114	%			
Fraction	Mass	Mass	Gold Grade (g/t)	Recovery	Recovery (Cumulative)
	g	%	Discrete	%	%
P. Conc 1	93.4	8.27	10.6	35.9	35.9
P. Conc 2	91.8	8.13	4.79	15.9	51.7
P. Conc 3	94.7	8.38	4.06	13.9	65.6
P. Conc 4	96.8	8.57	1.94	6.78	72.4
P. Conc 5	100	8.86	1.27	4.59	77.0
Tail	653	57.8	0.98	23.0	100
Total	1130	100	2.45		
Primary Conc	477	42.2	4.47		

Table 13-31: Gravity concentration results; Silver

Assayed Head	2.77	g/t					
Calc'd head	2.99	g/t					
Accountability	108	%					
Fraction	Sink/Float	Mass	Mass	Silver Grade (g/t)		Recovery	Recovery (Cumulative)
		g	%	Discrete	Combined	%	%
P. Conc 1	Sink	13.1		33.8			
	Float	86.8	9.29	2.30	6.42	19.9	19.9
P. Conc 2	Sink	21.4		20.8			
	Float	67.1	8.24	1.30	6.01	16.6	36.5
P. Conc 3	Sink	11.3		16.8			
	Float	94.0	9.81	1.40	3.05	10.0	46.5
P. Conc 4	Sink	7.45		14.3			
	Float	120	11.9	1.10	1.87	7.42	53.9
P. Conc 5	Sink	7.77		14.2			
	Float	85.5	8.68	1.30	2.38	6.89	60.8
Tail		560	52.1	2.25			
Total		1074	100	2.99			
Primary Conc		514	47.9	3.80		39.2	100

Table 13-32: Gravity concentration results; Sulphur

Assayed Head	3.50	%					
Calc'd head	3.54	%					
Accountability	101	%					
Fraction	Sink/Float	Mass	Mass	Sulphur Grade (%)		Recovery	Recovery (Cumulative)
		g	%	Discrete	Combined	%	%
P. Conc 1	Sink	13.1		35.7			
	Float	86.8	9.29	2.27	6.64	17.4	17.4
P. Conc 2	Sink	21.4		33.1			
	Float	67.1	8.24	2.05	9.55	22.3	39.7
P. Conc 3	Sink	11.3		34.5			
	Float	94.0	9.81	1.98	5.47	15.2	54.9
P. Conc 4	Sink	7.45		33.7			
	Float	120	11.9	1.27	3.17	10.6	65.5
P. Conc 5	Sink	7.77		35.5			
	Float	85.5	8.68	0.98	3.86	9.47	75.0
Tail		560	52.1	1.70			
Total		1074	100	3.54			
Primary Conc		514	47.9	5.54		25.0	100

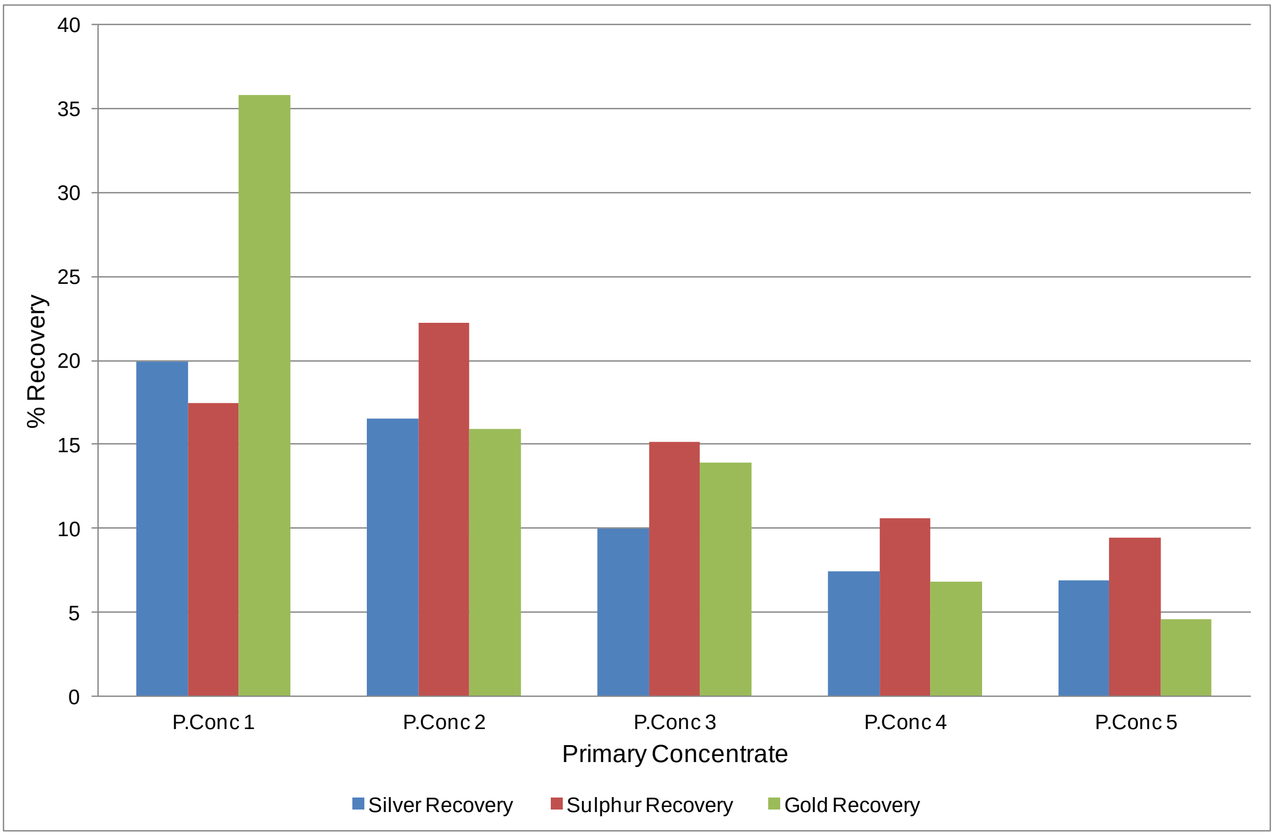


Figure 13-6: Gravity Recovery Comparison

13.3.4.2 Flash Flotation Test work

A scouting flash flotation test was conducted on the main material zone composite sample. The test was conducted on a 1 kg sample milled to 80%-425 µm. The test was conducted on pulp conditioned with $CuSO_4$ and PAX at natural pH and at 60% solids.

Rate samples were collected over a 5 minute flotation period. The rate samples collected were analysed for gold and sulphide Sulphur owing to mass limitations. A 4 kg bulk sample was used to generate a bulk concentrate and tailings for downstream processing.

The flash flotation test results (Table 13-33) indicated that the gold and sulphide sulphur kinetics were relatively fast, with as much as 12% mass recovered into a three minute concentrate. Gold and sulphide recovery from the three minute concentrate was 71.5% and 84.2% respectively.

The sulphur grades up to and including the three minute concentrate showed substantial upgrading (in relation to the feed grades), with the fourth and the fifth concentrate grades much lower and similar to the feed grades. The grades into the fourth and fifth concentrate indicate poor selectivity and that the recovery into these units could be as a result of entrainment instead of true flotation.

Table 13-33: Flash Flotation Test Results

Incremental Product	Time (mins)	Mass (g)	Mass (%)	Grade		Recovery	
				Au	S	Au	S
Conc 1	1	58.6	6.0	13.7	28.6	40.1	52.8
Conc 2	1	37.1	3.8	13.2	22.3	24.5	26.1
Conc 3	1	23.1	2.4	6.0	7.4	6.9	5.3
Conc 4	1	28.5	1.9	3.2	3.9	3.0	2.3
Conc 5	1	15.9	1.6	3.0	3.5	2.4	1.8
Tail		827.7	84.4	0.6	0.5	23.1	11.7
Total		980.9	100.0	2.0	3.2	100.0	100.0
Conc 1	1	58.6	6.0	13.7	28.6	40.1	52.8
Conc 1+2	2	95.7	9.8	13.5	26.2	64.6	78.9
Conc 1+2+3	3	118.8	12.1	12.0	22.5	71.5	84.2
Conc 1+2+3+4	4	137.3	14.0	10.9	20.0	74.4	86.5
Conc 1+2+3+4+5	5	153.2	15.6	10.0	18.3	76.9	88.3

13.3.5 Cyanidation Testwork on Gold and Silver Bearing Samples

Prior to leaching, ultrafine milling was conducted on the following CIL residue samples:

- Main HLS Floats;
- Main HLS Slimes;
- East HLS Floats;
- East HLS Slimes;
- Wadi HLS Floats;
- Wadi HLS Slimes;
- Main -25 µm CIL Residue;
- East -25 µm CIL Residue; and
- Wadi -25 µm CIL Residue.

Due to mass constraints it was not possible to conduct the milling in a conventional mill and thus a swing disk pulverizer was used to mill the samples. A Fristch PSD analysis was conducted on the CIL residue samples following the milling.

For each test, slurry was prepared at 45% solids by mass ahead of the preconditioning. The slurry preconditioning was conducted for a period of 1 hour, during which period the pH was adjusted to 10.5 using lime at a concentration of 100 g/l. On completion of the preconditioning, carbon and cyanide were added at concentrations of 20 g/l and 10 kg/t respectively. Cyanide was added in the form of NaCN. The dissolution tests were conducted with duration of 24 hours. The tests were agitated by way of bottle rolling.

On completion of the leaching, the slurry was poured through a screen to remove the carbon into the Buchner filter. The filtrate was titrated using silver nitrate and oxalic acid to establish the sodium cyanide and lime consumptions respectively. The residue, carbon and filtrate were analysed for gold and silver.

Fritsch PSD Analysis was conducted on the CIL residue samples following ultra-fine milling using the swing disk pulverizer. The Wadi sample milled readily and grind was at the target; however for the Main 1 and East 2 sample stickiness was a problem.

The sizing for the earlier milling durations has been included (Table 13-34 and Table 13-35). The initial readings and 30 min readings were not problematic. Based on the first two readings the milling times were determined as 52.5 min for the East sample and 67.5 min for Main (assuming linearity).

Both samples were already at about 70% passing 10 µm after 30 min. The samples were milled for a further 20 min to achieve a total milling time of 50 min. At this point the material was extremely bound to the pot. The material was de-agglomerated (as much as was possible) and the maximum ultra sound setting and duration used on the laser sizer, but it is clear that the material was not completely de-agglomerated; for this reason, and due to the restricted mass, the first two points and a linear curve were assumed. It is not possible that the curve is actually linear, so the samples were not quite at 80% passing 10 µm, but it is likely to be fairly close, certainly better than 70% passing 10 µm, but also slightly less than 80%.

Table 13-34: East 2 Fritsch PSD analysis

Sample	East		
Measurement	Reading 1	Reading 2	Average
Time (Min)	% passing 10 µm		
0			63.8
30	75.5	70.6	73.1
50	65.2	64.8	65.0

Table 13-35: Main 1 Fritsch PSD analysis

Sample	Main		
Measurement	Reading 1	Reading 2	Average
Time (Min)	% passing 10 µm		
0			60.2
30	69.6	68.4	69.0
50	65.4	65.9	65.7

Table 13-36 presents the gold head assay for each of the samples. Additional gold head assays from previous study have been included in the table. Table 13-37 presents the silver head assay.

Table 13-38 presents a summary of the gold cyanidation test results, while Figure 13-7 presents the gold dissolution comparison between the different samples. The slime samples yielded relatively higher gold dissolutions in comparison to both the Float and the CIL residue samples. Gold dissolutions of 91.3, 82.8, and 89.2% were attained for the Main 1 slime, East 2 slime and Wadi 3 slime respectively with cyanide consumptions of 6.11, 6.05, and 5.84 kg/t respectively.

Table 13-39 presents a summary of the silver cyanidation test results, while Figure 13-8 presents the silver dissolution comparison between the different samples.

As was the case with the gold dissolution, the slime samples yielded reasonably higher silver dissolutions in comparison to both the float and the CIL residue samples. Silver dissolutions of 69.4%, 67.7% and 62.3% were attained for Main 1 slime, East 2 slime and Wadi 3 slime respectively.

Table 13-36: Gold Head Assays

Sample ID	Report 16/379	Report 15/640 (Metallurgy)			Report 15/640 (Mineralogy)
	Au	Au	Au	Au	Au
	g/t	g/t	g/t	g/t	g/t
Main 1 Float	0.58				0.78
East 2 Float	0.48				0.50
Wadi 3 Float	1.04				1.36
Main 1 Slime	1.26				1.39
East 2 Slime	1.26				1.52
Wadi 3 Slime	2.46				2.79
Main 1 CIL Res	0.18	0.22	0.22	0.22	
East 2 CIL Res	0.27	0.29	0.31	0.28	
Wadi 3 CIL Res	0.40	0.38	0.42	0.43	

Table 13-37: Silver Head Assay

Sample ID	Report 16/379	
	Ag	Ag
	g/t	g/t
Main 1 Float	1.00	0.90
East 2 Float	1.10	1.20
Wadi 3 Float	7.40	7.60
Main 1 Slime	2.20	2.20
East 2 Slime	4.20	3.90
Wadi 3 Slime	16.8	17.7
Main 1 CIL Res	2.00	2.00
East 2 CIL Res	1.60	1.50
Wadi 3 CIL Res	4.10	4.10

Table 13-38: Gold Cyanidation Test Results

Sample ID	Assayed Head	Calc. Head	Reagent Consumption		Au Dissolution	Accountability
			NaCN	CaO		
	Au (g/t)	Au (g/t)	(kg/t Feed)	(kg/t Feed)	Calc Head (%)	Au %
Main 1 Float	0.68	0.68	3.97	0.01	80.5	101
East 2 Float	0.49	0.46	4.01	0.01	74.6	94.8
Wadi 3 Float	1.20	1.09	4.74	0.02	76.3	90.6
Main 1 Slime	1.33	1.34	6.11	0.02	91.3	101
East 2 Slime	1.39	1.34	6.05	0.02	82.8	96.5
Wadi 3 Slime	2.46	2.36	5.84	0.00	89.2	95.9
Main 1 CIL Res	0.21	0.22	3.07	0.02	32.3	104
East 2 CIL Res	0.31	0.35	2.59	0.00	54.2	114
Wadi 3 CIL Res	0.41	0.39	2.24	0.00	21.9	95.8

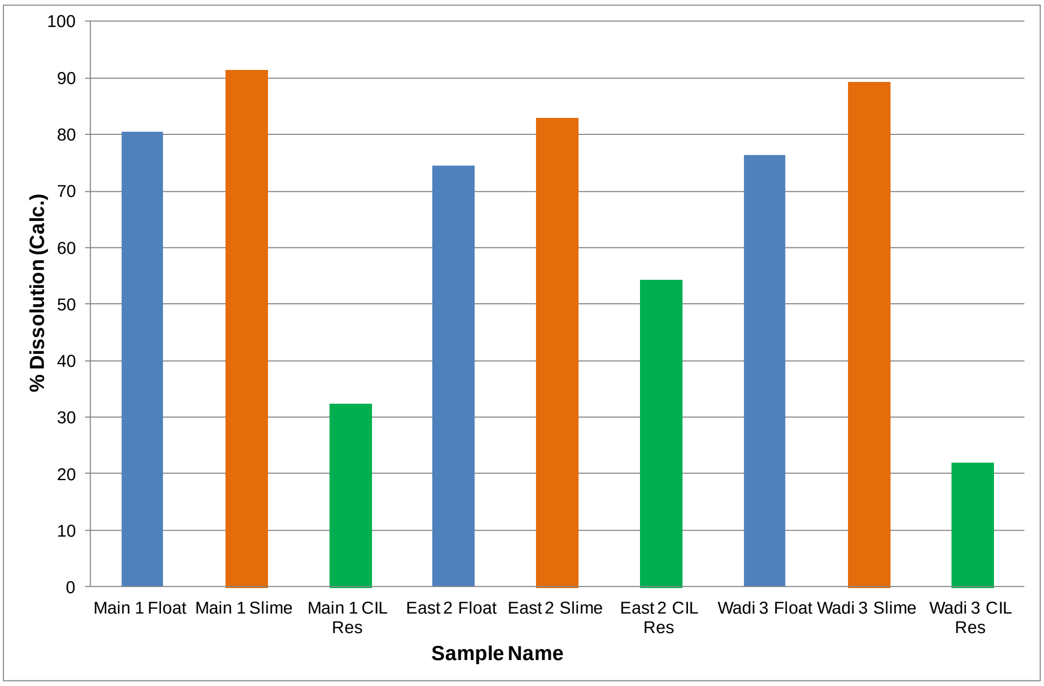


Figure 13-7: Gold Dissolution Comparison

Table 13-39: Silver Cyanidation Test Results

Sample ID	Assayed Head	Calc. Head	Reagent Consumption		Ag Dissolution	Accountability
			NaCN	CaO		
	Ag (g/t)	Ag (g/t)	(kg/t Feed)	(kg/t Feed)	Calc Head (%)	Ag %
Main 1 Float	1.00	1.47	3.97	0.01	22.6	147
East 2 Float	1.10	1.17	4.01	0.01	3.25	106
Wadi 3 Float	7.40	4.67	4.74	0.02	23.7	63.1
Main 1 Slime	2.20	1.90	6.11	0.02	69.4	86.5
East 2 Slime	4.20	2.88	6.05	0.02	67.7	68.6
Wadi 3 Slime	16.8	10.4	5.84	0.00	62.3	61.7
Main 1 CIL Res	2.00	1.72	3.07	0.02	5.94	86.2
East 2 CIL Res	1.60	1.16	2.59	0.00	24.2	72.6
Wadi 3 CIL Res	4.10	3.24	2.24	0.00	15.0	79.1

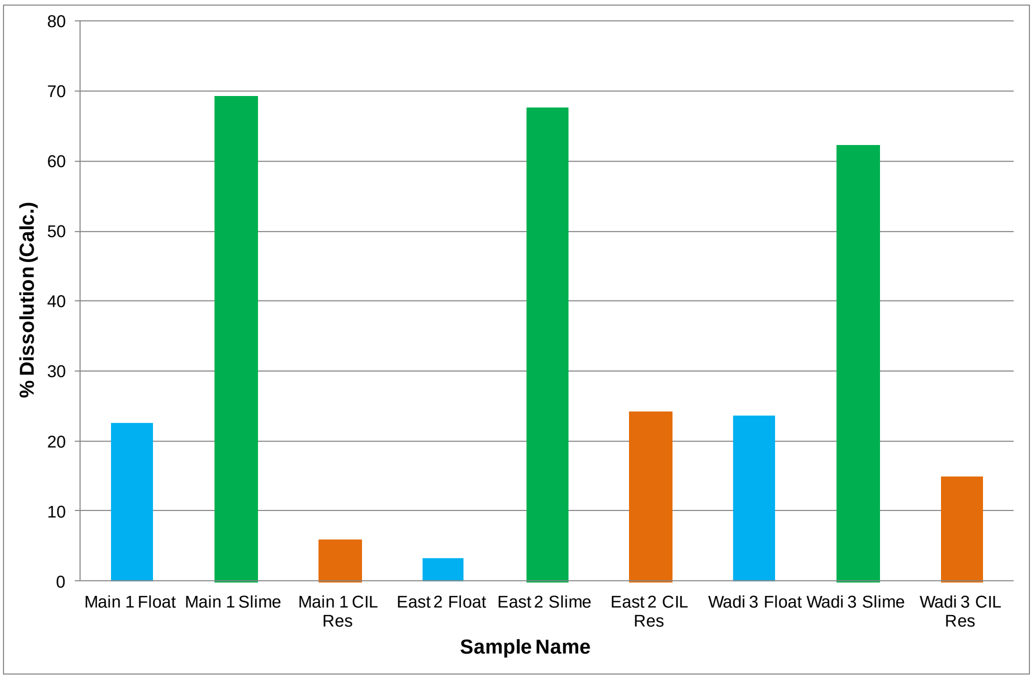


Figure 13-8: Silver Dissolution Comparison

A total of nine samples were made available for the test work. Ultrafine milling was required on three of the samples prior to the leach tests namely Main 1 CIL residue, East 2 CIL residue and the Wadi 3 CIL residue.

The slime samples yielded better leaching response, this as higher gold and silver dissolutions were obtained for the slime samples. It should be noted though that the silver cyanidation tests yielded poor metallurgical accountabilities. Due to mass constraints it was not possible to conduct either the re-assays or the repeat leach tests to attain the metallurgical accountability.

Table 13-40 presents a summary of the overall gold dissolution for the CIL tests conducted on the Main 1 ROM, East 2 ROM and Wadi 3 ROM. A regrind of the residues yielded gold dissolution increase of 3.2%, 8.8% and 1.8% for the Main 1, East 2 and Wadi 3 sample respectively.

Table 13-40: Overall Dissolution for the CIL Tests

Sample ID	ROM Head	Residue Head	ROM Dissolution	Residue Dissolution	Overall Dissolution
	Au g/t	Au g/t	Au %	Au %	Au %
Main 1	2.13	0.21	90.1	32.3	93.3
East 2	1.75	0.31	83.8	54.2	92.6
Wadi 3	4.53	0.41	91.5	21.9	93.3

13.3.6 Heap Leach Amenability Testwork

Heap leach amenability tests were conducted on gold and silver bearing composite oxide samples.

13.3.6.1 Simulated heap leach tests

Simulated heap leach tests using intermittently rolled bottles, see Figure 13-9, were conducted on the composite samples at three top sizes of -6, -12, and -25.4 mm. Maximum gold dissolutions of 76.6, 72.9, and 72.9% were attained for the -6, -12, and -25.4 mm top size respectively with maximum silver dissolutions of 9.68, 9.72, and 9.64% respectively. Results are summarised in Figure 13-10. It is important to note that these leach tests were conducted with excess cyanide and are not directly comparable with the column tests.

Figure 13-9: Bottle Roll Setup

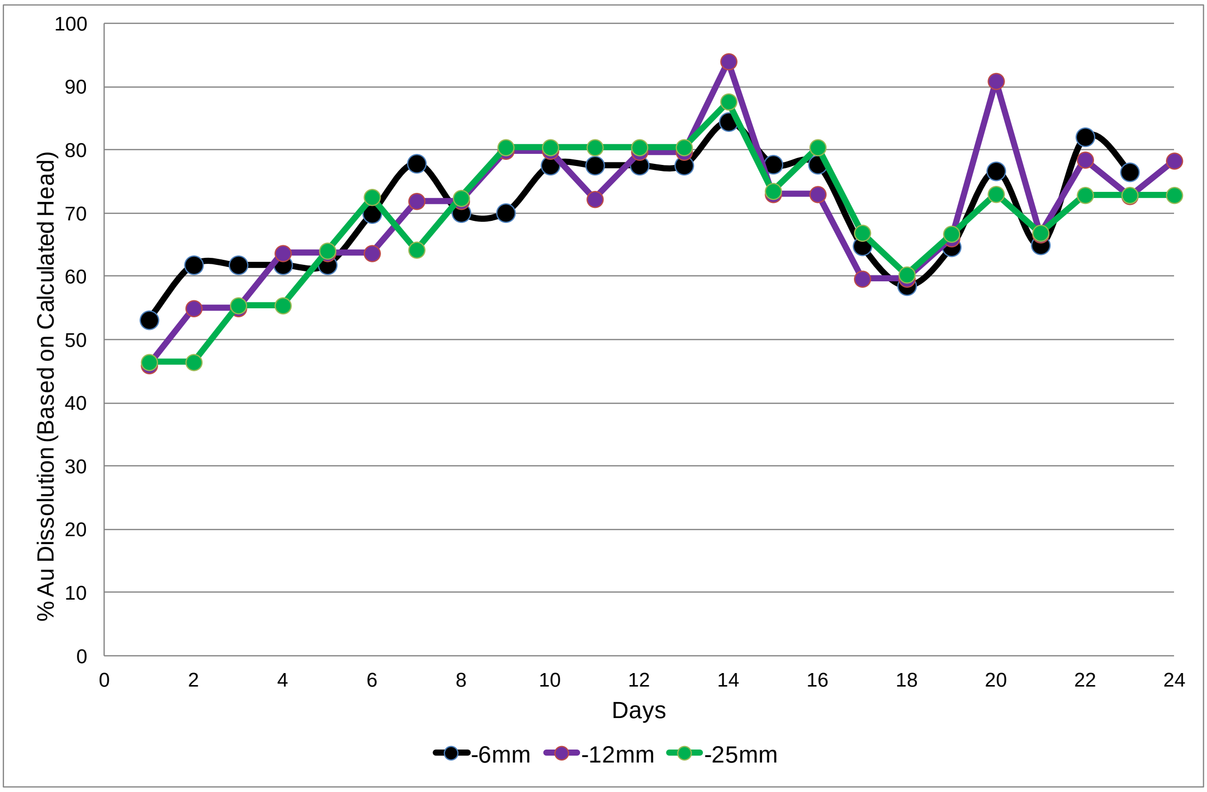


Figure 13-10: Simulated Heap Leach Gold Dissolution

13.3.6.2 Agglomeration

Agglomeration was conducted using cement as the binding agent at a concentration of 11, 12, 15, 18, and 20 kg/t, an additional test was conducted with without cement addition.

Percolation rate tests, see Figure 13-11, were conducted to determine the effect of agglomeration on ponding and the associated feed rate. Based on the percolation rate tests, a cement addition of 11 kg/t was selected for the further tests (pressure percolation and column heap leaching).

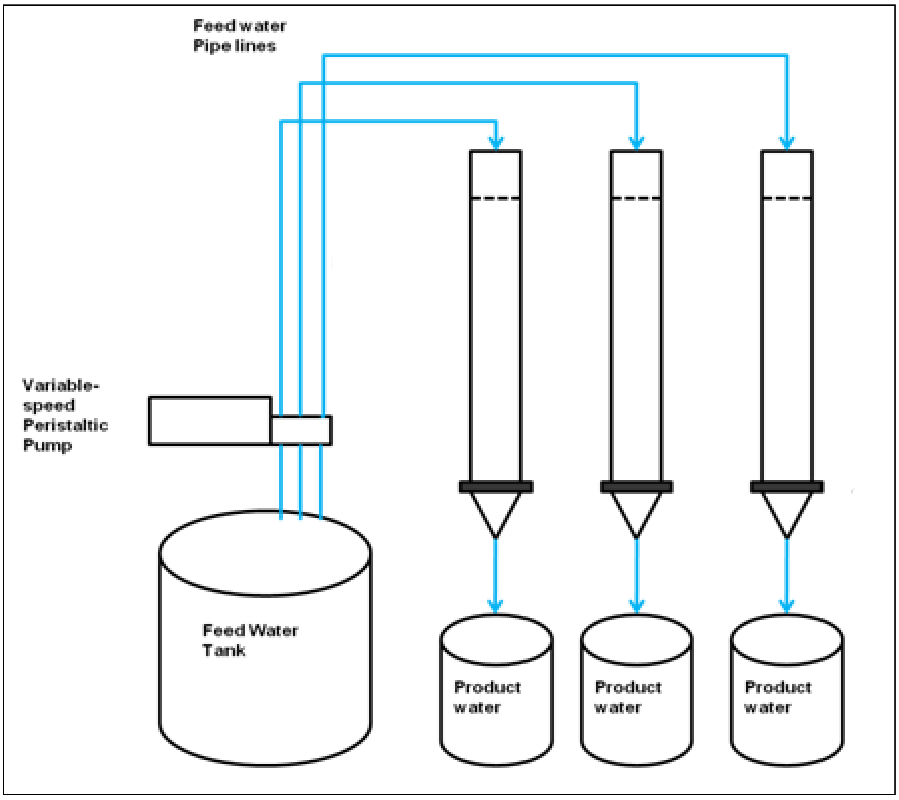


Figure 13-11: Percolation Test Setup Schematic

13.3.6.3 Pressure Percolation

A sample was loaded into the test rig vessel (see Figure 13-12) and subjected to increasing pressures ranging from 0 kPa to 4000 kPa with the irrigation rate being maintained at 10 $l/h/m^2$. The objective of this test was to determine whether ponding would occur and the corresponding pressure. The test also assists in the modelling of the heap height; the heap height is related to the pressure as described by Equation 1.

$$P = \rho g h \qquad \text{(Equation 1)}$$

Where:

P = pressure exerted on the sample

ρ = sample bulk density

g = gravitational acceleration

h = heap height

The test results yielded heap heights of 15.6 m.

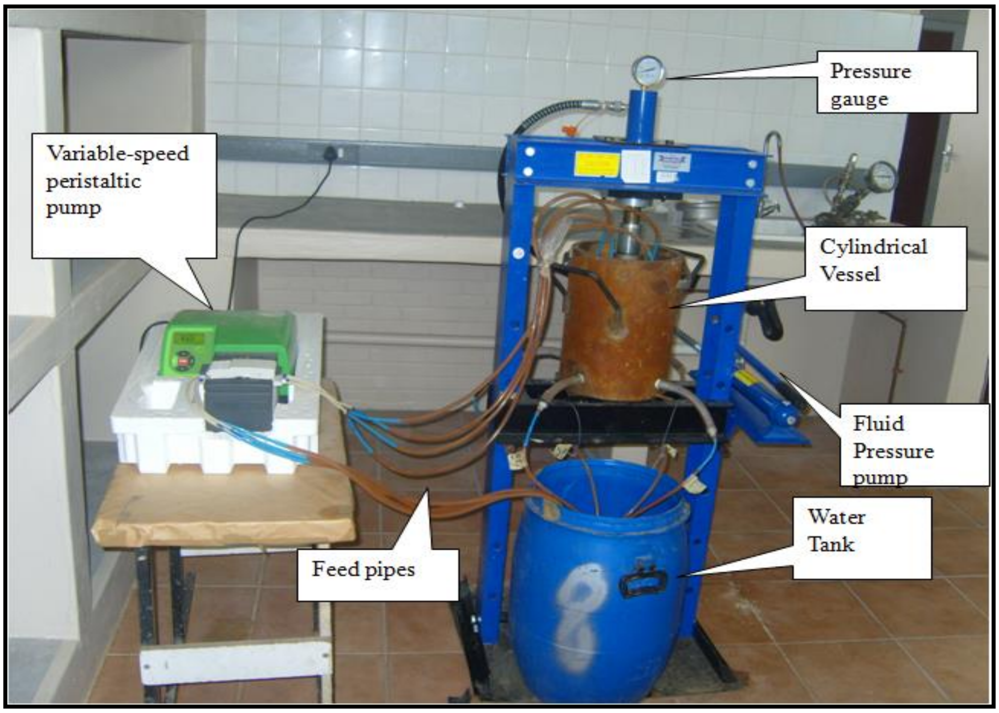


Figure 13-12: Pressure Percolation Test Rig

13.3.6.4 Column Heap Leach Trial

An exploratory column heap leach test was conducted at a crush size of -12 mm in order to verify the simulated heap leach results, this test yielded a gold dissolution of 60.6% and silver dissolution of 4.85% (based on residue). Results of the test are summarised in Figure 13-13.

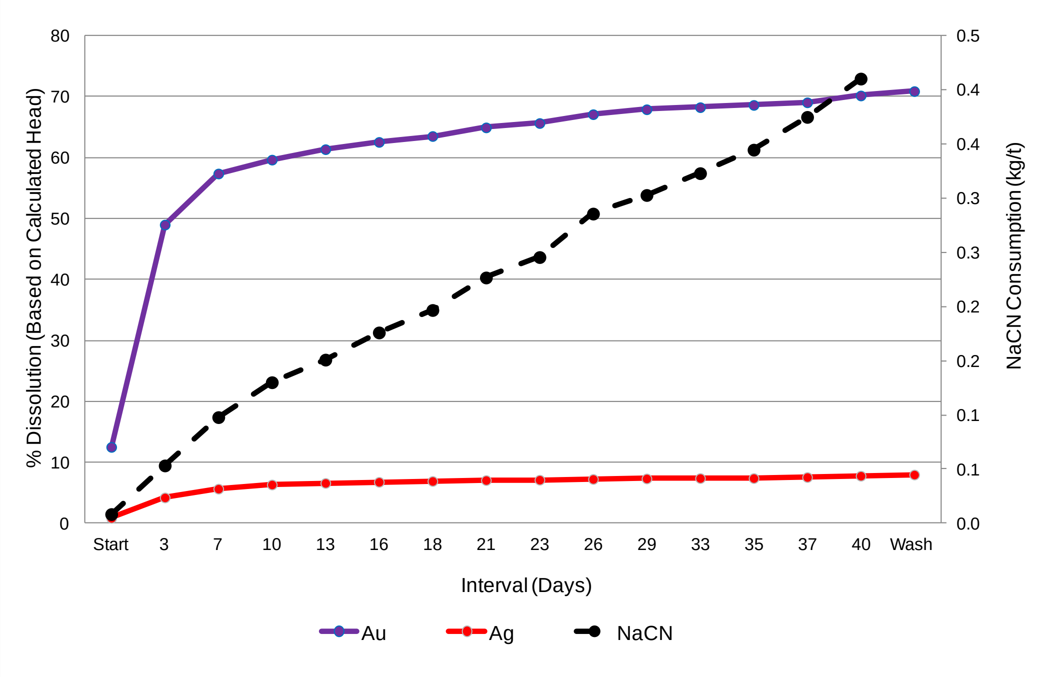


Figure 13-13: Exploratory column heap leach data including gold and silver dissolution and cyanide consumption.

Subsequent to the exploratory column heap leach test, column heap leach tests were conducted on the agglomerated samples using 15 kg/t cement. Columns of 150 mm diameter were loaded with mass up to a height of 4 m. The solution was fed to the columns using a peristaltic pump as shown in Figure 3. Sodium cyanide (NaCN) concentration of 200 ppm was used, with an irrigation rate of 10 l/h/m^2. The pregnant leach solution was removed twice a week and its volume measured and recorded. A sample was removed from the pregnant leach solution and titrated to monitor reagent consumptions and analysed for gold and silver.

The leaches were terminated after 48 days, following which a three day water wash was conducted and the columns were allowed to drain. The columns were decommissioned in three sections (top, middle and bottom), from each section residue aliquots were removed by way of rotary splitting and analysed for gold and silver.

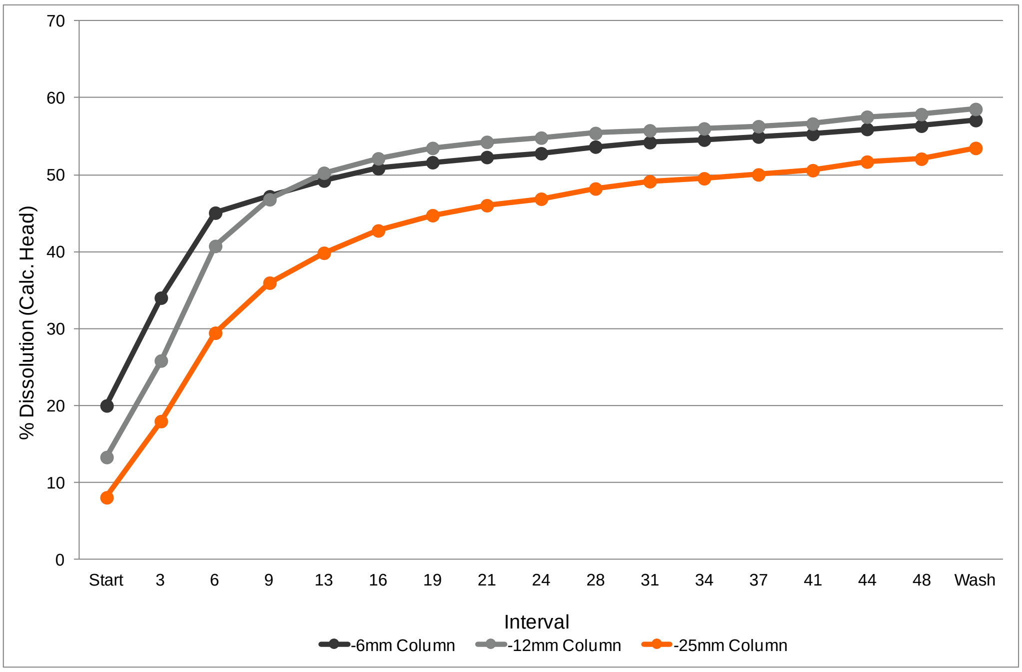


Figure 13-14: Column Heap Leach Gold Dissolution.

13.4 Metallurgical Summary

The extensive metallurgical test work conducted on various different composite samples from the deposit clearly indicates that the two main material types, at Gulat Safar (Main & East) and Wadi Doum are amenable to direct cyanidation.

The diagnostic leach and gold deportment studies conducted by SGS RSA clearly show that gold deportment is mainly with pyrite and at an extremely fine grain size. The gold dissolution in cyanide is sensitive to grind size.

Referring to Figure 13-15 below, note how the leaching of gold continues below 25 microns and this is supported by the gold deportment data generated by SGS RSA. Of the three lithologies, the East Zone Fresh clearly contains more fine grained refractory gold than Main Zone Fresh and the lowest refractory gold component is associated with Wadi Doum Fresh.

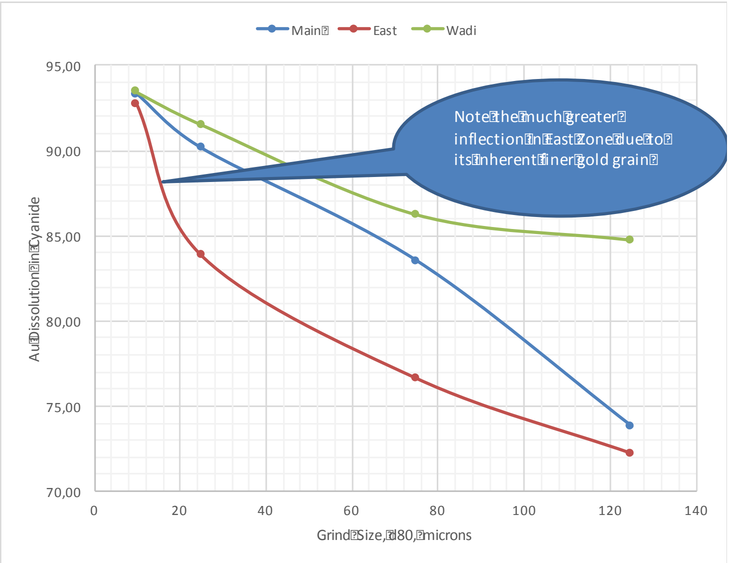


Figure 13-15 – Summary of all Diagnostic Leach Testwork

The terminal theoretical leach dissolution for all three material types is clearly 93-94% for Gold. It should be noted that the Flash Flotation of Main Zone Fresh followed by intensive cyanide leaching of reground (-10 µm) flash flotation concentrates gave rise to 94% overall recovery. This confirms the maximum potential for Main Zone Fresh.

In summary the following process circuit (Refer to Figure 13-16) would appear to offer Orca the best potential and will be the circuit that is further developed at Pre-Feasibility level.

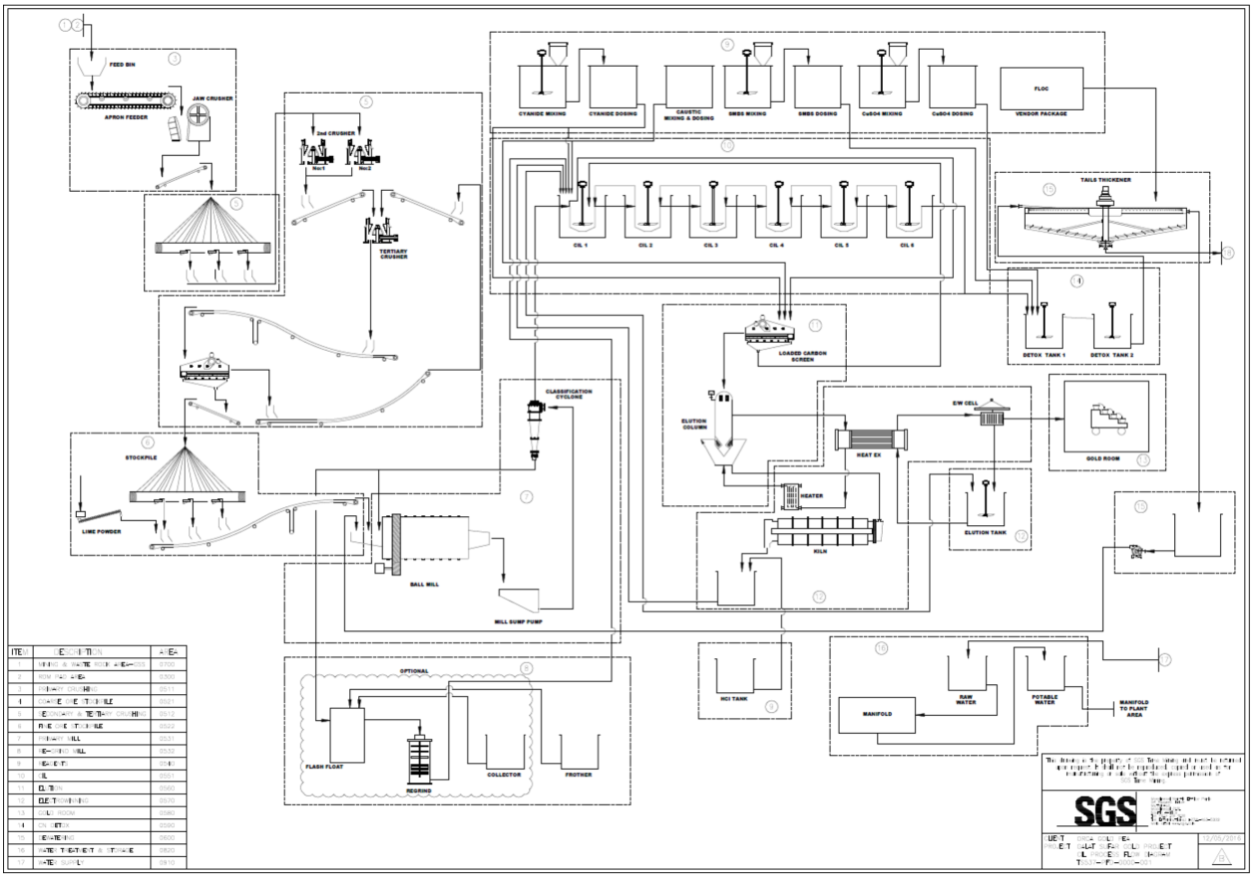


Figure 13-16 – Proposed Flowsheet for Orca Gold GSS and Wadi Doum Project

This circuit (excluding Flash Flotation) has produced the following gold recoveries as summarised in Table 41 below

Table 41 – Summary of Gold Recoveries used for PEA Study.

Pit	Material	% Recoveries
GSS	Oxide	92
	Transitional	87
	Fresh	80
Wadi Doum	All	83

The economic viability of heap leaching, specifically the east oxides have shown poor gold dissolution rates and this process option has been discounted on this basis.

Froth Flotation provides a potential for Orca to coarsen the primary grind size since the flotation response of East and Main was extremely encouraging.

The next stage of work will focus upon the optimisation of:

- Flash Flotation Reagents and Flotation time;
- Regrinding of Optimised Flash Flotation Concentrate;
- Cyanide Leach parameters on optimised reground flash flotation concentrates; and
- Variability of following parameters by material zones and lithologies:
  - Bond Ball mill work index
  - Head Assay and Clay content
  - Flash Flotation Response using optimised conditions from aforementioned program
  - CIL Response

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

In February 2016, MPR Geological Consultants Pty Ltd (MPR) estimated gold Mineral Resources for the GSS and WD deposits. The estimates incorporate results from additional RC and diamond drilling completed since MPR last estimated resources for these deposits in January 2015.

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 2016. 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 more than 50 degrees in 5 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 Qualified Person responsible for the Mineral Resources is 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 for seven days between 16th and 21st January, 2014.

14.2 Estimation of GSS Resources

14.2.1 Resource dataset

Relative to the January 2015 dataset, the GSS sampling database available for the current review incudes nine additional RC holes drilled during September and October 2015 comprising the following:

- 4 holes in filling the main mineralized area in the eastern part of the deposit increasing confidence in estimated resources for this area;
- 3 holes infilling an area of exploratory drilling in the far eastern part of the deposit allowing estimation of resources in this area; and
- 2 exploratory holes which lie to the south of the main mineralized zones and are too broadly spaced for inclusion in resource estimates.

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 drill holes not relevant to the current estimate.

The compiled resource dataset comprises 30,915 composites with gold grades ranging from 0.00 to 122.3 g/t and averaging 0.48 g/t. The dataset is dominated by composites from RC holes which represent 95% of mineralized domain composites. Holes completed during 2015 provide around 2% of the combined dataset including 5% 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 drill hole 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 drill hole traces coloured by drilling type and phase.

Orca supplied surfaces representing the base of oxidation and the top of fresh rock interpreted from drill hole logging. These surfaces were used for flagging of the resource composites into oxide, transition and fresh subdomains, 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 metres and averages approximately 26 metres. The interpreted depth to fresh rock ranges from around 3 to 113 metres depth and averages approximately 55 metres.

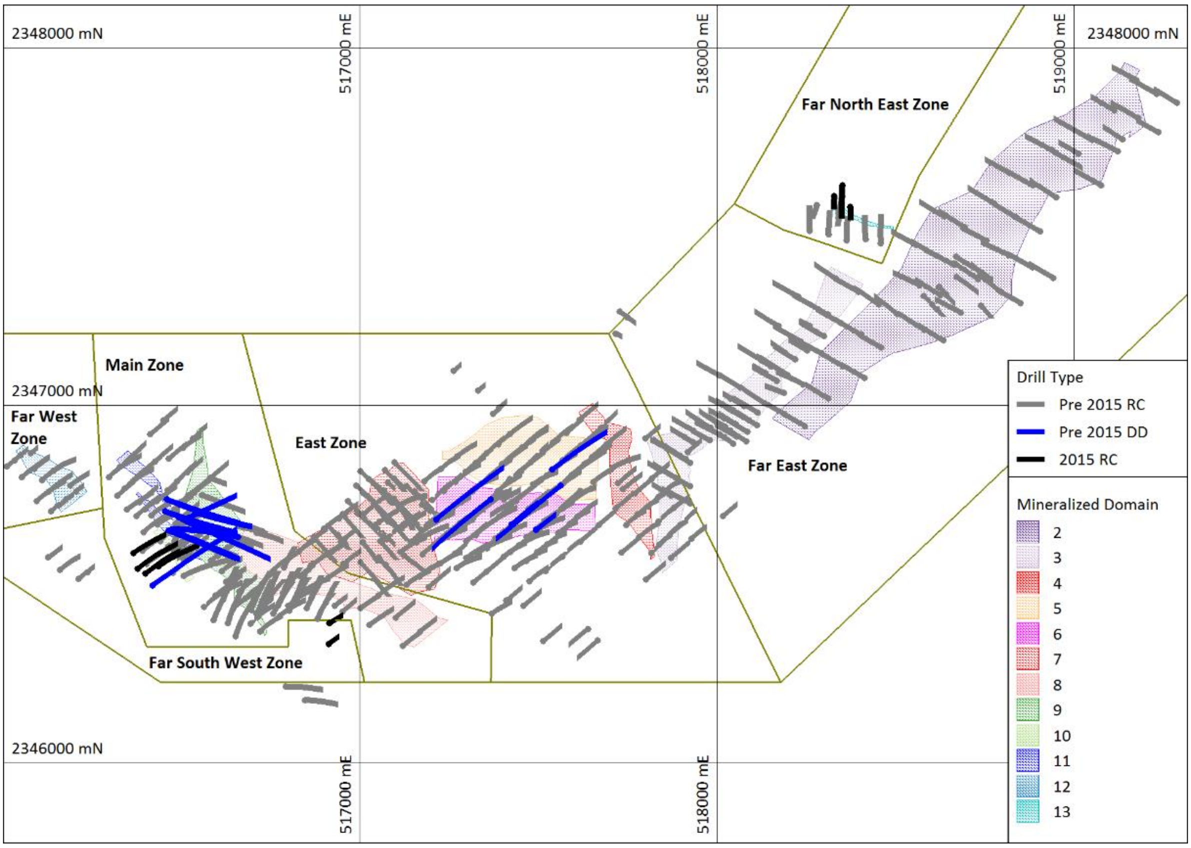


Figure 14-1: GSS mineralized domains and drill traces.

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 0.15 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	9,733	2,570	2,905	750	2,449	3,085	3,216
Mean	**0.08**	**0.30**	**0.49**	**0.43**	**0.69**	**0.88**	**0.50**
Variance	0.15	1.53	4.50	0.85	1.71	5.25	2.70
Coef. Var.	**4.77**	**4.08**	**4.34**	**2.15**	**1.91**	**2.60**	**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	0.06	0.13	0.26	0.37	0.14
3rd Quartile	0.04	0.13	0.39	0.40	0.77	1.06	0.45
Maximum	14.0	30.9	100.5	9.05	32.7	78.2	61.2
	Domain						
	8	**9**	**10**	**11**	**12**	**13**	**Total**
Number	1,572	2,983	256	938	363	95	30,304
Mean	**0.61**	**0.89**	**2.02**	**1.07**	**0.57**	**1.82**	**0.48**
Variance	12.3	5.69	35.8	7.28	5.53	11.5	3.50
Coef. Var.	**5.79**	**2.67**	**2.97**	**2.53**	**4.10**	**1.86**	**3.88**
Minimum	0.00	0.00	0.00	0.01	0.00	0.01	0.00
1st Quartile	0.02	0.04	0.06	0.06	0.01	0.16	0.01
Median	0.07	0.19	0.26	0.33	0.11	0.71	0.06
3rd Quartile	0.31	0.89	1.08	0.97	0.52	1.91	0.35
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 subdomains 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 subdomain 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 summarises 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	Subdomain	Upper bin (>99%ile) Au g/t			Source of bin grade
		Threshold	Maximum	Bin grade	
1	Oxide	0.785	4.995	1.255	Median
	Transition	1.370	4.735	2.125	Median exclud. comps. > 5.0 g/t
	Fresh	1.080	3.445	1.580	Median exclud. comps. > 5.0 g/t
2	Oxide	1.910	4.370	2.755	Median
	Transition	3.960	8.175	6.006	Mean
	Fresh	3.785	8.100	5.936	Mean exclud. comps. >10 g/t
3	Oxide	4.450	8.735	4.890	Median
	Transition	5.180	9.705	6.345	Median exclud. comps. >10 g/t
	Fresh	5.230	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	3.300	6.415	3.805	Median exclud. comps. >8 g/t
	Transition	5.740	8.755	7.055	Median exclud. comps. >10 g/t
	Fresh	5.280	9.635	6.135	Median
6	Oxide	5.110	9.960	6.090	Median
	Transition	4.590	6.425	5.365	Median exclud. comps. >8 g/t
	Fresh	6.120	19.580	8.975	Median exclud. comps. >20 g/t
7	Oxide	7.900	12.425	10.492	Mean
	Transition	4.145	17.500	9.861	Mean
	Fresh	4.590	8.190	6.600	Mean exclud. comps. >10 g/t
8	Oxide	6.605	9.080	7.992	Mean exclud. comps. >10 g/t
	Transition	7.650	11.230	9.585	Median exclud. comps. >16 g/t
	Fresh	5.810	14.175	9.110	Median
9	Oxide	10.935	20.625	14.470	Median
	Transition	10.870	48.700	17.945	Median
	Fresh	6.620	49.550	10.610	Median
10	All	7.005	19.975	18.450	Median exclud. comps. >20 g/t
11	Oxide	4.995	10.265	9.685	Mean
	Transition	15.675	7.140	5.515	Median
	Fresh	2.200	19.475	17.828	Mean exclud. comps. >20 g/t
12	Oxide	3.250	2.825	2.825	Median
	Transition	2.985	3.490	3.427	Mean exclud. comps. >4 g/t
	Fresh	9.285	8.315	8.315	Median
13	All	6.620	9.830	9.830	Mean exclud. comps. >10 g/t

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14.2.5 Variogram Models

Some less populous domains were combined for the purpose of variogram analysis. Table 14-3 summarises 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 Dom2&3
	Far North East	Use Dom8,10,11,12
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 Dom8,10,11,12

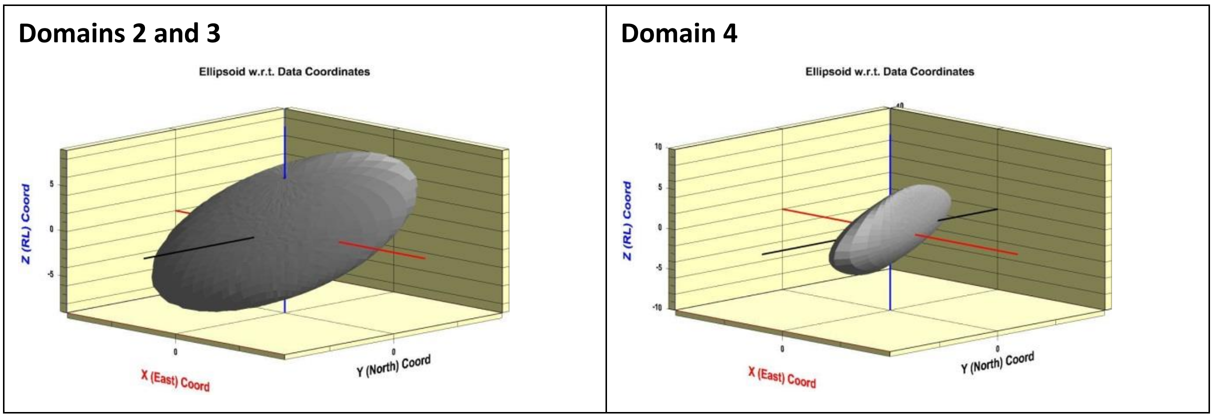


Figure 14-2: GSS three dimensional variogram plots.

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 metres east-west by 25 metres north-south by 5 metres vertical. The plan-view panel dimensions are consistent with drill hole 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 4 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)	Minimum	Minimum	Maximum
	(x,y,z)	Data	Octants	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 metres east by 5 metres north by 2.5 metres 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/	Information	Total
		Panel	Effect	Adjustment
1	Far West, Far SW, Main, East, Far East	0.177	0.633	0.112
	Far North East	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.

Panels within the Far West, Main, East, and Far East zones informed by search passes 1 and 2 are classified as Indicated. All other panels, including all panels informed by search passes 3 and 4, and all panels within the Far South West and Far North East zones are assigned to the Inferred category.

14.3 Estimation of WD resources

14.3.1 Resource dataset

Relative to the January 2015 dataset, the WD sampling database available for the current estimates incudes nine additional holes drilled during September and October 2015 comprising the following:

- 6 RC holes which infill central portions of the main mineralized domain.
- 1 RC hole testing the northern portion of the main mineralized domain.
- 2 diamond holes which infill central and southern portions of the main mineralized domain.

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

The resource dataset comprises 5,150 composites with gold grades ranging from 0.005 to 388.6 g/t and averaging 0.88 g/t. The dataset is dominated by composites from RC holes which represent 96% of mineralized domain composites. Holes completed during 2015 provide around 15% of the combined dataset, including 18% of composites within mineralized domains.

14.3.2 Geological Interpretation and Domaining

Drilling to date has delineated two distinct 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 drill hole 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 drill-hole traces coloured by drilling type and phase.

Orca supplied surfaces representing the base of oxidation and the top of fresh rock interpreted from drill hole logging. These surfaces were used for flagging of the resource composite into oxide, transition and fresh subdomains, density assignment and partitioning final resources by oxidation type.

Depth to the interpreted base of complete oxidation ranges from around 3 metres to approximately 55 metres and averages approximately 21 metres. The interpreted depth to fresh rock ranges from around 14 to 65 metres depth and averages around 28 metres.

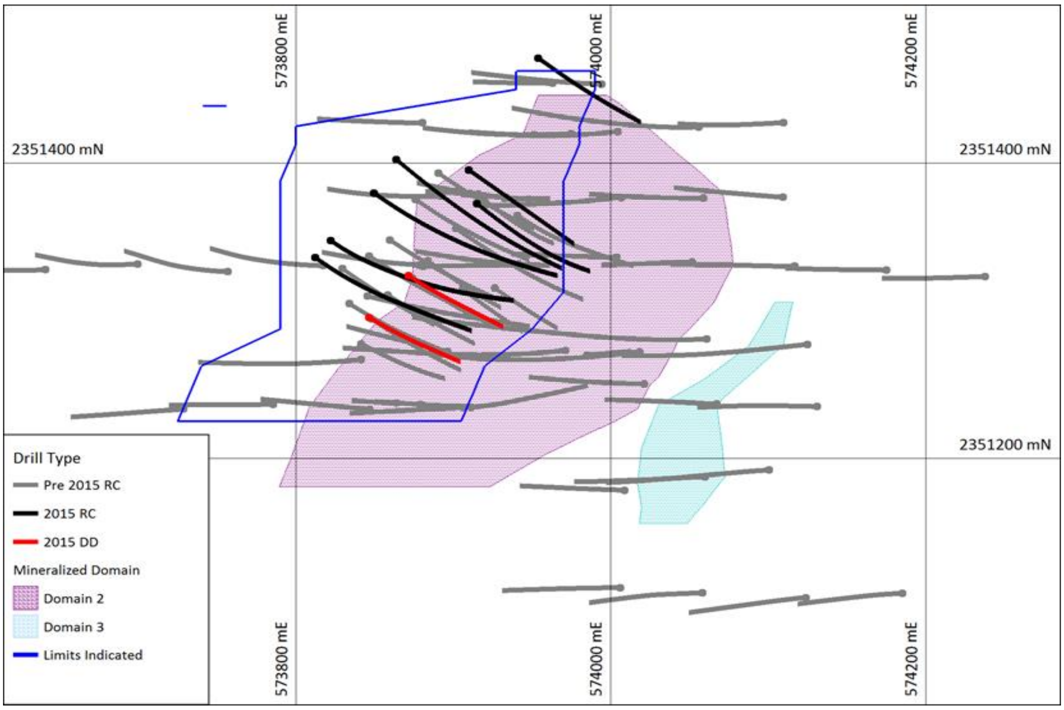


Figure 14-3: WD mineralized domains and drill traces.

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 of the 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 6.7 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 and 3**	**Total**
Number	2,013	3,000	137	3,137	5,150
Mean	**0.10**	**1.42**	**0.39**	**1.38**	**0.88**
Variance	0.05	90.0	0.22	86.2	52.9
Coef. Var.	**2.06**	**6.66**	**1.20**	**6.73**	**8.26**
Minimum	0.01	0.01	0.01	0.01	0.01
1st Quartile	0.02	0.11	0.10	0.11	0.05
Median	0.05	0.28	0.25	0.27	0.14
3rd Quartile	0.11	0.65	0.48	0.63	0.39
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 subdomains 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 summarises 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	Subdomain	Upper bin (>99%ile) Au g/t			Source of bin grade
		Threshold	**Maximum**	**Bin grade**	
1	Combined	0.98	5.270	0.505	97th percentile
2	Combined	19.40	388.6	30.43	Bin median
3	Combined	1.635	3.410	2.767	Bin mean

14.3.5 Variogram models

Domains 1 and 3 contain too few mineralised 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.

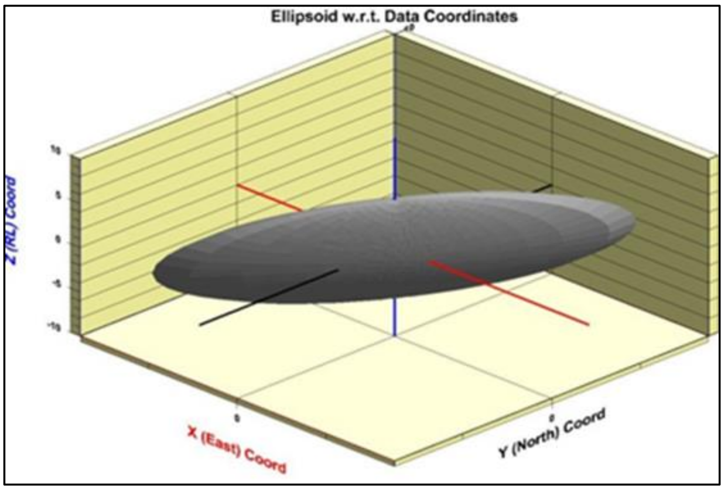


Figure 14-4: WD, Domain 2 three dimensional variogram plot.

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 metres east-west by 25 metres north-south by 5 metres vertical. The plan-view panel dimensions were selected on the basis of drill hole spacing in more closely drilled portion of the deposits.

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 metres east by five metres north by 2.5 metres 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)	Minimum	Minimum	Maximum
	(x,y,z)	Data	Octants	Data
1	37.5,37.5,10	16	4	48
2	50,50,13	16	4	32
3	50,50,13	8	2	32

Table 14-9: WD variance adjustment factors.

Domain	Block/	Information	Total
	Panel	Effect	Adjustment
1	0.270	0.635	0.171
2	0.270	0.635	0.171
3	0.270	0.635	0.171

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 mineralisation. Figure 14-3 shows the surface projection of this wire-frame.

Panels within the classification wire-frame informed by passes 1 and 2 are classified as Indicated. All other panels, including all panels informed by search pass 3 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 drill holes 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 drill holes. 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.

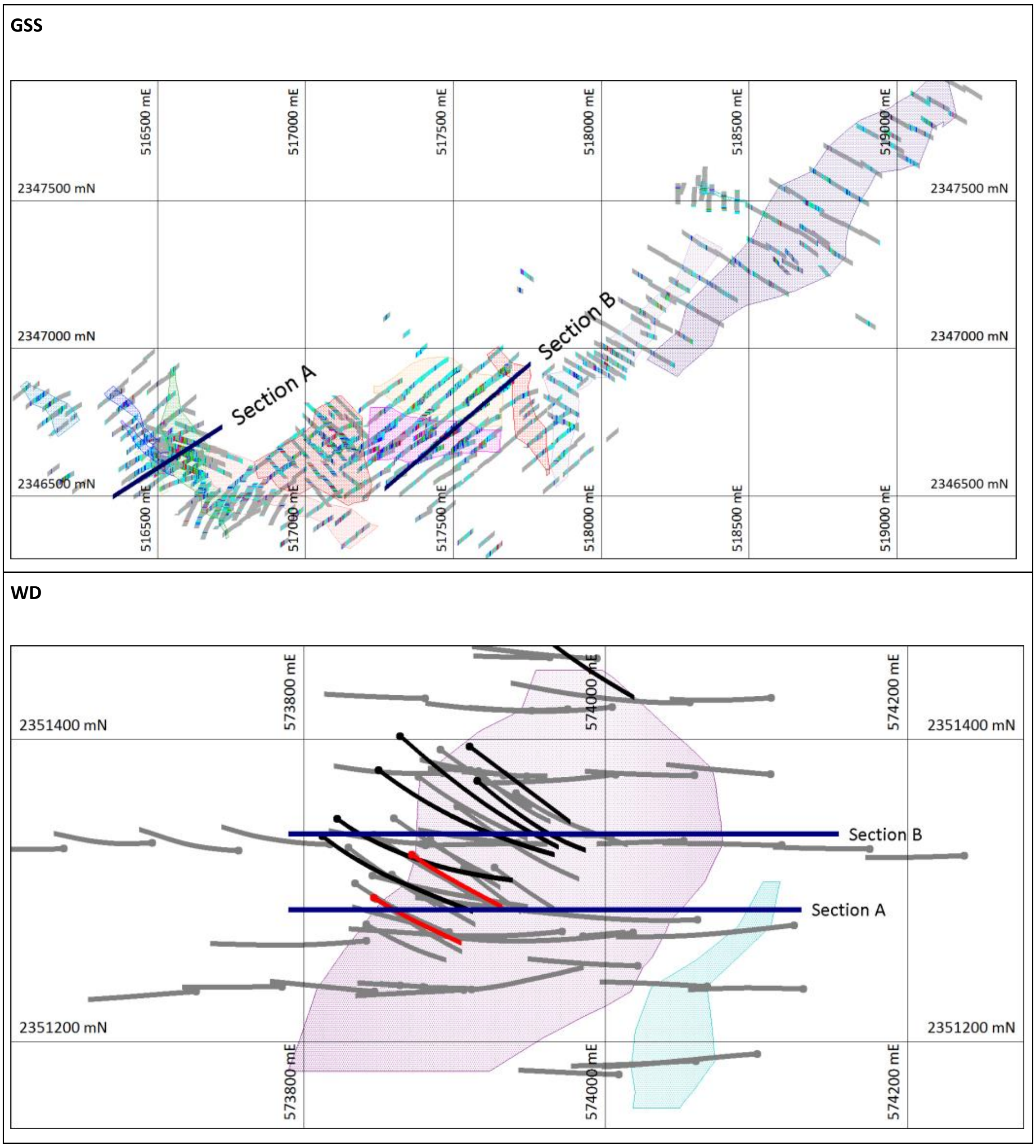
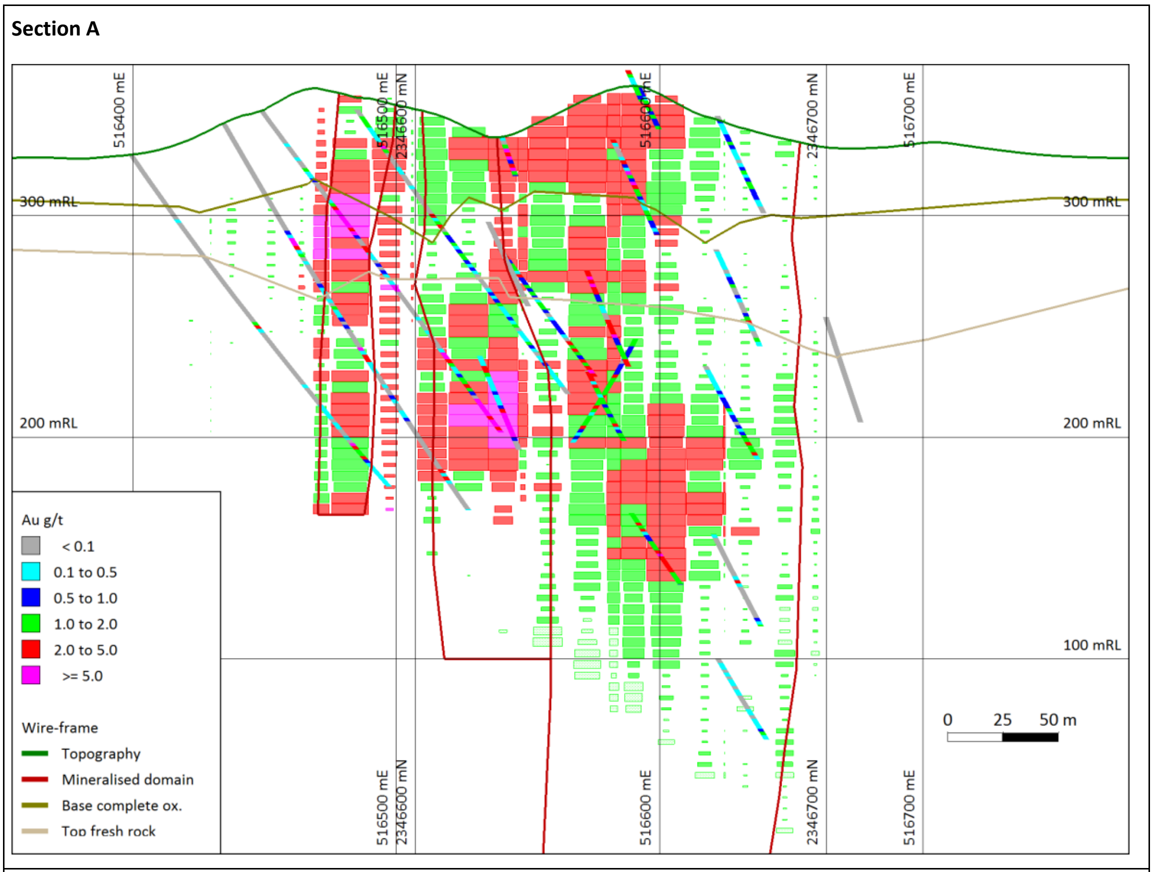


Figure 14-5: Plan view of section traces, top GSS South, bottom WD.



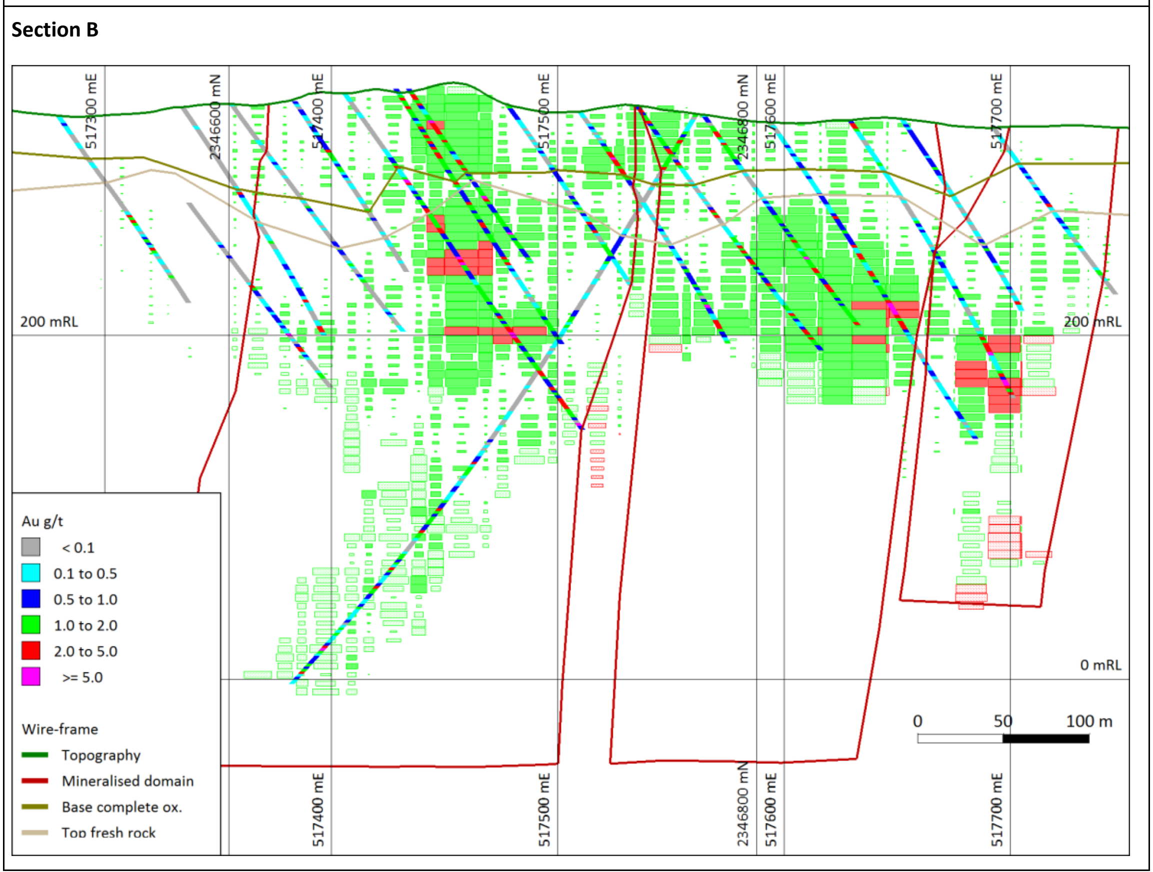
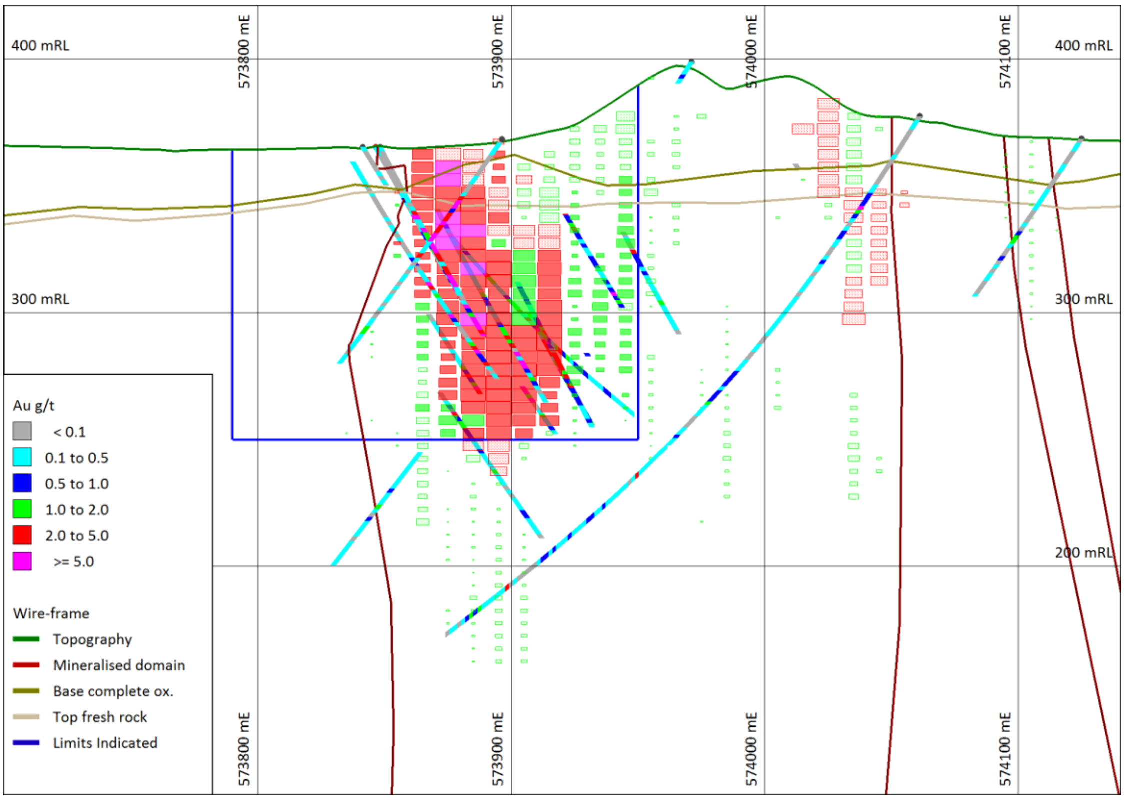


Figure 14-6: GSS resource model blocks at 1.0 g/t cut-off.

Section A



Section B

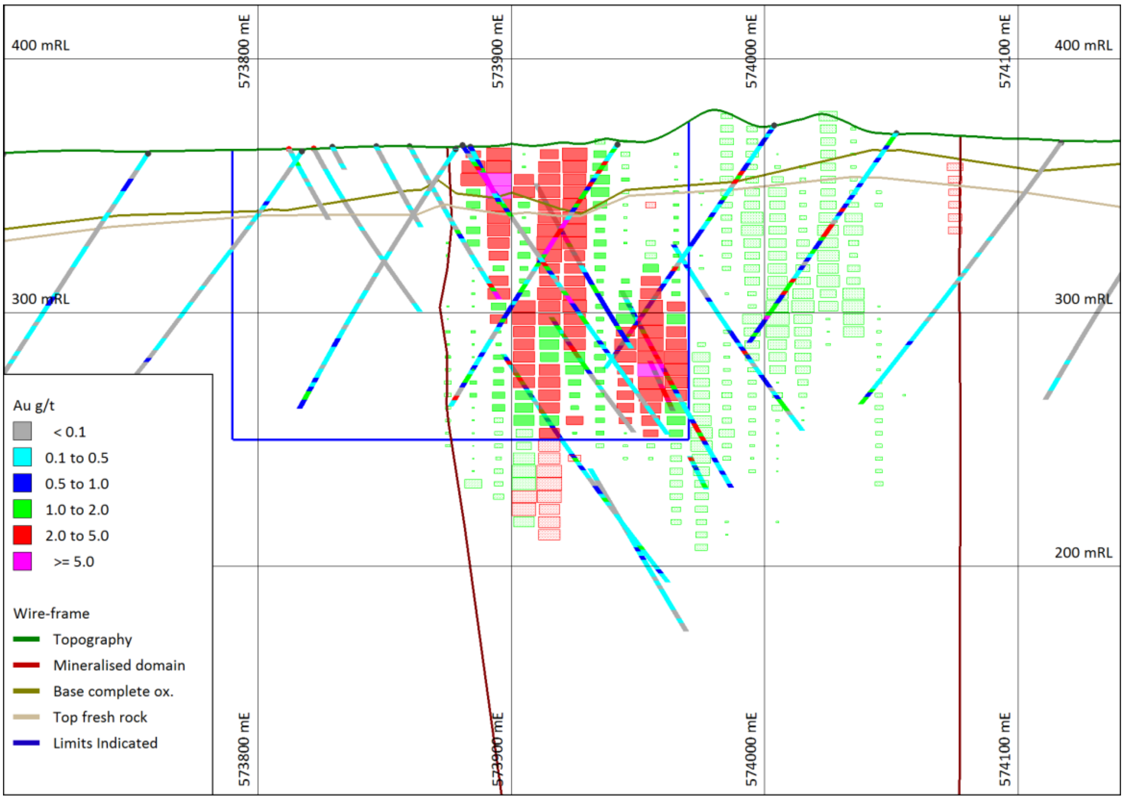


Figure 14-7: WD resource model blocks at 1.0 g/t cut-off.

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. For GSS composites from the Far South West and Far North East areas, where all estimates are classified as Inferred are excluded from the swath plots. 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, as expected the average block grades estimated by MIK are generally 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 due to 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; and
- 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.

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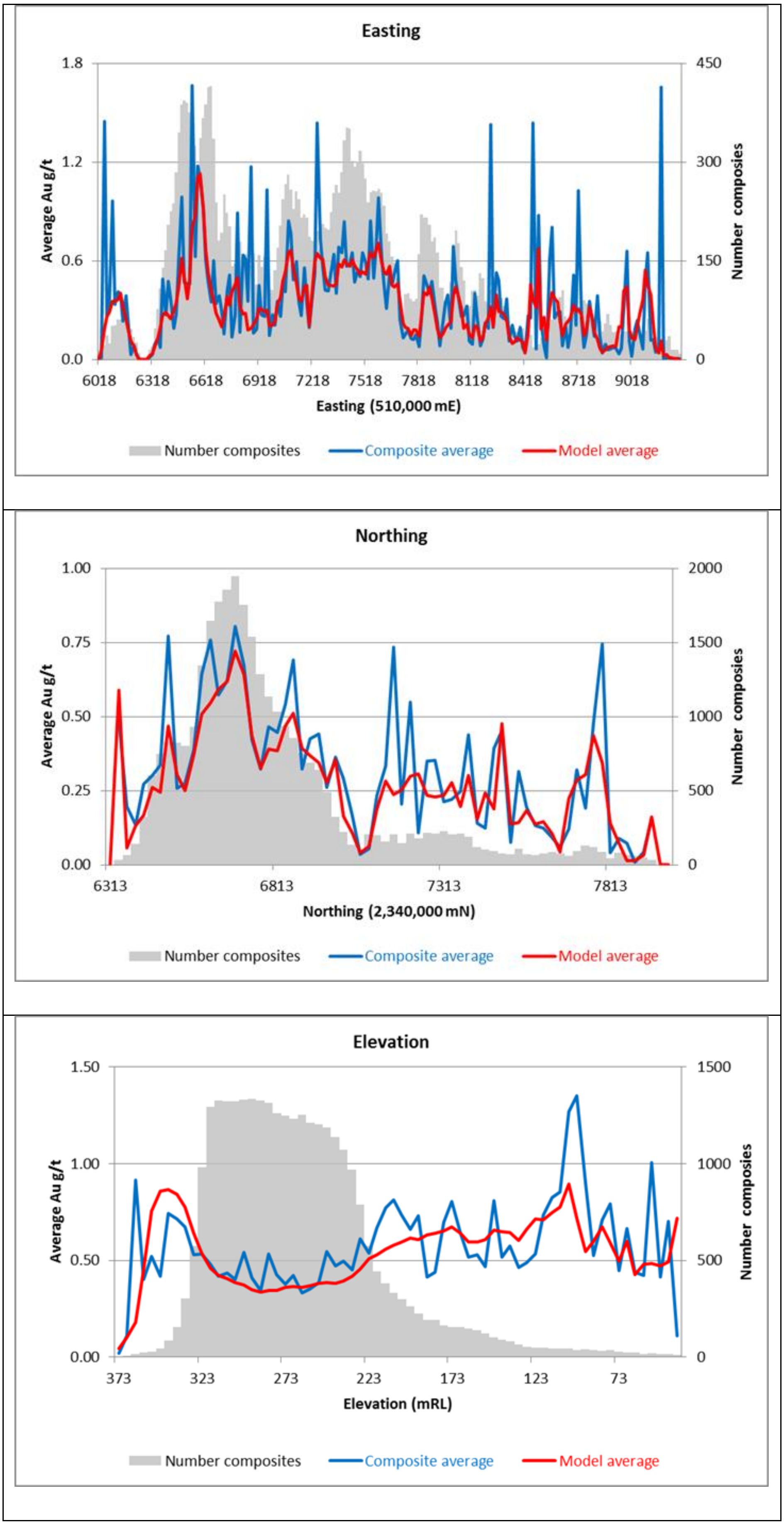


Figure 14-8: GSS average estimated panel grades versus composite grades.

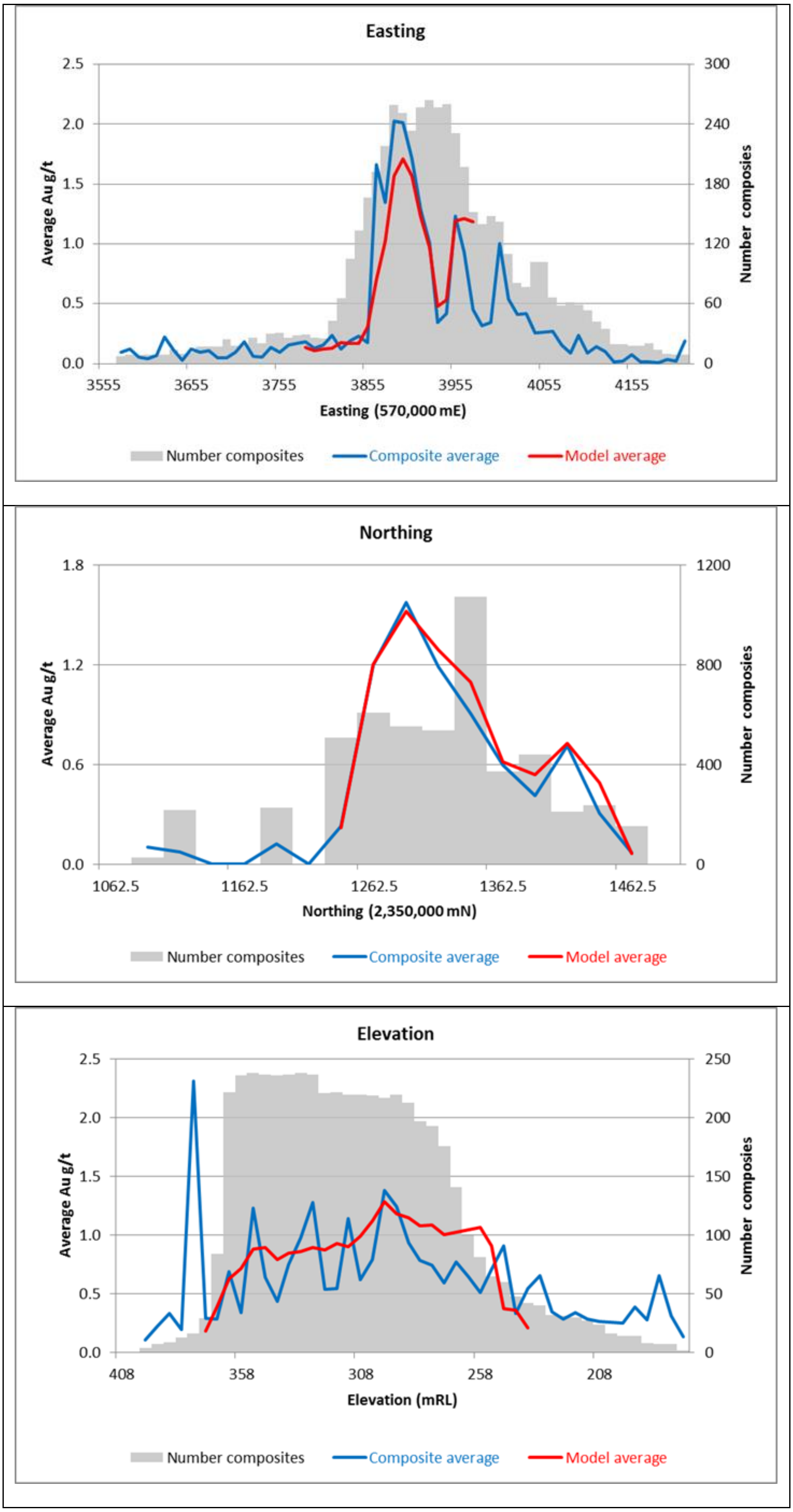


Figure 14-9: WD average estimated panel grades versus composite grades.

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 metres depth with around 90% of the Indicated Resources, and 70% of the Inferred resources occurring at depths of less than 160 metres. The WD estimates extend to around 210 metres depth with around 90% of the Indicated Resources, and 70% of the Inferred resources occurring at depths of less than 100 metres.

For GSS the combined oxidised and transitional material hosts around 38% and 20% of the Indicated and Inferred resources respectively with the remainder lying in fresh rock. For WD the combined oxidised and transitional material hosts around 15% and 21% of the Indicated and Inferred resources respectively.

Table 14-11: February 2016 Mineral Resource Estimates.

GSS						
Cut off	Indicated			Inferred		
Au g/t	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz
0.2	109	0.79	2,769	52	0.7	1,170
0.4	72.6	1.04	2,428	30	1.0	965
0.5	60.7	1.16	2,264	25	1.1	884
0.6	50.9	1.27	2,078	20	1.2	772
0.8	36.3	1.51	1,762	14	1.4	630
1.0	**26.3**	**1.74**	**1,471**	**10**	**1.7**	**547**
1.2	19.3	1.97	1,222	7	1.9	428
1.5	12.3	2.33	921	4	2.3	296

WD						
Cut off	Indicated			Inferred		
Au g/t	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz
0.2	3.31	1.48	158	12.5	0.6	241
0.4	2.42	1.91	149	5.6	1.0	180
0.5	2.15	2.10	145	4.2	1.2	162
0.6	1.94	2.27	142	3.3	1.3	138
0.8	1.60	2.60	134	2.1	1.7	115
1.0	**1.36**	**2.91**	**127**	**1.4**	**2.1**	**95**
1.2	1.17	3.19	120	1.0	2.4	77
1.5	0.96	3.59	111	0.7	3.0	68

Combined						
Cut off	Indicated			Inferred		
Au g/t	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz
0.2	112	0.81	2,926	64.5	0.7	1,411
0.4	75.0	1.07	2,576	35.6	1.0	1,145
0.5	62.9	1.19	2,409	29.2	1.1	1,046
0.6	52.8	1.31	2,220	23.3	1.2	910
0.8	37.9	1.56	1,896	16.1	1.4	745
1.0	**27.7**	**1.80**	**1,599**	**11.4**	**1.7**	**641**
1.2	20.5	2.04	1,342	8.0	2.0	505
1.5	13.3	2.42	1,032	4.7	2.4	363

Table 14-12: February 2016 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	5.3	1.79	305	1.1	1.6	57
Transition	4.7	1.76	266	0.8	1.6	41
Fresh	16.3	1.72	901	8.1	1.7	443
Total	**26.3**	**1.74**	**1,471**	**10**	**1.7**	**547**

WD						
Material	Indicated			Inferred		
	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz
Oxide	0.12	3.26	13	0.2	2.7	17
Transition	0.08	3.11	8	0.1	2.3	7
Fresh	1.16	2.86	107	1.2	2.0	77
Total	**1.36**	**2.91**	**127**	**1.4**	**2.1**	**95**

Combined						
Material	Indicated			Inferred		
	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz
Oxide	5.4	1.82	318	1.3	1.8	74
Transition	4.8	1.78	274	0.9	1.7	49
Fresh	17.5	1.80	1,008	9.3	1.7	520
Total	**27.7**	**1.80**	**1,599**	**11.4**	**1.7**	**641**

15 MINERAL RESERVE ESTIMATES

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

16 MINING METHODS

16.1 Geotechnical Considerations

16.1.1 Introduction and background

A review of existing geotechnical data was conducted by SRK Consulting (UK) Limited as part of the preliminary economic assessment of the Block 14 gold concession. The report from this review has been used as a basis for subsequent optimisation and pit design work discussed in this study.

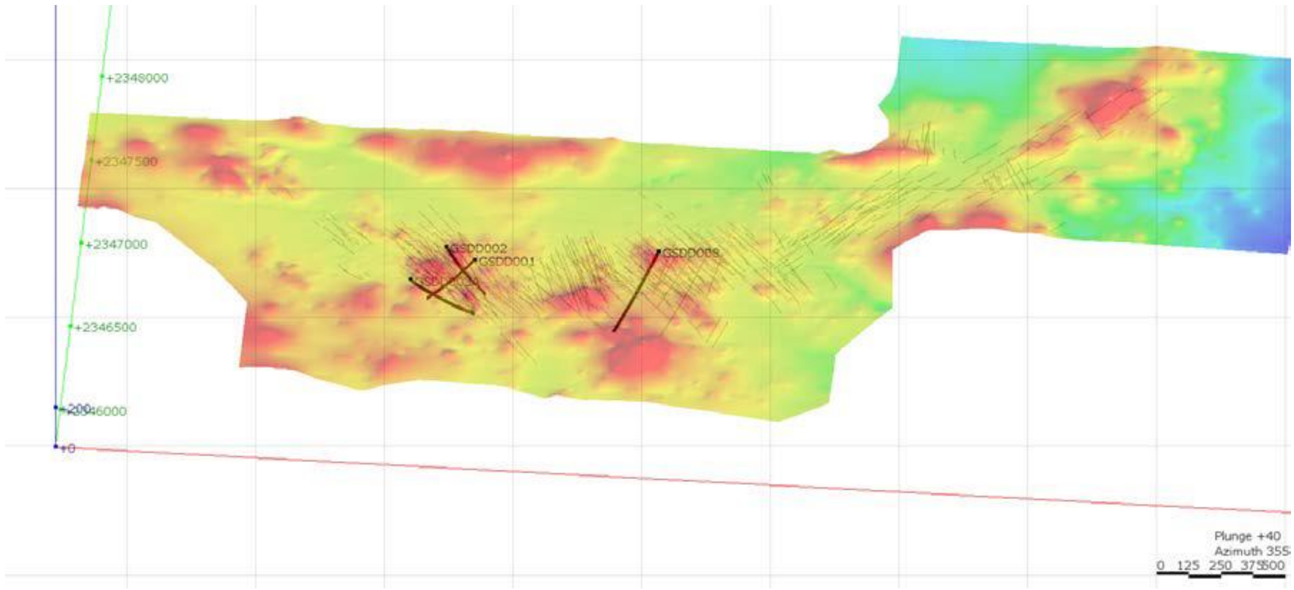
Basic geotechnical data from 18 cored boreholes were provided for the review. These consisted of lithology, rock strength, RQD, weathering state and rock fabric. A structural database was also provided, gathering mainly data from the vein and foliation orientation. At this stage, no systematic logging of natural open joint orientations and conditions has been recorded.

16.1.2 Development of Geotechnical Characterisation Parameters

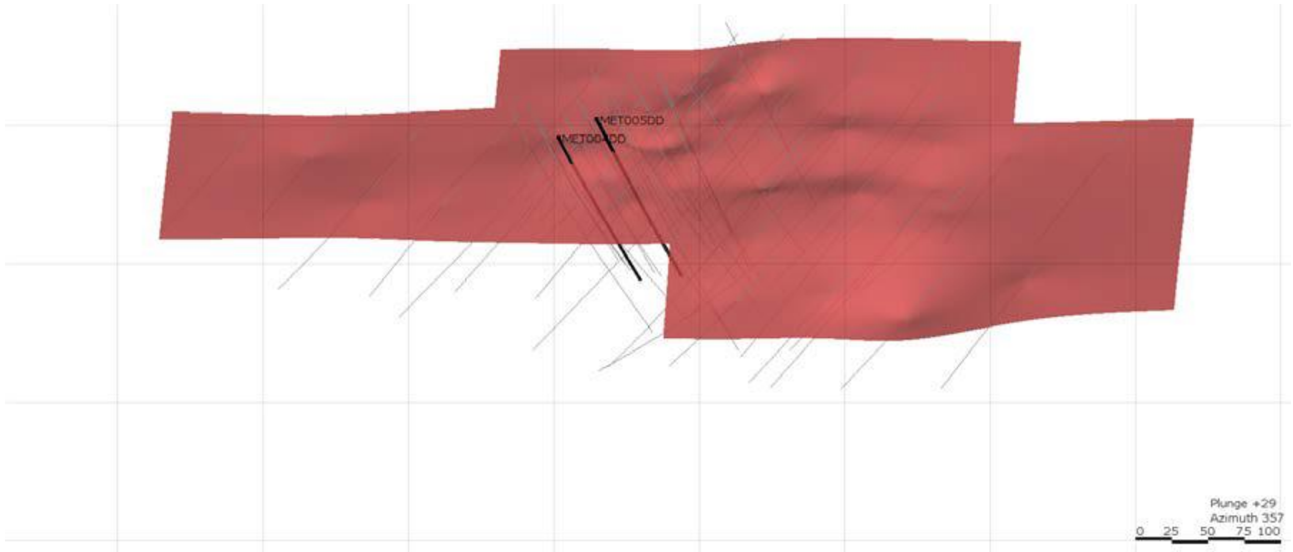
SRK calculated Bieniawski rock mass rating (RMR) values from the core photographs of six boreholes spread across WD and GSS. These six holes comprised over 1,500 m of borehole core. The boreholes considered in the present study are presented on Figure 16-1 and in Table 16-1.

Table 16-1: RMR logged borehole

BHID	X	Y	Z	Logged Length
GSDD001	516659.5	2346632	331.982	300
GSDD002	516542.0	2346701	330.712	255
GSDD003A	516419.3	2346497	328.537	408
GSDD008	517366.7	2346737	333.094	375
MET004DD	573846.5	2351295	365.346	120
MET005DD	573871.6	2351324	365.554	130



GSS



WD

Figure 16-1: Borehole Positions

The following five parameters are used to classify the rock mass using Bieniawski RMR system:

- Intact rock strength (IRS);
- Rock quality designation (RQD);
- Spacing of discontinuities;
- Conditions of discontinuities (large and small scale roughness, persistence, alteration, fill and aperture); and
- Groundwater conditions.

Each of the five parameters is assigned a value corresponding to the characteristics of the rock. The sum of the five parameters is the RMR value, which lies between 0 and 100. The following assumptions were made for the photo-logging:

- The ground is dry;
- The joint aperture is set to <0.1 mm unless clearly observed on the core photograph;
- The joint persistence is 3-10 m for foliated rock and 1-3 m for massive rock;
- The joint weathering is directly related to the rock mass weathering state; and
- The joint infill strength is defaulted as hard.

Figure 16-2 presents the distribution of RMR values with depth. Figure 16-3 presents the logged RMR89 values distribution according to RMR classes. A summary of the rock mass data generated from photo-logging of these holes is presented in Table 16-2. Statistics on intact rock strengths logged by the Client's geologists are presented on the histogram, Figure 16-4.

Only two boreholes were photo-logged for WD. No distinction between the rock mass quality at WD and GSS was observed.

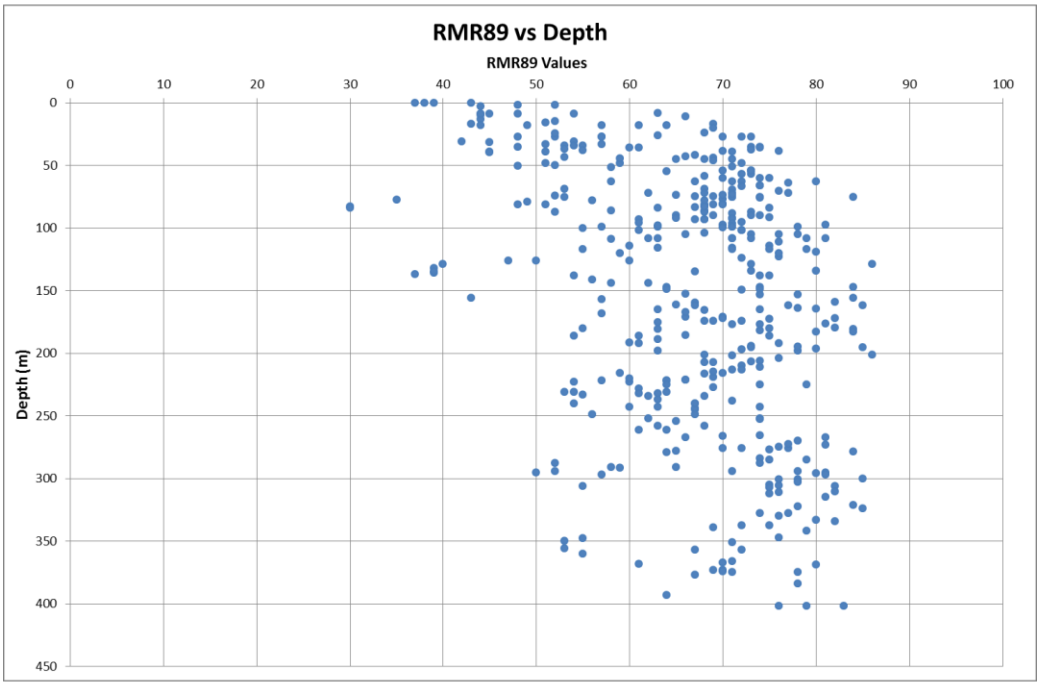


Figure 16-2: RMR89 vs Depth

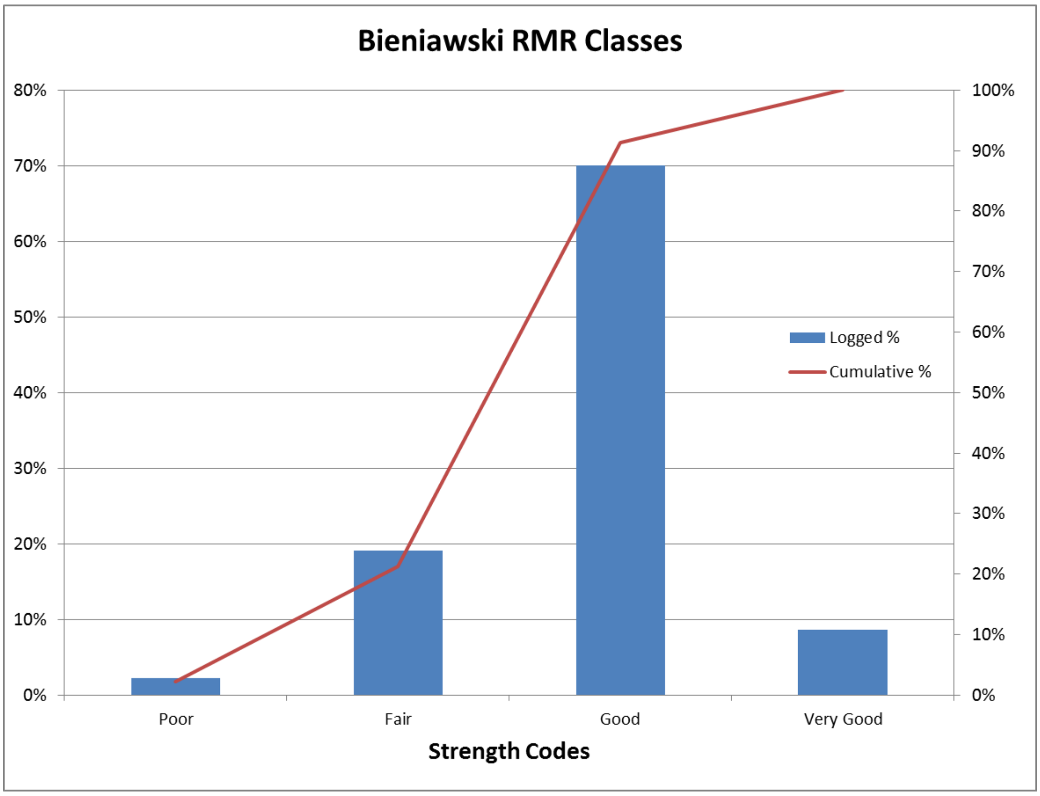


Figure 16-3: Logged RMR89 class distribution

Table 16-2: Rock mass logging summary

Lith Code	Lith Description	Logged Length (m)	Percentage of Total	Weighted RMR Average	Min	Max	Std Dev
QSP	Quartz sericite pyrite schist	599	37.75%	66	30	86	11.5
MDI	Sheared Altered Diorite	384	24.23%	69	44	86	9.8
QSS	Quartz sericite schist	187	11.83%	62	37	82	10.3
KDI	Potassium altered diorite	187	11.79%	67	42	80	8.8
BXH	Breccia - Hydrothermal	81	5.14%	78	66	85	4.2
VQ	Vein - Quartz	50	3.15%	66	35	85	12.7
VAN	Andesite	43	2.74%	66	43	74	8.3
BRD	Black red diorite	25	1.55%	68	60	75	4.8
MDY	Mafic dyke	12	0.74%	65	39	83	11.6
IDI	Diorite	10	0.63%	68	64	78	5.7
BXV	Breccia volcanic	7	0.44%	65	54	73	9.5
Average		1,585	100%	67	47	81	9.0

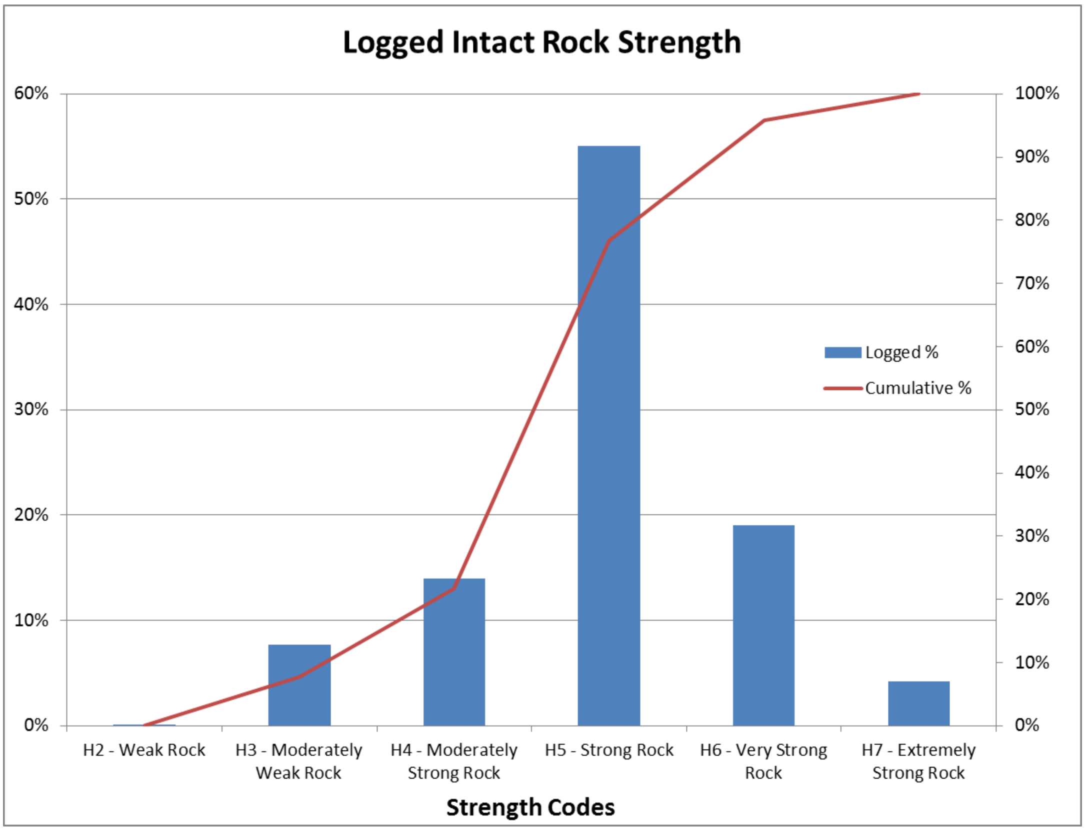


Figure 16-4: Logged intact rock strength

The rock mass data indicates that the fresh rock units can mostly be classified as fair to good rock, a typical "good" core example is shown on Figure 16-5. As observed on Figure 16-2 and Figure 16-3, a few intervals have lower RMR values, characterising poor and fair rock masses. These weaker zones at depth could illustrate potential fault intersections. Figure 16-6 gives an example of a weaker rock mass interval.

Figure 16-5: GSDD002 – Typical "Good" rock mass

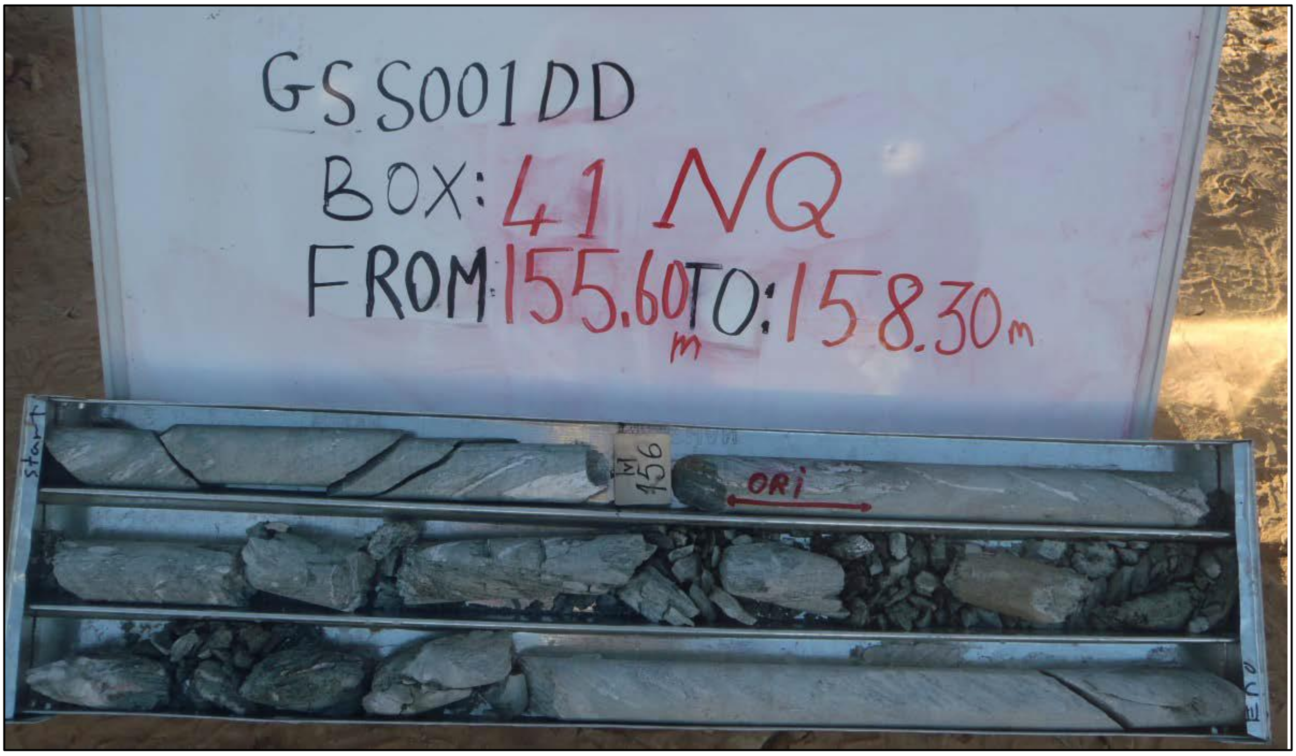


Figure 16-6: GSDD001 – Typical "poor to Fair" rock mass

There is a limited range of average RMR values between the rock types indicating that the fresh rock mass, for the PEA purposes, can be considered as a single geotechnical unit with a weighted RMR average of approximately 67. Most of the rock mass (55 %) is logged by the geologists as a strong rock, belonging to the H5 category, corresponding to an intact rock strength comprised between 50 and 100 MPa.

16.1.3 Preliminary Rock Mass Stability Analysis

For the inter-ramp stability analysis SRK has invoked the Generalised Hoek Brown empirical strength criterion. This requires as input the intact rock strength and the geologic strength index (GSI) of the lithologies being modelled. SRK has taken the GSI value to be equivalent to RMR-5, which is a reasonable approximation of GSI for RMR values greater than 40. Considering the low confidence in strength data, SRK has used the lower bound intact rock strength parameter in the stability analyses. The rock mass input parameters used for this sensitivity analysis are presented in Table 16-3.

Table 16-3: Design rock mass properties

Material	Density	UCS (Mpa)	GSI	mi	D
Fresh Rock	2.8	50	62	20	0.7

Sensitivity analyses with respect to inter-ramp slope angle and stack height have been carried out on a generic cross section formed within the fresh rock material. Using these rock strength input parameters, the results of the analyses for 50, 100, 150, and 200 m high inter-ramp stack cut at an angle to obtain a design factor of safety of 1.3 are presented graphically in Figure 16-7 below.

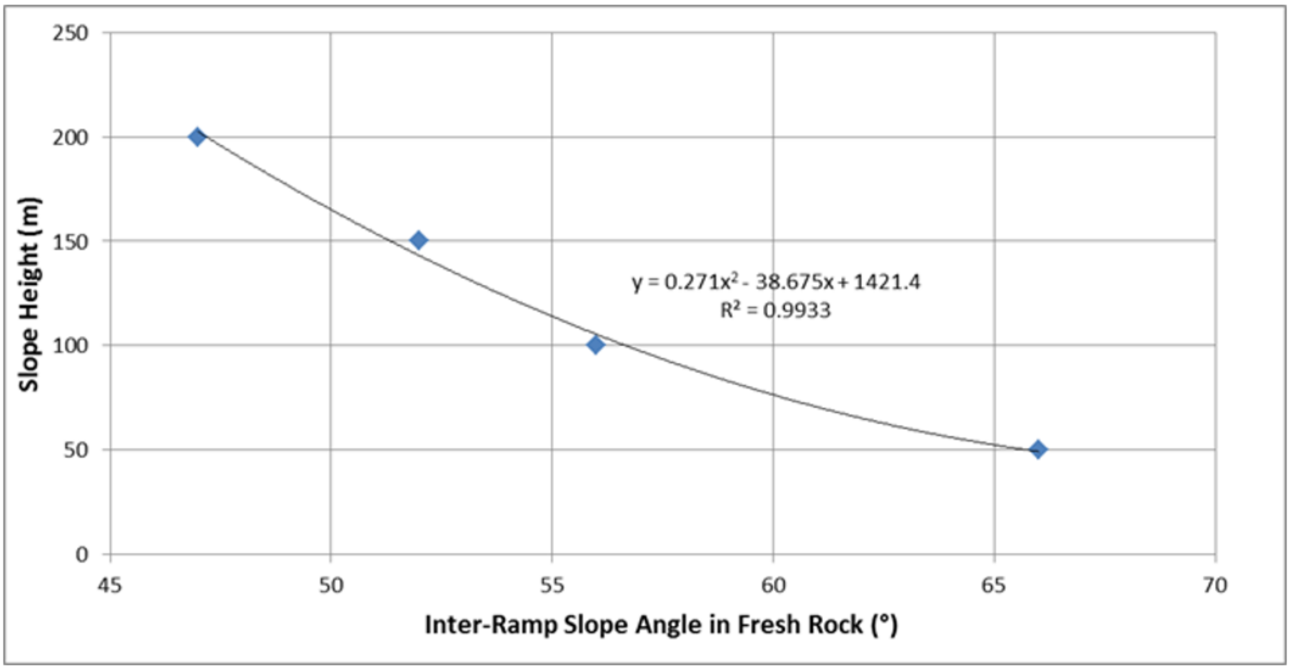


Figure 16-7: Results of inter-ramp slope stability analyses in fresh rock

Approximately 3% of the rock mass has been logged as W4, corresponding to a highly weathered rock. Rock mass within this overburden zone is weaker and presents an average RMR value of 46. Oxide overburden should not be cut at an angle steeper than 35°.

16.1.4 Structural Conditions

A structural database was provided to SRK as part of the review data. This comprises measurements of foliation, contact, veins and joints from orientated drill cores. Data from the West zone of GSS, recorded from boreholes GSDD001, GSDD002, GSDD003A, GSDD004, GSDD005, GSDD006, GSDD009, GSDD010, GSRD082 and GSDD011A have been plotted together and presented in Figure 16-8. Data from GSS East, collected from boreholes GSDD012, GSDD008, MET001DD, MET002DD and GSDD007A are presented on Figure 16-9. Both data sets show significant differences concerning the foliation orientation, dipping roughly sub vertically to the northeast (79/038 on average) for the eastern part and to the northwest (79/283 on average) for the western part. This could be interpreted as the presence of an anticline fold at GSS. It was not possible to isolate separate joint sets from the given dataset as most of the logged joints follow the foliation trend.

About 84% of the rock mass logged at GSS has been described as foliated. The foliation orientation relative to the pit wall will play a vital role on the overall wall stability and hence will have to be investigated further once the pit position is more precisely known. Even though this preliminary assessment shows that the foliation should be dipping towards the eastern and western pit wall, presenting the most favourable scenario, the foliation could potentially trigger the formation of major wedges on the southern and northern pit wall. The orientation and variability of the foliation across site and at the future pit wall location will therefore have to be studied closer.

For WD, only joint information has been recorded from MET004DD and MET005DD, as presented on Figure 16-10. No foliation orientation was taken from these two boreholes and yet 46% of the logged rock mass has been described as foliated. A number of sub perpendicular structures are observed on Figure 16-10 that could be responsible for potential toppling type instabilities depending on the joint persistence and spacing, parameters which are currently unknown.

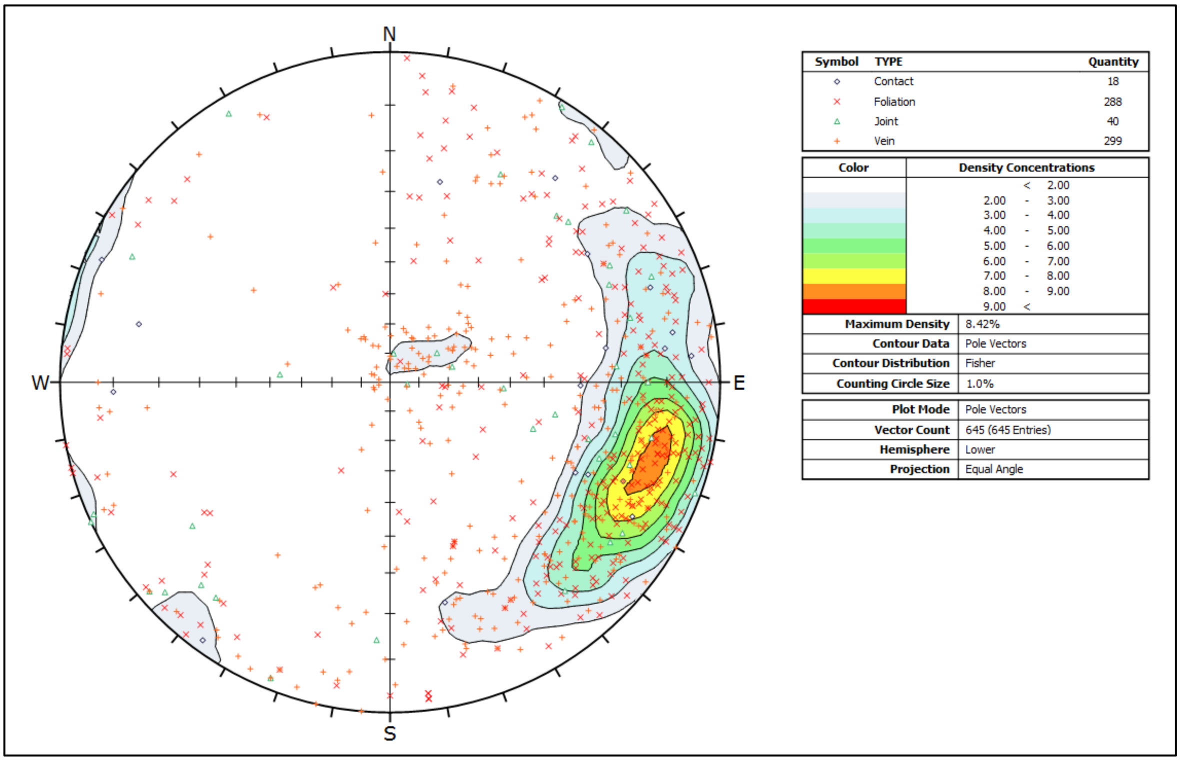


Figure 16-8: Structural data from GSS-West

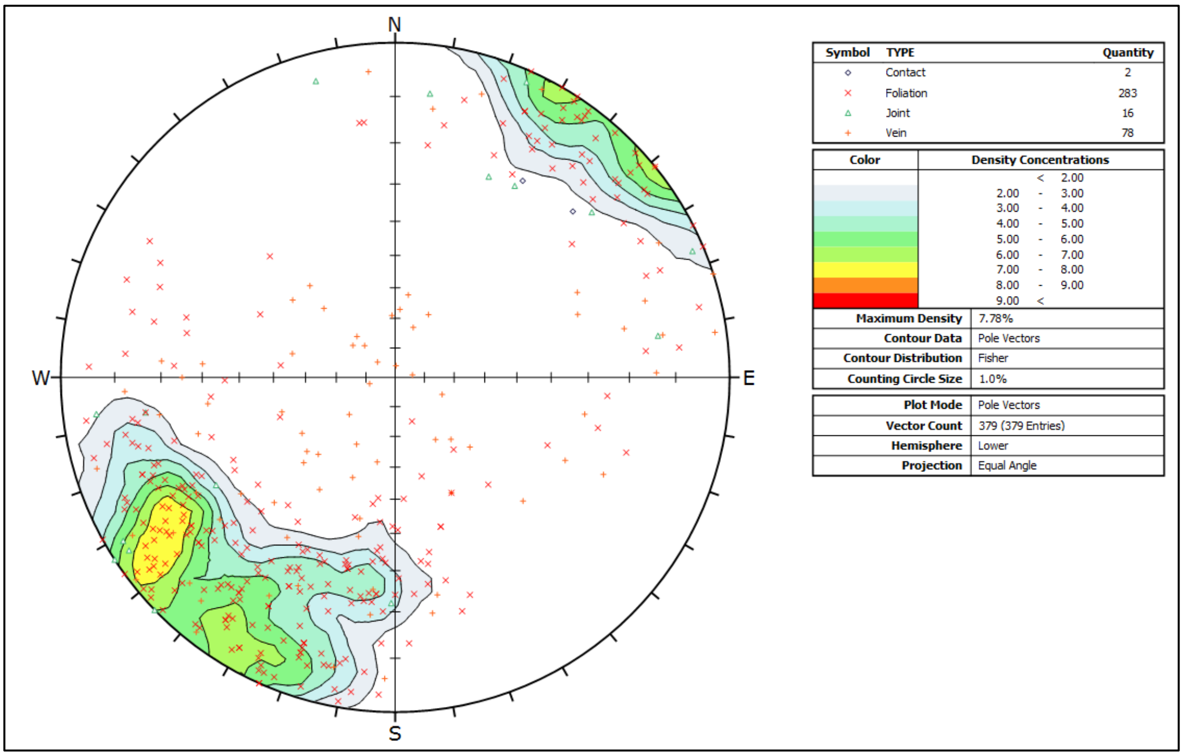


Figure 16-9: Structural data from GSS-East

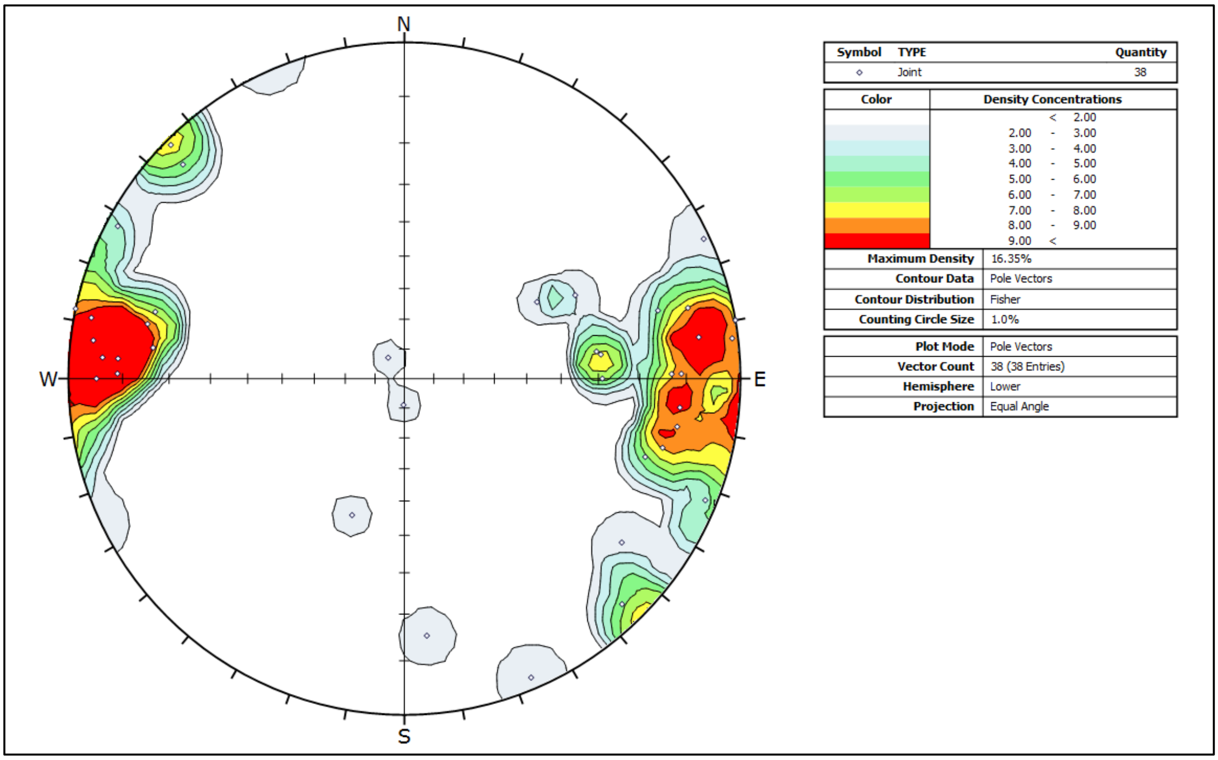


Figure 16-10: Structural data from WD

16.1.5 Geotechnical Conclusions and Recommendations

Because of limited geotechnical data, a high level of conservatism has been placed on the recommendations for this geotechnical assessment. For optimisation purposes pit slope angles should not exceed 35° in the near surface oxide zone. Inter-ramp pit slopes in the fresh rock should not exceed the recommended angles plotted in Figure 16-7 for both GSS and WD deposits.

This study did not include any information related to the structural orientation. Orientation data are necessary for kinematics analyses in order to determine the potential for planar, toppling and wedge failures on a bench scale and therefore give a detailed bench and berm configuration.

No large scale structural information was available for this assessment; however potential fault zones have been observed on core photographs. These zones will have to be investigated further as their relative orientation to the pit wall could have a crucial impact on the wall stability.

A more detailed geotechnical investigation programme would be required for a PFS including:

- Drilling of orientated boreholes;
- Confirmation of groundwater conditions;
- Detailed geotechnical logging of the core;
- Laboratory strength testing of selected core samples;
- Development of a geotechnical domain model;
- Undertaking more detailed rock mass and kinematic stability analysis; and
- Determination of large scale structures.

16.2 Hydrogeology and Mine Dewatering

Groundwater was only encountered in one hole during exploration drilling at either deposit. With only rare annual precipitation in the region, there is no requirement to consider routine mine dewatering. Refer to "Groundwater Exploration and Groundwater Resource Assessment for the GSS Project, Sudan – Block 14 Area Summary Report for PEA Supplement, dated July 2016. GCS Water & Environmental Consultants", and a summary of the report in Section 18.1.3

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 130 metres deep. The GSS deposit will be exploited by seven separate pits, five of which are shallow oxide/transitional pits only. The other two are deeper than the WD pit at 150 and 205 metres.

While the mineralisation 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.

16.4 Mine Optimisation

16.4.1 Cut-off grade determination

The original block model was a partial percentage or proportional model. Grade bins were spaced at 0.1 g/t intervals from 0.2 g/t to 1.0 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 optimisations.

Cut-off grades were calculated for oxide, transitional and fresh material for WD, GSS Main and GSS East zones. In the case of WD, the option of processing locally or at the GSS CIL plant was also considered. Table 16-4 to Table 16-6 below show the parameters used for the cut-off grade calculation while Table 16-7 shows the calculated cut-off grades and associated bins used within the model to determine tonnes and grade of potential crusher feed.

Table 16-4: Selling Parameters

Item	Value
Gold Price	\$1,100/oz
Refining & Selling Cost and Royalty	\$82/oz

Table 16-5: Processing Recoveries

Pit	Material	% Recovery
GSS	Oxide	92
	Transitional	87
	Fresh	80
WD	Oxide	92
	Transitional	87
	Fresh	83

Table 16-6: Processing Costs ($/t processed)

	WD (local)	GSS Main (WD remote)	GSS East
Oxide (Heap Leach)	N/A	$9.84	$9.84
Oxide (CIL)	$23.39	$17.83	$17.83
Transitional	$23.51	$17.92	$17.92
Fresh	$25.10	$19.13	$20.71

Haulage for WD material to the GSS ROM pad has been estimated at $8.48/tonne of crusher feed.

Table 16-7: Cut-off grades and grade bins

	WD (local)		WD (remote)		GSS Main		GSS East	
	Cut-off	Model Bin	Cut-off	Model Bin	Cut-off	Model Bin	Cut-off	Model Bin
Oxide (Heap Leach)	N/A	N/A	N/A	N/A	0.4	0.4	0.4	0.4
Oxide (CIL)	0.78	0.8	0.87	0.9	0.59	0.6	0.59	0.6
Transitional	0.83	0.8	0.93	0.9	0.63	0.6	0.63	0.6
Fresh	0.96	1.0	1.05	1.0	0.73	0.7	0.79	0.8

16.4.2 Cost data for optimisation

It is 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.

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

Unit costs were determined for the following items:

- Loading;
- Fixed hauling component;
- Drill & Blast;
- Ancillary; and
- 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 optimisation process to account for vertical haulage; see section 0 for a breakdown of the cost estimates.

16.4.3 Optimisation Scenarios

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

16.4.3.1 GSS

There were two main scenarios examined for the GSS deposit, based around the treatment of the oxide. The base case was to treat oxide through a CIL plant, along with the transitional and fresh material. An alternative case was proposed where the oxide would be treated using a heap leach plant prior to the commencement of the CIL processing of transitional and fresh material.

Table 16-8 below shows the comparison between the base case and the Heap Leach case, considering the pit shell with revenue factor 1.

Table 16-8: Comparison of GSS optimisation scenarios

	Base Case (CIL)	Heap Leach Case	Difference
Oxide tonnes (million)	10.5	14.4	-3.6
Oxide grade	1.37	1.15	0.22
Transitional tonnes (million)	8.2	8.4	-0.2
Transitional grade	1.33	1.32	0.01
Fresh tonnes (million)	12.2	12.3	-0.1
Fresh grade	1.61	1.61	0
Total crusher feed tonnes (million)	30.8	35.0	-4.2
Total crusher feed grade	1.46	1.35	0.11
Waste tonnes (million)	55.7	54.6	1.1
Total mined tonnes (million)	86.5	89.6	-3.1
Strip Ratio	1.81	1.56	0.25
Total cost (million)	$851.3	$821.8	$29.5
Revenue (million)	$1,261.1	$1,242.9	$18.2
Value (million)	$409.8	$421.1	-$11.3
Recovered Ounces	1,232,762	1,214,921	17,841

Despite the fact that the Heap Leach option has a lower processing cost for oxide, and generates more crusher feed tonnes, the relative processing recoveries for heap leach and CIL means that more ounces are generated from the CIL option. For this reason, the Revenue Factor = 0.95 shell from the CIL optimisation was chosen as the basis for design.

16.4.3.2 Wadi Doum

The WD optimisation considered the use of a local or remote processing plant as two options.

Table 16-9 provides a comparison of the WD optimisations (examining the Revenue Factor = 1 shells) while Figure 16-11 shows the undiscounted value of the two scenarios. The effect of the $8.48 transport cost can be clearly seen by the fact that the scenarios using the GSS plant have lower values.

The decision was made to use the GSS processing plant for WD material. The additional operating margin from a local processing plant is insufficient to cover the additional capital costs. In addition, benefits can be gained by treating the WD feed at a higher throughput in the GSS plant providing a grade boost to GSS feed while the GSS pits are developed. The Revenue Factor = 0.95 shell for the GSS plant scenario has been chosen as the basis of design for WD.

Table 16-9: Comparison of WD Optimisation Scenarios

	Local Plant	GSS Plant
Oxide tonnes ('000)	539.8	348.1
Oxide grade	2.01	2.72
Transitional tonnes ('000)	271.5	172.7
Transitional grade	1.93	2.59
Fresh tonnes ('000)	2,424.9	1,530.8
Fresh grade	2.15	2.74
Total feed tonnes ('000)	3,236.1	2,051.7
Total crusher feed grade	2.11	2.73
Waste tonnes ('000)	8,864.7	7,796.4
Total mined tonnes ('000)	12,100.8	9,848.0
Strip Ratio	2.74	3.80
Total cost (million)	$96.7	$84.9
Revenue (million)	$184.9	$151.9
Value (million)	$88.2	$67.0
Recovered Ounces	180,743	148,477

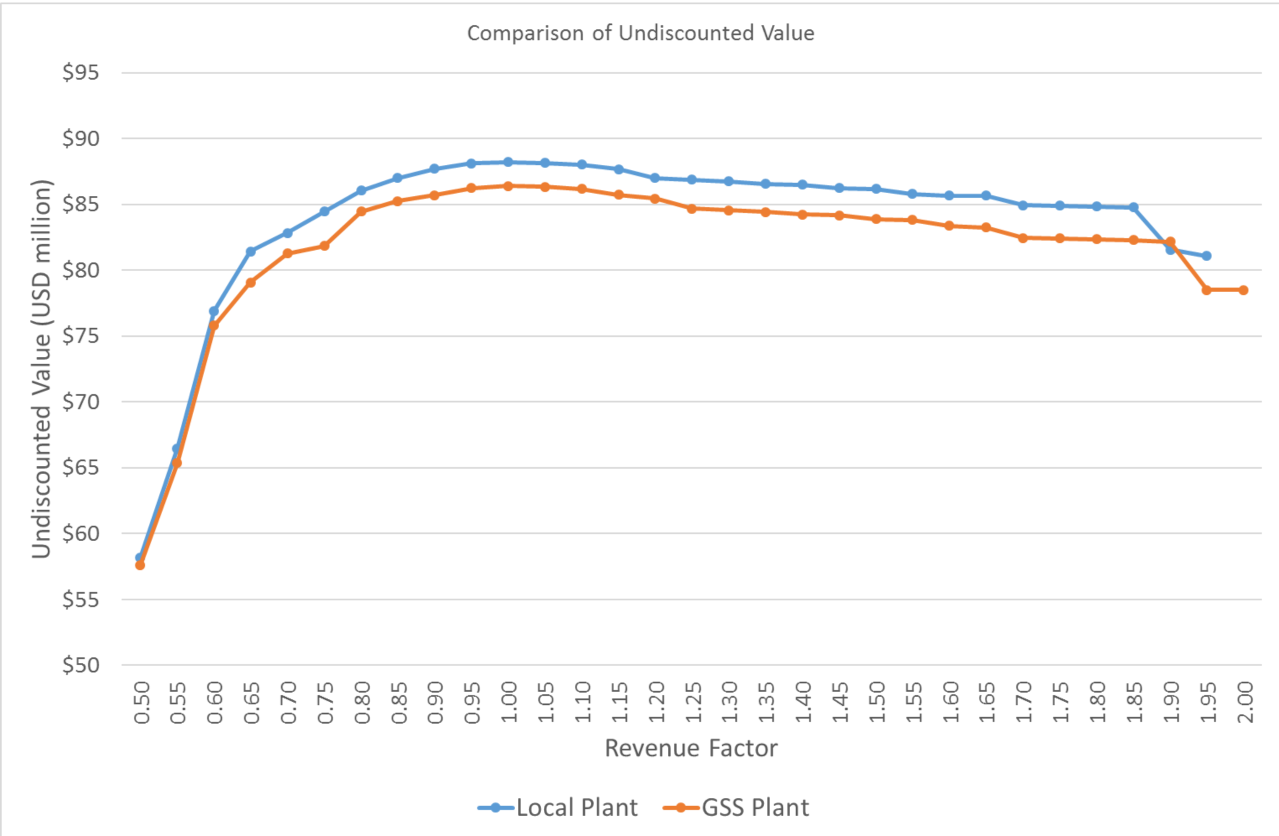


Figure 16-11: WD optimisation comparison

16.5 Mine design and sequencing

16.5.1 Mine Design Parameters

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

Wall angles, batter and berm 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 metres was assumed, 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.

16.5.2 Pit Statistics

16.5.2.1 GSS

The GSS deposit contributes over 90% of the total crusher feed and 87% of the contained ounces. The deposit is exploited through 7 individual pits extending along strike for nearly 3 kilometres (Figure 16-12). The overall strip ratio for the pits is 1.97:1. Table 16-10 shows the material contained within the GSS pits while Figure 16-13 shows the relative proportions of resource and material categories for the crusher feed.

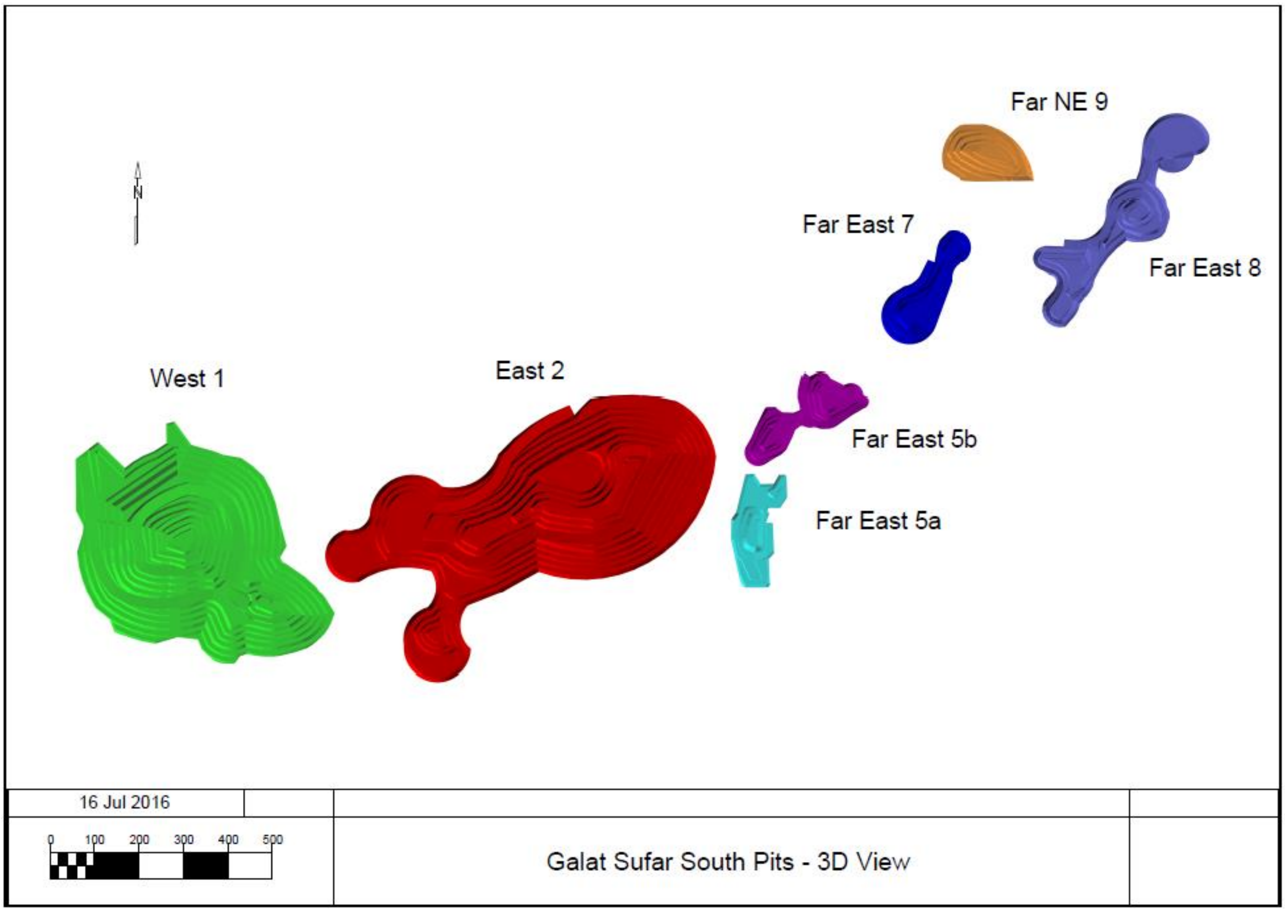


Figure 16-12: 3D view of the GSS pits

Table 16-10: GSS Pit Inventory

	Oxide		Transitional		Fresh		Total	
	Tonnes	*Grade*	*Tonnes*	*Grade*	*Tonnes*	*Grade*	*Tonnes*	*Grade*
Inferred	1,501,435	1.22	416,701	1.41	182,741	1.40	2,100,876	1.27
Indicated	8,341,659	1.40	7,095,822	1.32	8,335,416	1.59	23,772,898	1.44
Measured	-	-	-	-	-	-	-	-
Waste	19,731,346		20,995,912		10,166,018		50,893,276	
Total Material	***29,574,440***		***28,508,435***		***18,684,175***		***76,767,050***	

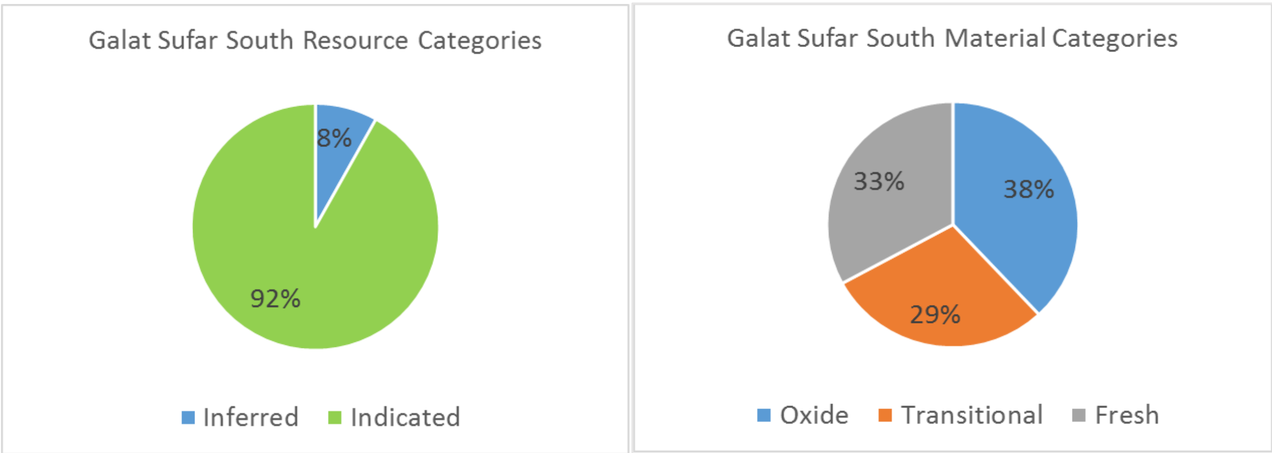


Figure 16-13: GSS Resource and Material categories for crusher feed

16.5.2.2 WD

Although much smaller than the GSS pits to the west, the WD deposit contains mineralisation at nearly double the grade. The deposit is exploited through a two stage 130m deep pit (Figure 16-14). The overall strip ratio for the WD pit is 3.91:1.

Table 16-10 shows the material contained within the GSS pits while Figure 16-13shows the relative proportions of resource and material categories for the crusher feed.

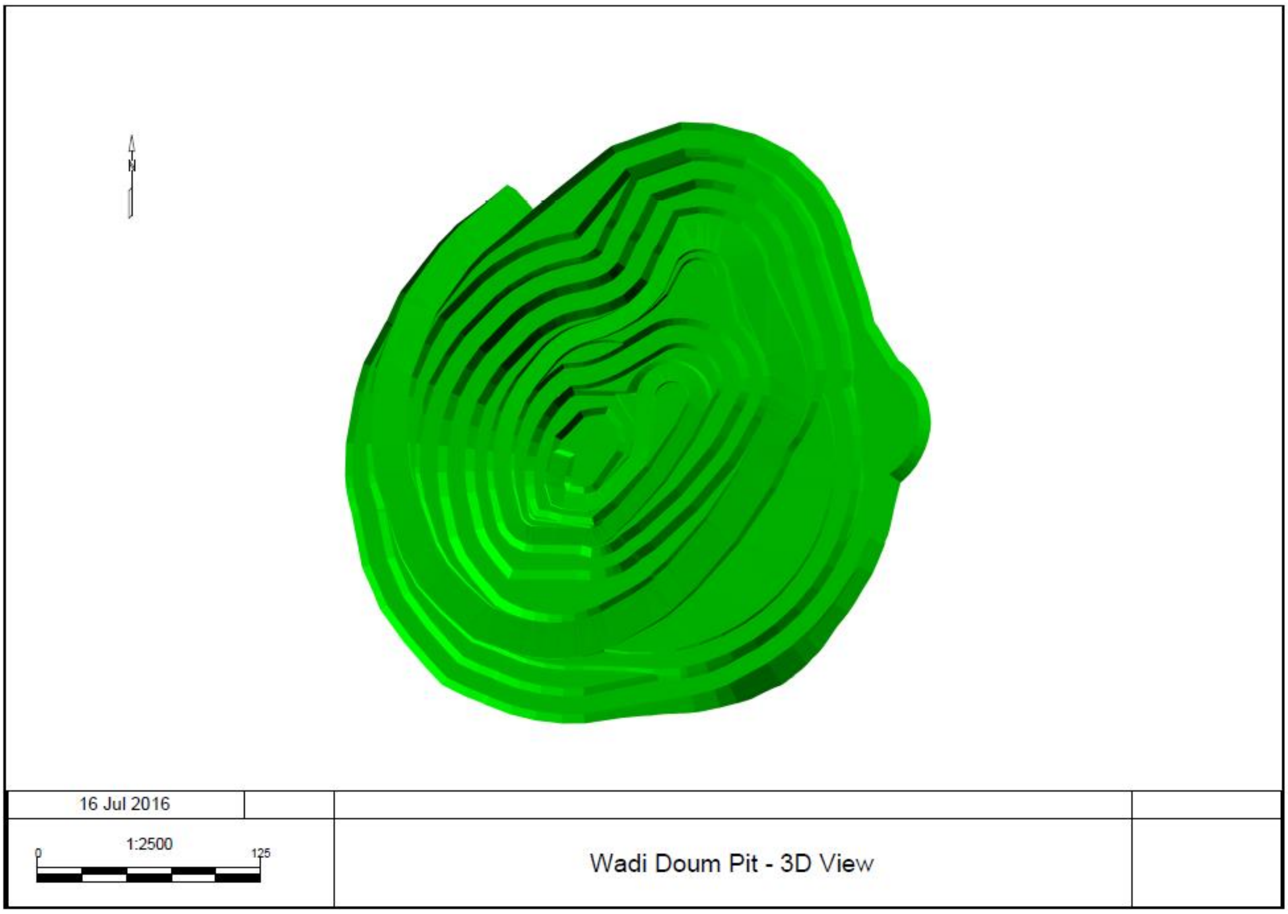


Figure 16-14: 3D view of the WD designed pit

Table 16-11: WD Pit Inventory

	Oxide		Transitional		Fresh		Total	
	Tonnes	***Grade***	***Tonnes***	***Grade***	***Tonnes***	***Grade***	***Tonnes***	***Grade***
Inferred	239,293	2.25	93,285	1.95	367,786	2.25	700,364	2.21
Indicated	135,882	2.93	96,259	2.82	1,066,028	2.90	1,298,169	2.90
Waste	3,116,245		832,289		3,862,342		7,810,876	
Total Material	***3,491,420***		***1,021,833***		***5,296,157***		***9,809,409***	

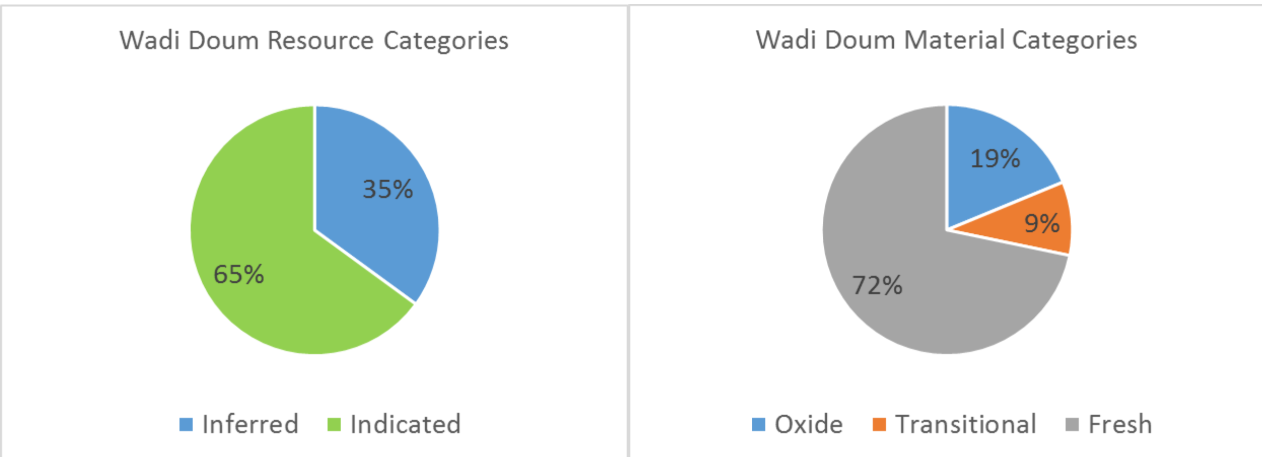


Figure 16-15: WD Resource and Material Categories for crusher feed

16.5.2.3 *Combined Summary*

The two deposits produce a total of 27.9 Mt @ 1.52 g/t Au of treatable material, 90% of which is in the Indicated category, as shown in Table 16-12 and Figure 16-16.

Table 16-12: Combined Project Inventory

	Oxide		Transitional		Fresh		Total	
	Tonnes	*Grade*	*Tonnes*	*Grade*	*Tonnes*	*Grade*	*Tonnes*	*Grade*
Inferred	1,740,727	1.36	509,985	1.51	550,527	1.97	2,801,240	1.51
Indicated	8,477,541	1.42	7,192,081	1.34	9,401,445	1.74	25,071,067	1.52
Measured	-	-	-	-	-	-	-	-
Waste	22,847,591		21,828,201		14,028,360		58,704,152	
Total Material	***33,065,859***		***29,530,268***		***23,980,332***		***86,576,459***	

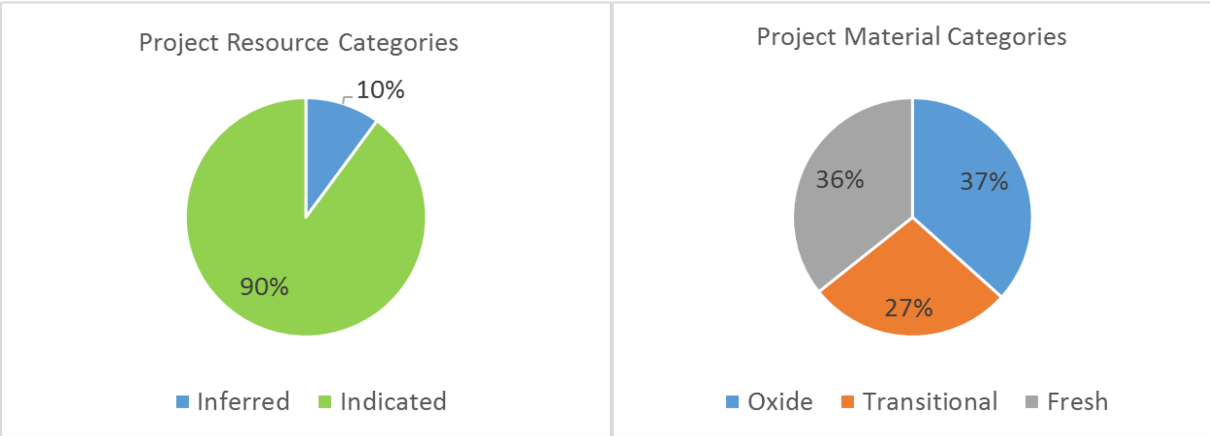


Figure 16-16 : Project Resource and Material Categories for crusher feed

16.5.3 Waste Rock Dumps, Haul Roads and ROM Pads

While no waste dump technical designs have been completed, areas have been set aside for waste dumps at both deposits, and a ROM pad at GSS. Given the topography and lack of surface water, construction of these is unlikely to create major issues. A second ROM pad will be required at WD to act as a transfer pad and space is available for this.

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.

16.5.4 Mining Schedule

The WD pit was divided into two stages to assist with waste stripping and crusher feed exposure. The GSS pits were treated as single pits for the purposes of scheduling. Further refinement of the schedule would include the development of cutbacks for the West 1 and East 2 pits. The other pits are too small to warrant phased extraction.

A ramp up period of 12 months was assumed at the start of the schedule. The target for Year 1 was 1.5 Mt of crusher feed, with all subsequent years targeting 1.8 Mt. Mining dilution and recovery were not included in the schedule.

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 16-year mine life, with WD being completed in Year 6.

The mine plan is presented in Table 16-13. Figure 16-17: Scheduled Crusher Feed, Figure 16-17 shows the source of crusher feed and corresponding Au grade while Figure 16-18 shows the pit extraction sequence.

Table 16-13: Mine Plan

	Units	Total	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	Year 15	Year 16
GSS Oxide Feed Tonnes	t	9,843,094	1,189,291	1,403,231	1,392,251	1,340,850	1,134,080	1,061,770	839,002	413,856	281,122	194,053	47,202	140,713	340,687	64,618	367	-
GSS Oxide Feed Grade	g/t	1.37	1.62	1.65	1.39	1.21	1.22	1.38	1.09	1.43	1.50	1.53	1.21	1.08	1.06	1.19	1.45	-
GSS Transitional Feed Tonnes	t	7,512,523	10,709	50,769	61,749	113,584	319,697	438,844	912,455	1,286,488	1,165,581	1,118,761	855,668	634,692	284,516	229,343	29,667	-
GSS Transitional Feed Grade	g/t	1.32	1.10	1.31	1.32	1.02	1.06	1.22	1.09	1.08	1.25	1.48	1.64	1.66	1.43	1.63	1.47	-
GSS East Fresh Feed Tonnes	t	5,273,723	-	-	-	-	223	885	48,543	99,656	353,177	482,544	858,326	823,932	624,371	862,190	993,475	126,401
GSS East Fresh Feed Grade	g/t	1.42	-	-	-	-	1.05	1.11	1.40	1.38	1.34	1.35	1.39	1.43	1.46	1.48	1.42	1.46
GSS Main Fresh Feed Tonnes	t	3,244,434	-	-	-	-	-	-	-	-	120	4,641	38,805	200,663	550,426	643,849	776,492	1,029,439
GSS Main Fresh Feed Grade	g/t	1.86	-	-	-	-	-	-	-	-	1.90	1.61	1.49	1.81	1.63	1.72	1.92	2.05
GSS Total Feed Tonnes	t	25,873,774	1,200,000	1,454,000	1,454,000	1,454,434	1,454,000	1,501,500	1,800,000	1,800,000	1,800,000	1,800,000	1,800,000	1,800,000	1,800,000	1,800,000	1,800,000	1,155,840
GSS Total Feed Grade	g/t	1.43	1.61	1.64	1.38	1.19	1.19	1.33	1.10	1.18	1.31	1.45	1.51	1.52	1.43	1.58	1.64	1.99
WD Oxide Feed Tonnes	t	375,175	269,166	100,492	5,517	-	-	-	-	-	-	-	-	-	-	-	-	-
WD Oxide Feed Grade	g/t	2.50	2.32	2.97	2.50	-	-	-	-	-	-	-	-	-	-	-	-	-
WD Transitional Feed Tonnes	t	189,544	29,704	136,675	23,165	-	-	-	-	-	-	-	-	-	-	-	-	-
WD Transitional Feed Grade	g/t	2.39	1.67	2.49	2.72	-	-	-	-	-	-	-	-	-	-	-	-	-
WD Fresh Feed Tonnes	t	1,433,815	1,130	112,833	321,318	350,000	350,000	298,533	-	-	-	-	-	-	-	-	-	-
WD Fresh Feed Grade	g/t	2.74	1.40	2.02	3.01	2.84	2.68	2.68	-	-	-	-	-	-	-	-	-	-
WD Total Feed Tonnes	t	1,998,533	300,000	350,000	350,000	350,000	350,000	298,533	-	-	-	-	-	-	-	-	-	-
WD Total Feed Grade	g/t	2.66	2.25	2.48	2.98	2.84	2.68	2.68	-	-	-	-	-	-	-	-	-	-
Total Oxide Feed Tonnes	t	10,218,268	1,458,457	1,503,723	1,397,768	1,340,850	1,134,080	1,061,770	839,002	413,856	281,122	194,053	47,202	140,713	340,687	64,618	367	-
Total Oxide Feed Grade	g/t	1.41	1.75	1.74	1.39	1.21	1.22	1.38	1.09	1.43	1.50	1.53	1.21	1.08	1.06	1.19	1.45	-
Total Transitional Feed Tonnes	t	7,702,067	40,413	187,443	84,914	113,584	319,697	438,844	912,455	1,286,488	1,165,581	1,118,761	855,668	634,692	284,516	229,343	29,667	-
Total Transitional Feed Grade	g/t	1.35	1.52	2.17	1.70	1.02	1.06	1.22	1.09	1.08	1.25	1.48	1.64	1.66	1.43	1.63	1.47	-
Total Fresh Feed Tonnes	t	9,951,972	1,130	112,833	321,318	350,000	350,223	299,418	48,543	99,656	353,297	487,186	897,130	1,024,595	1,174,797	1,506,039	1,769,966	1,155,840
Total Fresh Feed Grade	g/t	1.75	1.40	2.02	3.01	2.84	2.68	2.67	1.40	1.38	1.35	1.36	1.39	1.50	1.54	1.58	1.64	1.99
Total Feed Tonnes	t	27,872,307	1,500,000	1,804,000	1,804,000	1,804,434	1,804,000	1,800,033	1,800,000	1,800,000	1,800,000	1,800,000	1,800,000	1,800,000	1,800,000	1,800,000	1,800,000	1,155,840
Total Feed Grade	g/t	1.52	1.74	1.80	1.69	1.51	1.48	1.55	1.10	1.18	1.31	1.45	1.51	1.52	1.43	1.58	1.64	1.99
GSS Total Waste Tonnes	t	50,893,276	2,587,248	3,393,700	3,066,548	2,790,575	3,222,142	3,497,198	3,619,787	4,394,175	4,635,088	4,264,275	3,951,033	3,500,132	3,272,054	2,222,041	1,694,398	782,883
GSS Total Mined Tonnes	t	76,767,050	3,787,248	4,847,700	4,520,548	4,245,010	4,676,142	4,998,698	5,419,787	6,194,175	6,435,088	6,064,275	5,751,033	5,300,132	5,072,054	4,022,041	3,494,398	1,938,722
GSS Strip Ratio		1.97	2.16	2.33	2.11	1.92	2.22	2.33	2.01	2.44	2.58	2.37	2.20	1.94	1.82	1.23	0.94	0.68
WD Total Waste Tonnes	t	7,810,876	2,178,323	2,150,000	1,597,863	834,479	677,951	372,260	-	-	-	-	-	-	-	-	-	-
WD Total Mined Tonnes	t	9,809,409	2,478,323	2,500,000	1,947,863	1,184,479	1,027,951	670,793	-	-	-	-	-	-	-	-	-	-
WD Strip Ratio		3.91	7.26	6.14	4.57	2.38	1.94	1.25	-	-	-	-	-	-	-	-	-	-
Total Waste Tonnes	t	58,704,152	4,765,571	5,543,700	4,664,410	3,625,054	3,900,093	3,869,458	3,619,787	4,394,175	4,635,088	4,264,275	3,951,033	3,500,132	3,272,054	2,222,041	1,694,398	782,883
Total Mined Tonnes	t	86,576,459	6,265,571	7,347,700	6,468,410	5,429,488	5,704,093	5,669,491	5,419,787	6,194,175	6,435,088	6,064,275	5,751,033	5,300,132	5,072,054	4,022,041	3,494,398	1,938,722
Global Strip Ratio		2.11	3.18	3.07	2.59	2.01	2.16	2.15	2.01	2.44	2.58	2.37	2.20	1.94	1.82	1.23	0.94	0.68

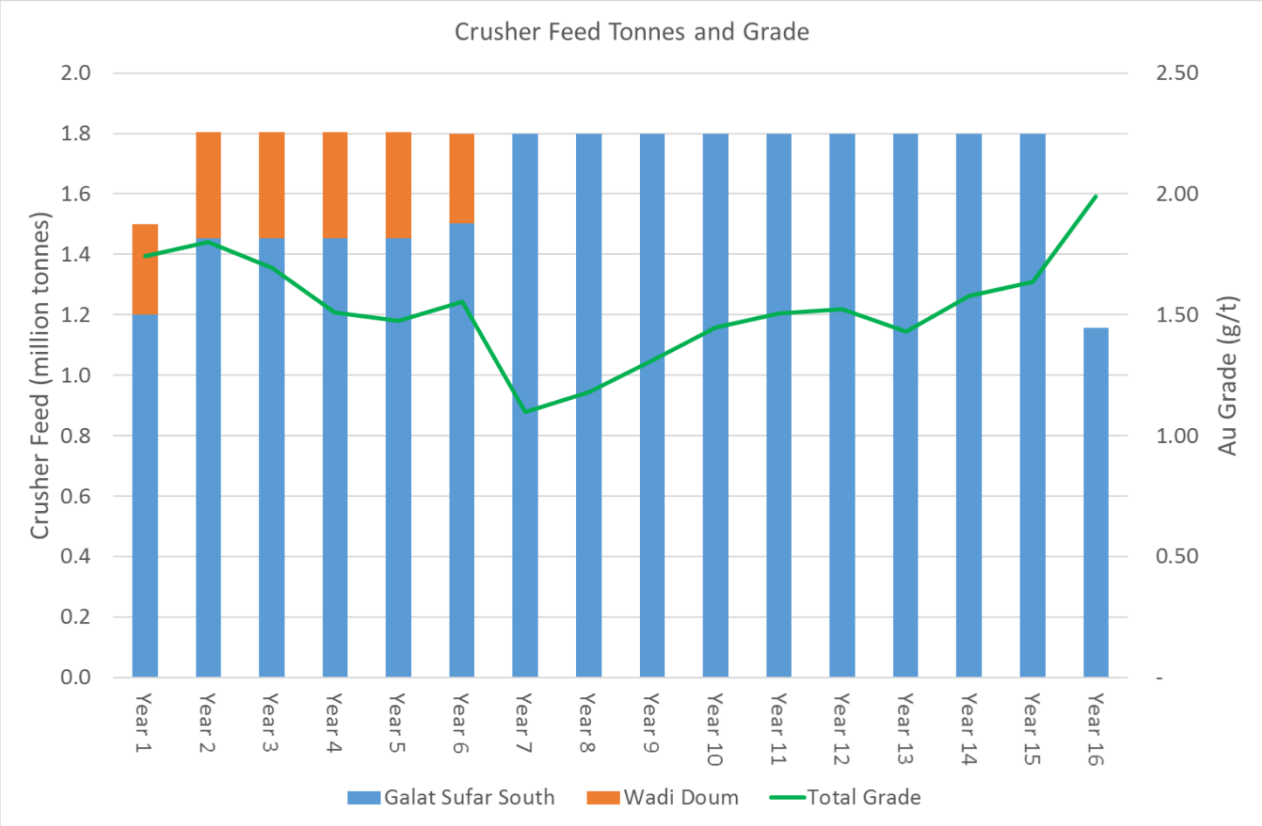


Figure 16-17: Scheduled Crusher Feed

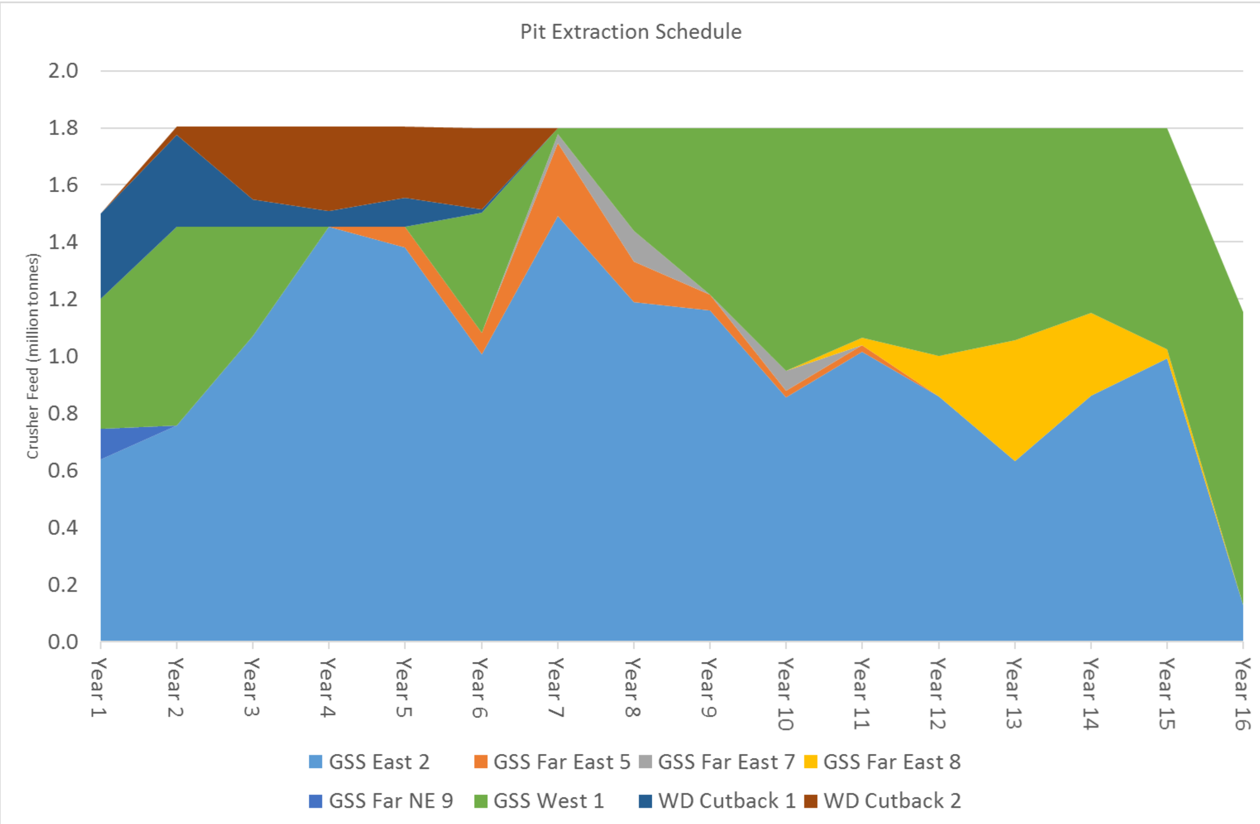


Figure 16-18: Pit Extraction Sequence

16.6 Operating Strategy

The project has been developed around the assumption that the mining operations will be carried out by a 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 the Principal.

The Principal 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; and
- Contract management personnel (including supervisors).

The Principal 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 will be required for the haulage from WD to GSS.

17 RECOVERY METHODS

17.1 Overview

Metallurgical test work conducted to date, with specific reference to Gold Deportment and Diagnostic Leach Test Work, validate that the GSS and WD material is amenable to gold recovery via cyanidation.

The most economically effective process scheme identified is the adsorption of gold onto activated carbon, through the carbon-in-leach (CIL) process preceded by a comminution circuit.

The design of the comminution circuit and the metal recovery plant is based on a nominal capacity of 1.8 Mtpa.

Figure 17-1 shows a simplified block flow diagram and Figure 17-2 is the Process Flow Diagram of the overall process facility, and provides the basis for the process description that follows. The Plant Layout and GSS Site Plan are included as Figure 17-3 and Figure 17-4. Figure 17-5 and Figure 17-6 provides perspective views of the GSS mine site and a typical plant, based on the proposed plant layout.

The GSS facility is demarcated into different areas, as per Table 17-1.

Table 17-1: Project Facility Areas

Description	Area No.
Project Wide	0000
Tailings Storage Facility Area	0100
Plant Area	0500
Crushing	*0510*
Stockpile	*0520*
Coarse Ore Stockpile	*0521*
Fine Ore Stockpile	*0522*
Milling	*0530*
Reagents	*0540*
Adsorption	*0550*
CIL	*0551*
Elution	*0560*
Electrowinning	*0570*
Gold Room	*0580*
CN Detox	*0590*
Dewatering	0600
ROM Storage Area	0300
Mining & Waste Rock Area - GSS	0700
On Site Infrastructure	0800
Utilities & Services	*0810*
Power Generation	*0820*
Fuel Storage	*0830*
Water Treatment	*0840*
Sewage Treatment	*0850*
Off Site Infrastructure	0900
Water Supply	*0910*

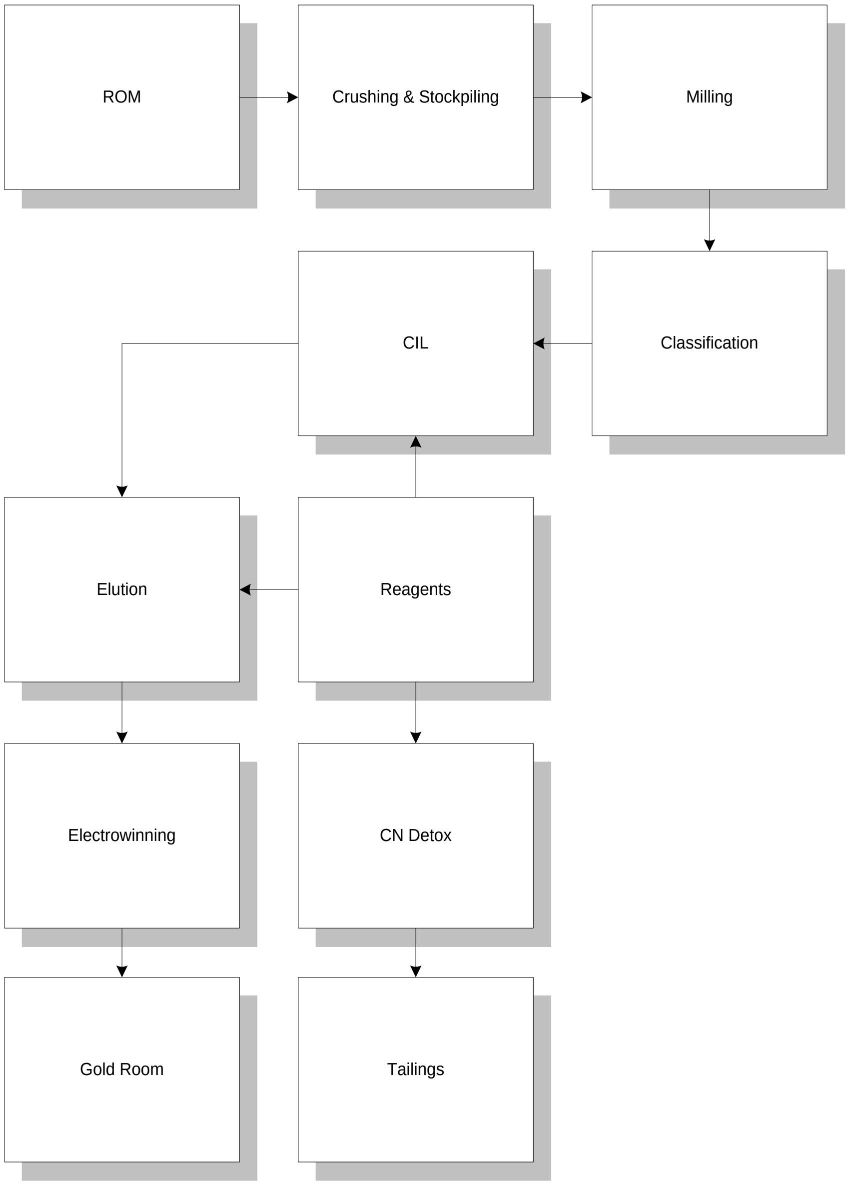


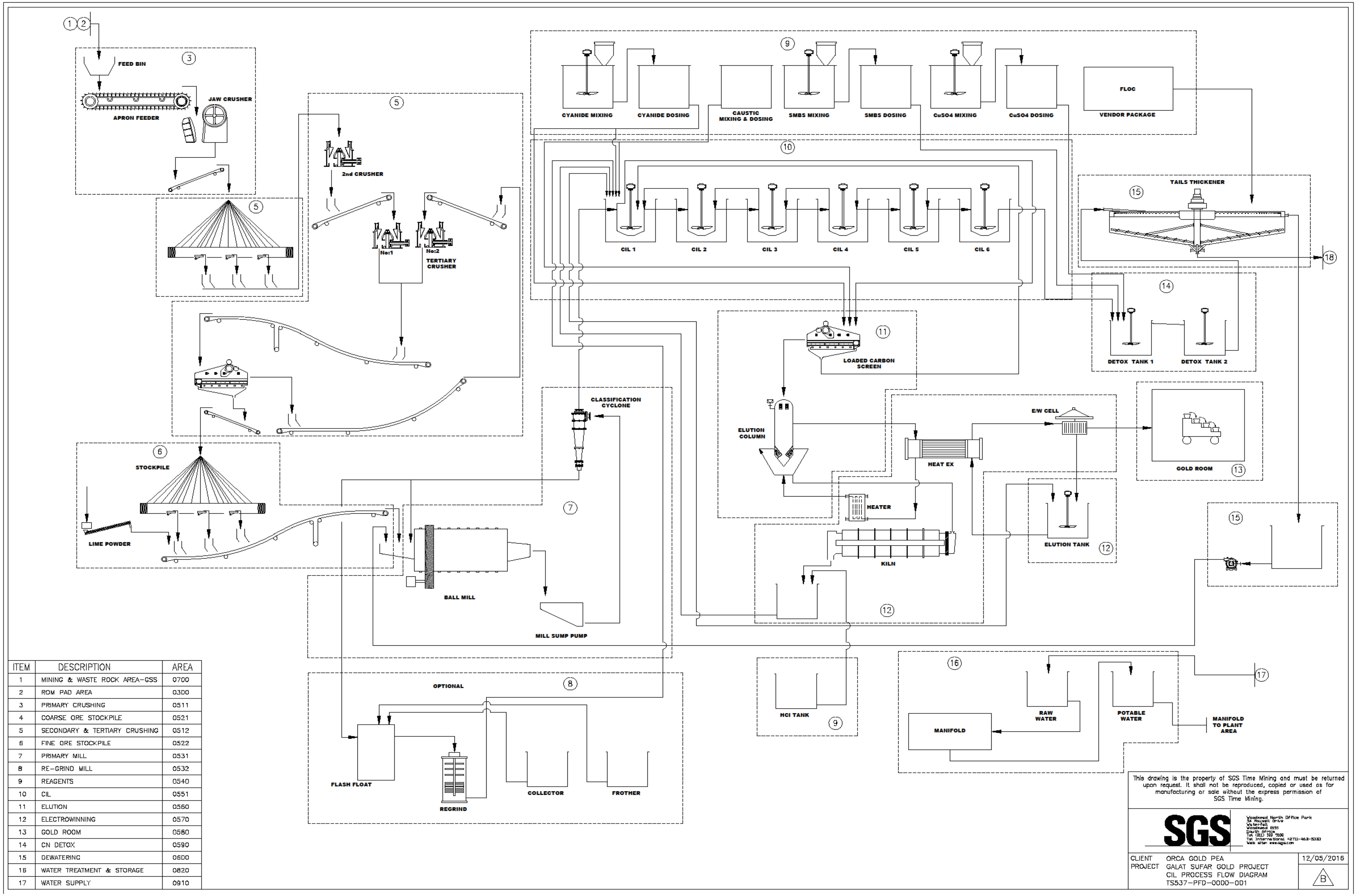
Figure 17-1: Process Block Flow Diagram

17.2 Process Description

Oxide, transitional and fresh Run-of-Mine (ROM) material from GSS and WD is fed into primary crusher and secondary & tertiary crusher circuit. The material is reduced in size from the top size of 600 mm to 80% passing 8 mm before it is fed into the ball mill. It is then ground to 80% passing 75 µm feed to the CIL circuit for leach and adsorption onto carbon. The cyclone underflow stream returns to the mill for further grinding.

The CIL tails is directed to a detoxification circuit where reagents such as SMBS and copper sulphate are added in high dissolved oxygen slurry to dissociate free cyanide to a level of less than 50 ppm.

After detoxification the slurry is pumped to the tailings thickener. The thickener overflow is distributed back into the process circuit and the underflow is pumped onto the tailings storage facility (TSF) in Area 0100. No TSF design was done at this stage of the project, and an existing design for conventional surface thickened tailings storage facility was used with an approximate footprint of 57 hectares. For surface thickened and paste storage the tailings are generally discharged from a central location either through risers or from point sources that are raised over the life of the facility.



ITEM	DESCRIPTION	AREA
1	MINING & WASTE ROCK AREA-GSS	0700
2	ROM PAD AREA	0300
3	PRIMARY CRUSHING	0511
4	COARSE ORE STOCKPILE	0521
5	SECONDARY & TERTIARY CRUSHING	0512
6	FINE ORE STOCKPILE	0522
7	PRIMARY MILL	0531
8	RE-GRIND MILL	0532
9	REAGENTS	0540
10	CIL	0551
11	ELUTION	0560
12	ELECTROWINNING	0570
13	GOLD ROOM	0580
14	CN DETOX	0590
15	DEWATERING	0600
16	WATER TREATMENT & STORAGE	0820
17	WATER SUPPLY	0910

Figure 17-2: Process Flow Diagram - CIL

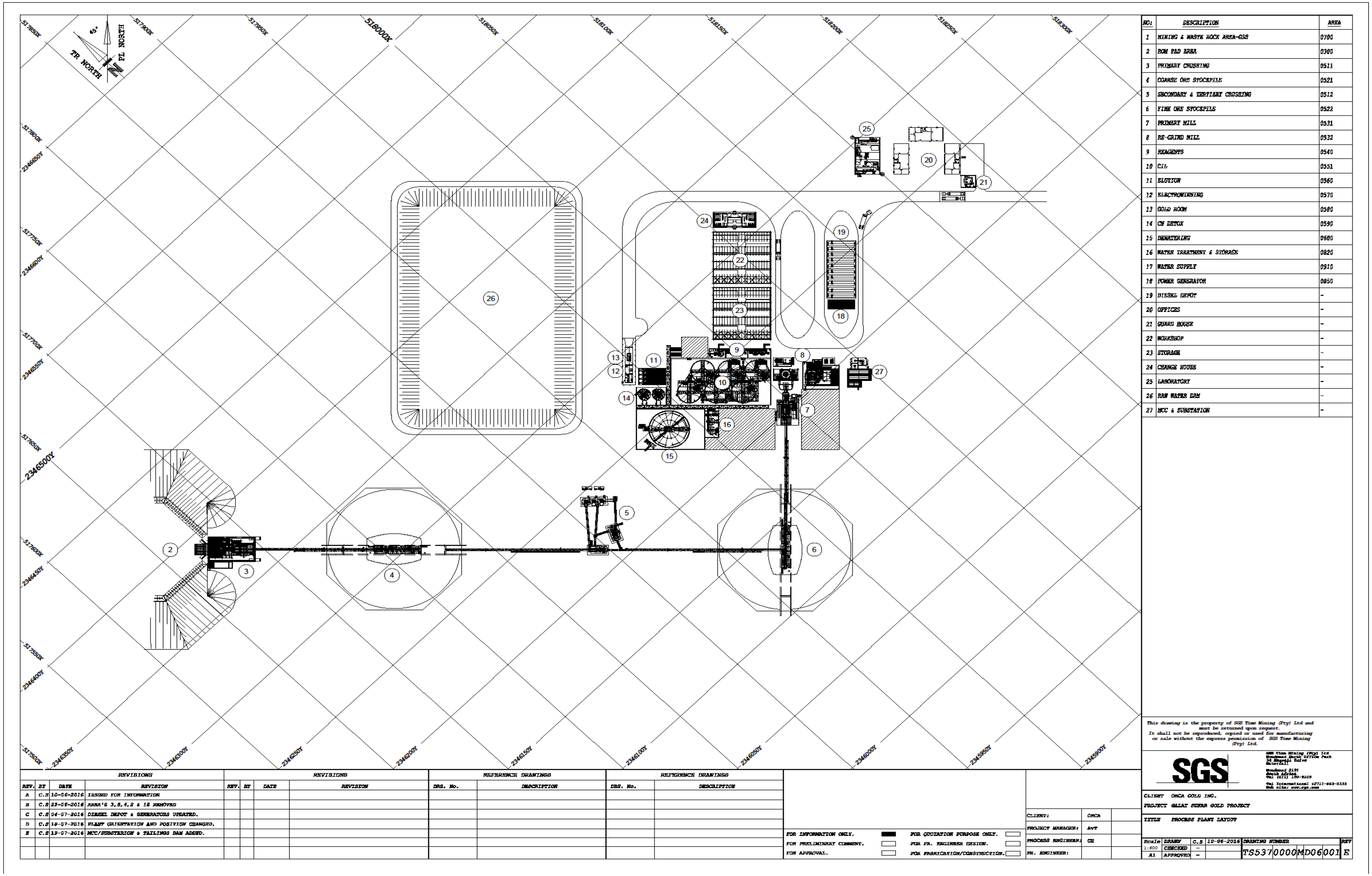


Figure 17-3: Plant Layout

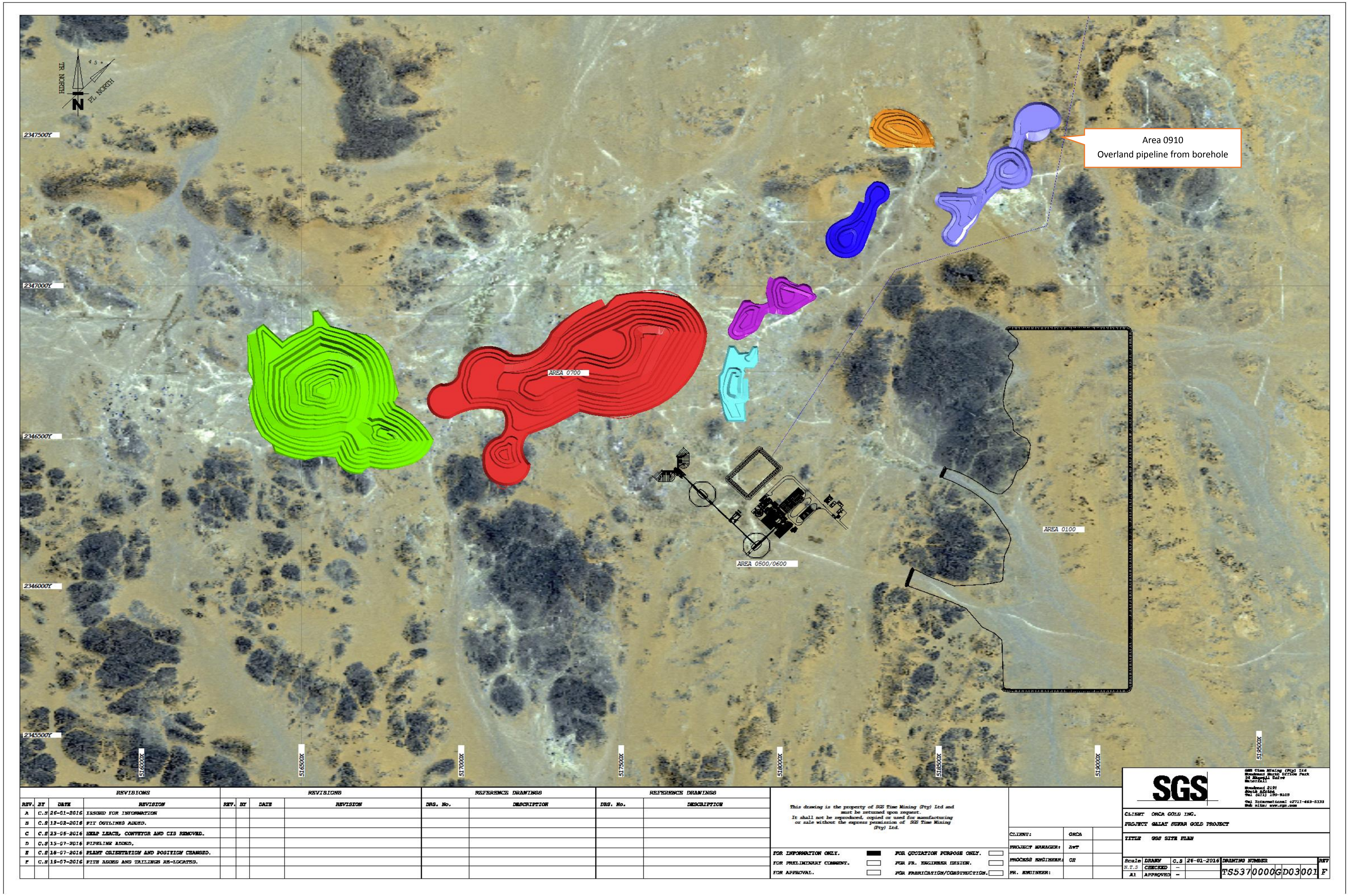


Figure 17-4: GSS Site Plan

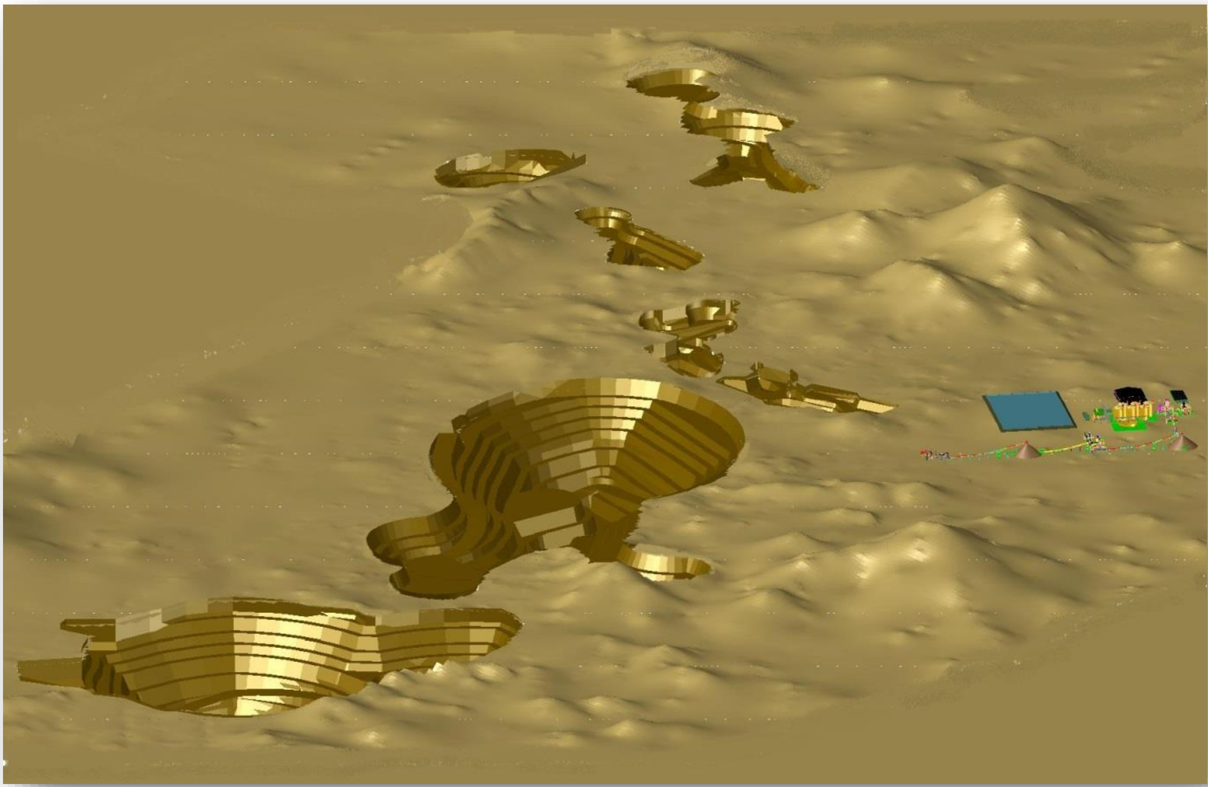


Figure 17-5: GSS Site Plan 3D View

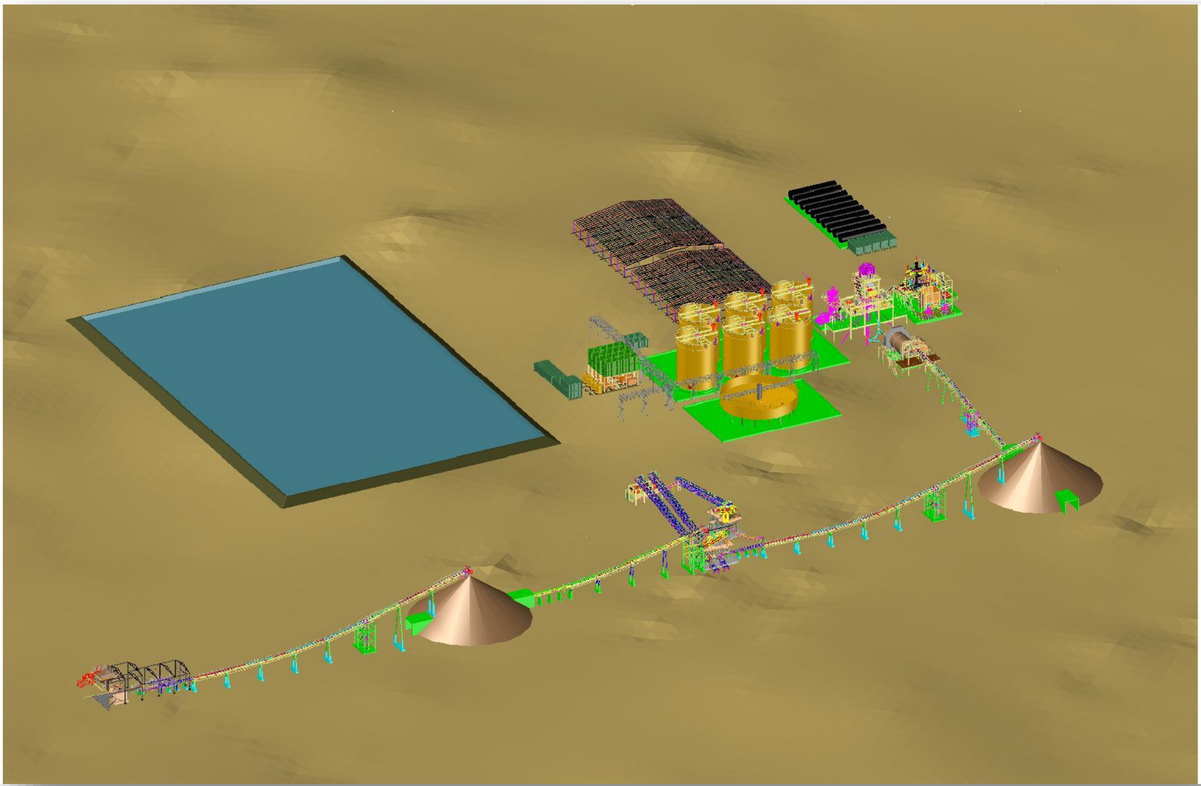


Figure 17-6: GSS Plant 3D View

17.2.1 Crushing & Stockpiling

The crushing circuit design was dictated by the maximum mill circuit of 80% passing 8 mm feed size to produce the required 80% passing 75 µm discharge size. A crusher simulation was carried out; refer to Figure 17-7, in order to size the crushing equipment.

Run-of-Mine (ROM) material will be trucked from the local GSS open pits and from the WD open pit. ROM material will be tipped from the trucks into a ROM bin. The material is withdrawn at a controlled rate by an apron feeder. The nominal feed rate of the apron feeder is 400 tph. The feeder discharges into a primary jaw crusher. The jaw crusher reduces material top size from 600 mm to 80% passing 125 mm. This material is then stockpiled on the coarse stockpile. The material is then reclaimed under the stockpile tunnel by variable speed vibrating feeders and delivered onto a conveyor to feed the secondary cone crusher. The secondary cone crusher feed conveyor will have a magnet installed over the conveyor for removal of any tramp steel that may damage the cone crusher. Coarse material is fed onto a double deck screen; +40 mm material is fed into the secondary cone crusher and the -40 mm +10 mm size fraction material reports to the tertiary crusher circuit. The discharge from the secondary cone crusher is fed onto another conveyor which feeds two tertiary cone crushers. The products from the tertiary crushers are combined and conveyed to a vibrating dry screen. The screen oversize is returned back to the tertiary crushers and the -10 mm undersize material is fed onto the fine material stockpile.

Ore is reclaimed from under the fine stockpile by vibrating feeders. The material is feed onto a conveyor where lime is proportionally dosed before being fed into the mill. If the crushing circuit is offline for an extended period, the dead portion of the stockpile can be dosed towards the centre to provide additional mill feed.

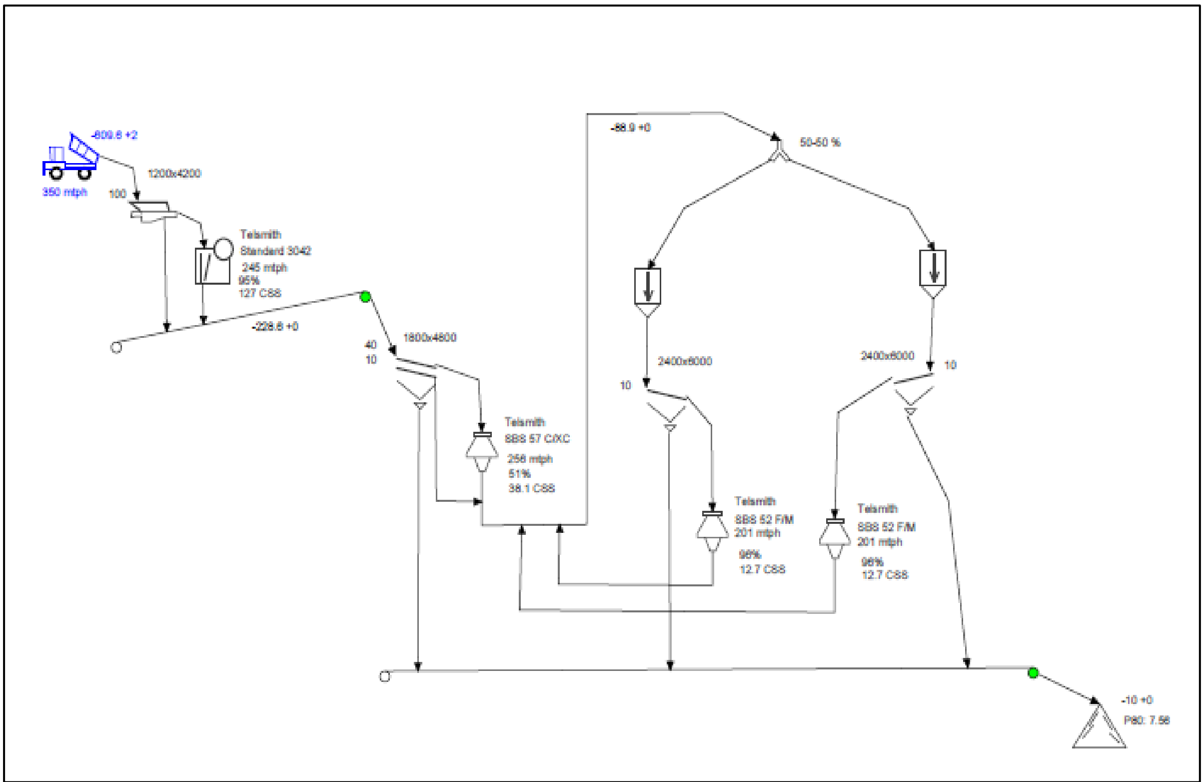


Figure 17-7: Crusher Circuit Simulation

17.2.2 Milling & Classification

Results from the comminution test work in section 13.3.1 were used to develop a comminution circuit simulation, see section 13.3.2. The current design is based on the results of the circuit simulation utilising a single primary mill with nominal dimensions of 16.0' x 28.5' with installed power of 3,900 kW to produce the required 80% passing 75 µm discharge size.

Ore from the fine material stockpile is fed to the ball mill. Inlet dilution water is added to the mill feed chute to control mill slurry density. The ball mill discharges the resultant slurry via a trommel screen into a launder. Oversize from the trommel, mostly ball scats, drops from the trommel into a bunker. Dilution water is added to the mill discharge launder. Slurry is pumped from the ball mill sump to the cyclone cluster for size classification. At the cyclone clusters, coarse particles pass through the cyclone spigots as dense slurry that gravitates to the underflow distributor and fine particles pass through the cyclone overflow at about 40% solids slurry to the CIL.

The two outlets of the cyclone underflow distributor divert slurry where about 75% portion is returned to the mill and the remainder flows to the CIL circuit.

Spillage pumps are provided in the mill bunded spillage area. Spillage is pumped to the mill sumps. The spillage pumps are situated in a drop out sump arrangement. Coarse solids settle out, and the spillage pump returns only excess water and slimes to the mill sump. Periodically a loader or Bobcat is used to remove accumulated settled solids from the drop out sump.

17.2.3 Carbon-in-Leach (CIL)

Slurry from the cyclone overflow gravitates to the CIL section for cyanidation. A splitter box is installed equipped with a plug valve. This plug valve is normally open and weir plates installed to direct the slurry into the Leach tank. If the leach tank needs to be bypassed then the plug valve is closed and weir plates removed to divert slurry to the first CIL tank. Slurry overflows from the mechanically agitated leach tank and then flows through six subsequent mechanically agitated CIL tanks, to ensure complete dissolution of gold as a cyanide complex and adsorption onto activated carbon.

Cyanide solution is added from a ring main to the leach tank. Compressed air is injected into the leach tank with the facility to add air to the first two CIL and fourth tanks, should it be required, to provide oxygen for the cyanidation reaction. All the tanks are equipped with two launder valves on the outlet. The normally open valve passes slurry to the next tank downstream. The second valve allows diversion of slurry to the successive tank downstream, in order for the slurry flow to bypass any single tank, should the need arise.

The CIL area is bunded to retain any overflow or spillage, and is provided with a spillage pump.

Each CIL tank is equipped with an inter-stage screen mechanism, with a cylindrical basket-type stainless steel wedge-wire screen surface. The mechanism drive turns wiper blades mounted on the outside of the basket. The wiper blades keep the screen surface clear and allow slurry to flow through, while the coarser carbon is retained in the tank. The screen has a pumping mechanism to allow slurry in the tank to flow through the screen surface whilst the tanks are on the same level. Tailings slurry from the last CIL tank gravitates to the tailings section.

A recessed impeller type pump, located in the first CIL tank transfers slurry containing loaded carbon from the first CIL tank to the loaded carbon screen. The screen separates and rinses the loaded carbon from the slurry and discharges it into the elution section. Slurry drains back from the screen to first CIL tank. Tanks are fitted with manholes and drain valves.

Interstage screens require regular cleaning to remove scaling from the CIL slurry and remove platelets of carbon that peg the wedge wire apertures. A spare interstage screen and a frame (maintenance bay) are provided. The spare screen is exchanged with an installed dirty screen and cleaned using wire brushes and a high pressure water spray gun. The cleaned screen is then available as a spare for exchange with the next screen that requires cleaning. Interstage screening is carried out using 700 mm slotted aperture wedge wire pumping screens. Using slotted screen apertures makes the circuit sensitive to platelet carbon, as this carbon can be lost to tails. The mechanical strength is excellent and the screens are normally replaced when the slots open up to 1.0 mm. It is necessary to periodically clean interstage screens to maximise flowrate through them. The screens are cleaned regularly using high-pressure water sprays. If a unit should fail for whatever reason or require cleaning it can be removed and replaced by a standby unit within 20 minutes. The correct period between cleaning screens will be found from trial and error, but should be done at the first sign of carbon being lost from any CIL tank to the next or to the tailings screens.

Table 17-2 provides a summary of the CIL design criteria.

Table 17-2: Concept Design Criteria - CIL

Description	Value	Unit
Feed slurry flow	433	m^3/h
Residence time-Total	30	h
Total tank volume Required	12,981	m^3
Volume per tank - Total	2164	m^3
Volume per tank - Live	2031	m^3
No. of leach tanks	6	
Freeboard	1.00	m
Height : Diameter ratio	1.25	
Tank diameter	13.00	m
Tank Height	16.300	m
Cyanide consumption	0.75	kg/t

17.2.4 Cyanide Detoxification & Filtration

The discharge from the final CIL tank flows by gravity through a carbon catchment screen to the cyanide detoxification tanks. The cyanide detoxification tanks are mechanically agitated tanks with airflow to assist with the detoxification reaction. Copper sulphate is pumped from a dedicated copper sulphate storage/mixing plant and added to the pulp. Sodium metabisulphite is pumped from a dedicated sodium metabisulphite storage/mixing plant and added to the pulp. Caustic is used to increase pH as required.

The residue is pumped to a thickener. The water is reused as process water and the thickened slurry is sent to the final tails dam.

17.2.5 Elution

The elution section uses a pressurised 4 tonne Zadra system. Loaded carbon is eluted by pumping a hot caustic cyanide solution, typically 1.0 - 2.0 % NaOH and 0.2 - 0.6 % NaCN, through the column at 120°C under pressure. Gold adsorbed onto the loaded carbon is eluted off the carbon and collected in the eluate solution. The eluate is passed through electrowinning cells to remove gold from the circulating eluate stream. The electrowinning tails return to the elution tank.

To perform an elution, the caustic and cyanide concentrations of the eluate solution in the elution tank are checked and if required, corrected by adding fresh reagents (caustic and/or cyanide). The contents of the elution column and elution tank are heated to operating temperature by circulating eluate through one side of the recuperative heat exchanger, then through the thermic oil heat exchanger where solution is heated by contacting with hot thermic oil into the column. The solution exits the column via the other side of the recuperative heat exchanger and back to the eluate tank.

Once the column, its contents and the eluate flowing through the column reach a temperature of 120°C, the eluate is diverted to the electrowinning cells. In the electrowinning cells, the eluted gold is recovered by electrowinning, as described in section 17.2.6, and then the eluate flows back to the elution tank in a closed circuit.

The column operates under a pressure of typically 300 to 350 kPa. Solution exiting the column passes through the recuperative heat exchanger to cool the hot eluate to below boiling point and preheat eluate entering the column.

Eluate solution is reused for a number of elution cycles until the level of contamination becomes unacceptable. The spent eluate is pumped to the CIL tanks. Thus, the benefit of any available caustic and cyanide in the spent eluate is used. To make a new batch of eluate solution, the elution tank is then filled with raw water and concentrated cyanide solution and caustic soda is added to reach the desired strength. The caustic soda solution is added in batches from the caustic tank by the caustic pump. Cyanide is delivered from the CIL cyanide ring main.

The pregnant electrolyte flows out of the recuperative heat exchanger to the electrowinning cells in the gold room. Once the gold has been stripped from a batch of loaded carbon, the eluted carbon is transferred hydraulically from the elution column to the eluted carbon tank in the regeneration section. The cyanide strength of the eluant should be maintained initially between 0.3% and 0.5 % NaCN. Elution efficiency maybe greatly reduced if the concentration is too low. If the cyanide concentration is above 0.5 % the problem is a higher rate of cyanide usage (wastage). It may be possible after the plant is commissioned to reduce or even eliminate the cyanide requirement for elution. This is partially dictated by operating conditions and somewhat cyanide free elution is unlikely to be effective in particular if the carbon becomes loaded with copper to over 1,000 ppm.

Due to the use of caustic and cyanide at the elution area, a safety shower is provided in this area. The caustic strength of the eluant should be set between 0.5 and 1.5 % NaOH.

Table 17-3 provides a summary of the concept process design criteria for the unit process.

Table 17-3: Process Design Criteria - Elution

Description	Value	Units
Elution method	Zadra	
Design carbon loading	1900	g/t
Design barren carbon loading	50	g/t
Strip Batch Size	4.0	t
Estimated number of elution per month	26	
Eluant Strength - Design	1% NaCN & 2% NaOH	
Carbon bulk density	0.50	t/m^3
Volume of carbon in the column	8.0	m^3
Bed expansion	3.0	%
Required volume as a result of expansion	8.2	m^3
Column length:diameter ratio	5.0	
Column diameter	1.1	m
Column height: tan-tan	5.3	m
Height of conical bottom & top	0.45	
Volume of conical top & bottom	0.5	m^3
Volume of tan-tan (straight part)	4.7	
Total volume	8.7	m^3
Type of carbon in use	6 x 12	mesh
Flow Through the Column	2.5	BV
Flow Through The Column	20.0	m^3/h
Time required to empty C from column	1.5	h
Time required for eluant to fill the column	1.0	h
Time required to fill eluant tank with water	0.5	h
Minimum required volume of eluant tank	14.0	m^3
Eluant tank extra capacity	43.0	%
Total required volume of eluant tank	20.0	m^3
Tank height:diameter ratio	1.15	
Column diameter	1.25	m
Column height (tan-tan)	7.40	m
Top-dish height	0.31	m
Bottom-cone height	1.06	m
Column total height	8.77	m
Eluant tank diameter	2.8	m
Eluant tank height	3.2	m
%C in transfer water (m/m)	20.0	%

17.2.6 Regeneration & Electrowinning

Carbon is withdrawn from the eluted carbon tank to feed the kiln by a screw feeder, which discharges the carbon into the rotary kiln for thermal regeneration. Regenerated carbon from the kilns is quenched in the quench pan and passed over a screen to remove fines, before discharge into the CIL to replace loaded carbon transferred out of the adsorption circuit.

Carbon fines passing through the quench screen deck gravitate to the fine carbon catchment bag, where any fugitive carbon can be recovered for reuse.

Eluate from the elution column is delivered to a flash tank. The tank acts as a steady head tank feeding the electrowinning cells.

Gold in the eluate is plated out onto cathodes. Periodically the cathodes are removed. A hoist is provided for removal of the cathodes and maintenance purposes.

A high pressure water gun is used to wash the cathode. The resultant slurry is dropped into the laboratory filter press. The filter press cake is stripped out and packed into trays for calcining.

The calcined material is smelted with fluxes into gold doré bullion in a furnace. The molten furnace charge is poured into moulds and the bullion bars are allowed to cool before cleaning.

Provision is made for storage of fluxes in the gold room before mixing with the furnace charge to prevent traffic into the gold room for security reasons.

Due to the heat and fumes generated by the electrowinning cells in the gold room, a ventilation fan is provided. A safety shower is also provided in this area.

Bullion bars are cleaned, weighed, stamped, sampled and then stored in a bullion safe while awaiting despatch.

Table 17-4 provides a summary of the unit process concept design criteria.

Table 17-4: Regeneration & Electrowinning Design Criteria Summary

Description	Value	Units
Type kiln	Electric	
Regeneration Kiln capacity	400	kg/h
Eluant solution e/w cell model	14 / 12	
No. of eluant solution e/w cell required	1 running & 1 s/by	
Eluant solution e/w cell capacity	500	l/min
Eluant solution electrowinning cycle type	20	h

17.2.7 Reagents

17.2.7.1 Lime

Dry powdered lime is added onto the mill feed belt from a silo by a variable speed rotary vane and screw feeder arrangement. The silo is filled from road tanker deliveries.

17.2.7.2 Hydrochloric Acid

Hydrochloric acid stored in drums, is transferred to an acid storage tank. From the tank acid is transferred with an acid transfer pump to the acid soak tank in the regeneration area.

Due to the fumes generated with the dilution of hydrochloric acid, an extraction fan is provided

Also located within this area is a dedicated spillage pump which returns spillage to the storage tank.

17.2.7.3 Cyanide

Sodium cyanide stored in bags is discharged through a bag breaker into a cyanide mixing tank equipped with a mechanical mixer. Raw water is added and mixed for a set period of time to achieve proper mixing. From the tank cyanide is transfer with a cyanide transfer pump through an inline strainer to the cyanide storage tank. From the storage tank two cyanide dosing pumps with a duty / standby arrangement transfer cyanide to the plant on a ring main.

17.2.7.4 Caustic

Caustic soda also stored in bags is discharged through a bag breaker into a caustic mixing tank equipped with a mixer. Raw water is added and mixed for a set period of time to achieve full hydration. From the tank caustic is transferred with a caustic dosing pump through an inline strainer to the respective elution and leach circuits.

17.2.7.5 SMBS

SMBS stored in bags is discharged through a bag breaker into a SMBS mixing tank equipped with a mixer. Raw water is added and mixed for a set period of time to achieve proper mixing. From the tank SMBS is transferred with a transfer pump through an inline strainer to the SMBS storage tank. From the storage tank a diaphragm pump doses the reagent at a controlled rate into the detox tanks.

17.2.8 Plant Utilities

17.2.8.1 Air Services

Plant and instrument air is produced by a compressor. Plant air utilised in the leach and gold room areas is stored in a plant air receiver. Compressed instrument air is first sent through an air dryer unit and then stored in an instrument air receiver before is distributed to all relevant areas

17.2.8.2 Potable water

A potable water tank is equipped with high pressure pumps to distribute drinking water to communal services and safety showers.

17.2.8.3 Raw water

Raw water stored in a raw water dam and buffer tank and is distributed via a header through the plant with pumps Also distributed through the plant from this tank is gland service water with duty/standby arrangement pumps and fire water services; the tank shall be designed as such to provide sufficient fire water supply.

18 PROJECT INFRASTRUCTURE

18.1 Water Supply

The estimated water demand is in the order of 4,000 m^3/day (46 l/s) initially, see Table 18-1.

Table 18-1: Average & Daily Water Demand

Year	Total Feed Tonnes	Average Annual Water Demand (m^3/annum)	Average Daily Water Demand (m^3/day)
1	1 500 000	1 200 000	3287.7
2	1 804 000	1 443 200	3954.0
3	1 804 000	1 443 200	3954.0
4	1 804 434	1 443 548	3954.9
5	1 804 000	1 443 200	3954.0
6	1 800 033	1 440 026	3945.3
7	1 800 000	1 440 000	3945.2
8	1 800 000	1 440 000	3945.2
9	1 800 000	1 440 000	3945.2
10	1 800 000	1 440 000	3945.2
11	1 800 000	1 440 000	3945.2
12	1 800 000	1 440 000	3945.2
13	1 800 000	1 440 000	3945.2
14	1 800 000	1 440 000	3945.2
15	1 800 000	1 440 000	3945.2
16	1 155 840	924 672	2533.3
Total	**27 872 307**	**22 297 846**	

A water supply infrastructure concept study for the PEA phase was conducted by ProPipe Process & Pipeline Projects. A specialist conducted a reconnaissance of the site and surrounding areas in order to gather information and prepare the study.

Two alternatives sources of water, as shown in Figure 18-1 identified:

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

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. The selected area is characterized by 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 is a risk of damage to the pipeline due to the high circulation and movement of people and equipment from these sites

The pipeline has a total length of 197 km and has been designed as 630 mm diameter in 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 220kW.

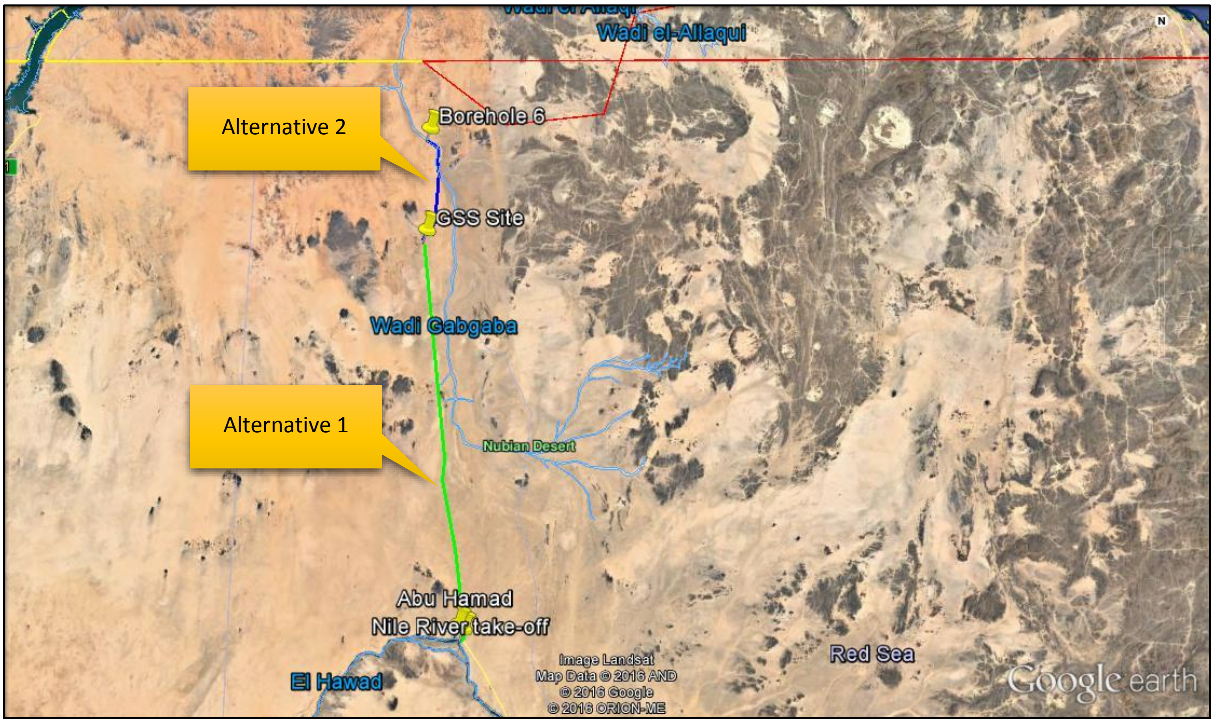


Figure 18-1: Water Supply Options

18.1.2 Alternative 2: Borehole Well Field

As an alternative to the water supply from the Nile, a concept design of an extraction and pumping system from Borehole 6 area to the GSS area was completed. The well field is located 50 km north of the GSS project area.

In order to verify the viability of Alternative 2, regional and local scale groundwater resource related investigations, involving remote sensing, ground geophysics and drilling, have been carried out in selected areas of the Project area since 2012. Refer to "Groundwater Exploration and Groundwater Resource Assessment for the GSS Project, Sudan – Block 14 Area Summary Report for PEA Supplement. Dated July 2016. GCS Water & Environmental Consultants" for additional information.

Based on geophysical surveys completed in late 2015, a follow up drill programme, testing two low resistivity anomalies hosted within the Nubian Sandstone basin has been completed in early July 2016.

The primary objective of the study was to determine if groundwater resources exist within the GSS with special reference to the Hydrogeological Areas HA8 and HA9. The two areas can be viewed in the locality maps below (Figure 18-3 and Figure 18-2). Landsat Images as background were obtained from Orca in 2013 and according to the latest geophysical report (Harwood M, Feb 2016), covers areas in the order of:

- HA8 – 71 757 500 m^2, and
- HA9 – 67 910 000 m^2.

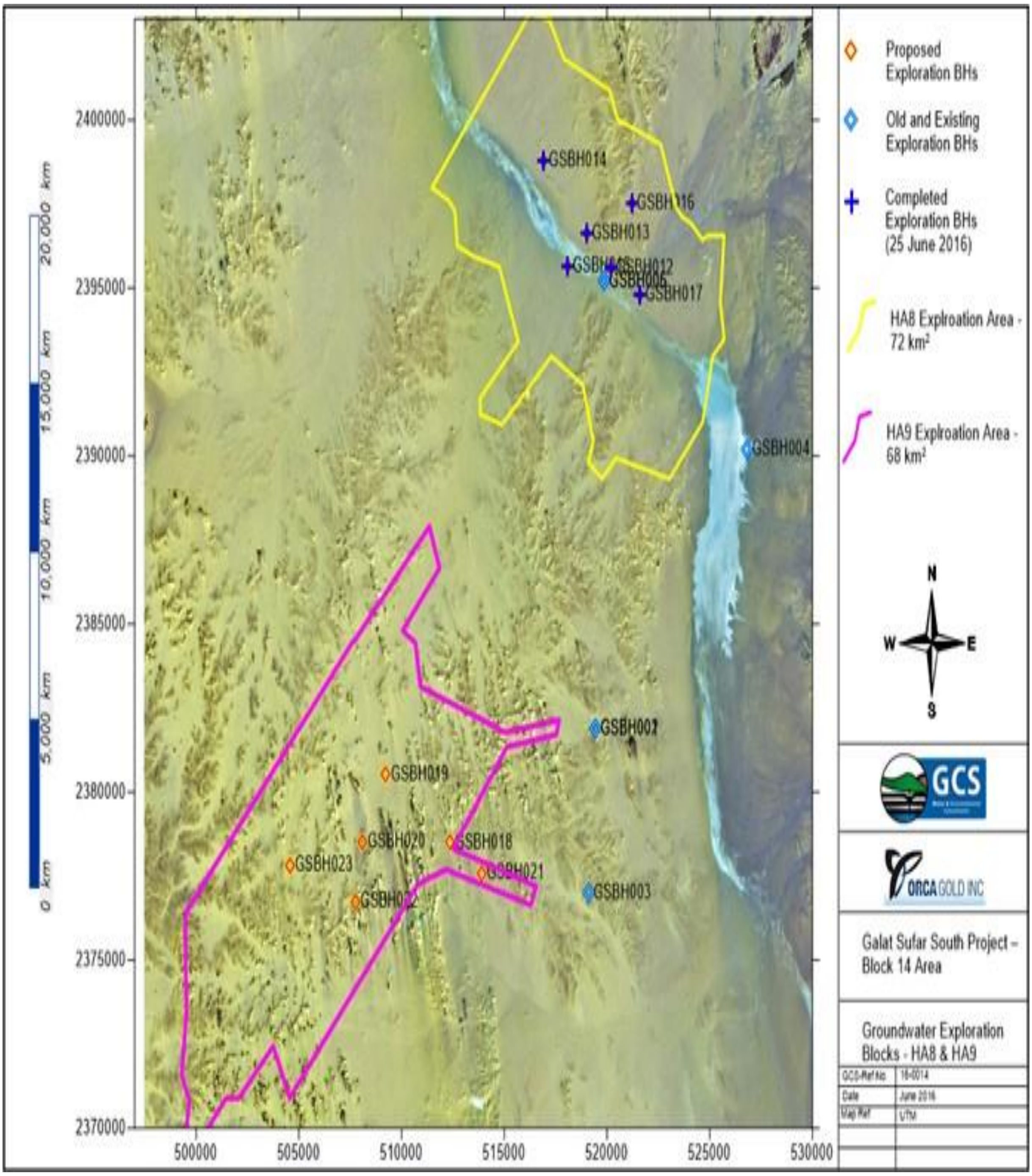


Figure 18-2: Groundwater Exploration Target Areas HA8 and HA9

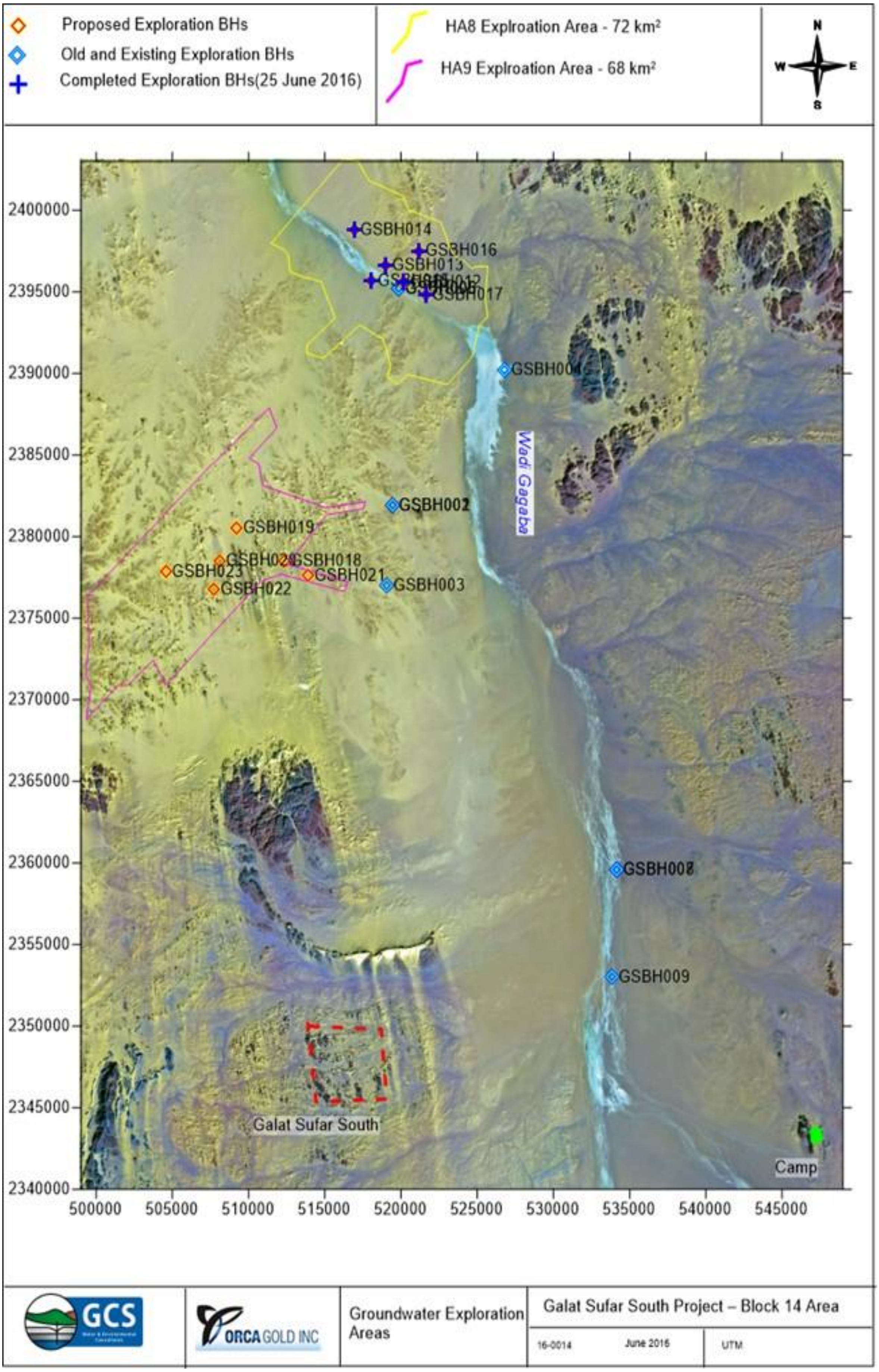


Figure 18-3: Block 14 Groundwater Exploration Drilling Areas

18.1.3 Preliminary Groundwater Resource Indicators

18.1.3.1 Indications for HA8

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 and it appears if aquifer yield generally increases with depth;
- It is fair to assume that yields may increase further for boreholes GSBH006, 12 and 13 if drilled deeper. No definite basement was reached in the boreholes and it is assumed that the basement floor may reach depths of 200m+. This further would suggest that yields may increase because the basement floor may act as an aquiclude (Figure 18-4);
- Figure 18-5 supplies an overview of the main area where groundwater was intersected. This area is approximately 13.8 km^2 in size and demarcated around the successful boreholes. Future drilling may reveal a bigger area;
- Figure 18-6 supplies a graphical presentation of the groundwater strike depths and approximate final blow yields;
- The primary aquifer in HA8 consists of interbedded sequence of fine to course grained sandstone and sandy clays. These layers are bounded by layers of mud-stone, siltstone and clay with fine sandstone; leaky conditions may exist from these and may act as a secondary aquifer in some cases. It appears if the thickness of the primary aquifer is between 4 and 20m and that the average is around 16m;
- If one considers the available drilling observation data, it can be conservatively assumed that aquifer storage in this identified zone is between 22M m^3 and 42M m^3. This range is based on 16 to 20m of aquifer thickness and 10 to 15% net specific yield for the observed strata (refer to Table 18-2);
- Based on the proposed daily 4000 m^3/day water demand, this range can supply the mine with water for the entire LOM; and
- No aquifer recharge has been considered and only the available storage capacity of the aquifer has been considered in the calculation made above. Any recharge that may occur from the Wadi during flash flood events which may occur every 4 to 6 years will increase groundwater availability. During the drilling observations it was clear that low TDS occurred in the upper zones which suggest that recent groundwater recharge took place.

18.1.3.2 Indications for HA9

No indications of groundwater occurred during the recent exploration drilling.

18.1.3.3 Proposed Mining Areas - GSS and WD

Limited data is available but very few mineral exploration boreholes intersected groundwater at the GSS and WD areas. It is fair to assume that these boreholes can develop in the order of 1 to 5 l/s for local water consumption purposes, like site offices, dust suppression and workshops.

18.1.4 Typical Well Field Design Indicators

A well field is usually developed and constructed for the abstraction of groundwater to a central distribution point after confirmation that the source is feasible and adequate to exploit. The proposed follow-up test work will provide further information on what will be required to design a well field.

The following indicators will be considered:

- 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 316L stainless Steel V Wire Screens be used;
- For the water demand of about 4000 m^3/day between 8 and 11 production boreholes will be required. The estimated cost per production borehole is US$85,000 and will be updated during the PFS; and
- To estimate the investment and operating costs a system consisting of a battery of deep wells and a main pumping station, which will drive the water to future facilities at GSS, through a 500 mm diameter HDPE pipeline has been designed. The energy required for pumps will be supplied by a diesel generator located at the main pump station.

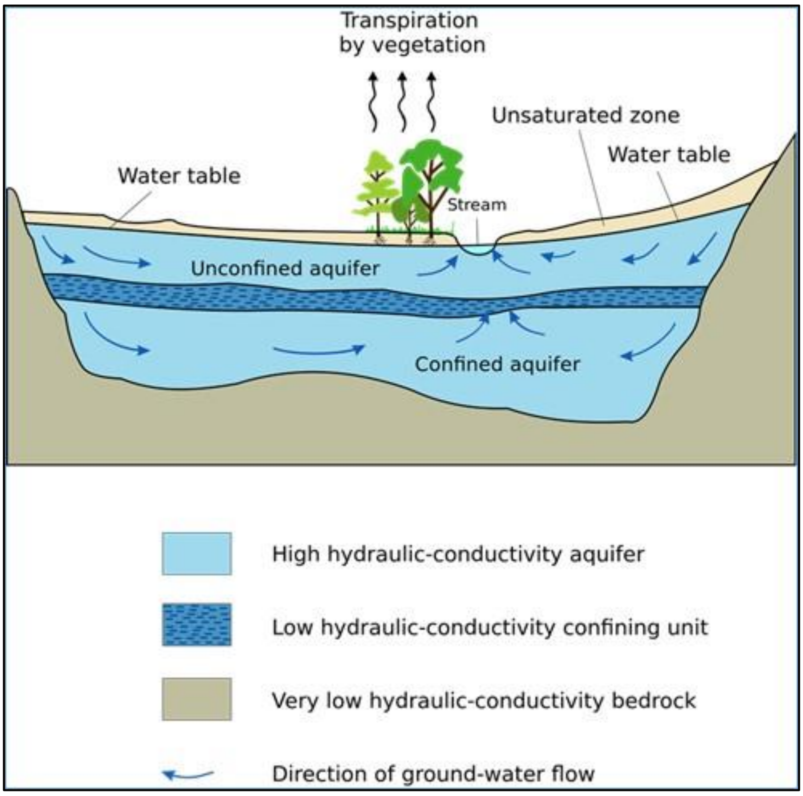


Figure 18-4: Simplified and graphical depiction of the aquifer concept for HA8

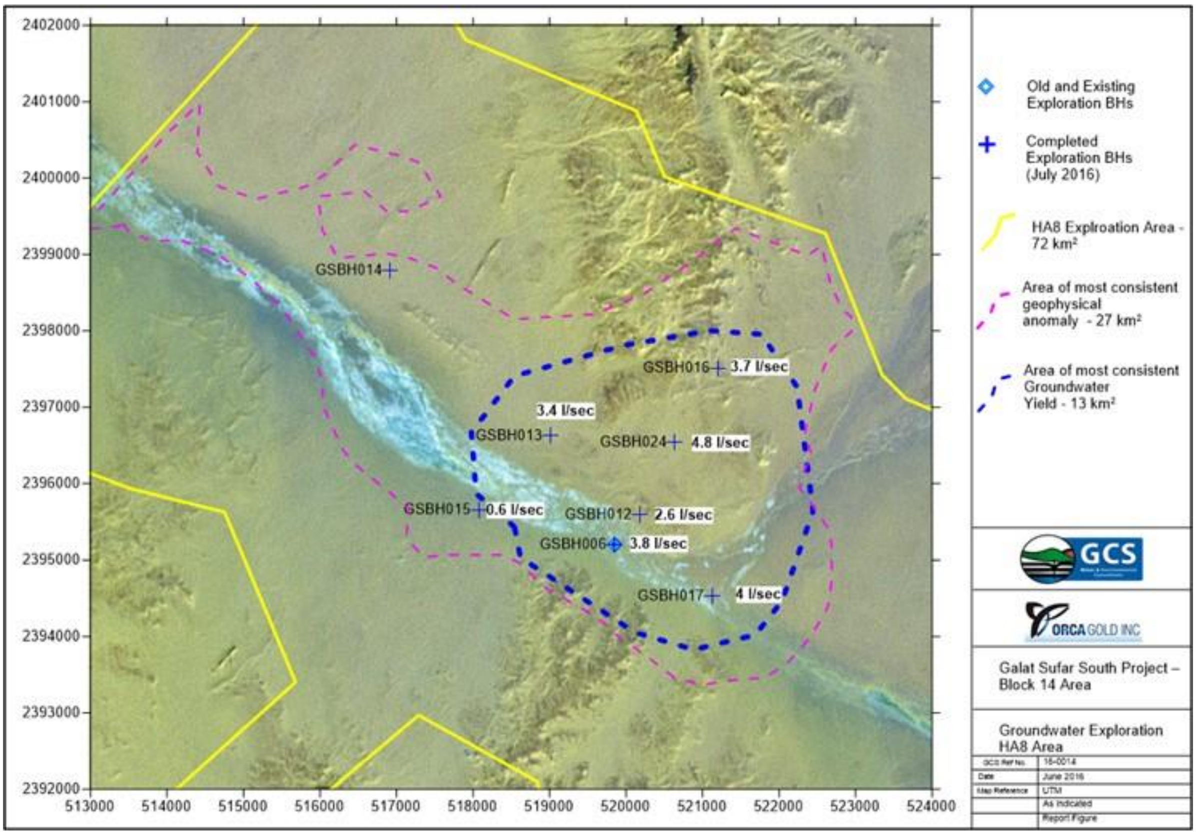


Figure 18-5: Observed Blow Yields for Area HA8

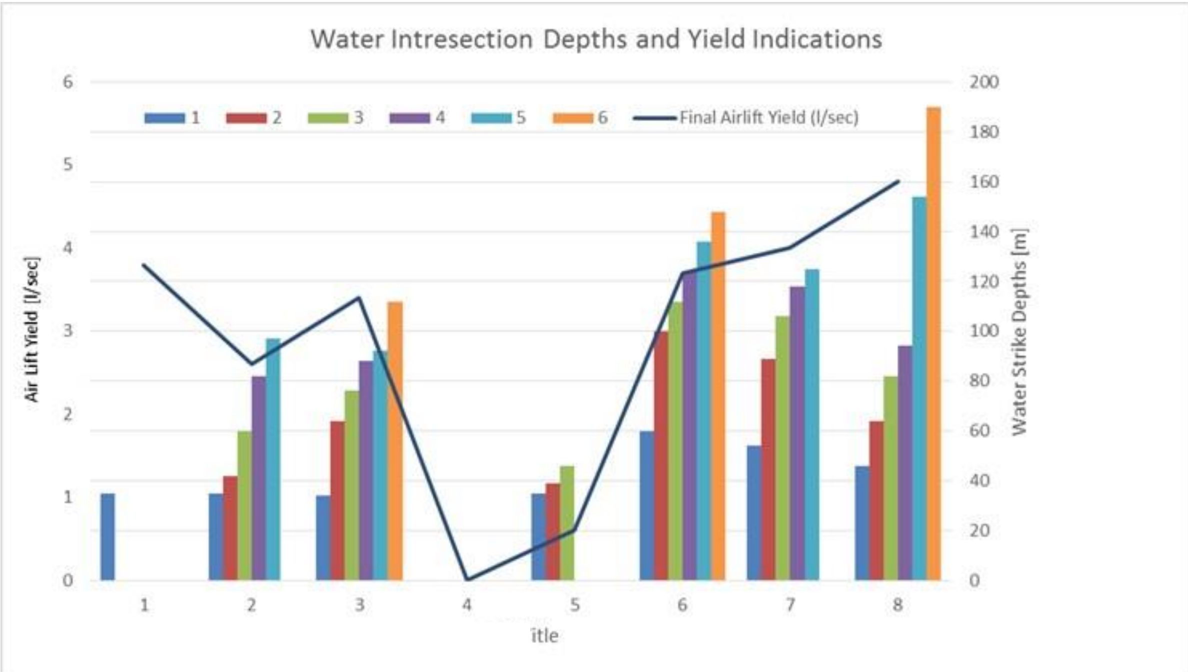


Figure 18-6: Observed Water Strike Depths and Final Blow Yields

Table 18-2: Range of indicative groundwater resource available in HA8

Block	Available Aquifer Area (m²)	Aquifer thickness (m)	Net Spec Yield (%)	Groundwater in storage (million m³)
HA8	13 874 541	16.0	0.10	**22**
HA8	13 874 541	20.0	0.10	**28**
HA8	13 874 541	16.0	0.15	**33**
HA8	13 874 541	20.0	0.15	**42**

18.2 Electric Power

The Ministry of Water Resources and Electricity divided the National Electricity Cooperation (NEC) into two different companies, i.e. Sudanese Electrical Transmission Company Ltd (SETCO) and Sudanese Electrical Distribution Company Ltd (SEDC), to manage power supply and distribution infrastructure and networks in Sudan.

18.2.1 Sudanese Electrical Transmission Company Ltd (SETCO)

SETCO is responsible for the electrical power grid & power supply. Web site: http://setco-sd.com/index.php/en/

18.2.2 Sudanese Electrical Distribution Company Ltd (SEDC)

SEDC is responsible for distribution of the electricity. Web site: http://www.sedc.com.sd/

18.2.3 Current Power Supply Infrastructure

The electrical power in the Sudan produced from both hydro and thermal generation, the network is depicted in Figure 18-7.

The total output is approximately 1500 MW. The actual consumption of domestic & bulk users in Sudan is approximately 1750 MW, so there is a shortage in the power production in Sudan of approximately 250 MW. The shortfall in power generation is supplemented from the Ethiopian grid.

Due to the shortfall in supply, every factory or an industrial project connected to the local power grid utilises standby diesel generators in case of sudden power cuts.

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

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

18.2.4 Project Power Supply Demand

The expected power demand for the process plant, based on installed motor sizes, is listed in Table 18-3.

18.2.5 Project Power Supply Options

Two possible supply options were investigated:

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

18.2.5.1 Option 1: Supply from the local power grid

The closet take-off point substation or electrical line to GSS is located in the town of Abu Hamad, approximately 200 km south from GSS site.

The cost of construction of a 220 kV power line to connect GSS to the local power grid is 221 thousand $/km and a 220 to 33 kV sub-station at GSS is USD22 million (as per recent data supplied by SETCO). The total estimated cost to connect to the grid will be USD66.2 million

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.

18.2.5.2 Option 2: Self generation

The project will utilise 3 x 3 MW and 3 x 1 MW containerised diesel engine driven turbine generators. The estimated investment is USD8.1 million, excluding ancillary electrical equipment and installation of the system. The current cost of diesel fuel, including transportation to GSS site is: $ 0.65/l .Based on the consumption figures of 0.23 l/kWh, a power cost of $ 0.15/kWh was used in the operational cost analysis.

18.2.5.3 Selected Option

In order to select the most viable power supply option, a capital and process power cost trade-off was carried out for the two identified options over the Life-of-Mine production schedule. For the purpose of the trade-off the grid supply option includes the capital cost for back-up generation and excludes maintenance and replacement costs for capital equipment. The results, as per Figure 18-8, shows that there is no break-even point, and if one considers the fact that it might be possible to move or sell the generator sets, while ownership of the distribution network is transferred to SETCO, Option 2 (Self generation) appears to be the most economical option at this stage.

Table 18-3: Expected Installed Power

Area Description	Total kW
Primary Crushing Total	240
Secondary & Tertiary Crushing Total	674
Material Handling Total	96
Coarse Ore Stockpile Total	73
Fine Ore Stockpile Total	94
Primary Mill Circuit Total	4,654
Re-grind Mill Circuit Total (Not currently included in demand calculation)	
CIL Total	1,045
Elution Total	1,102
Electrowinning Total	166
Gold Room Total	103
Reagents Total	61
Water Treatment & Storage Total	299
Utilities & Services Total	668
Water Treatment & Storage Total	275
Fuel Storage Total	-
Dewatering Total	821
Tailings Storage Facility Area Total	-
Auxiliary	1,037
Grand Total	11,408

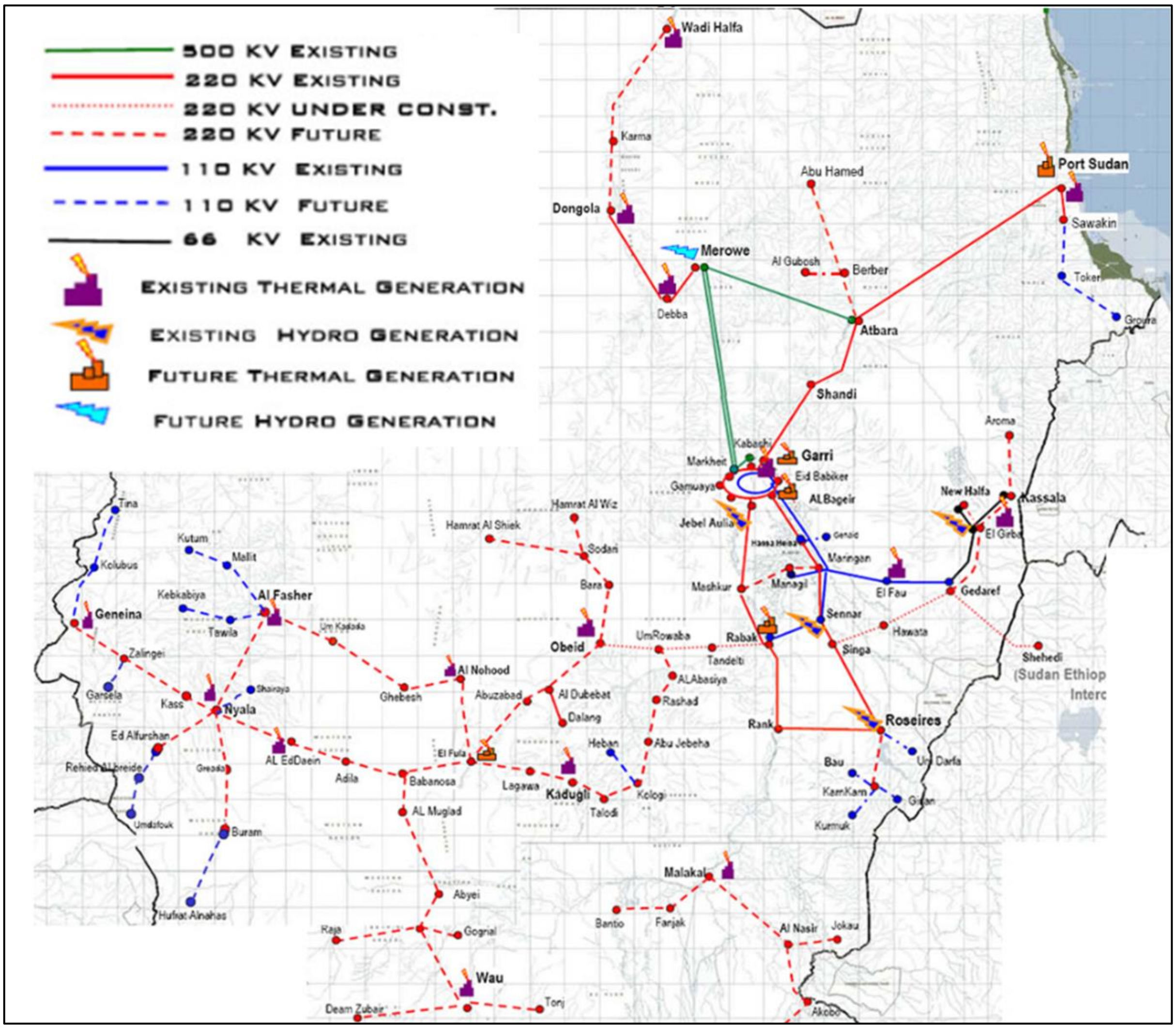


Figure 18-7: Sudanese Power Generation & Distribution Network

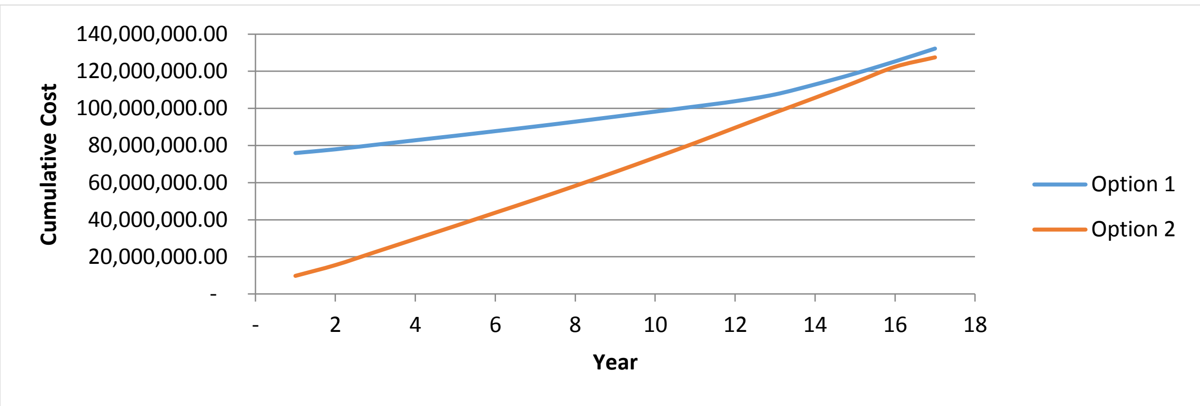


Figure 18-8: Power Supply Options Trade-off

18.3 Fuel Supply & Storage

The two principal consumers of diesel will be the mining contractor and diesel for power generation. The mining contractor will be responsible for its own fuel storage and fuel deliveries.

Very limited fuel storage will be available on site; an allowance was made for nine 100 m^3 above ground storage tanks, as per Figure 18-9, that will provide approximately 15 days of storage at full design capacity. Table 18-4 provides a summary of the storage and diesel delivery cycle requirements. In order to continuously operate at full capacity and have at least 25% or 4 day buffer capacity available, two 25 m^3 tankers needs to supply the facility every day or four 25 m^3 tankers every other day. There is adequate fuel supply available in Sudan and local transport contractors in the area, in order to supply the plant with the required fuel, the supply of fuel to GSS is considered to be a low risk.

Figure 18-9: Typical Diesel Storage Tank

Table 18-4: Fuel Consumption & Storage Analysis

Description	Value	
Generator Consumption	0.27	l /kVAh
	0.23	l /kwh
	16,7 million	l /year
	16,7 thousand	m^3/year
No of tanks on site	9	
Available	900	m^3
Delivery cycles/year	18.54	
Required	57.14	m^3/day
Storage	15.75	days
Storage Buffer	25%	
	3.94	days
Refill cycle	11.81	days
Refill vol	675	m^3
25 m^3 tankers / cycle	27	
Tankers / day	2	

18.4 General Infrastructure

18.4.1 Warehousing

An allowance for on-site reagents and general stores was made.

18.4.2 Workshops / Maintenance

An allowance for an on-site plant workshop was made.

The project plan does not allow for any truck maintenance workshops for the mining contractor. It is anticipated that the mining contractor will provide a concrete pad area with a clean-up sump for vehicle fuelling, and light maintenance.

18.4.3 Camp / Accommodations

An allowance was made for an on-site camp; although this facility will need to be designed once all labour requirements have been determined.

18.4.4 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.

18.4.5 Security

The project will be monitored 24-hours a day by a contract security service. Access to the plant will be only through the main security gate.

18.4.6 Health & Safety

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

The Tailings Storage facility, ponds and other facilities containing cyanide may have secondary fencing to restrict access to these areas.

The open pit will be bunded off with an earth bund to prevent accidental entry.

18.4.7 Laboratory

An allowance for an on-site containerised laboratory was made, excluding laboratory equipment. The requirements of the laboratory need to be addressed during the next phase of the project.

18.4.8 Sewage

There is no access to sewage treatment works on the GSS site and sewage shall be managed on site via septic tanks and soakaways.

18.4.9 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.

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 negotiated in the future should the project progress.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

Mineesia Ltd, a UK based consultancy, was engaged in 2014 to undertake a preliminary assessment of the environmental and social conditions relating to the Block 14 project and oversee the planning and collection of environmental and social baseline data. This study is aimed at presenting an overview of the current environmental and social issues that should be considered in the Environmental Impact Assessment (EIA) to be completed for the Project. Currently the Project is at exploration stage. The study is based on a site visit between November and December 2014; 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 International Best Practice

The Project EIA will be undertaken in compliance with Sudanese regulations, as well as in compliance with the standards of the International Finance Corporation (IFC) and other international environmental and social development agreements that Sudan is party to. Baseline data collection and assessment has been designed to meet best international practice, as exemplified by the following:

- Equator Principles III;
- IFC Performance Standards on Social and Environmental Sustainability;
- Air, noise and water quality standards adopted by the IFC, as established by the World Health Organization; and
- International Council of Mining and Metals best industry practice guidance documents on community development planning and mine closure.

20.3 Legal Setting

Sudan has developed an integrated permitting infrastructure for formal mining operations. Permits are provided in the form of a Concession Agreement. Current exploration activities are undertaken under the auspices of this Concession Agreement, which gives MSMCL the right to search for and win minerals, through the establishment of a mine operation. This right is to be exercised in a responsible manner, and the mining company has the responsibility to manage the effects of the mining activity in such a way as to mitigate any negative impact. The Concession Agreement is a governing document which allows for environmental management at all stages of the Project, and includes input from the local authorities. The scope and nature of each stage will determine the level of detail required for an Environmental Protection and Management Programme (EPMP).

During the prospecting and exploration stage, the EPMP should address those activities directly and indirectly impacting on the environment. As such, MSMCL implemented an Exploration Statement for their activities. This statement 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.

In addition, the Project has a number of procedures relating to emergency response, fire and driver safety, risk assessment and document control. A management system is being developed to include environmental and social components in conjunction with EIA work. Terms of Reference for the EIA for the Project have been developed, identifying key environmentally and socially sensitive areas.

It is intended that all environmental parameters are documented in the EIA report, with potentially significant impacts arising from the proposed Project examined in further detail. This Terms of Reference was presented to the SMRC (the local authorities) for comment in May 2015 (as part of the permit renewal process), and establishes the data to be collected and potential impacts that are to be addressed in the EIA. The final EIA will be submitted to SMRC for comment prior to obtaining approval from the Higher Council for Environment and Natural Resources prior to the commencement of mineral exploitation.

20.4 Baseline Project Description

The project layout is presented in Figure 17-4. The Terms of Reference submitted to SMRC has been developed 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) or Carbon-in-Pulp (CIP) extraction methods;
- 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/CIP 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 between 1.5 million m^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; and
- Accommodation camp – to house employees.

The Project Design is still being developed, and feedback from the EIA process will be incorporated into the design and site layout. This will include site selection methods that include environmental and social criteria in the selection process, as well as technical and financial criteria.

20.5 Environmental and Social Setting

The site is located in a remote location, with no human settlements nearby. There is numerous artisanal and small scale mining operations in the vicinity of the Project, although many of these are illegal and unlicensed. 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. 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 and around Abu Hamad, which is the closest community and currently used as a base of operations for artisanal and small scale mining operations. Abu Hamad is likely to become a staging post for the Project, which would result in potential effects, which will be determined through the social component of the EIA.

The Project engages with the artisanal and small scale miners on a regular basis, in accordance with the Community Consultation Procedure.

Community consultation plays an important role in the overall operation of the Project, and the purpose of the procedure is to establish a framework for open, honest and informative communications with interested parties.

Consultation is a two-way discussion enabling communities and stakeholders to make informed decisions and for managing relationships with artisanal miners in the area. It also provides a method of recording any concerns or issues as they may arise. This information will be used to inform the social baseline for the EIA, and develop suitable mitigation measures for potential adverse impacts on legal activities. It is anticipated that the consultation process will continue as the Project develops, becoming more formal and frequent through the EIA process. The scope of the social assessment will be developed once more details of the Project development are available.

The following sections provide a brief overview of the baseline environmental and social conditions. These conditions will be described further in the EIA and used to assess potential impacts resulting from the Project.

20.5.1 Climate

The Project is located in the Nubian Desert, characterised by a hot dry climate. Climatic data has been collected from Wadi Halfa (approximately 200 km west of site), and is considered to be representative of the site conditions. Weather data is currently being collected from the site to validate the regional climatic data collected from Wadi Halfa.

Regional data shows a maximum summer temperature of 52°C. June is the warmest month with an average temperature of 40.8°C, while January is coolest with an average temperature of 11.2°C. Highest temperatures occur between April to October (summer) and lower temperatures averaging 17°C during the winter. The annual average rainfall for Sudan overall is 40 mm, although the northern regions experience a normal annual average rainfall of 0 mm. There are occasional periods of rainfall, with the maximum daily rainfall recorded at Wadi Halfa being less than 7 mm in a 24 hour period. Site staff has experienced rare, heavy rainstorms since commencing work in the region in 2011, particularly in August 2013 and October 2014.

20.5.2 Air Quality

The nearest established communities to the project area are Abu Hamad, approximately 200 km south of the exploration camp, and Wadi Haifa, approximately 200 km west. There are no anthropogenic sources of air pollution (NOx, COx, SOx) that could affect the site. However, dust is an impact that has both natural and anthropogenic sources. Dust varies widely in its physical and chemical composition, source and particle size. Small dust particles (less than 10 µm) are of concern due to their potential impact on human health, although they are predominantly the result of anthropogenic factors such as mechanical processes and fuel combustion. Larger mineral dust particles (between 10 and 75 µm) do not pose the same health effects and are generally referred to as nuisance dust.

Directional dust data is being collected from the site to demonstrate the site conditions, and will be used to inform the EIA. The results to date demonstrate that there is always a high or very high risk of nuisance dust present at the site, as a result of loose and semi-consolidated sediments with sparse vegetation and windy conditions. Ongoing monitoring is being used to further develop the baseline conditions.

20.5.3 Surface Water

There are no permanent surface watercourses within the Project area. The Nile River is the dominant geographic feature of the Sudan and most of the country lies within the Nile's catchment basin. However, there is a wadi (intermittent, usually dry watercourse) system that runs through the area. Due to sudden rain together with low soil permeability, regional flooding can occur, most recently in 2014. After significant rainfall, the wadis flow for a short period, from few days to a few weeks.

Water quality data for these 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.

20.5.4 Groundwater

The Project area is within an arid zone with no flowing water. There is a well at Talat Abda, which is currently being monitored to establish baseline water quality for the EIA. 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. Historically insufficient financial resources have prevented investment in infrastructure and capacity development and quantitative and qualitative monitoring data are unavailable (Abdo, et al., 2012).

Ongoing monitoring of the groundwater is being conducted on the site. Water quality analysis to date within the project area indicates that groundwater is not suitable for human or agricultural use, and further analysis will be used to detect trends in water quality. Further investigations are underway to locate a source of water supply for the mine operation.

20.5.5 Flora and Fauna

The Project area includes the following habitats:

- Wadis – An intermittent watercourse that rarely flows and delineated by the presence of trees (such as Acacia spp.) that grow along the better defined channels;
- Rocky outcrops – barren outcrops with sparse vegetation that potentially provides sheltered areas for birds, small mammals, reptiles and insects; and
- Desert area – comprising sand dunes and scrubby vegetation. Vegetation includes a mixture of grasses and small shrubs.

Numerous birds have been observed in the area. Camels roam the area (Figure 20-1a) and jackals, wildcats and Ruppell's foxes have been recorded as being present in the vicinity of the camp. No protected species such as gerbils have been identified in the exploration area, although details of smaller mammals and insects (Figure 20-1 b, c) are not well known.

Ongoing faunal data collection is being conducted through the use of wildlife cameras, set to capture images of wildlife at night (Figure 20-1c). To date, these have recorded jackals using the waste facilities of the camp. The cameras have also recorded significant human presence in the area, due to the artisanal and illegal small scale miners. Data will continue to be collected to further inform the EIA.

Flora studies may be required at the locations of facilities, although the barren conditions mean that little vegetation is present. Any land clearance will be managed through an internal permit system to ensure the protection of any vegetation found. This system will be used to mitigate potential disturbance of flora through mine operations.

Figure 20-1: Fauna in the project area. a: Roaming camels. b: Scarab beetle. c: Golden Jackal captured by remote camera. (Source: Orca Gold).

20.5.6 Soils and Land Use

Soils in the area are predominantly sand and gravel, comprising of transported sediments with low agricultural potential. The soil is dry and extremely fragile, generally degraded, unproductive and easily eroded by both wind and occasional flooding. In places the sands are weakly cemented and form an adequate base for vehicles. Agriculture is not viable and land use is limited to seasonal grazing of livestock as well as artisanal and small scale mining.

Potential impacts on soils and land use relate to limiting land use and potential soil contamination from operations. These impacts are likely to be insignificant given the limited existing land use and the magnitude of area affected by the Project. The Project footprint will be the key area impacted, and closure plans will be developed to rehabilitate the footprint prior to abandonment.

20.5.7 Noise

Noise can travel long distances given the flat terrain and noise levels are likely to increase once the operation commences. Potential receptors of noise and vibration are limited to artisanal and illegal small scale miners in the area, which are existing sources of noise and vibration, and exploration drilling operations. These receptors should already have measures to protect their workers from noise and vibration impacts. No communities are currently affected by noise and vibration from the Exploration activities, and none are expected to be affected by mining operations. Workers will be provided with personal protective equipment (PPE) and trained in its correct use. Wildlife is scarce in the vicinity of the project, and no vulnerable species have been identified at the site. Noise and vibration will therefore be managed through the Health and Safety system for the Project.

20.5.8 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.

20.5.9 Infrastructure

Traffic impacts due to exploration are considered to be insignificant, although information of road use will be required to confirm this. Further data will be collected to assess the impact of Project construction and operation on road use, once the Project details are more defined.

Due to the remoteness of the location, there is no other support infrastructure that will be adversely affected by the Project.

20.5.10 Archaeology and Cultural Heritage

The region has been worked historically, with workings dating to circa 4,000BC. Sites of interest have been recorded by Klemm et al, 2013 and 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.

Figure 20-2: Historic artefacts. Left: Pharaonic tomb. Right: Grinding stone (Source: Orca Gold).

Chance finds are defined as archaeological or cultural heritage sites or assets found unexpectedly during the life of a project, from exploration and feasibility study, through to construction and/or operation. Types of cultural heritage that may need to be considered include:

- traditional artefacts on the surface, below ground or in rock shelters;
- axe-grinding or tool sharpening grooves; paintings;
- rock engravings; or
- burial sites.

Cultural heritage can represent irreplaceable sources of life and inspiration and are to be safeguarded. 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.

20.5.11 Land acquisition and Resettlement

There are no local communities in the vicinity of the project, thus 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.

20.5.12 Environmental Management Plan

MSMCL is committed to limiting adverse environmental impacts arising from exploration and project development, in line with international best practice. The company uses best practice procedures to manage impacts and rehabilitate exploration areas after closure. These procedures are part of an Environmental Protection and Management Plan (EPMP) for the site, with the following priorities:

- Protect the health of workers, the public, flora and fauna;
- Manage all waste generated by operations in a responsible manner; and
- Minimise emissions generated by the project, particularly dust.

The EPMP is a live document that can be reviewed and updated on a systematic basis in line with the principles of continual improvement.

20.5.13 Monitoring and Reporting

An integrated monitoring plan will be developed as part of the EIA. However, some environmental monitoring is ongoing on the site. Weather data are continuously collected from the site, with data downloaded on a monthly basis. Directional dust pads are being used to assess the extent of natural dust in the atmosphere, and at least 1 year of data will be available for analysis in the EIA. Passive infrared cameras have been installed on the site at key potential wildlife refuges, such as waste pits, to capture evidence of the presence of wildlife (such as presented in Figure 20.1).

Exploration personnel also take photographs of wildlife when possible.

Groundwater levels are recorded on a monthly basis, and water quality samples have been collected and sent for analysis on a 6 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.

The Project reports its environmental progress internally on a monthly basis, and provides information to the SMRC (National authority) on a quarterly basis. Through the EPMP and EIA report, a number of plans and procedures will be developed for the Project, including erosion control, waste management and closure plans. The results of any studies conducted on the site will be used to inform the Project design, so as to minimise adverse impacts from the Project and reduce long-term costs of environmental compliance.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

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.

For the purpose of a PEA the capital cost estimate needs to be developed to an accuracy level range of -25% to +30% and address 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 1.8 million tonnes of material per annum. For the purpose of this PEA, a contract mining scenario has been assumed.

All costs are estimated in United States dollars (US$) and, unless otherwise stated, are referred to as "US$, $, or USD" or is indicated without a specific currency.

21.2 Capital Cost Summary

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 reported as EPCM/Home Office Cost and Field Costs, and contingencies have also been estimated, based on percentages of the Direct Capital Cost Estimate. Risk amounts are specifically excluded from this estimate.

Sustaining costs were not included in the Capital Cost Estimate, but an annual percentage allowance of the initial installed capital cost was included in the Financial Model cash flow and will be financed from the project revenue.

The total capital cost (CAPEX) is estimated at US$138 million. This estimate is inclusive of US$101 million direct costs, US$17 million indirect costs and US$20 million contingency. The direct cost estimate is inclusive of the total estimated TSF cost; this cost will however be phased over the life of the project, see Table 22-3. The total pre-production capital cost required is US$122 million.

A summary of the estimated capital requirements is shown in Table 21-1.

21.3 Direct Capital Costs - Mining

Due to the use of mining contractors, who will provide the mining fleet, the capital costs only include a single lump sum value of US$5 million has been included in Year 1 to cover items such as:

- Principal mine office, including fittings, furniture and computer systems;
- Fuel Storage and Dispensing Facility; and
- Light vehicles for principal mining team.

Table 21-1: Capital Cost Summary ($'000)

Direct Cost		
0000	General / Plant Wide Infrastructure	$ 3,548
0100	Tailings Storage Facility Area [a]	$ 18,665
0500	Plant Area	$ 37,867
0300	ROM Storage Area (Included above)	$ -
0700	Mining & Waste Rock Area - GSS	$ 5,000
0800	On Site Infrastructure	$ 16,403
0900	Off Site Infrastructure	$ 20,122
Sub -Total (Direct Cost)		**$ 101,494**

Indirect Cost	
EPCM Fees / Home Office Cost	$ 11,619
Field Cost	$ 5,461
Sub -Total (Indirect Cost)	**$ 17,081**

Total Installed Cost (TIC)	
Direct Cost + Indirect Cost	$ 122,713

Contingency	
@ 19.0% of Direct Cost	$ 20,070

Total Capital Cost [c]	
TIC + Contingency	**$ 138,646**

Total Capital Cost (Pre-production)	
TIC + Contingency – Post Production TSF Cost [b]	**$ 122,580**

(a) Total TSF Cost over LOM
(b) 70% of TSF direct cost estimate
(c) Excludes SiB costs

21.4 Direct Capital Cost Estimate – Plant Facility & Infrastructure

The plant facility and infrastructure was demarcated into different areas, with unique area numbers allocated to the areas, as per Table 17-1. Cost centres, as per Table 21-2, were allocated to each of the areas.

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. The balance of the cost centre prices were generally factorised from equipment costs, as per Table 17-1.

The estimated cost was benchmarked against similar sized projects and was found to be within the cost range of the various projects.

The two high cost items, as per Figure 21-1 are related to Area 0100 (Tailings Storage Facility) & Area 0910 (Water Supply Infrastructure); the basis of estimate of these two items is based on the following:

- Area 0100 (Tailings Storage Facility): Bill of Quantities & rates based on similar size facility; and
- Area 0910 (Water Supply Infrastructure): Bill of Quantities & rates based on the concept study, as per section 18.1.

Table 21-2: Cost Centres & Basis of Estimate

Cost Centre	Basis of Estimate	Typical Factors Applied (Fraction of Equipment)		
		Material	Labour	Total
Equipment	Priced Equipment List	1	0.2	1.2
Piping	Factored	0.2	0.4	0.6
Instrumentation	Factored	0.1	0.06	0.16
Electrical	Factored	0.15	0.25	0.4
Site Preparation	Factored	0.01	0.05	0.06
Concrete	Factored	0.06	0.29	0.35
Structural Steel	Factored	0.23	0.17	0.4
Buildings	Factored	0.01	0.06	0.07
Painting	Factored	0.02	0.03	0.05

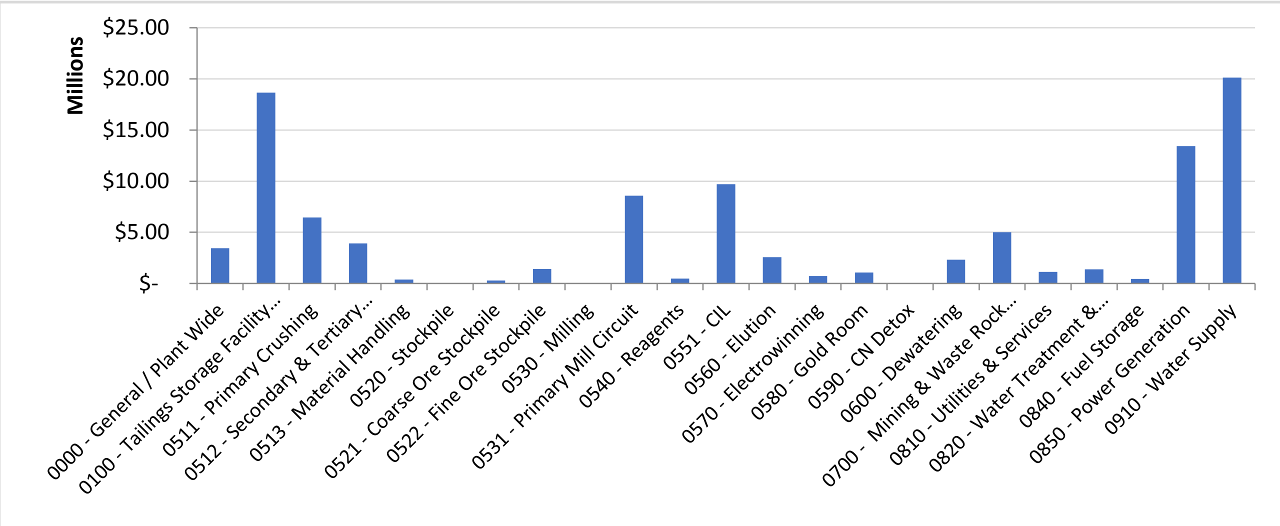


Figure 21-1: CAPEX Breakdown per Area

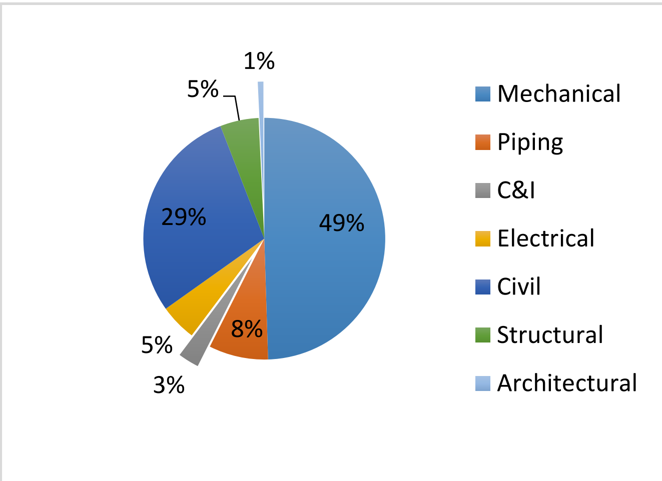


Figure 21-2: Direct Cost Split per Discipline

21.5 Indirect Capital Cost Estimate

A breakdown of the pre-production project Indirect Costs is provided in Table 21-1.

21.5.1 Engineering, Procurement, Construction Management (EPCM) & Home Office Cost

EPCM services and Owner's Costs were factored from the direct costs and include project management & project controls, engineering services, and procurement services. The cost is estimated as 11.5% of the Direct Cost.

21.5.2 Field Cost / Construction Indirect

Construction indirect costs were factored from the total direct labour costs at 21.5%.

Construction indirect costs typically include security, health and safety, environmental compliance, room and board, construction management staff (owner's team), training, recruiting, lodging of employees, mobile and communication, internet, fuel cost for owners' vehicles, repairs and ancillary equipment rental for construction.

21.6 Estimating Accuracy

Where applicable, risk, engineering maturity and price source estimate adjustment factors (see Table 21-3) were applied to each line item, based on the level of engineering and perceived risks associated with the specific unit process. The factors were applied to the Direct Cost estimate in order to obtain a low and high cost range for each unit process or area.

Table 21-3: Estimate Adjustment Factors

Risk / Opportunity	Factor	Best	Most Likely	Worst
Plot Plan Clearance / Real Estate availability	1.11	5%	10%	20%
Non-proven Scope Items	1.16	5%	15%	30%
Contractor dependency	1.10	5%	10%	15%
Re-use existing facilities / Retrofitting	0.91	5%	-15%	0%
Demolition Works	1.15	5%	15%	25%
Soil Data	1.07	0%	5%	20%
Availability of Utilities	1.10	5%	10%	15%
Environmental Aspects	1.20	10%	20%	30%
Imports, Exports, Duties	1.15	10%	15%	20%

Engineering Maturity Level of Project Definition	High	Low
0%	1.30	0.90
30%	1.20	0.91
40%	1.15	0.92
50%	1.12	0.94
60%	1.10	0.95
70%	1.08	0.96
75%	1.06	0.97
80%	1.04	0.98
90%	1.02	1.00

Price Source	Factor
Supplier Quote	8%
Estimated from take-offs & rates	10.0%
Escalated rates	10.0%
Budget Price	15.0%
Factorised	25.0%

The low and high cost totals for each area were used to generate a set of random values between the low and high cost totals. The random values were used to generate a Monte Carlo Histogram, see Figure 21-3. Based on the normal distribution there is an 80% probability that the Direct Cost will be between $ 99 million and $132 million, which equates to an estimated accuracy range of -1.7 +30.1 %.

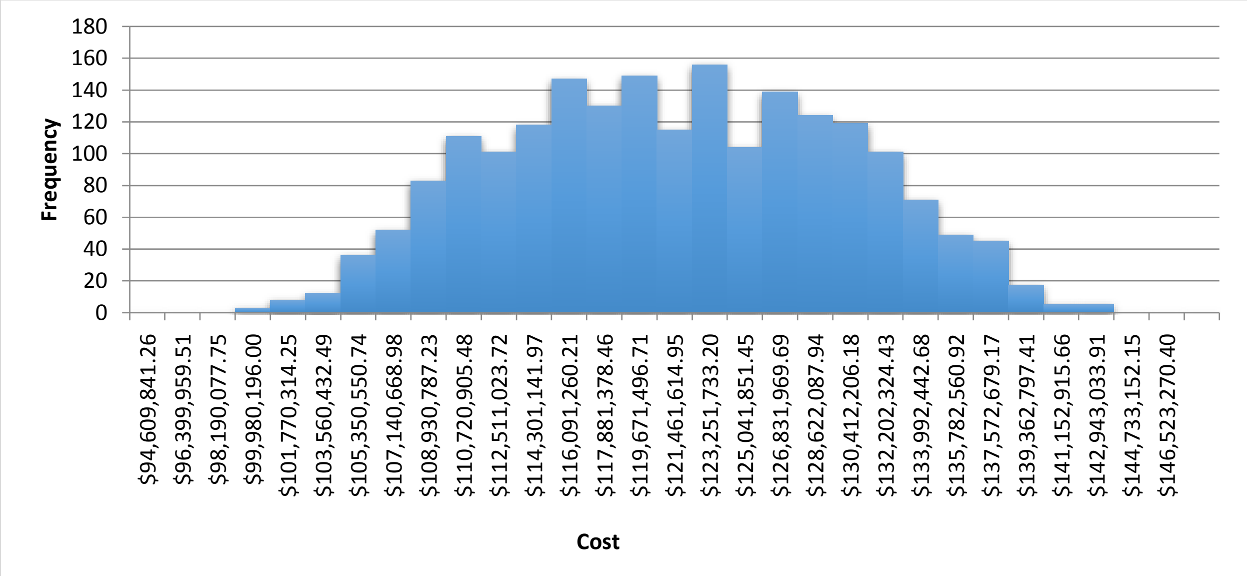


Figure 21-3: Monte Carlo Histogram

21.7 Contingency

For the purpose of this PEA, a normal distribution was developed using the low-high Direct Cost estimates as determined in section 21.6. The median value of the range of values was used to calculate the contingency at 19% of the Direct Cost, refer to Figure 21-4. Indirect Costs were not included in the contingency analysis. The purpose of the contingency is not to account for the estimating accuracies or error, but is rather an allowance for scope items not included or not sufficiently developed during the PEA.

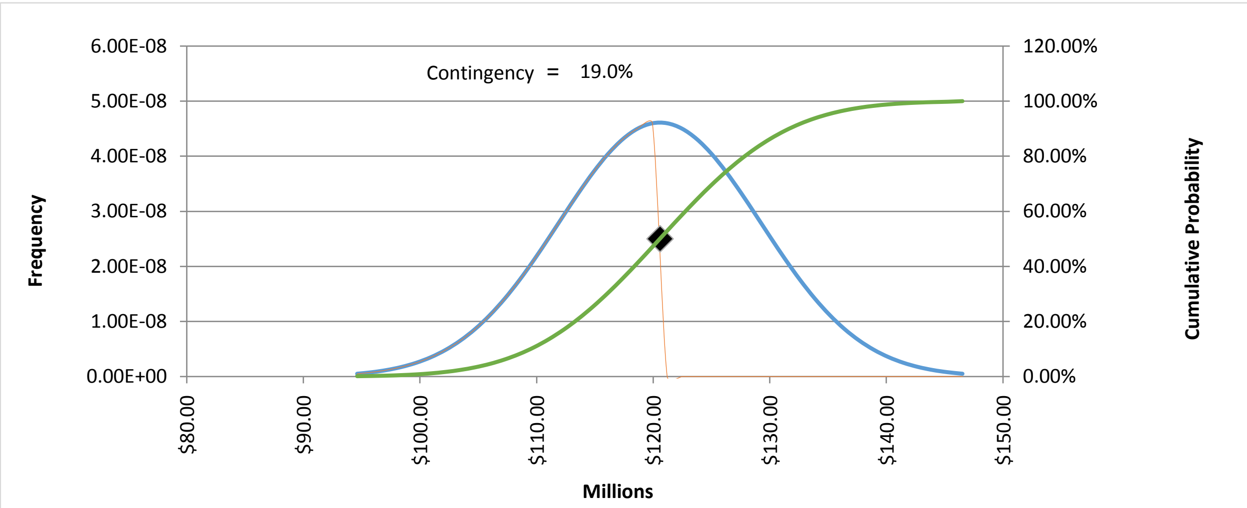


Figure 21-4: Contingency Analysis – Normal Distribution

21.8 Operating Costs – Mining

As the bulk of the mining costs would be related to a mining contract, the mine operating costs were derived from three existing mining contracts using similar equipment awarded in West Africa in the last 3 years.

Base mining costs were calculated for the assumed fleet and dependant on material type (Crusher Feed/Waste and Oxide + Transitional/Fresh) and destination.

The cost included components for:

- Loading costs;
- Fixed hauling costs;
- Drill & Blast costs;
- Ancillary costs; and
- Mine adminastration costs.

Base mining costs do not include any time haul trucks spend travelling up or down in-pit ramps.

Table 21-4 shows the breakdown of fixed mining costs for the assumed fleet.

Table 21-4: Fixed Mining Costs

Deposit	Material	Loading	Hauling	D & B	Ancillary	Mine Admin	Total
GSS	Oxide CF	$0.26	$0.38	$0.89	$0.49	$0.83	$2.86
GSS	Fresh CF	$0.29	$0.38	$1.00	$0.49	$0.83	$3.00
GSS	Oxide Waste	$0.26	$0.49	$0.74	$0.49	$0.83	$2.81
GSS	Fresh Waste	$0.29	$0.49	$0.74	$0.49	$0.83	$2.84
WD	Oxide CF	$0.26	$0.32	$0.89	$0.49	$0.83	$2.79
WD	Fresh CF	$0.29	$0.32	$1.00	$0.49	$0.83	$2.93
WD	Oxide Waste	$0.26	$0.37	$0.74	$0.49	$0.83	$2.69
WD	Fresh Waste	$0.29	$0.37	$0.91	$0.49	$0.83	$2.90

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 $0.034/t/10m vertical lift.

The fuel price used for mining calculations was $0.60/l.

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 $8.48/t for crusher feed contributed by WD.

Figure 21-5 shows the unit mining cost for the life of the operation compared to the total tonnes mined in each period. This does not include crusher feed re-handle from WD to the GSS processing plant. The significant increase in the last three years is due to the completion of most of the shallow oxide pits, resulting in material being sourced from progressively deeper benches in the two larger GSS pits.

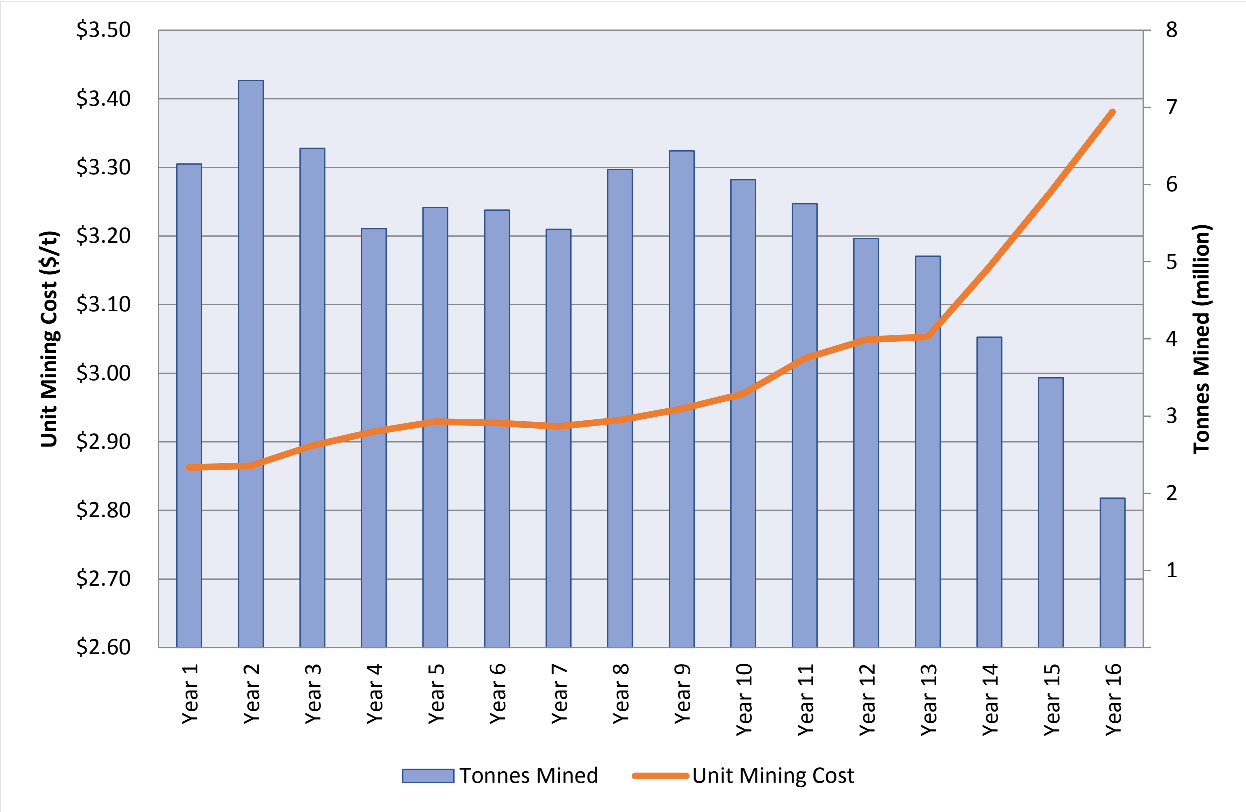


Figure 21-5: Unit Mining Cost & Tonnes Mined by Year

21.9 Operating Cost – Plant & Infrastructure

Operating Cost (OPEX) was divided into different cost centres and Fixed and Variable were applied to the costs centres for each of the different processed material type, as per Table 21-5.

For the purpose of the PEA, processing operating costs for similar operations were used; the exception to this was the variable power costs. The theoretical power draw was calculated from each of the processed material type Work Indices values, as per section 13.3.1, and installed power from the mechanical equipment list. The power draw was used calculate the power cost / tonne based on the generator consumption and a diesel fuel price of $0.65/l, see Table 21-6.

21.10 Exclusions

The following were not included in this estimate.

- Costs associated with scope changes;
- Escalation;
- Financing costs;
- Foreign exchange fluctuations;
- Cost associated with schedule delays such as those caused by:
  - scope changes
  - unidentified ground conditions
  - labour disputes
- Environmental permitting activities;
- Sustaining capital, an allowance for sustaining costs is included in the financial model; see Table 22-3; and
- Mine closure and rehabilitation cost; it is assumed that this will be financed from the sustaining cost.

Table 21-5: Operating Cost per Ore Type

COST CENTRE	Fixed US$/y	Oxide	Trans	East Fresh	Main Fresh	Wadi Fresh
		Variable	Variable	Variable	Variable	Variable
		US$/t	US$/t	US$/t	US$/t	US$/t
Power	542,298	5.06	5.39	6.14	5.55	5.18
Operating Consumables	0	5.50	5.11	5.11	5.11	5.11
Maintenance Materials	2,170,004	0.19	0.19	0.19	0.19	0.19
Laboratory	883,200	0.00	0.00	0.00	0.00	0.00
Process & Maintenance Labour	3,766,616	0.00	0.00	0.00	0.00	0.00
Administration Labour	3,059,705	0.00	0.00	0.00	0.00	0.00
General & Administration Costs	3,861,344	0.00	0.00	0.00	0.00	0.00
TOTAL	**14,283,166**	**10.75**	**10.69**	**11.44**	**10.85**	**10.48**

Table 21-6: Power Draw Calculations

Area	Total Installed	Mill	Total (excl Milling)	Absorbed (Factor = .90)	kWh/t (excl Milling)	Oxide	Trans	East Fresh	Main Fresh	Wadi Fresh
Primary Crushing Total	239.80		239.80	215.82	0.84	0.84	0.84	0.84	0.84	0.84
Secondary & Tertiary Crushing Total	674.00		674.00	606.60	2.36	2.36	2.36	2.36	2.36	2.36
Material Handling Total	96.00		96.00	86.40	0.34	0.34	0.34	0.34	0.34	0.34
Coarse Ore Stockpile Total	72.55		72.55	65.30	0.25	0.25	0.25	0.25	0.25	0.25
Fine Ore Stockpile Total	93.65		93.65	84.29	0.33	0.33	0.33	0.33	0.33	0.33
Primary Mill Circuit Total	4,654.30	3,900.00	754.30	678.87	2.64	12.46	14.72	19.84	15.80	13.24
CIL Total	1,045.48		1,045.48	940.93	3.66	3.66	3.66	3.66	3.66	3.66
Elution Total	1,102.07		1,102.07	991.86	3.86	3.86	3.86	3.86	3.86	3.86
Electrowinning Total	165.75		165.75	149.18	0.58	0.58	0.58	0.58	0.58	0.58
Gold Room Total	103.10		103.10	92.79	0.36	0.36	0.36	0.36	0.36	0.36
Reagents Total	60.99		60.99	54.89	0.21	0.21	0.21	0.21	0.21	0.21
Water Treatment & Storage Total	574.00		574.00	516.60	2.01	2.01	2.01	2.01	2.01	2.01
Utilities & Services Total	668.00		668.00	601.20	2.34	2.34	2.34	2.34	2.34	2.34
Fuel Storage Total	-		-	-	-	-	-	-	-	-
Dewatering Total	821.37		821.37	739.23	2.88	2.88	2.88	2.88	2.88	2.88
Tailings Storage Facility Area Total	130.00		130.00	117.00	0.46	0.46	0.46	0.46	0.46	0.46
Water Supply					1.67	1.67	1.67	1.67	1.67	1.67
Grand Total	**11,758.06**			**5,974.25**	**24.93**	**34.62**	**38.96**	**44.07**	**40.03**	**37.48**
					Power Cost / t (No Re-grind)	**5.06**	**5.39**	**6.14**	**5.55**	**5.18**

22 ECONOMIC ANALYSIS

22.1 Introduction

The economic analysis is based on Indicated and Inferred Mineral Resources and mine schedule as per Table 16-13.

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. The process recoveries, metal prices, operating costs and refining charges, royalties and capital expenditures (both initial and sustaining) were also taken into account.

The price forecast of gold is given in US$. The financial assessment of the project is carried out on a "100% equity" basis and the debt and equity sources of capital funds are ignored. No provision is made for the effects of inflation. Results are given after taxation. Current Sudan tax regulations are applied to assess the tax liabilities. All amounts in this section are presented in US$. Discounting has been applied from the first year of operation (Year 2 onward).

The model reflects the base case and technical assumptions shown in the foregoing sections of this report.

This PEA is based on the economic analysis for a Contract Miner Operated scenario.

22.2 Model Inputs and Assumptions

The model inputs and assumptions used in the economic analysis are summarised in Table 22-1, and unless otherwise stated is used in the model.

Table 22-1: Model Inputs & Assumptions

Model Inputs	Unit / Value
Base Currency :	USD / US$
Base Date :	July 2016
Money Terms :	July 2016
Sudan Royalty (charged against Revenue)	7%
Sudan Tax Rate	15%
NPV Discount Rate	7%
Required IRR	20%
Metal Price Scenario – Fixed	$1200 / oz
Refinery Charges & Shipping	$5/oz
Oxide Metal Recovery	92%
Transitional Metal Recovery	87%
GSS Fresh Metal Recovery	80%
Wadi Fresh Metal Recovery	83%
Money Terms :	July 2016

Assumptions
ton or t = metric tonne (1,000 kg)
oz or troy ounce = 31.1034768 gram or g
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

Capital expenditures for the project have been scheduled with the bulk of the expenditure happening within the first two years prior to production, see Table 22-2.

Sustaining capital or SiB Plant, Mining and TSF expansion costs were 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. It is assumed that the rate is fixed and not linked to the gold spot price.

22.2.3 Cost of Sales

For the purpose of this PEA, Cost of Sales includes freight and refining costs. A value of US$5.00/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 (Year 2), and can be summarised as follows:

- Initial pre-production capex depreciated over the total LOM, based on units recovered.
- 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$33.75 million up to date (Jun 30, 2016).
- The 1% annual sustaining capital is assumed to be largely
  - o repairs and maintenance, or
  - o 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 section 0 and 21.9, will be included in the cash flow.

22.3 Financial Model

The operating costs, capital costs, mining & 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 summarised in Table 22-4. On a pre-tax basis, the project has a Net Present Value (NPV) of US$156 million at a discount rate of 7%, an Internal Rate of Return (IRR) of 25%, and undiscounted payback after 3 years of production; on a post-tax basis the NPV is US$128 million at a discount rate of 7%, the IRR is 22%, and the and the undiscounted payback period is 4 years after start 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: Capital Cost Expenditure

Direct Cost	Year 0	Year 1
0000 - General / Plant Wide	33%	67%
0100 - Tailings Storage Facility Area		30%
0500 - Plant Area		
0510 - Crushing		
0511 - Primary Crushing	57%	43%
0512 - Secondary & Tertiary Crushing	55%	45%
0513 - Material Handling	83%	17%
0520 - Stockpile	0%	0%
0521 - Coarse Ore Stockpile	59%	41%
0522 - Fine Ore Stockpile	59%	41%
0530 - Milling	0%	0%
0531 - Primary Mill Circuit	55%	45%
0540 - Reagents	55%	45%
0550 - Adsorption	0%	0%
0551 - CIL	57%	43%
0560 - Elution	57%	43%
0570 - Electrowinning	57%	43%
0580 - Gold Room	57%	43%
0590 - CN Detox	0%	0%
0600 - Dewatering	57%	43%
0300 - ROM Storage Area	0%	0%
0700 - Mining & Waste Rock Area - GSS		40%
0800 - On Site Infrastructure	0%	0%
0810 - Utilities & Services	57%	43%
0820 - Water Treatment & Storage	54%	46%
0830 - Sewage Treatment	0%	0%
0840 - Fuel Storage	62%	38%
0850 - Power Generation	73%	27%
0900 - Off Site Infrastructure	0%	0%
0910 - Water Supply	40%	60%

Indirect Cost	Year 0	Year 1
EPCM Fees / Home Office Cost	50%	50%
Field Cost		100%
Contingency	20%	80%

Table 22-3: Sustaining Cost

LOM	Sustaining Cost Phasing – Plant (% of Direct Cost)	TSF Phasing (% Balance of Pre-Production Direct Cost)	Sustaining Cost Phasing Mining (% of Direct Cost)	Sustaining Cost (incl. Plant Mining cost , TSF expansion)
Year 0				-
Year 1				-
Year 2	1%		60%	4,014,944.83
Year 3	1%			1,014,944.83
Year 4	1%	20%		4,748,049.86
Year 5	1%			1,014,944.83
Year 6	1%			1,014,944.83
Year 7	1%	20%		4,748,049.86
Year 8	1%			1,014,944.83
Year 9	1%			1,014,944.83
Year 10	1%	10%		2,881,497.35
Year 11	1%			1,014,944.83
Year 12	1%	10%		1,014,944.83
Year 13	1%			2,881,497.35
Year 14	1%			1,014,944.83
Year 15	1%			1,014,944.83
Year 16	1%			1,014,944.83
Year 17		10%		1,866,552.52
Year 18				-

Table 22-4: Financial Indicators

Description	Pre-tax	Post-tax
Au Spot Price	1,200	1,200
$NPV_{7\%}$ million	156	128
IRR[(a)]	25%	22%
PVR[(b)]	1.28	1.05
PBP (payback period) [(c)]	3.2	3.7

(a) Required hurdle rate = 20%

(b) PVR > 0 project is economically satisfactory

PVR = 0 project is in an economic breakeven

PVR <0 project is not economically satisfactory

(c) Period in years from start of production

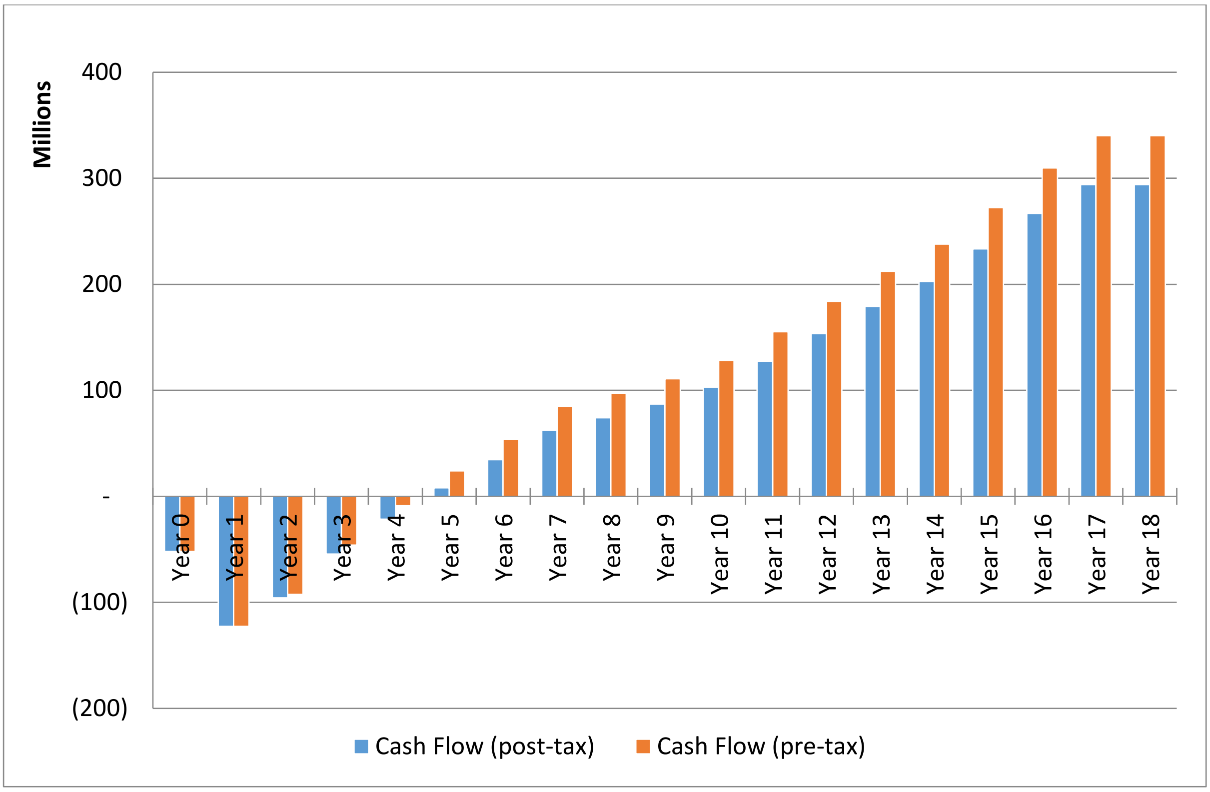


Figure 22-1: Cumulative Cash Flow

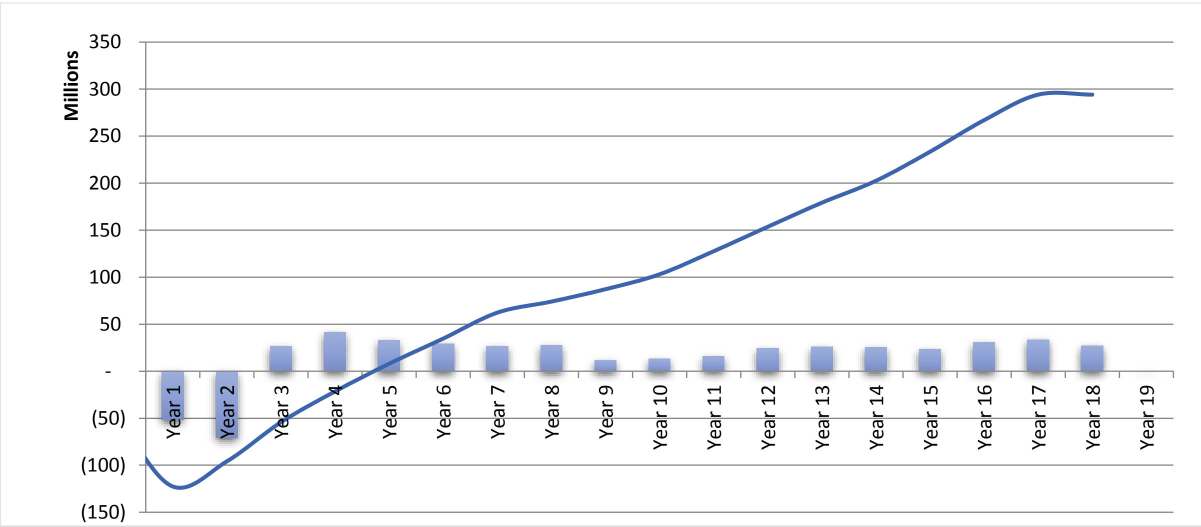


Figure 22-2: Annual vs. Cumulative Post-tax Cash Flow

Table 22-5: Financial Model (post-tax)

Description	Unit	Notes / Source	Tot / Ave	Year 0	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	Year 15	Year 16	Year 17	Year 18
GSS East Fresh Feed Tonnes	t ('000)			-	-	-	-	-	-		1	49	100	353	483	858	824	624	862	993	126	-
GSS Main Fresh Feed Tonnes	t ('000)			-	-	-	-	-	-	-	-	-	-		5	39	201	550	644	776	1,029	-
GSS Total Feed Tonnes	t ('000)			-	-	1,200	1,454	1,454	1,454	1,454	1,502	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,156	-
WD Oxide Feed Tonnes	t ('000)					269	100	6														
WD Oxide Feed Grade	g/t			-	-	2.32	2.97	2.50	-	-	-	-	-	-	-	-	-	-	-	-	-	-
WD Transitional Feed Tonnes	t ('000)			-	-	30	137	23	-	-	-	-	-	-	-	-	-	-	-	-	-	-
WD Transitional Feed Grade	g/t			-	-	1.67	2.49	2.72	-	-	-	-	-	-	-	-	-	-	-	-	-	-
WD Fresh Feed Tonnes	t ('000)					1	113	321	350	350	299											
WD Fresh Feed Grade	g/t			-	-	1.40	2.02	3.01	2.84	2.68	2.68	-	-	-	-	-	-	-	-	-	-	-
WD Total Feed Tonnes	t ('000)					300	350	350	350	350	299											
WD Total Feed Grade	g/t			-	-	2.25	2.48	2.98	2.84	2.68	2.68	-	-	-	-	-	-	-	-	-	-	-
Total Oxide Feed Tonnes	t ('000)					1,458	1,504	1,398	1,341	1,134	1,062	839	414	281	194	47	141	341	65			
Total Oxide Feed Grade	g/t			-	-	1.75	1.74	1.39	1.21	1.22	1.38	1.09	1.43	1.50	1.53	1.21	1.08	1.06	1.19	1.45	-	-
Total Transitional Feed Tonnes	t ('000)					40	187	85	114	320	439	912	1,286	1,166	1,119	856	635	285	229	30		
Total Transitional Feed Grade	g/t			-	-	1.52	2.17	1.70	1.02	1.06	1.22	1.09	1.08	1.25	1.48	1.64	1.66	1.43	1.63	1.47	-	-
Total Fresh Feed Tonnes	t ('000)					1	113	321	350	350	299	49	100	353	487	897	1,025	1,175	1,506	1,770	1,156	
Total Fresh Feed Grade	g/t			-	-	1.40	2.02	3.01	2.84	2.68	2.67	1.40	1.38	1.35	1.36	1.39	1.50	1.54	1.58	1.64	1.99	-
Total Feed Tonnes	t ('000)					1,500	1,804	1,804	1,804	1,804	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,156	
Total Feed Grade	g/t			-	-	1.74	1.80	1.69	1.51	1.48	1.55	1.10	1.18	1.31	1.45	1.51	1.52	1.43	1.58	1.64	1.99	-
GSS Total Waste Tonnes	t ('000)					2,587	3,394	3,067	2,791	3,222	3,497	3,620	4,394	4,635	4,264	3,951	3,500	3,272	2,222	1,694	783	
GSS Total Mined Tonnes	t ('000)					3,787	4,848	4,521	4,245	4,676	4,999	5,420	6,194	6,435	6,064	5,751	5,300	5,072	4,022	3,494	1,939	
GSS Strip Ratio				-	-	2.16	2.33	2.11	1.92	2.22	2.33	2.01	2.44	2.58	2.37	2.20	1.94	1.82	1.23	0.94	0.68	-
WD Total Waste Tonnes	t ('000)					2,178	2,150	1,598	834	678	372											
WD Total Mined Tonnes	t ('000)					2,478	2,500	1,948	1,184	1,028	671											
WD Strip Ratio				-	-	7.26	6.14	4.57	2.38	1.94	1.25	-	-	-	-	-	-	-	-	-	-	-
Total Waste Tonnes	t ('000)		58,704			4,766	5,544	4,664	3,625	3,900	3,869	3,620	4,394	4,635	4,264	3,951	3,500	3,272	2,222	1,694	783	
Total Mined Tonnes	t ('000)		86,576			6,266	7,348	6,468	5,429	5,704	5,669	5,420	6,194	6,435	6,064	5,751	5,300	5,072	4,022	3,494	1,939	
Global Strip Ratio			1.5	-	-	3.18	3.07	2.59	2.01	2.16	2.15	2.01	2.44	2.58	2.37	2.20	1.94	1.82	1.23	0.94	0.68	-
Processing																						
Oxide Tonnes Processed	t ('000)		10,218			1,458	1,504	1,398	1,341	1,134	1,062	839	414	281	194	47	141	341	65			
Oxide Grade Processed	g/t		1.41	-	-	1.75	1.74	1.39	1.21	1.22	1.38	1.09	1.43	1.50	1.53	1.21	1.08	1.06	1.19	1.45	-	-

Description	Unit	Notes / Source	Tot / Ave	Year 0	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	Year 15	Year 16	Year 17	Year 18
Oxide Recovered Metal	g ('000)	92%	13,285			2,345	2,402	1,788	1,487	1,275	1,346	838	546	389	272	52	140	332	71			
Transitional Tonnes Processed	t ('000)		7,702			40	187	85	114	320	439	912	1,286	1,166	1,119	856	635	285	229	30		
Transitional Grade Processed	g/t		1.35	-	-	1.52	2.17	1.70	1.02	1.06	1.22	1.09	1.08	1.25	1.48	1.64	1.66	1.43	1.63	1.47	-	-
Transitional Recovered Metal	g ('000)	87%	9,040			54	354	126	101	295	466	868	1,213	1,272	1,436	1,222	916	354	326	38		
GSS East Fresh Tonnes Processed	t ('000)		5,274								1	49	100	353	483	858	824	624	862	993	126	
GSS East Fresh Grade Processed	g/t		1.42	-	-	-	-	-	-	1.05	1.11	1.40	1.38	1.34	1.35	1.39	1.43	1.46	1.48	1.42	1.46	-
GSS East Fresh Recovered Metal	g ('000)	80%	5,992								1	54	110	380	523	954	941	730	1,022	1,130	147	
GSS Main Fresh Tonnes Processed	t ('000)		3,244	-	-	-	-	-	-	-	-	-	-	120.33	4,641.32	38,804.52	200,662.98	550,425.77	643,848.68	776,491.78	1,029,438.93	-
GSS Main Fresh Grade Processed	g/t		1.86	-	-	-	-	-	-	-	-	-	-	1.90	1.61	1.49	1.81	1.63	1.72	1.92	2.05	-
GSS Main Fresh Recovered Metal	g ('000)	80%	4,827												6	46	290	716	887	1,190	1,691	
WD Fresh Tonnes Processed	t ('000)		1,434			1	113	321	350	350	299											
WD Fresh Grade Processed	g/t		2.74	-	-	1.40	2.02	3.01	2.84	2.68	2.68	-	-	-	-	-	-	-	-	-	-	-
WD Fresh Recovered Metal	g ('000)	83%	3,258			1	189	802	824	778	664											
Total Tonnes Processed	t ('000)		27,872			1,500	1,804	1,804	1,804	1,804	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,156	
Average Process Grade	g/t		1.52	-	-	1.74	1.80	1.69	1.51	1.48	1.55	1.10	1.18	1.31	1.45	1.51	1.52	1.43	1.58	1.64	1.99	-
Total Recovered Metal	g ('000)		36,401			2,399	2,946	2,716	2,412	2,348	2,476	1,760	1,869	2,042	2,237	2,274	2,287	2,132	2,306	2,358	1,838	
Total Recovered Metal	oz ('000)		1,170			77	95	87	78	76	80	57	60	66	72	73	74	69	74	76	59	
Cumulative Recovered Metal	oz ('000)					77	172	259	337	412	492	548	609	674	746	819	893	961	1,035	1,111	1,170	1,170
Average Recovery	%		0.86	-	-	0.92	0.91	0.89	0.89	0.88	0.88	0.89	0.88	0.86	0.86	0.84	0.83	0.83	0.81	0.80	0.80	-

INCOME STATEMENT																						
Revenue (USD)	$ ('000)	USD 1200.00 / oz	**1,404,402**			92,573	113,651	104,782	93,047	90,605	95,534	67,914	72,096	78,776	86,315	87,752	88,231	82,271	88,955	90,976	70,925	
Less:																						
Royalty @ 7%	$ ('000)		**(98,308)**	-	-	(6,480)	(7,956)	(7,335)	(6,513)	(6,342)	(6,687)	(4,754)	(5,047)	(5,514)	(6,042)	(6,143)	(6,176)	(5,759)	(6,227)	(6,368)	(4,965)	-
Operating Cost - Mining	$ ('000)	OPEX	**(274,961)**	-	-	(21,148)	(23,918)	(21,640)	(18,795)	(19,679)	(19,128)	(15,839)	(18,159)	(18,971)	(18,009)	(17,378)	(16,160)	(15,486)	(12,688)	(11,409)	(6,555)	-
Operating Cost - Process (Fixed)	$ ('000)	OPEX	**(228,531)**	-	-	(14,283)	(14,283)	(14,283)	(14,283)	(14,283)	(14,283)	(14,283)	(14,283)	(14,283)	(14,283)	(14,283)	(14,283)	(14,283)	(14,283)	(14,283)	(14,283)	-

Description	Unit	Notes / Source	Tot / Ave	Year 0	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	Year 15	Year 16	Year 17	Year 18
Operating Cost - Process (Variable)	$ ('000)	OPEX	**(302,783)**	-	-	(16,129)	(19,358)	(19,306)	(19,301)	(19,283)	(19,248)	(19,334)	(19,345)	(19,526)	(19,618)	(19,896)	(19,901)	(19,820)	(19,995)	(20,110)	(12,614)	-
Operating Profit	$ ('000)		**499,820**	**-**	**-**	**34,532**	**48,136**	**42,218**	**34,155**	**31,018**	**36,187**	**13,705**	**15,262**	**20,482**	**28,362**	**30,053**	**31,711**	**26,923**	**35,762**	**38,806**	**32,509**	**-**
Cost of Sales	$ ('000)		**(5,852)**	**-**	**-**	**(386)**	**(474)**	**(437)**	**(388)**	**(378)**	**(398)**	**(283)**	**(300)**	**(328)**	**(360)**	**(366)**	**(368)**	**(343)**	**(371)**	**(379)**	**(296)**	**-**
Freight & Refining	$ ('000)	$ 5.00/oz	**(5,852)**	-	-	(386)	(474)	(437)	(388)	(378)	(398)	(283)	(300)	(328)	(360)	(366)	(368)	(343)	(371)	(379)	(296)	-
OPcost/t	*$ ('000)*		*(19.06)*	*-*	*-*	*(20.27)*	*(18.65)*	*(18.62)*	*(18.61)*	*(18.61)*	*(18.63)*	*(18.68)*	*(18.68)*	*(18.78)*	*(18.83)*	*(18.99)*	*(18.99)*	*(18.95)*	*(19.04)*	*(19.11)*	*(23.27)*	*-*
Total Cost / oz recovered	*$ ('000)*		*(777.93)*			*(757.37)*	*(696.74)*	*(721.50)*	*(764.52)*	*(794.19)*	*(750.46)*	*(962.85)*	*(950.97)*	*(893.00)*	*(810.70)*	*(794.03)*	*(773.72)*	*(812.30)*	*(722.57)*	*(693.14)*	*(654.98)*	
EBITDA	$ ('000)		**493,968**	**-**	**-**	**34,146**	**47,663**	**41,781**	**33,767**	**30,640**	**35,789**	**13,422**	**14,962**	**20,154**	**28,002**	**29,687**	**31,343**	**26,580**	**35,392**	**38,427**	**32,213**	**-**
Depreciation	$ ('000)	Depreciation	**(187,620)**	-	-	(11,517)	(13,909)	(13,229)	(11,861)	(11,576)	(12,543)	(9,210)	(9,715)	(10,739)	(11,669)	(11,847)	(12,297)	(11,535)	(12,389)	(12,648)	(10,936)	-
EBIT			**306,348)**	**-**	**-**	**22,629)**	**33,754)**	**28,552)**	**21,906)**	**19,064)**	**23,246)**	**4,211)**	**5,247)**	**9,415)**	**16,333)**	**17,840)**	**19,046)**	**15,046)**	**23,002)**	**25,779)**	**21,278)**	**-**
Interest on Overdraft	$ ('000)		**-**																			
Interest - Loan	$ ('000)	Loan	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Profit Before Tax	$ ('000)		**306,348**	**-**	**-**	**22,629**	**33,754**	**28,552**	**21,906**	**19,064**	**23,246**	**4,211**	**5,247**	**9,415**	**16,333**	**17,840**	**19,046**	**15,046**	**23,002**	**25,779**	**21,278**	**-**
Tax @ 15%	$ ('000)	Tax	**(45,952)**	-	-	(3,394)	(5,063)	(4,283)	(3,286)	(2,860)	(3,487)	(632)	(787)	(1,412)	(2,450)	(2,676)	(2,857)	(2,257)	(3,450)	(3,867)	(3,192)	-
Net Profit	$ ('000)		**260,396**	**-**	**-**	**19,235**	**28,691**	**24,269**	**18,620**	**16,204**	**19,759**	**3,580**	**4,460**	**8,002**	**13,883**	**15,164**	**16,189**	**12,789**	**19,552**	**21,912**	**18,086**	**-**
Retained profit b/f	$ ('000)		**-**	-	-	-	19,234,771	47,925,732	72,195,223	90,815,268	107,019,461	126,778,441	130,358,181	134,818,165	142,820,551	156,703,477	171,867,465	188,056,723	200,845,485	220,397,432	242,309,718	260,395,666
Profit available for dividend	$ ('000)		**260,396**	**-**	**-**	**19,235**	**47,926**	**72,195**	**90,815**	**107,019**	**126,778**	**130,358**	**134,818**	**142,821**	**156,703**	**171,867**	**188,057**	**200,845**	**220,397**	**242,310**	**260,396**	**260,396**
Dividend	$ ('000)	Dividend	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Retained Profit	$ ('000)		**260,396**	**-**	**-**	**19,235**	**47,926**	**72,195**	**90,815**	**107,019**	**126,778**	**130,358**	**134,818**	**142,821**	**156,703**	**171,867**	**188,057**	**200,845**	**220,397**	**242,310**	**260,396**	**260,396**
FREE CASH FLOW																						
Revenue (USD)	$ ('000)	Income Statement	**1,404,402**	-	-	92,573	113,651	104,782	93,047	90,605	95,534	67,914	72,096	78,776	86,315	87,752	88,231	82,271	88,955	90,976	70,925	-
SiB Capital expenditure financed out of operations	$ ('000)	CAPEX - Finance Split	**(31,290)**	-	-	(4,015)	(1,015)	(4,748)	(1,015)	(1,015)	(4,748)	(1,015)	(1,015)	(2,881)	(1,015)	(1,015)	(2,881)	(1,015)	(1,015)	(1,015)	(1,867)	-
Royalty	$ ('000)	Income Statement	**(98,308)**	-	-	(6,480)	(7,956)	(7,335)	(6,513)	(6,342)	(6,687)	(4,754)	(5,047)	(5,514)	(6,042)	(6,143)	(6,176)	(5,759)	(6,227)	(6,368)	(4,965)	-
Operating cost	$ ('000)	Income Statement	**(806,275)**	-	-	(51,560)	(57,559)	(55,229)	(52,379)	(53,245)	(52,659)	(49,456)	(51,787)	(52,780)	(51,911)	(51,557)	(50,344)	(49,589)	(46,965)	(45,802)	(33,452)	-
Cost of Sales	$ ('000)	Income Statement	**(5,852)**	-	-	(386)	(474)	(437)	(388)	(378)	(398)	(283)	(300)	(328)	(360)	(366)	(368)	(343)	(371)	(379)	(296)	-
Interest on overdraft	$ ('000)	Income Statement	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tax payment	$ ('000)	Income Statement	**(45,952)**	-	-	(3,394)	(5,063)	(4,283)	(3,286)	(2,860)	(3,487)	(632)	(787)	(1,412)	(2,450)	(2,676)	(2,857)	(2,257)	(3,450)	(3,867)	(3,192)	-
Total Free Cash Flow	$ ('000)		**416,726**	**-**	**-**	**26,737**	**41,585**	**32,750**	**29,466**	**26,766**	**27,554**	**11,775**	**13,160**	**15,860**	**24,537**	**25,996**	**25,605**	**23,309**	**30,926**	**33,545**	**27,155**	**-**
FINANCING CASH FLOW																						
Loan receipts	$ ('000)	CAPEX - Finance Split	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Equity Receipts	$ ('000)	CAPEX - Finance Split	**122,580**	51,830	70,750	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Project Finance Receipts	$ ('000)		**122,580**	**51,830**	**70,750**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Loan Payments	$ ('000)	Loan	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Description	Unit	Notes / Source	Tot / Ave	Year 0	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	Year 15	Year 16	Year 17	Year 18
Interest on loan	$ ('000)	Loan	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Loan Payments	$ ('000)		**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Capital Expenditure (Pre-Production)	$ ('000)	CAPEX - Project Cost	**(122,580)**	(51,830)	(70,750)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Financing Net Cash Flow	$ ('000)		**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**

TOTAL CASH FLOW																						
Cash Flow	$ ('000)		**294,146**	(51,830)	(70,750)	26,737	41,585	32,750	29,466	26,766	27,554	11,775	13,160	15,860	24,537	25,996	25,605	23,309	30,926	33,545	27,155	-
Free cash flow + Financing Cash Flow	$ ('000)		**416,726**	-	-	26,737	41,585	32,750	29,466	26,766	27,554	11,775	13,160	15,860	24,537	25,996	25,605	23,309	30,926	33,545	27,155	-
Cash flow b/f	$ ('000)			-	(51,830)	(122,580)	(95,843)	(54,258)	(21,508)	7,958	34,724	62,278	74,053	87,213	103,072	127,610	153,606	179,210	202,519	233,445	266,991	294,146
Cash available for dividends	$ ('000)		**294,146**	**(51,830)**	**(122,580)**	**(95,843)**	**(54,258)**	**(21,508)**	**7,958**	**34,724**	**62,278**	**74,053**	**87,213**	**103,072**	**127,610**	**153,606**	**179,210**	**202,519**	**233,445**	**266,991**	**294,146**	**294,146**
Dividends	$ ('000)	Dividends	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cash Balance	$ ('000)		**294,146**	**(51,830)**	**(122,580)**	**(95,843)**	**(54,258)**	**(21,508)**	**7,958**	**34,724**	**62,278**	**74,053**	**87,213**	**103,072**	**127,610**	**153,606**	**179,210**	**202,519**	**233,445**	**266,991**	**294,146**	**294,146**
Cumulative Cash Flow	$ ('000)			**(51,830)**	**(122,580)**	**(95,843)**	**(54,258)**	**(21,508)**	**7,958**	**34,724**	**62,278**	**74,053**	**87,213**	**103,072**	**127,610**	**153,606**	**179,210**	**202,519**	**233,445**	**266,991**	**294,146**	**294,146**

22.4 Financial Summary

The results of the financial model are summarised in Table 22-6.

Revenue generated per lithology is shown in Figure 22-3; where total oxide provides 37%, total transition provides 25% and total fresh provides 38% of the total revenue.

Figure 22-5 is a summary of the total mining cost; waste mining costs contribute 52% and 8% to the cost of mining at GSS and at WD respectively.

A breakdown of the total cash costs is shown in Figure 22-4. The combined variable and fixed plant operating costs account for 58% of the total cash cost. Figure 22-6, Figure 22-8 and Figure 22-7 provides a breakdown of the plant operating cost, fixed costs account for 43% and variable costs account for 57% of the plant operating costs.

From inspection the following items contribute the most (77%) to the plant operating cost:

- Labour (21%);
- Power (28%); and
- Process Consumable / Reagents (28%).

Table 22-6: Financial Model Summary

Description	Units	LOM
Tonnage Feed	t'000*	27,872
Feed Grade Processed (average)	g/t	1.52
Gold Recovery (average)	%	86
Production Period	Years	15
Waste Rock	t'000*	58,704
Total Mined	t'000*	86,576
Stripping Ratio		1.5
Gold Production	oz	1,170,335.37
Annual Gold Production (average)	oz/y	78,022.36
Gross Revenue	US$'000*	1,404,403
Operating Profit (e)	US$'000*	499,820
Total Operating Costs (a)	US$'000*	904,583
Total Cash Cost (b)	US$'000*	910,435
Pre-production Capital Cost	US$'000*	122,581
Sustaining Capital Cost	US$'000*	31,290
Total Capital Cost (c)	US$'000*	153.871
Operating Cost / t Ore Processed (a)	US$/t	28.93
Process Cost / t Ore Processed	US$/t	19.06
Cash Cost / oz Au recovered (d)	US$/oz	777.93

(a) Includes Mining & Processing Cost
(b) Operating Cost + Freight & Refining Costs
(c) Pre-production Capital Cost + Sustaining Capital Cost
(d) Royalties + Operating Cost + Freight & Refining Costs
(e) Excludes Freight & Refining

* Values rounded up

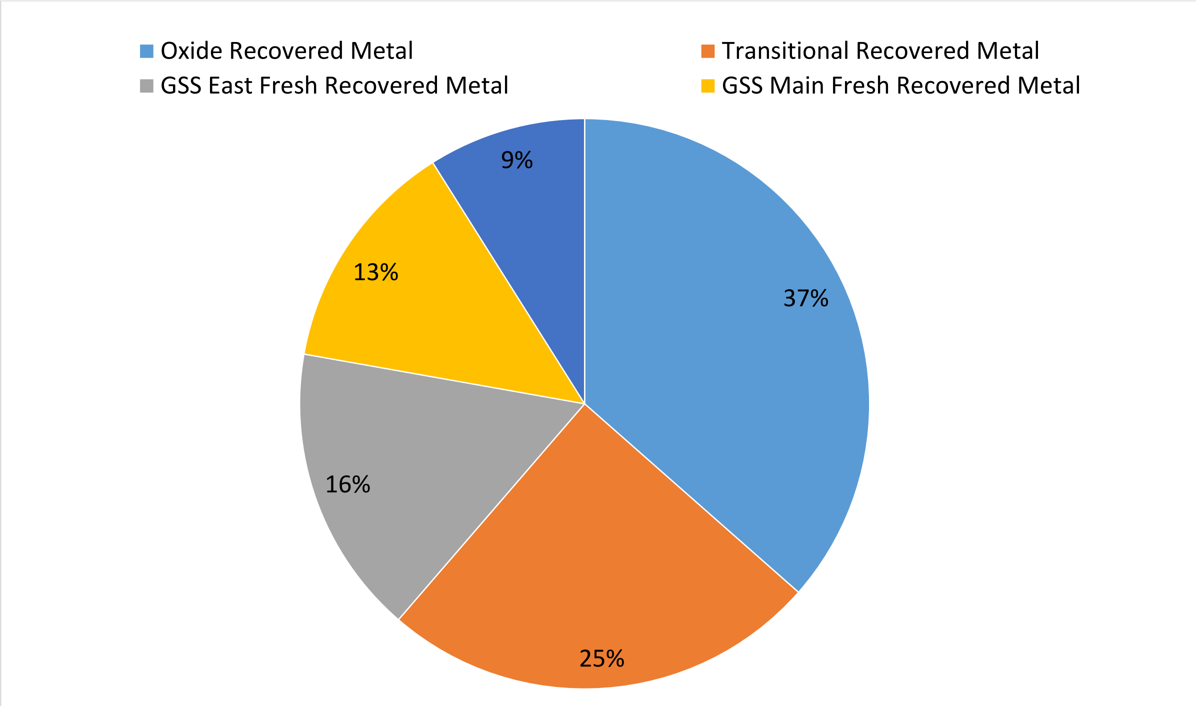


Figure 22-3: Revenue Generated per Material Type

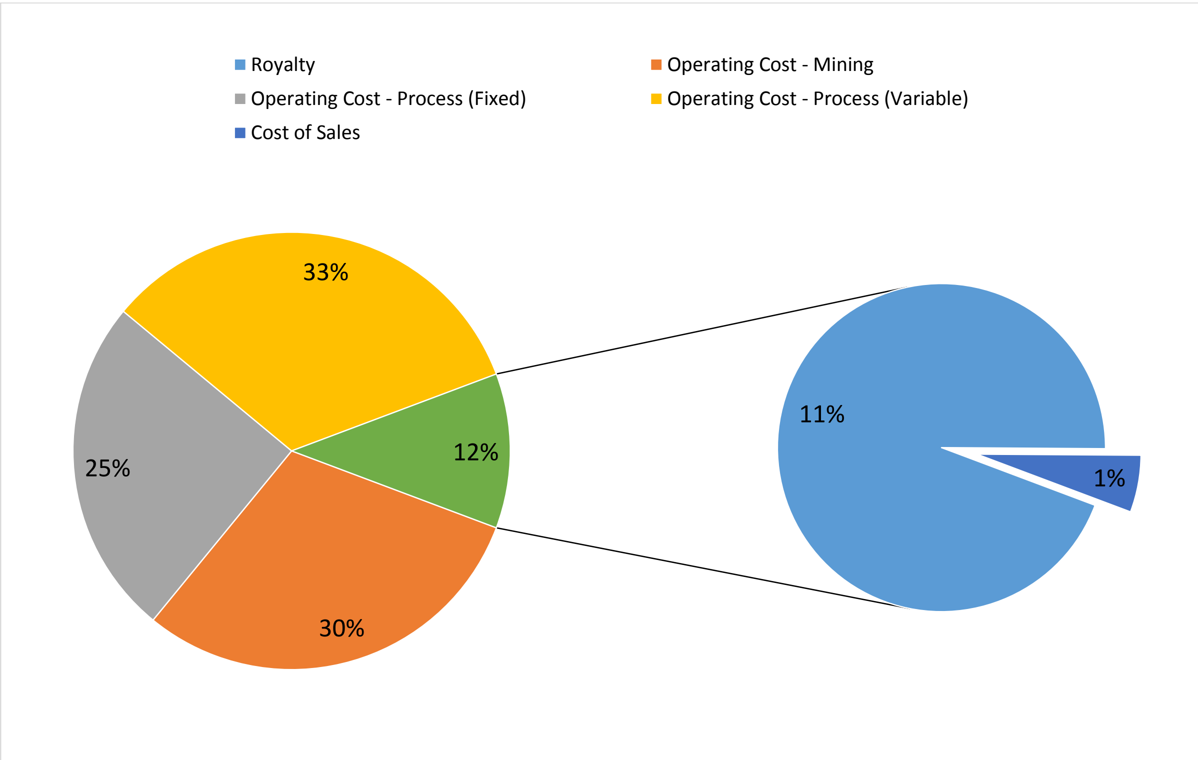


Figure 22-4: Total Cash Cost Split

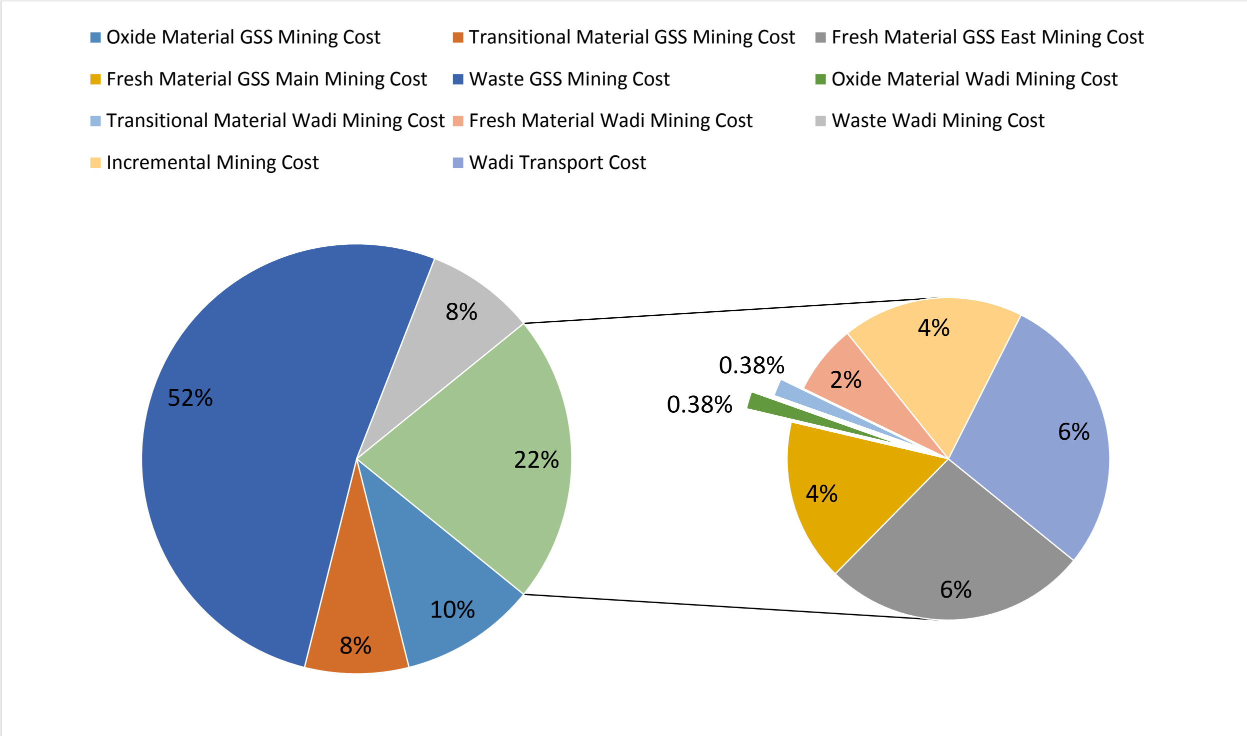


Figure 22-5: Total Mining Cost Split

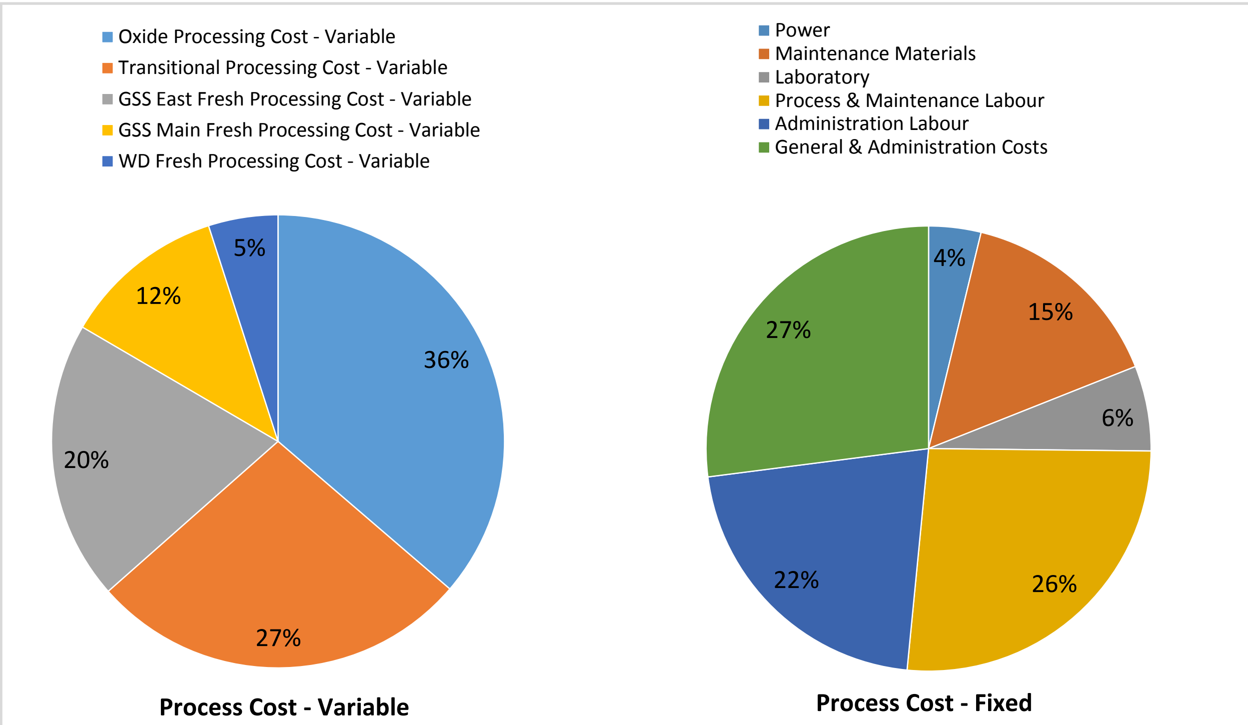


Figure 22-6: Total Process Operating Cost Split

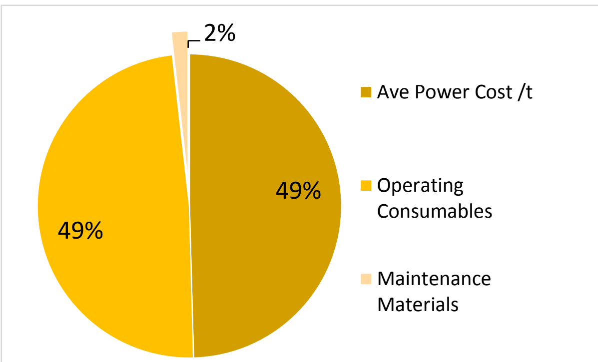


Figure 22-8: Variable Operating Cost Split

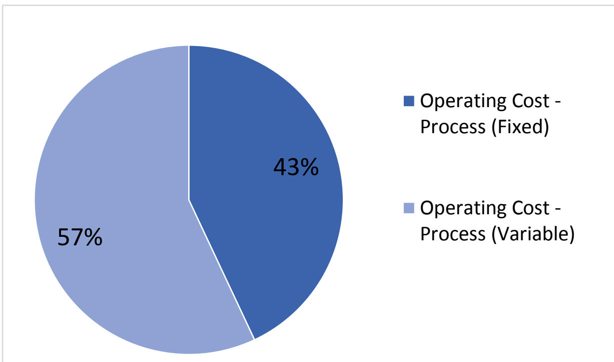


Figure 22-7: Operating Cost Split - Fixed vs. Variable

22.5 Single Parameter Sensitivities

Figure 22-9 shows the changing post-tax NPV for varying single parameter sensitivities at a 100% base case of 7% discount rate for revenue, capital cost, and plant operating costs, mining operating cost – GSS, mining operating cost – WD, and gold recoveries for the various material types processed. Figure 22-9 also shows the post-tax IRR sensitivity to parameters that the NPV is most sensitive to i.e. revenue, capital cost, plant operating cost and GSS mining cost.

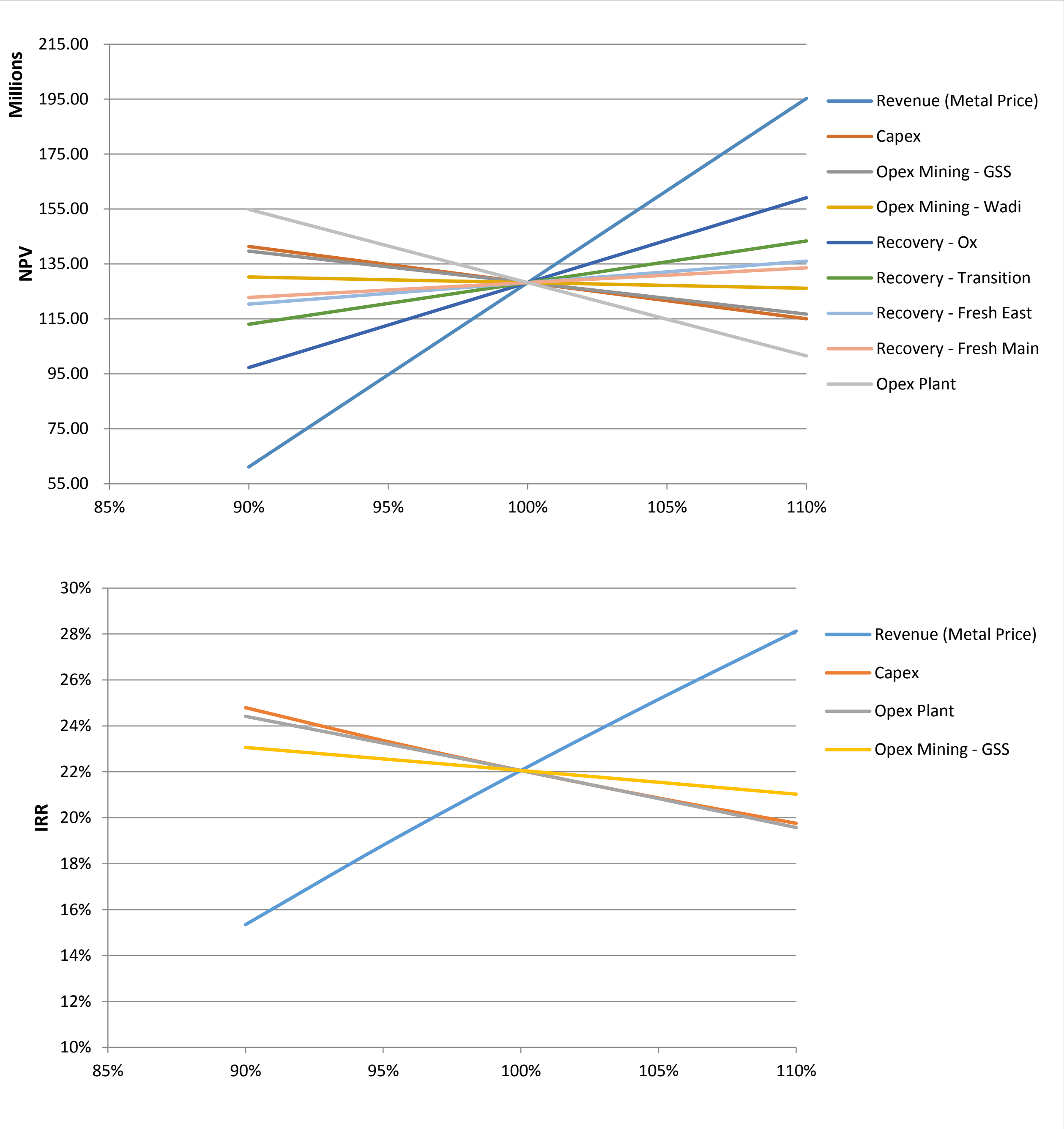


Figure 22-9: NPV & IRR Sensitivity

22.6 Two Parameter Sensitivities

NPV and IRR sensitivities to varying two parameters are shown in Table 22-7, Table 22-8, Table 22-9, Table 22-10, Table 22-11, and Table 22-12. Parameters chosen are based on the single parameters that the NPV and IRR are most sensitive to i.e. revenue, capital cost and plant operating cost, as per Figure 22-9.

Table 22-7: NPV Sensitivity - Change in Revenue vs. Change in CAPEX

		Revenue Change						
		85%	90%	95%	100%	105%	110%	115%
CAPEX Change	90%	40,663,663.99	74,314,459.69	107,855,993.40	141,374,920.41	174,893,847.42	208,412,774.92	241,931,701.09
	95%	34,069,618.59	67,729,580.79	101,279,187.24	134,798,114.02	168,317,041.59	201,835,967.83	235,354,894.04
	100%	27,467,778.36	61,142,376.17	94,702,380.23	**128,221,306.98**	161,740,235.40	195,259,162.00	228,778,087.66
	105%	20,865,938.38	54,555,172.03	88,125,572.45	121,644,500.35	155,163,429.04	188,682,355.73	222,201,282.07
	110%	14,264,099.05	47,967,968.25	81,548,766.17	115,067,693.10	148,586,621.93	182,105,548.10	215,624,475.92
	115%	7,656,518.54	41,380,765.70	74,971,960.22	108,490,885.89	142,009,814.84	175,528,740.72	209,047,669.93
	120%	1,040,299.94	34,793,561.62	68,394,346.34	101,914,079.47	135,433,007.83	168,951,935.09	202,470,861.58
	125%	- 5,580,698.78	28,206,356.21	61,815,116.99	95,337,273.71	128,856,202.25	162,375,128.00	195,894,055.73
	130%	- 2,201,697.54	21,613,376.76	55,233,009.86	88,760,467.03	122,279,395.98	155,798,321.94	189,317,248.23

Table 22-8: IRR Sensitivity - Change in Revenue vs. Change in CAPEX

		Revenue Change						
		85%	90%	95%	100%	105%	110%	115%
CAPEX Change	90%	14%	18%	21%	25%	28%	31%	34%
	95%	13%	16%	20%	23%	27%	30%	33%
	100%	12%	15%	19%	**22%**	25%	28%	31%
	105%	11%	14%	18%	21%	24%	27%	30%
	110%	10%	13%	17%	20%	23%	25%	28%
	115%	9%	13%	16%	19%	22%	24%	27%
	120%	8%	12%	15%	18%	21%	23%	26%
	125%	8%	11%	14%	17%	20%	22%	25%
	130%	7%	10%	13%	16%	19%	21%	24%

Table 22-9: NPV Sensitivity - Change in Revenue vs. Change in OPEX

		Revenue Change						
		85%	90%	95%	100%	105%	110%	115%
OPEX Change	90%	54,289,201.90	87,847,101.71	121,366,027.95	154,884,955.54	188,403,883.58	221,922,811.17	255,441,737.90
	95%	40,897,900.85	74,511,543.60	108,034,205.55	141,553,132.93	175,072,059.03	208,590,986.51	242,109,912.53
	100%	27,467,778.36	61,142,376.17	94,702,380.23	**128,221,306.98**	161,740,235.40	195,259,162.00	228,778,087.66
	105%	13,970,303.32	47,754,109.19	81,364,356.20	114,889,482.70	148,408,410.15	181,927,336.00	215,446,265.12
	110%	385,763.37	34,330,716.64	67,995,551.18	101,557,658.00	135,076,584.61	168,595,511.46	202,114,439.97
	115%	-590,439.58	20,852,115.16	54,607,283.11	88,217,166.30	121,744,759.74	155,263,689.39	188,782,615.75
	120%	- 98,551.26	7,279,609.21	41,189,218.80	74,848,726.61	108,412,936.43	141,931,864.70	175,450,790.01
	125%	-659,476.29	- 450,741.24	27,727,468.58	61,460,458.94	95,069,978.09	128,600,039.94	162,118,966.45
	130%	- 44,161.13	-,956,831.13	14,173,457.85	48,045,844.95	81,701,901.18	115,268,214.04	148,787,141.81

Table 22-10: IRR Sensitivity - Change in Revenue vs. Change in OPEX

OPEX Change	Revenue Change 85%	90%	95%	100%	105%	110%	115%
90%	15%	18%	21%	24%	27%	30%	33%
95%	13%	17%	20%	23%	26%	29%	32%
100%	12%	15%	19%	**22%**	25%	28%	31%
105%	10%	14%	17%	21%	24%	27%	30%
110%	8%	12%	16%	20%	23%	26%	29%
115%	6%	11%	15%	18%	22%	25%	28%
120%	4%	9%	13%	17%	20%	24%	27%
125%	No IRR	7%	12%	16%	19%	22%	25%
130%	No IRR	5%	10%	14%	18%	21%	24%

Table 22-11: NPV Sensitivity - Change in OPEX vs. Change in CAPEX

CAPEX Change	OPEX Change 85%	90%	95%	100%	105%	110%	115%
90%	89,538,712.24	173,854,212.9	158,169,713.5	142,485,214.2	126,800,714.9	111,116,215.5	95,431,716.2
95%	182,406,758.60	166,722,259.27	151,037,759.94	135,353,260.61	119,668,761.28	103,984,261.95	88,299,762.62
100%	175,274,804.97	159,590,305.64	143,905,806.31	**128,221,306.98**	112,536,807.65	96,852,308.32	81,167,808.99
105%	168,142,851.34	152,458,352.00	136,773,852.67	121,089,353.34	105,404,854.01	89,720,354.68	74,035,855.35
110%	161,010,897.70	145,326,398.37	129,641,899.04	113,957,399.71	98,272,900.38	82,588,401.05	66,903,901.72
115%	153,878,944.07	138,194,444.74	122,509,945.40	106,825,446.07	91,140,946.74	75,456,447.41	59,771,948.08
120%	146,746,990.43	131,062,491.10	115,377,991.77	99,693,492.44	84,008,993.11	68,324,493.78	52,639,994.45
125%	139,615,036.80	123,930,537.47	108,246,038.14	92,561,538.81	76,877,039.47	61,192,540.14	45,508,040.81
130%	132,483,083.16	116,798,583.83	101,114,084.50	85,429,585.17	69,745,085.84	54,060,586.51	38,376,087.18

Table 22-12: IRR Sensitivity - Change in OPEX vs. CAPEX

CAPEX Change	OPEX Change 85%	90%	95%	100%	105%	110%	115%
90%	29%	28%	26%	25%	23%	22%	20%
95%	28%	26%	25%	23%	22%	20%	19%
100%	26%	25%	23%	**22%**	21%	19%	18%
105%	25%	23%	22%	21%	19%	18%	16%
110%	24%	22%	21%	20%	18%	17%	15%
115%	22%	21%	20%	19%	17%	16%	14%
120%	21%	20%	19%	18%	16%	15%	14%
125%	20%	19%	18%	17%	15%	14%	13%
130%	19%	18%	17%	16%	15%	13%	12%

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 mineralisation 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 summarised in the PEA Report and where possible included as Appendices to this report.

25 INTERPRETATION AND CONCLUSIONS

Based on the review of the available information and observations made during the site visit, SGS Time Mining concludes the following, in no particular order of importance:

25.1 Title and Geology

10% 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. The PFS will declare project reserves for the first time.

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 optimisations show that both deposits have economically viable material under the assumed economic and physical parameters.

A total of 27.9 million tonnes of crusher feed at a gold grade of 1.52 g/t is contained within 8 pits across the two deposits, along with some 58.7 million tonnes of waste, resulting in a strip ratio of 2.11:1. 90% of this crusher feed is contained within the Indicated category.

A combined mining schedule demonstrates that a processing rate of 1.8 Mt can be sustained for 14 years after an initial ramp up period in Year 1 and a ramp down period in Year 16. The pit at WD is mined for the first six 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 improve the extraction of crusher feed material and reduce waste movement thereby lowering overall operating costs.

25.3 Mineral Processing & Recovery Methods

The most economically effective process scheme identified is the adsorption of gold onto activated carbon, through the carbon-in-leach (CIL) process preceded by a comminution circuit.

The design of the comminution circuit and the metal recovery plant is based on a nominal capacity of 1.8 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.

Based on the test work results, there is upward potential in gold recovery using flash flotation followed by ultra-fine grinding and cyanide leaching of the flash flotation concentrates to recover some of the more refractory gold associated with Pyrite. This will be investigated as part of the Pre-Feasibility Study.

25.4 Major Infrastructure

25.4.1 Selected Water Supply Option

Based on the capital and operating cost comparison and the fact that HA8 well field shows potential of being able to supply the project over the life of mine, Alternative 2 was selected as the preferred alternative for the purposes of the PEA design and economic assessment.

The following concluding remarks can be made:

- Drilling a total of 8 exploration boreholes within the HA8 Area indicated promising results and the occurrence of groundwater within interbedded sequences of pale fine to coarse grained sandstone and sandy clays.
- The results suggest that further drilling and pump testing should be feasible and that a definite source of groundwater can be exploited.
- The full lateral and vertical extents of the aquifer system are yet to be established.
- This report is based on information that is indicative only and depends on further testing, the identification of the aquifer boundaries and delineation of the full aquifer extents. The follow up drilling and pump testing will supply more detail in this regard.

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 major cities like Khartoum, Port Sudan, or 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 impact, which they have begun doing through the implementation of an Exploration Statement. This statement 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.

There are a few improvements that the Project should undertake in the near future, namely:

- Conduct social baseline surveys of Abu Hamad and record primary economic activities of people there and those working in the vicinity of the Project; and
- Develop a grievance procedure to identify and pre-empt potential tensions with artisanal and small scale mining operations.

25.6 Economic Analysis

Based on a gold price of US$1,200 per troy ounce, the Project has a pre-tax internal rate of return (IRR) of 25% and pre-tax net present value (NPV) of US$156 million, at a 7% discount rate.

Based on a gold price of US$1,200 per troy ounce, the Project has a post-tax internal rate of return (IRR) of 22% and a post-tax net present value (NPV) of US$128 million, at a 7% discount rate.

The Pre-tax undicoounted payback period is 3.2 years and Post-tax, undiscounted payback period is 3.7 yearsfromstartofproduction.

The NPV and IRR are most sensitive to the metal price, capital cost and plant operating cost.

Based on the current analysis there exists significant up-side potential in the event that the metal prices increase above 10% of the current base case.

25.7 Risks & Opportunities

25.7.1 Risks

The following risks were identified; a formal Risk Response Plan shall be developed during the Pre-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.
- A lack of water may cause dust issues, placing restriction on some operating practices. Use of road sealants will be considered for permanent or long term roads, especially the haul road between GSS and WD.
- Due to the location of the plant, transport and logistics of fuel and reagents might be problematic and sufficient buffer capacity and storage must be designed into the plant in the event of logistical problems and non-delivery of items to the project. The development of the test production wells may not produce the anticipated water volumes, whereby it would be necessary to pump water from the Nile.
- The current Direct Capital Cost estimate has been shown to be within a level of accuracy range of -1% +30 %. The risk is that the direct cost might increase by 30% once more detail engineering and development is completed. Based on Table 22-8 it can be seen that the project will be able to absorb this increase and still meet the Company's 20% hurdle rate at the current (July 2106) metal spot price.

25.7.2 Opportunities

25.7.2.1 Mining

Work is being planned to improve the geotechnical understanding of the deposits. This would potentially allow for the steepening of the pit walls.

A more detailed review of the GSS pits shows that the crusher feed extraction can be further optimised for some pits. This would include possible steepening of the pit walls in conjunction with optimal ramp placement.

Both feed grade and operating costs can be refined by further optimising the extraction sequence of the various GSS pits.

An understanding of the effects of oxide and transitional material on the CIL process would guide better blending of the crusher feed, possibly leading to a slight increase in recovery and/or throughput as the plant feed could be more consistent.

25.7.2.2 Mineral Processing

Develop a refined Process Design Criteria based on consolidated historical and future test work results in order to optimise the plant design.

Optimise reagent consumptions in order reduce the reagents usage which contributes 28% of the total operating costs, and any savings will contribute to more favourable economic results.

Test work during the PEA has indicated that gold deportment is very closely related to sulphides (+90% of which is pyrite). Preliminary testing has shown positive results from the use of flash flotation within the grinding circuit with subsequent regrinding of the concentrate. This has the potential to increase overall LOM recoveries by 2-3%.

An understanding of the effects of oxide and transitional material on the CIL process would guide better blending of the crusher feed, possibly leading to a slight increase in recovery and/or throughput as the plant feed could be more consistent.

The PEA set processing throughput at 1.8Mtpa pending confirmation of water production rates from the HA8 bore field, north of Block 14. The PFS will look at the economics of increasing throughput and reducing the mine life from 16 years to 12-15 years.

25.7.2.3 Flash Flotation & Regrind Circuit

Preliminary flash flotation test work conducted as per section 13.3.4.2 indicates that an opportunity exists to increase the gold recoveries of the transitional and fresh sulphide material through an additional flash flotation and regrind circuit. Slurry from the cyclone underflow distribution box would feed a flash flotation cell. This cell would be utilized to recover fast floating material. The residence time in the cell would be about 3 minutes. Air together with flotation reagents such as collector and frother would be added in a controlled manner to assist in selective flotation of host transition and fresh sulphide material.

The sulphide minerals would be recovered as concentrate and pumped by a dedicated sump pump into the regrind mill. The mass pull would be a maximum of 15%. This would potentially result in concentrate flowrate of less than 30 tph. The concentrate would be reground to 80% passing 25 µm and pumped to the CIL circuit or a dedicated intensive leach circuit. The tails of the flash float cell would be returned to the mill for further grinding.

The current process plant capital cost and economical model does not make allowance for this circuit and should be investigated further once more detailed test work is conducted in future Pre-Feasibility phase of the project.

25.7.2.4 Tailings Storage Facility

The current capital cost is based on a conventional surface impoundment. There is an opportunity to optimise the design during the next phases of the project and utilise the natural topography and areas in identified wadis or valleys in between the natural outcrops as starter walls in order to reduce the upfront development cost of the facility.

25.7.2.5 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 28% of the plant operating costs and any savings will contribute to more favourable economic results.

26 RECOMMENDATIONS

26.1 General

The PEA has demonstrated a strong project with several opportunities for improvement. Accordingly, the Company has approved the decision to commence a Pre-Feasibility Study ("PFS") of the Block 14 project focused on optimizing the Project towards a development decision in 2017.

The estimated cost of the PFS inclusive of resource and geotechnical drilling, test work, trade off studies, engineering and PFS engineering to be $4.5 million, a breakdown is provided in Table 26-1

Table 26-1: PFS Cost Estimate Summary

Description	Cost ($'000)
Consultants	917
Test Work	651
Mining & Geotechnical	792
EIA	154
Hydrology	707
Resource Drilling	1,250
Total	**4,471**

26.2 Additional Recommendations

26.2.1 Environmental

MSMCL is still to finalize the final Project design, 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:

- Conduct social baseline surveys of Abu Hamad and record primary economic activities of people there and those working in the vicinity of the Project; and
- Develop a grievance procedure to identify and pre-empt potential tensions with artisanal and small scale mining operations.

26.2.2 Mining and Geology

The following follow up work is required:

- Review geotechnical information and complete additional geotechnical investigations in order to determine if wall steepening is possible;
- Complete waste dump and haul road design to allow for more accurate estimate of haulage requirements; and
- 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

Froth Flotation provides a potential for Orca to coarsen the primary grind size since the flotation response of East and Main was extremely encouraging.

The next stage of work will focus upon the optimisation of:

- Flash Flotation Reagents and Flotation time;
- Regrinding of Optimised Flash Flotation Concentrate;
- Cyanide Leach parameters on optimised reground flash flotation concentrates; and
- Variability of following parameters by material zones and lithologies:
  - Bond Ball mill work index
  - Head Assay and Clay content
  - Flash Flotation Response using optimised conditions from aforementioned program
  - CIL Response

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 delivered with 8 "casing;
- Detailed pump tests at an optimal constant discharge rate for at least 24 to 48 hours per borehole;
- It is also recommended to drill an additional 4 exploration boreholes to determine the full extents of the aquifer to the north and south; and
- Detailed assessment for PFS purposes based on the data received. This must include a preliminary numerical groundwater model to illustrate the zone of depression and aquifer boundaries.

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28 QUALIFIED PERSONS CERTIFICATES

*I, **Keith Leonard Bright** do hereby certify that:*

1. I am a Study Manager at SGS Time Mining (Pty) Ltd with an office at Woodmead North Office Park, 54 Maxwell Drive, Johannesburg, South Africa.
2. This certificate applies to the technical report with an effective date of 25th July 2016, and titled “PRELIMINARY ECONOMIC ASSESSMENT NI 43-101 TECHNICAL REPORT BLOCK 14 GOLD PROJECT”.
3. I am a graduate with a B.Sc (Engineering) degree in metallurgy from the Royal School of Mining (London) and I have practised my profession continuously since that time. Since graduating I have worked as a consultant at SGS Time Mining and other process engineering companies on a wide range of mineral projects, specializing in precious and rare metals. I have undertaken many geological investigations, resource estimations, mine evaluation technical studies and due diligence reports. I am a Fellow of the South African Institute of Mining & Metallurgy (Membership Number 41729).
4. 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.
5. I visited the Block 14 property between 10th and 15th January 2016.
6. I am responsible for all Sections of the report, reliant on expert inputs into Sections 7, 8, 9, 10, 11, 12, 13, 14, 16, and 20.
7. I am independent of the issuer as described in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. 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.
10. 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 30th day of August, 2016.

KLBright

____

Keith Bright, B.Sc (Eng) A.R.S.M FSAIM
Manager: Studies

*I, **Michael Peter Hallewell** do hereby certify that:*

1. I am Director and Owner of MPH Minerals Consultancy Ltd with an office at Tremough Innovation Centre, Penryn, Cornwall, UK, TR10 9TA.
2. This certificate applies to the technical report with an effective date of 25th July 2016, and titled "PRELIMINARY ECONOMIC ASSESSMENT NI 43-101 TECHNICAL REPORT BLOCK 14 GOLD PROJECT".
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. Since graduating I have worked as a Global Manager with SGS Minerals Business team, leading all plant and metallurgical accounting audits, designing and overseeing scoping, prefeasibility and feasibility test work programs in a wide variety of metals in different regions. I was also Regional (Eurasia incl. Russia, Middle East & Africa) Geometallurgical Manager. Prior to this I spent 20 years as an Operational Manager, mainly as Plant Manager, Consulting or Senior Metallurgist in precious metals, base metals and ferrous metals.
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 Section 13 of the report.
7. I am independent of the issuer as described in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. 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.
10. 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 25th day of August, 2016.

Mike Hallewell, B.Sc, FIMMM, FSAIM,FMES,C.Eng
Consulting Metallurgist

*I **Nicolas 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 25th July 2016, and titled "PRELIMINARY ECONOMIC ASSESSMENT NI 43-101 TECHNICAL REPORT BLOCK 14 GOLD PROJECT".
3. I am a practising Geologist and registered Member of the Australian Institute of Geoscientists.
4. I am a graduate of Latrobe University, Melbourne, Australia with a Bachelor of Science (Honours) degree in Geology (1988). I have practiced my profession continuously since 1988.
5. I am a "Qualified Person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
6. I visited the Galat Sufar South Project between 17th January and 21nd January 2014. The purpose of the visit was to review the exploration practices and project geology.
7. I am responsible for authoring Sections 11, 12 and 14.
8. As of the effective date of the technical report, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
9. I am independent of Orca pursuant to Section 1.5 of the instrument.
10. I have read the technical report and the National instrument and Form 43-101F1 (the "Form") and the technical report has been prepared in compliance with the Instrument and the Form
11. I do not have nor do I expect to receive a direct or indirect interest in Orca, and I do not beneficially own, directly or indirectly, any securities of Orca or any associate or affiliate of such company.
12. My involvement with the Block 14 project is limited to work on mineral resource estimates since August 2013 and the preparation of this technical report.

Dated this 30th day of August 2016 at Perth.

Nicolas James Johnson, B.Sc (Hons) (Geol), MAIG
Consulting Geologist