FEASIBILITY STUDY

NI 43-101 TECHNICAL REPORT

BLOCK 14 GOLD PROJECT

REPUBLIC OF THE SUDAN

Report Date: 14 September, 2020
Effective Date: 31 August, 2020

Prepared for:

ORCA GOLD INC
2000 – 885 West Georgia Street
Vancouver, BC
V6C 3E8

Prepared by:

Lycopodium
Leve 2, 60 Leichhardt Street, Spring Hill
Brisbane Queensland 4000
Australia

Revision History

Rev	Date	Description
0	14 September 2020	Issued for Use

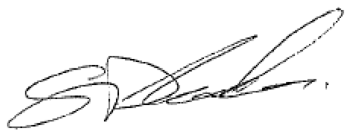
Date and Signature Page

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

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

The overall report entitled Feasibility Study Technical Report, Block 14 Gold Project Republic of Sudan, was collated and signed by the following author:

Signed, this 14th September, 2020.



____

Dr. Geoff Duckworth B.Eng (Chem), M.Eng.Sc, PhD, FIChemE, MIEAust, FAusIMM, RPEQ 2702
Study Manager
Lycopodium Minerals Pty Ltd

Other discipline specific QPs were:

- **Mike Hallewell**: Consultant, MPH Minerals Consultancy Ltd., B.Sc, FIMMM, FSAIMM, FMES, C.Eng.
- **Nicholas James Johnson**: Consulting Geologist, MPR Geological Consultants Pty Ltd, B.Sc (Hons) (Geol), MAIG.
- **Pieter Labuschagne**: Consultant, GCS (Pty) Ltd, M.Sc Hydrogeology (Pr.Sci.Nat 400386/11).
- **Carl Nicholas**: Consultant, Mineesia Ltd., M.Sc, B.Sc (Hons), DIC, CEnv, MIMMM.
- **Chris Reardon**: Country Manager, Orca Gold inc., B.Sc. (Previously Consultant, Deswik Europe Ltd.)
- **Tim Rowles**: Consultant, Knight Piésold Pty. Ltd., B.Sc, M.Sc, MAusIMM (CP), RPEQ (10166)

Signed certificates for the above mentioned are located at the end of the report.

CONTENTS

List of Figures

List of Tables

1. EXECUTIVE SUMMARY

1.1 Introduction

This report comprises a Revised Feasibility Study (RFS) of Orca Gold's Block 14 Project (Project) located in the Republic of Sudan, incorporating power generation using LNG fuel with associated operating costs and revising the capital estimate to Q3 2020 base.

The RFS has been prepared by Lycopodium Minerals Pty Ltd (Lycopodium) 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 Reliance on Other Experts

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

Lycopodium has relied on the advice of other experts in the preparation of this report.

1.3 Property Description and Ownership

1.3.1 Property Description

The Block 14 mineral exploration project covers an area of 1,000 km^2, located in the Nubian Desert in the north of the Republic of the Sudan, close to the international border with Egypt. The property is situated 700 km north of the national capital Khartoum and straddles the boundary between the Red Sea and Nile States. The nearest population centre is the town of Abu Hamad, located 180 km due south of the property.

1.3.2 Ownership

The Exclusive Prospecting License (EPL) for Block 14 was originally granted to Meyas Nub under a Concession Agreement dated 19 May 2010. A purchase agreement dated 1 March 2012 (the "Purchase Agreement"), granted Sand Metals Company Ltd (SMCL), a wholly owned subsidiary of Orca, the right to acquire a 70% interest in Meyas Sand Mineral Company Ltd (MSMCL), a Sudanese joint venture company incorporated to hold the Block 14 EPL. Under the terms of the Purchase Agreement, SMCL have paid Meyas Nub US$9.5M and own 70% of MSMCL.

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

The EPL 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 EPL key dates and terms are detailed below in Table 1.1.

Table 1.1 EPL Key Dates and Terms

Start Date	Expiry Date	Area (km^2)	Terms
May 19, 2010	May 18, 2014	7,046	Initial exploration period.
May 19, 2014	May 18, 2015	7,046	Additional one-year extension period granted, extending the expiry date of the initial exploration period.
May 19, 2015	May 18, 2017	3,747	Relinquishment of 50% of the Initial Exploration area (excluding Mineral Discovery Areas).
May 19, 2017	May 18, 2018	2,170	Relinquishment of 50% of remaining area (excluding Mineral Discovery Areas).
May 19, 2018	November 18, 2018	2,170	Extension of the final period of the permit.
December 18, 2018	November 18, 2022	1,000	Initial term of an exploration permit over an area of 1,000km^2.

1.4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

A number of international airlines with routes to Europe and neighbouring countries provide daily flights from Khartoum. The airport at Port Sudan also has international flight handling capability, although services are limited to neighbouring countries. Daily flights between Khartoum and Port Sudan are available.

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 where 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.

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

The northern desert of the Republic of the Sudan has infrequent precipitation and regional records indicate average annual precipitation to be nil. However, infrequent rainstorms do occur, with five events of rain >1 mm and one event producing 17.5 mm, have been recorded since baseline studies were initiated.

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.

The infrastructure requirements for the development of the project are minimal. Operational requirements can be satisfied using regionally available materials and services, mainly from Abu

Hamad, Atbara (where the company operates a small logistics office) and Khartoum (administration office).

The landscape is characterised by rocky hills separated by wide, flat sand filled drainage channels known as wadis. Elevation varies between 225 and 775 meters above sea level (masl), with the bulk of the higher ground in the east of the project area. Wadi Gabgaba, which runs north-south through the centre of the project area, is the main drainage system in the area. The western part of Block 14 contains more subdued topography.

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

1.5 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.

A number of exploration programmes have been carried out by various teams including Bureau de Research Geologique et Minieres (BRGM), Geosurvey International, Cominor, Robertson's Research International (RRI), La Source and Bundesanstalt fur Geowissenschaften un Rohstoffe (BGR) during the period from 1977 to 2004.

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 six years.

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 January 2012. 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 Galat Sufar South (GSS) confirmed the presence of gold in wide intersections of a major shear zone at the prospect and the first drill hole was completed in November 2012. In 2014, a second deposit was discovered at Wadi Doum (WD) through systematic follow up and evaluation of artisanal mining sites.

1.6 Geology and Mineralization

The geological framework of Block 14 is dominated by two distinct geological domains - the andesite dominated Gabgaba terrain of the Arabian Nubian Shield (ANS) to the east and the marine sediments of the Keraf Suture in Western Gabgaba; these are separated by the Eastern Gabgaba Fault System (EGFS). The central part of the licence is dominated by the Northern Gabgaba Graben, a downthrown portion of the Keraf that has been infilled by clastic sediments.

The geological framework of the Gabgaba terrane is relatively simple; the andesitic sequence is un-foliated and preserves its original mineralogy and textures and does not display evidence of folding. Where recognised, the primary layering in the volcano-sedimentary rocks is east-west.

The Keraf sediments are a thick sequence of folded / thrusted marine sediments dominated by pelites, marls and limestone units, with localised occurrences of coarser grained siliciclastic sediments. This is a significant geological formation in the region occurring along the length of the Keraf Suture.

The Galat Sufar Andesite domain is an anomalous exposure of andesitic volcanic rocks within the thick sequence of Keraf sediments. It is interpreted as a doubly plunging antiformal fold created by complex inference folding, which exposes the andesitic volcanic-sedimentary underlying the shallow marine sediments (the exact relationship between the andesites and sediments is unclear and is potentially structural). It is within this andesite domain that the Galat Sufar South deposit is located.

1.6.1 Galat Sufar South (GSS)

The GSS deposit is located in the central portion of the Galat Sufar Andesite Domain. The GSS deposit is located just south of the contact between marine sediments to the north (a remnant of the Keraf sediments) and an andesitic volcanic sequence to the south. The andesitic sequence is heterogeneous comprising lava flows, pyroclastic deposits and primary volcanic breccias.

Of importance to deposit formation, the andesite sequence contains a discrete 80 – 200 m wide volcaniclastic-sedimentary horizon which contains dioritic sills / dykes. Mineralisation and alteration are concentrated in this unit, which is bordered to the north and south by increasingly unaltered andesitic flows and further volcaniclastics.

The host unit has been sequentially and intensely altered by the addition of albite, sericite, silica and lastly carbonate. Alteration grades from largely unaltered andesitic lavas and volcaniclastic host rocks to strongly altered and foliated silica – sericite schists in which the protolith cannot be identified.

Pyrite is by far the most dominant sulphide with chalcopyrite, sphalerite, galena, tennantite / tetrahedrite occasionally seen in core and confirmed in petrological investigation. Gold is fine grained, typically <40 µm. With 95% of the gold being free gold, the remainder occurs as Petzite (Ag_3AuTe_3). The pyrite contains ± 20% silver.

The dominant foliation at the prospect scale (S1) is pervasively developed throughout the GSS deposit area. It is sub-vertical and strikes towards the NW (330˚ - 340˚) at moderate to high angles to the orientation of the mineralised unit.

1.6.2 Wadi Doum (WD)

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

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

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

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

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

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

1.7 Deposit Types

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

Evidence of orogenic gold mineralization is present throughout the project area. It is generally associated with narrow gash veins, shear type veins and quartz veinlet swarms in well foliated schistose rocks within the volcano-sedimentary domains which are intruded by stocks and sheets of diorites, syenites and granitoids.

The Nubian volcanic sequences are also prospective for Volcanogenic Massive Sulphide (VMS) mineralization and examples of economic VMS deposits within the Arabian Nubian Shield include the Bisha Project in Eritrea, the Jabal Sayid Mine in Saudi Arabia and the Ariab Project in Sudan.

1.8 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. Exploration in Block 14 is currently focussed on geological mapping in the wider resources area and on target generation within the 1,000km^2 EPL.

1.9 Drilling

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

During the second half of 2017, as part of the ongoing a Feasibility Study, a major resource infill and expansion drill programme was undertaken comprising 12,449 m of diamond drilling in 34 holes and 2,806 m of reverse circulation drilling in 14 holes. In addition, 1,395 m of geotechnical diamond drilling was also completed (19 Holes) in 2017.

Resource drilling continued between January and August 2018, with a further 3,101 m of diamond drilling in 10 holes and 13,593 m of RC drilling in 87 holes completed. To date 102 holes for 24,884 m of diamond drilling, and 804 holes for 102,217 m of reverse circulation drilling have been completed on Block 14.

1.10 Sample Preparation Analysis and Security

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

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

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

When samples are dispatched to the laboratory, a completed sample submission form accompanies the samples. The submission form details the sample number sequences and instructs the laboratory on the elements required for analysis and the analytical methods to be used.

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.

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

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

1.11 Data Verification

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

1.12 Minerals Processing and Metallurgical Testing

The flowsheet development has culminated in the following flowsheet concept:

- Single Stage Crushing followed by SAG mill – ballmill "SAB" comminution circuit:
  - addition of a minimum of 0.60 kg/t lime to the SAG mill feed is recommended to minimise solubilisation of cyanide soluble iron minerals prior to cyanide addition
  - D80 53 µm for Oxides and Wadi Doum Transition and Fresh

- D80 75 µm for Transition
- D80 90 µm for East and Main Fresh.

- High Rate Thickening prior to and immediately after CIP circuit:
  - thickeners to be run at >60% solids underflow density
  - 20 – 30 g/t Anionic Polyacryamide flocculant is recommended
  - 10 - 12% weight feed well pulp density.

- Whole Rock Leaching in CIP mode with Kemix adsorption technology to minimise carbon tonnage per day to elution:
  - the following flowsheet and operational features are recommended:
    - one 4 hr pre-aeration tank to oxidise Fe2+ to Fe3+
    - 45 hr Leaching residence time followed by six stage Kemix adsorption circuit
    - maintain pH 10.5 during the leaching process
    - 55% solids leach feed
    - 0.5 g/L CN
    - min. 3 ppm DO.

- Conventional tailings storage.

The main thrust of the FS testing was to:

- Develop sufficient comminution testing to enable a Domain designed SAG mill – ball mill circuit.
- Generate a lithological understanding of leach response using standard conditions in each of the major domains so that gold and silver recoveries, along with cyanide and lime consumptions, can be determined.
- Mercury and copper metal deportment has also been tracked for elution and gold room design purposes.

Table 1.2 below summarises the overall metal recoveries and reagent consumptions by domain and lithology. The gold, cyanide and lime data has been reported as 45 hr leach residence time data by extrapolation of the 24, 32 and 48 hr kinetic data. Silver has been reported as 48 hr due to the absence of kinetic data.

Table 1.2 Estimated LOM Recovery and Reagent Consumptions

Pit	Domain	Tonnes (millions)	Ozs Au (000's)	45 hr Au Recovery (%)	48 hr Ag Recovery (%)	45 hr CN (kg/t)	45 hr CaO (kg/t)
Main	Oxide	4.3	117.0	88.65	9.4	0.22	1.35
	Trans	5.1	195.1	82.74	19.7	0.29	1.03
	Fresh	13.5	569.9	77.75	44.5	0.18	0.72
East	Oxide	8.3	238.7	90.03	8.3	0.19	0.94
	Trans	11.2	322.9	85.04	26.7	0.23	0.91
	Fresh	30.7	1,039.3	81.02	38.0	0.22	0.65
NEZ	Oxide	1.6	43.3	90.76	6.0	0.18	1.22
	Trans	2.2	60.0	83.39	34.7	0.28	0.94
	Fresh	0.4	10.6	82.45	34.8	0.27	0.60
Wadi Doum	Oxide	0.5	32.3	87.21	1.3	0.55	1.57
	Trans	0.1	9.1	84.74	22.0	1.22	0.75
	Fresh	1.9	155.3	73.56	37.3	0.45	0.65
LOM Weighted Average		**79.9**	**2,853**	**82.0**	**32.2**	**0.23**	**0.81**

The overall life of mine gold recovery is estimated to be 82.0%, silver recovery - 32.2% and cyanide and lime consumptions, 0.23 kg/t and 0.81 kg/t respectively. The lower silver recovery is due to poor adsorption of silver onto carbon.

The reagent dosages highlighted in colour above have been adjusted to allow for the pre-aeration stage ahead of leaching. These were the only domains that showed consistent trends.

Detailed commentary and trends in metal recovery rates and reagent consumptions variability are expanded within Section 13.

1.13 Mineral Resource Estimates

In September 2018, MPR Geological Consultants Pty Ltd (MPR) estimated gold and silver Mineral Resources for the Galat Sufar South and Wadi Doum 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 and silver resources recoverable by open pit mining for a wide range of mineralization styles.

The estimates are based RC and diamond drilling data supplied by Orca in September 2018. Details of this sampling and assaying are described in relevant sections of this report.

Relative to the dataset used for previous Mineral Resources, the current sampling database includes an additional 74 RC holes and 13 diamond cored holes at Galat Sufar South, and eight RC holes and three diamond holes at Wadi Doum.

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. Table 1.3 below shows the estimate at 0.6 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 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.

Galat Sufar South Mineral Resource estimates are truncated at 350 m depth, with around 90% of Indicated and Inferred resources occurring at depths of less than 240 and 300 m respectively. Wadi Doum estimates extend to around 255 m depth, with around 90% of Indicated and Inferred resources occurring at depths of less than 115 m and 190 m respectively.

For Galat Sufar South combined oxidized and transitional material hosts around 30% and 15% of Indicated and Inferred resources respectively with the remainder lying in fresh rock. For Wadi Doum combined oxidized and transitional material hosts around 19% and 9% of Indicated and Inferred resources respectively.

Table 1.3 January 2018 GSS & WD Mineral Resource Estimates Estimate at 0.6 g/t Cut-off

GSS										
	Indicated					**Inferred**				
	Mt	**Au g/t**	**Ag g/t**	**Au koz**	**Ag koz**	**Mt**	**Au g/t**	**Ag g/t**	**Au koz**	**Ag koz**
Oxide	9.6	1.32	1.41	409	437	1.0	1.0	1.1	31	35
Trans.	13.2	1.22	1.31	517	557	1.5	1.0	1.1	49	55
Fresh	52.7	1.27	1.54	2,153	2,613	14.4	1.2	1.4	567	655
Total	**75.6**	**1.27**	**1.48**	**3,080**	**3,607**	**16.9**	**1.2**	**1.4**	**648**	**745**
WD										
	Indicated					**Inferred**				
	Mt	**Au g/t**	**Ag g/t**	**Au koz**	**Ag koz**	**Mt**	**Au g/t**	**Ag g/t**	**Au koz**	**Ag koz**
Oxide	0.6	1.90	2.82	34	50	0.1	1.0	1.6	3	6
Trans.	0.2	2.02	3.69	10	18	0.04	0.9	1.6	1	2
Fresh	3.6	1.89	5.93	218	683	1.5	1.2	3.8	59	183
Total	**4.3**	**1.90**	**5.45**	**262**	**751**	**1.6**	**1.2**	**3.6**	**63**	**191**
Combined										
	Indicated					**Inferred**				
	Mt	**Au g/t**	**Ag g/t**	**Au koz**	**Ag koz**	**Mt**	**Au g/t**	**Ag g/t**	**Au koz**	**Ag koz**
Oxide	10.2	1.35	1.49	443	487	1.1	1.0	1.2	34	41
Trans.	13.4	1.22	1.33	527	575	1.5	1.0	1.2	50	57
Fresh	56.3	1.31	1.82	2,371	3,296	15.9	1.2	1.6	626	838
Total	**79.9**	**1.30**	**1.70**	**3,342**	**4,358**	**18.5**	**1.2**	**1.6**	**711**	**936**

1.14 Mineral Reserve Estimates

The Mineral Reserve estimate is shown in Table 1.4 and described in Section 15. The first Mineral Reserve estimate for the Block 14 project was issued in November 2018, based on drilling up to and including that completed in 2018. The Mineral reserve estimate used a gold price of $1,100/oz. The figures in this table are rounded to reflect the precision of the estimates and include rounding errors.

Table 1.4 Summary of Mineral Reserves for Block 14

	Classification	**Oxide**		**Transitional**		**Fresh**		**Total**	
		'000 tonnes	**Au g/t**	**'000 tonnes**	**Au g/t**	**'000 tonnes**	**Au g/t**	**'000 tonnes**	**Au g/t**
Main	Probable	4,347	1.27	5,088	1.19	13,488	1.31	22,923	1.28
East	Probable	8,302	0.89	11,236	0.89	30,729	1.05	50,267	0.99
North East	Probable	1,606	0.84	2,192	0.85	367	0.90	4,166	0.85
Total GSS	***Probable***	***14,255***	***1.00***	***18,516***	***0.97***	***44,584***	***1.13***	***77,356***	***1.07***
Wadi Doum	***Probable***	***527***	***1.90***	***119***	***2.37***	***1,941***	***2.49***	***2,588***	***2.36***
Block 14 Total	**Probable**	**14,783**	**1.03**	**18,635**	**0.98**	**46,525**	**1.19**	**79,943**	**1.11**

1.15 Mining Methods

Three geotechnical reviews of the project have been conducted by SRK Consulting (UK) Ltd over the last several years, with the latest being completed at the end of 2017.

A series of dedicated geotechnical boreholes have been drilled at Galat Sufar South and Wadi Doum for the geotechnical study in order to gather rock mass and structural information to derive slope parameters. This data complimented the basic geotechnical data from 18 cored boreholes which was provided for the 2016 PEA study.

A structural database was created from the logging of downhole geophysics OTV / ATV images, in addition to foliation and joint orientation measurements made in 2016. Structural domains were identified, presenting their own joint set orientations and attributes. Point load testing and laboratory strength testing of drill core was carried out to characterise the rock mass strength. This work has determined that inter-ramp pit slope angles range from 36.7° in the near surface weathered oxide rock mass to between 55° and 65° in the fresh rock, depending on structural geology controls and vertical inter-ramp height.

Given the gold grades and proximity to surface, the deposits will be mined by open pits using conventional trucks and excavators. Pit optimisations were run on both GSS and Wadi Doum using updated processing cost and recovery data. Mining costs were broken into base and incremental mining costs. Costs were built from first principles using knowledge of recent mining contracts operating under similar conditions in West Africa, and then validated through a contract budget pricing exercise involving several international mining contractors.

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

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

Based on the assumed mining equipment, a bench height of 5 m was used, although geotechnical conditions would allow 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 96% of the total crusher feed and 93% of the contained ounces and is exploited through eight separate pits, six of which are shallow oxide / transitional pits only. The two main pits are 260 m in depth. Containing considerable amounts of fresh rock, these will be exploited through a number of cutbacks. The overall strip ratio for the pits is 1.44:1.

Although much smaller than the GSS pits to the west, the WD deposit contains mineralization at 2.2 times the grade of GSS. The deposit is exploited through a three stage 120 m deep pit. The overall strip ratio for the WD pit is 2.89:1.

There is scope for larger pits with increased resources, improved geotechnical or financial conditions.

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

A ROM pad for GSS was also designed and can cater for six separate ROM fingers. The fingers will enable blending of the feed from the ROM pad to the primary crusher. A skyway has been designed at the back of the ROM pad which allows the construction of 10 m high fingers, allowing approximately 20,000 t of mill feed to be stockpiled on each finger.

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

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

It has been assumed that mining will be conducted by a mining contractor. The mine plan assumed that 91 t capacity rigid body haul trucks would be used at both GSS and Wadi Doum. As Wadi Doum has quite a short life compared to GSS, it will be possible to then use the equipment in the oxide pits at GSS or for other tasks such as the TSF lifts. During the contract tender process, the option of using smaller equipment for Wadi Doum should be investigated as this may add benefits to the economics through requiring smaller ramps and less waste stripping, as well as providing a more versatile fleet for other tasks around the operation.

A throughput trade off study determined an optimal processing rate of 6 Mtpa. A pre-production period was included which consists of six months' production prior to the commissioning of the processing plant. During this time, waste at both GSS and Wadi Doum will be targeted for mining, with any ore being stockpiled for reclaim when the plant is operational. A ramp up period of two years was also included.

A mining schedule was prepared which utilised an elevated cut-off grade. This strategy used artificial cut-off grades to allow higher grade material to be processed through the plant, while lower grade material is stockpiled. When compared with a simple "feed-all as it is available" schedule, the economic benefits were considerable. This approach was supported by a separate study which investigated the optimal extraction sequence for the operation.

Under this schedule, GSS has a mine life of eight years excluding the pre-production period and Wadi Doum has a mine life of under four years. The processing schedule extends for an additional six years after GSS is completed as low-grade stockpiles are rehandled and processed.

Table 1.5 shows the annualised mine production schedule.

Table 1.5 Mine Production Schedule

	Units	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
GSS Main Ore Tonnes	'000 t	22,923.0	14.5	2,835.0	1,250.4	3,434.0	5,908.1	5,068.3	3,896.5	516.1	
GSS Main Ore Au Grade	g/t	1.28	0.68	1.24	1.52	1.20	1.13	1.35	1.37	1.87	
GSS Main Waste Tonnes	'000 t	47,756.7	106.9	5,996.0	4,495.3	9,863.9	16,839.9	7,533.3	2,873.6	47.8	
GSS East Ore Tonnes	'000 t	50,266.9	205.8	3,303.4	7,454.8	5,902.8	3,818.2	6,769.8	6,018.2	12,164.9	4,628.8
GSS East Ore Au Grade	g/t	0.99	0.80	0.93	0.93	0.96	0.91	0.95	0.96	1.06	1.18
GSS East Waste Tonnes	'000 t	58,850.2	250.1	4,373.0	8,056.6	8,228.2	6,981.0	13,376.4	8,582.7	7,680.2	1,322.0
GSS NEZ Ore Tonnes	'000 t	4,165.7	10.3	797.5	530.2	0	0.9	194.6	2,261.8	370.5	
GSS NEZ Ore Au Grade	g/t	0.85	0.97	0.85	0.75	0.00	0.40	1.87	0.80	0.78	
GSS NEZ Waste Tonnes	'000 t	5,152.9	0.6	695.1	451.5	158.1	51.8	657.7	2,778.1	360.1	
Wadi Doum Ore Tonnes	'000 t	2,587.6	120.0	528.9	600.0	600.0	600.0	138.6			
Wadi Doum Ore Au Grade	g/t	2.36	2.51	1.85	2.38	2.37	2.73	2.55			
Wadi Doum Waste Tonnes	'000 t	7,488.3	578.7	3,188.2	1,984.0	1,006.2	647.8	83.3			
Total Ore Tonnes	'000 t	79,943.2	350.6	7,464.9	9,835.4	9,936.9	10,327.2	12,171.3	12,176.5	13,051.6	4,628.8
Total Ore Au Grade	g/t	1.11	1.38	1.10	1.08	1.13	1.14	1.15	1.06	1.08	1.18
Total Waste Tonnes	'000 t	119,248.1	936.3	14,252.3	14,987.4	19,256.4	24,520.5	21,650.7	14,234.4	8,088.1	1,322.0
Overall Strip Ratio		1.49:1	2.67:1	1.91:1	1.52:1	1.94:1	2.37:1	1.78:1	1.17:1	0.62:1	0.29:1

1.16 Recovery Methods

The metallurgical testwork conducted at SGS has provided the basis for the plant design. The key process design criteria for the plant are:

- Nominal throughput of 6.0 Mtpa with a grind size of 80% passing (P_{80}) 53 to 90 µm depending on ore type.
- Process plant availability of 91.3% supported by the selection of standby equipment in critical areas, reputable western vendor supplied equipment and on-site LNG generated power supply.
- Sufficient automated plant control to minimize the need for continuous operator interface but allow manual override and control if and when required.

The treatment plant design incorporates the following unit process operations:

- Single stage primary crushing with a gyratory crusher to produce a crushed product size P_{80} of approximately 110 mm.
- A crushed ore stockpile with a nominal live capacity of 10,000 t. Three apron feeders will be provided underneath the stockpile in the reclaim tunnel to deliver ore to the milling circuit.
- A grinding circuit configured as a two stage circuit with a SAG mill and closed circuit ball mill (SAB). The circuit will produce a P_{80} grind size of 53 µm for GSS oxide and Wadi Doum ores, approximately 75 µm for GSS transition ore and 90 µm for GSS fresh ores. In later years, pebble crushing will be required due to increasing ore competency; provision for this equipment has been allowed in the design.
- Pre-leach thickening to increase the slurry density feeding the leach and carbon in pulp (CIP) circuit to minimise tankage and reduce overall reagent consumption.

- Leach circuit incorporating a dedicated pre-aeration tank for passivation of cyanide consuming minerals and 10 leach tanks in series to provide 45 hrs leach residence time.
- A Kemix Pumpcell CIP circuit for recovery of gold onto carbon, to minimise carbon inventory, gold in circuit and operating costs. The CIP and elution circuit design is based on daily carbon harvesting.
- 12.5 t split AARL elution circuit, including optional cold cyanide strip, electrowinning, mercury retorting and gold smelting to recover gold from the loaded carbon to produce doré and safely remove mercury.
- Tailings thickening to recover and recycle process water from the CIP tailings.
- Tailings pumping to the tailings storage facility (TSF).

Table 1.6 shows the annualised processing schedule.

Table 1.6 Processing Schedule

	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
Galat Sufar South																
GSS Total Processed Tonnes	'000 t	77,356	4,200	5,200	5,400	5,400	5,812	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	3,343
GSS Total Processed Grade	g/t	1.07	1.34	1.34	1.35	1.37	1.63	1.48	1.49	1.14	0.72	0.68	0.68	0.62	0.55	0.49
Wadi Doum																
WD Total Processed Tonnes	'000 t	2,588	600	600	600	600	188	0	0	0						
WD Total Processed Grade	g/t	2.36	2.04	2.37	2.34	2.74	2.23	0.00	0.00	0.00						
	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
Total Block 14																
Total Processed Tonnes	'000 t	79,943	4,800	5,800	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	3,343
Total Processed Grade	g/t	1.11	1.43	1.44	1.45	1.51	1.64	1.48	1.49	1.14	0.72	0.68	0.68	0.62	0.55	0.49

1.17 Project Infrastructure

1.17.1 Water Supply

The groundwater resource required to operate a 6.0 Mtpa plant is:

- Constant water supply of an average of 145 l/s, 12,500 m^3/day or 4.6 million cubic meters (Mm^3) of water per annum.
- Minimum volume over the LOM of 64 Mm^3.

Exploration has been carried out in Area 5, where two existing wells had for many years reliably supplied water. An airborne conductivity survey was carried out by SkyTEM covering an area of 1,080 km^2 in conjunction with geological mapping.

Drilling of 17 exploration boreholes in Area 5 and four test production boreholes commenced in April 2017 and was completed in September 2017. Boreholes were drilled between 90 and 200 m deep. Groundwater was intersected at an average depth of 75 m below surface or 437 mamsl and the main aquifer system (higher yielding zone) around 100 m below surface or 410 mamsl. Static groundwater elevation is in the order of 443 mamsl. The main airlift yields ranged between 3.3 l/s and 11.9 l/s (the maximum flow able to be measured by the V notch). Multiple groundwater strikes were observed in a well sorted gravel zone at various depths and the average aquifer thickness of the main zone appears to be 40 m. No basement (bedrock) was intersected and the depth of the identified aquifer basin at its deepest central portion is uncertain; limited increases in airlift yields were observed in depth.

A total of nine aquifer tests consisting of conventional constant discharge rate and recovery tests were evaluated to obtain initial estimates of hydraulic properties and determine aquifer hydraulic parameters. The range of pumping rates was found to be 3.1 to 19.0 l/s.

A numerical model was constructed using Visual Modflow Pro and calibrated to match the pump tests. A conceptual well field design was used to evaluate the aquifer potential, where twenty boreholes drilled to a depth of 120 – 150 m at a spacing of 850 m and average yields of 7.5 l/s per borehole were simulated. Pumping of boreholes was simulated for a period of 10, 12 and 15 years with positive results.

A classification methodology as prescribed by Green, et al, 2005 has been used to evaluate the Water Resource. The aquifer contains 100 Mm^3 in the Measured Resource category and a further 40 Mm^3 in the Indicated Resource category.

1.17.2 Power Supply

The power supply will be provided by an on-site generation plant fuelled by LNG. The commercial execution model used for the study was a power station funded under a Build Own Operate Transfer (BOOT) arrangement, transferred after five years.

The load profile estimates a 24.3 MW peak and 23.5 MW average demand (Table 1.7). This can be serviced using the twelve (12) 2.5MW Rolls Royce MTU 16V4000 Gas engines with Stamford LVS1804T 400V alternators to generate power at 11kV, 50Hz.

Table 1.7 Project Electrical loads

Area	Connect Load (MW)	Maximum Demand (MW)	Average Demand (MW)	Annual Energy Consumption (GWh)
Process Plant	31.07	23.78	23.01	184.08
Site Infrastructure	1.01	0.47	0.46	3.68
Borefield Infrastructure	0.44	0.35	0.32	2.56
Total	**32.52**	**24.6**	**23.79**	**190.32**

LNG will be transported to the site (in liquid form) from the Port of Sudan. Once onsite, the LNG will be stored in cryogenic vessels. Waste heat recovered from the operation of the Power Station will be utilized to re-gas the LNG for utilization by the gas generators. All power generation equipment associated with this Power Station will be LNG fueled.

The Power Plant will operate in an N+2 arrangement.

- 10 + 2 Generators (N+2 Configuration) rated at 2,484 kWe each at site.
- 24,840 kWe + 4,968 kWe installed thermal Generation Capacity.
- 28,340 kWth + 5,668 kWth potentially recoverable thermal output.

A power cost of US$0.1086 /kWh was used in the operational cost analysis based on the LNG price $9.70/MBTU delivered to site.

The distribution of onsite power from the power station is designed around an 11 kV, 50 Hz reticulation network including both underground and overhead power lines. The plant loadings are broadly dived between two main switchboards, one for the front-end crushing / milling and the second supplying the leach / desorption back end of the process. Site infrastructure services are provided from a high voltage kiosk located adjacent to the plant administration facilities which reticulates via overhead powerlines to the mine services, tailings decant and camp facilities.

Remote site power infrastructure of the water supply at the borefield and transfer water pumping station will consist of two high-speed diesel generators in a duty / standby arrangement at the pump station, power is stepped up to an 11 kV, 50Hz overhead powerline which reticulates out to the multiple boreholes. Small pole mounted step-down transformers and local bore pump controls panels complete the system and provide integrated control of the pumping system.

1.17.3 Tailings Storage Facility

The design of the tailings storage facility (TSF) was undertaken, to international standards, to provide a facility to safely contain all the the tailings solids and reduce the potential effect thereof on the environment in the form of dusting, seepage, or run-off from the tailings surface during operation and post closure. Provision was made for the effects of seismic events and probable maximum precipitation events during operation and post closure. To support the design and improve the safety of the facility, seepage analysis and stability analysis were conducted on the embankments. A water balance model was prepared to determine the water volumes retained in the TSF and the available recycle volumes to the plant. If built and operated in accordance with the principles and design concepts outlined in this document, this facility would contain the tailings generated from the project and the effects on the environment would be within acceptable limits as defined by international standards.

The proposed TSF is located within a valley located north of the unnamed Wadi running north of the process plant. The TSF has been designed as two cell paddock facility to allow alternate deposition and subsequent upstream raising.

The facility will have a partial HDPE basal liner employed over the base of the valleys with an extensive underdrainage system and subliner drainage to recover water and reduce seepage. The liner will extend up the embankment batter slopes when constructed by downstream techniques. Water will be recoved from the facility from decant towers constructed in each cell with water pumped back to the process for recycling.

At the final stage of the construction, the total embankment length of the East Cell would be approximately 2.8 km (excluding the divider wall which is approximately 1.0 km long), with a maximum embankment height of 38 m at the southeast corner. The total embankment length of the West Cell would be approximately 1.3 km (excluding the divider wall), with a maximum embankment height of 31 m at the southern extremity. The tailings beach surface at full capacity will cover an area of approximately 280 ha. At end of operations the tailings beach will be covered by a waste rock cover to prevent loss of tailings post closure.

1.18 Market Studies and Contracts

No formal market studies have been undertaken. Block 14 will produce gold doré which is readily marketable on an 'ex-works' or 'delivered' basis to a number of refineries in Europe and Africa. There are no indications of the presence of penalty elements that may impact the price or render the product unsalable.

1.19 Environmental

There are currently no objections to the development of the Project. There are no receptors in the area, with no human settlements in proximity. The Project has undertaken baseline studies for the Environmental Impact Assessment (EIA) and data collection is continuing. The hyperarid conditions and sparse vegetation of the concession provide a difficult environment for wildlife, which is primarily nocturnal in movement. Using remote infrared cameras provides the opportunity to record those nocturnal activities. Other wildlife records are captured through daily observations to the extent possible. Climate data and weather data has been collected and compared, to provide reliable data for the EIA and design teams. Water data from the existing boreholes and Talat Abda well has been collected. There are no other known sources of potable water with few potential water users in the vicinity. Social data continues to be collected, through monitoring of artisanal miner activities. All information collected will be used in managing social and environmental impacts associated with the Project.

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 mining activities, including exploration, in such a way as to mitigate the negative impacts. To achieve this, MSMCL initiated environmental measures through their Exploration Statement, which is specific to exploration work and includes an Environmental Protection and Management Plan (EPMP) to mitigate impacts associated with the work. In accordance with the Concession Agreement, the EPMP is a dynamic document that is revised and updated as the Project progresses.

The EIA was approved by the Higher Council of Enviroment and Natural Resources in February 2019. The council determined that while the development of the Project may give rise to some environmental impacts, these can be minimised. Project development and operations will cause several changes to some aspects of the local environment, e.g. visual, air and water quality, use of water resources and removal of wildlife habitat among others. The absence of any human community in the concession means that social impact will be minimal and restricted to within the mine complex itself. Assuming the implementation of mitigation measures proposed in the EIA, any changes are considered manageable and controllable. Therefore, the development, operation and closure of the Project could be undertaken in an effective environmental and social manner.

1.20 Capital and Operating Costs

The capital estimate is summarized in Table 1.8 and 1.9. The initial project capital cost is estimated at US $321.2M, including a contingency allowance of US$33.6M.

Table 1.8 Capital Estimate Summary (3Q20, ±15%)

Main Area	**US$'000s**
Mine	$15,443
Treatment Plant	$193,492
Generator Plant	$ 7,100
TSF	$17,301
Camp	$2,506
EPCM	$26,853
Owners Costs	$24,970
Subtotal	**$287,664**
Contingency	$33,557
Grand Total	**$321,222**

The duration of the detailed design and construction phase of the project has been estimated to be 27 months, with the construction of the raw water pipeline the dominant critical path activity. The site accommodation camp size has been selected, by taking the manning demands for the various overlapping activities into account. A program of early engineering works has been completed, which will enable detailed design to commence on completion of financing.

The total LOM cost is estimated at US$499.8M, including sustaining capital costs of US$178.6M, as shown in Table 1.9. The Camp and Generator Plant will be financed under a Build Own Operate Transfer (BOOT) contract. The duration of the contracts will be five years.

Table 1.9 Sustaining Capital Estimate Summary (3Q20, ±15%)

Main Area	US$'000s
Camp	$14,198
TSF	$53,733
Generator Plant	$63,112
Process Plant	$35,000
Closure	$12,546
Grand Total	**$178,590**

Contract open pit mining costs were derived by Deswik Europe from first principles based on equipment required and include pit and dump operations, road maintenance, mine supervision and technical services cost. These costs were then compared with several contractor responses for budget pricing followed by validation against a more detailed budget estimate provided by a well established African based mining contractor. The average open pit operating cost (US$ /t mined) is shown in Table 1.10.

Table 1.10 Mining Costs

Mine Area	Mineralized Rock (US$ /t)	Waste Rock (US$ /t)
Main	2.85	2.48
East	2.87	2.52
NEZ	2.36	2.09
Wadi Doum	4.67	4.41
Total	**2.89**	**2.60**

A grade control drilling cost of $0.26 /t has been included in the above cost for ore tonnes. In addition, a transfer cost of US$7.59 /t will be incurred on ore from Wadi Doum. The average unit rate for rehandle is US$0.87 /t and amounts to approximately US$34.9over the life of the project. A diesel price of US$0.60 /l was used.

The process operating cost estimate has been compiled from a variety of sources, including metallurgical testwork, Orca advice, OMC comminution modelling, first principle calculations, vendor quotations and the Lycopodium database.

The process estimate comprises the following major cost centres:

- Plant and related infrastructure power.
- Plant consumables, including mill media and liners, reagents and diesel for fixed plant equipment and plant mobile equipment.
- Plant maintenance materials, including mobile equipment parts.
- Laboratory.
- Plant and administration labour.
- General and administration costs.

The process operating cost includes all direct costs to produce gold bullion for the Project. The battery limits are the ROM feed into the primary crusher (ROM loader by Mining), production of dore and mercury in the gold room and discharge of tailings at the TSF.

Operating costs are presented in United States Dollars (US$), to an accuracy of ±15% and are based on pricing obtained during the third quarter of 2020.

Process operating costs have been developed for each major domain and lithology. Operating costs were developed using the plant parameters specified in the process design criteria. Tables 1.11 to Table 1.14 present the operating cost summary by material type. Costs are presented as fixed and variable for ease of presentation.

Table 1.11 Operating Cost per Material Type, Main Zone (3Q20, ±15%)

Cost Centre	Fixed US$'000/y	MZF - PGSS	MZF - FQSP	MZF - VAN	MZT - PGSS	MZT - FQSP	MZT - VAN	MZOX - PGSS	MZOX - FQSP	MZOX - VAN
		Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t
TOTAL	19,977	5.55	7.32	5.81	6.5	6.46	5.88	5.93	6.04	5.43

Table 1.12 Operating Cost per Material Type, East Zone (3Q20, ±15%)

Cost Centre	Fixed US$'000/y	EZF - FQSP	EZF - PGSS	EZF - SRR	EZF - VAN	EZT - FQSP	EZT - PGSS	EZT - SRR	EZT - VAN	EZOX - FQSP	EZOX - PGSS
		Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t
TOTAL	19,977	7.68	6.38	9.06	7.76	6.78	6.06	6.15	5.83	5.36	5.43

Table 1.13 Operating Cost per Material Type, East Zone & Wadi Doum (3Q20, ±15%)

Cost Centre	Fixed US$'000/y	EZOX - SRR	EZOX - VAN	WDF - QPR	WDF - VAN	WDF – VC1	WDF – VC3	WDT	WDOX
		Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t
TOTAL	19,977	5.38	5.45	7.04	7.32	7.49	7.56	9.20	6.51

Table 1.14 Operating Cost per Material Type, North East Zone (3Q20, ±15%)

Cost Centre	Fixed US$'000/y	NET - PGSS	NET - VAN	NEOX - FQSP	NEOX - PGSS	NEOX - VAN	LOM
		Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t
TOTAL	19,977	5.56	5.31	5.60	5.53	5.54	6.41

1.21 Economic Analysis

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

Unless otherwise stated, all currencies refer to US$.

The financial analysis used a base price of US$1,350 /oz. The financial assessment of the project is carried out on a "100% equity" basis and the debt and equity sources of capital funds are ignored. No provision has been made for the effects of inflation. Current Sudan tax regulations are applied to assess the tax liabilities. Discounting has been applied from the first year of operation.

The results of the financial model are summarized in Table 1.15. A breakdown of the annualised operating and economic details can be found in Tables 22.4 and 22.5.

Table 1.15 Financial Model Summary

Description	Units	LOM
Feed Tonnage	Mt	79.9
Waste Rock	Mt	119.3
Total Mined	Mt	199.2
Stripping Ratio	W:O	1.49:1
Feed Grade Processed (average)	g/t	1.11
Gold Recovery (average)	%	82.0
Production Period	Years	13.6
Gold Production	'000 oz	2,341
Annual Gold Production (Yr 1-7)	'000 oz/y	228
Annual Gold Production (average)	'000 oz/y	167
Silver Production	'000 oz	1,195
Pre-production Capital Cost	US$M	321
Sustaining Capital Cost	US$M	179
Total Capital Cost	US$M	500
Gross Revenue	US$M	3,167
Total Operating Costs	US$M	1,604
Operating Profit	US$M	1,563
Mining Cost / t Mined	US$ /t	2.67
Process Cost / t Processed	US$ /t	8.16
Cash Cost / oz Au recovered	US$ /oz	676
AISC / oz Au recovered	US$ /oz	751

Table 1.16 shows the sensitivity of the NPV and IRR with gold price.

Table 1.16 NPV and IRR Gold Price Sensitivity

	Low	Base	3Yr Trailing		Spot[1]
Au Price	1,250	1,350	1,410	1,750	1,950
$NPV_{5\%}$ ($million)	467	607	691	1,166	1,445
IRR	27.5%	33.3%	36.8%	55.4%	66.0%

[1]31 August, 2020

1.22 Adjacent Properaties

Managem International operate a pilot plant scale operation 100 km south of Block 14 at Gabgaba and are known to be exploring for gold mineralization in the Block 15 exploration licence.

Tahe Minerals are operating a small gold mine 200 km to the west of Block 14 in the Wadi Halfa area. There is no publicly available information on the project.

1.23 Interpretation and Conclusions

1.23.1 Mining

The mining of the GSS and WD deposits has been shown to be technically feasible through conventional open pit mining methods. Pit optimizations show that both deposits have economically viable material under the assumed economic and physical parameters.

A combined mining schedule demonstrates that a processing rate of 6.0 Mt can be sustained for 14 years, including an initial ramp up period in Years 1 and Year 2. A pre-production period of six months is utilized to strip waste from both deposits and stockpile minor amounts of crusher feed, ready for the commissioning of the processing plant. The pit at WD is mined for the first five years and blended with material from GSS. An elevated cut off grade over eight years maximises early returns by blending the significantly higher-grade material from the WD deposit and gaining access to the higher grades at depth in the Main and East pits. The last six years rehandle the low-grade stockpiles.

1.23.2 Mineral Processing and Recovery Methods

A comprehensive program of metallurgical testwork was conducted at SGS to identify the optimum processing route, generate recovery equations and provide data for plant design. Mineralogy, comminution, leach, carbon modelling, thickening and rheology tests were conducted on a large range of ore lithologies and domains.

The Block 14 ores are amenable to a conventional gold processing scheme incorporating crushing, SAG and ball milling, cyanide leach, carbon adsorption, elution, electrowinning and gold smelting. The comminution circuit and gold recovery plant design is based on a throughput rate of nominally 6.0 Mtpa.

Trade-off studies have identified that there is economic benefit in including a pre-aeration stage ahead of cyanide leaching to reduce cyanide consumption rates and that the Kemix Pumpcell CIP circuit provides the optimum carbon adsorption circuit design.

The proposed process route is a proven and robust concept with very little associated risks, apart from the typical 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.

1.23.3 Major Infrastructure

Water Supply

Seventeen exploration boreholes and four test production holes have been drilled within the Area 5 aquifer, which indicated high and constant airlift yields with the occurrence of groundwater within an inter-layered basin type sequence of course to medium grained sandstone.

A classification methodology prescribed by Green, et al, 2005, has been used to evaluate the Water Resource. The aquifer contains 100 Mm^3 in the Measured Resource category and a further 40 Mm^3 in the Indicated Resource category.

Power Supply

Based on the trade-off carried out, self-generation using LNG fuel is the preferred option with the least associated risk.

1.23.4 Environmental

There are currently no objections to the development of the Project. There are no receptors in the area, with no officially recognised human settlements in proximity.

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 mining activities, including exploration, in such a way as to mitigate the negative impacts.

The EIA has been approved by the Higher Concil of Enviroment and Natural resources. This determined that while the development of the Project may give rise to some environmental impacts and identified ways these can be minimised. The absence of any current social habitat in the concession means that social impact would be minimal and restricted to within the mine complex itself. Furthermore, the implementation of mitigation measures proposed in the EIA, any changes are considered manageable and controllable. Therefore, the development, operation and closure of the Project could be undertaken in an effective environmental and social manner.

1.23.5 Economic Analysis

Based on a gold price of US$1,350 /oz, the Project has a pre-tax IRR of 38.5% and pre-tax $NPV_{5\%}$ of US$723M. The Project has a post-tax IRR of 33.3% and a post-tax $NPV_{6\%}$ of US$607M.

The Post-tax, undiscounted payback period is 2.9 years from start of production.

The NPV and IRR are most sensitive to the metal price / recovery, and to a lesser extent pre-production capex, plant and mine operating cost.

1.23.6 Opportunities

Mining

Material haulage from Wadi Doum could be improved if long distance haulage units which can be loaded at the face are utilized. This would negate the need for ROM stockpiles and re-handling of the crusher feed, potentially saving US$0.40 – 0.50 /t of crusher feed from Wadi Doum.

A review of the hydrogeology will lead to a greater understanding of the effects of groundwater on the pit walls (if any) and could allow for further steepening.

Mineral Processing

Additional test work will be undertaken to support an economic trade-off / optimization study on increasing silver recoveries by increasing carbon movement to investigate the potential to reduce operating costs or improve recovery.

Tailings

There is the potential to construct the starter embankment from overhauled Run of Mine waste if the mining schedule and truck availability permits. This could reduce the cost of the initial construction of the tailings storage facility.

1.24 Recommendations

The Feasibility Study has demonstrated a strong project. During the period of financing a range of early works will reduce the timeframe required between project award and project completion for a relatively minor investment.

During the financing period, Lycopodium will carry out early works engineering services covering:

- Specifications for additional long lead equipment packages. The mills, crusher and raw water pipeline are complete.
- Geotechincal investigations.

ISI Prime would complete the power station design to enable the long lead item specifications to be completed.

1.24.1 Mining

- Complete hydrogeological study for GSS to identify presence, volume and effects of ground water and follow up with updated geotechnical review.
- Review and revise stockpile management practices to allow better grade definition within stockpiles.

1.24.2 Mineral Processing and Metallurgical Testing

- Conduct a Trade-Off Study to ascertain the benefits of increasing silver adsorption stage efficiencies onto carbon by moving more carbon per day to reduce gold loadings, which will allow more silver to adsorb.

1.24.3 Water Supply

- Undertake a long duration pump test, in the order of 30 days, from two of the strong yielding boreholes and from two of the medium yielding boreholes. This will supply more information to confirm the hydrogeological flow parameters and how the aquifer behaves during longer term abstraction.

1.24.4 Tailings Storage Facility

The following work is recommended prior to or during the detailed design stage of the tailings storage facility:

- Complete a geophysical survey (or geotechnical investigations) along the alignment of the main embankment to confirm the depth of aeolian sand below the embankment footprint to allow refinement in the location of drainage systems and the cut-off trench design.
- Conduct geological mapping of the TSF basin to define the main structural features within the basin to ensure they are accounted for in the detailed design.
- Conduct additional drilling and packer testing to confirm the in-situ permeability and variability of the permeability of the foundation rocks in the areas not covered by the HDPE liner in the TSF.
- Complete any outstanding sterilisation drilling within the TSF basin to confirm that there are no exploration targets below the facility.

1.24.5 Process Plant Geotechnical Design

The following work is recommended prior to or during the detailed design stage of the process plant:

- Complete geotechnical investigations and geotechnical design checks at the actual location of the key process infrastructure including the crusher, mills, tanks, and thickener.
- Finalise the concrete aggregate testing which has commenced to ensure suitable aggregate sources (both fine and coarse) have been located and develop and test potential concrete mix designs for the early construction works.

2. INTRODUCTION

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

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

2.1 Basis of Technical Report

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

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

- Geoff Duckworth (Lycopodium Minerals Pty Ltd), responsible for report Sections: 1.16, 1.17, 1.20, 1.21, 1.23, 1.24, 17, 18.2, 18.4, 18.5, 18.6, 21, 22, 25 and 26.
- Michael Hallewell (MPH Minerals Consultancy Ltd), responsible for report Sections: 1.12, 13, 25.6.2 and 26.2.3.
- Nicholas Johnson (MPR Geological Consultants Pty Ltd), responsible for report Sections: 1.13 and 14.
- Pieter Labuschagne (GCS), responsible for report Sections: 1.17.1, 18.1 and 26.2.4.
- Carl Nicholas (Mineesia Ltd), responsible for report Sections: 1.19, 20, 25.4 and 26.2.1.
- Chris Reardon (Orca Gold Inc, previously Deswik Europe Ltd.), responsible for report Sections: 1.14, 1.15, 15, 16, 21.3, 21.5, 25.1, 25.6.1, 25.6.2 and 26.2.2.
- Tim Rowles (Knight Piésold Pty. Ltd.) responsible for report Sections: 1.17.3, 18.3, 25.6.2, 26.2.5 and 26.2.6.
- Other sections have been provided by the Company.

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

2.2 Property Inspections by Authors

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

Table 2.1 Summary of QP Site Visits

Qualified Person	Site Visit
Geoff Duckworth	05/10/16 – 07/10/16
Nicholas Johnson	17/01/14 – 21/01/14
Pieter Labuschagne	23/11/16 – 02/12/16
Carl Nicholas	24/11/14 – 03/12/14
Chris Reardon	23/07/18 – 25/07/18
Tim Rowles	05/10/16 – 07/10/16

2.3 Effective Dates

The Effective Date of this report is August 31, 2020. There were no material changes to the scientific and technical information of the Project between the Effective Date and signature date of this Report.

2.4 Abbreviations

a	annum
AAS	Atomic Absorption Spectrometry
ANS	Arabian Nubian Shield
Area 5	Hydrological Area for potential project water supply
As	Arsenic
Au	Gold
B14WC	Block 14 Water Concession
BOO	Build Own Operate
BOOT	Build Own Operate Transfer
CA	Concession Agreement
CAE Fusion	Geological Data Management System
CIL	Carbon-in-Leach
CIP	Carbon-in-pulp
CRM	Certified Reference Material
˚C	Degree Celsius
EGFS	Eastern Gabgaba Fault System
EIA	Environmental Impact Assessment
EPL	Exclusive Prospecting Licence
EPMP	Environmental Protection and Management Programme
F_{80}	80% of a unit process feed particle size is below a given size, based on particle size distribution (PSD)
g	grams
g/l	grams per litre
g/t	grams per tonne
HA8	Hydrological Area for potential project water supply
HQ	Exploration drill size (96 mm OD / 63.5 mm ID)
GAT	Gravity Amenability Test
GDMS	Geographical Data Management System
GED	General Exploration Drilling
GPS	Global Positioning System
GSS	Galat Sufar South
ha	Hectare
HARD	Half Normal Distribution
HLS	Heavy Liquid Separation
hr	Hour
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
IRR	Internal Rate of Return
Km	kilometer
km^2	square kilometres
Kv	kilovolt
kWh	kilowatt hour
LNG	Liquefied Natural Gas
l	litre
l/s	litre per second
M	million
masl	metres above sea level

MDA	Mineral Discovery Area
Meyas Nub	Previously known as Emdehan Multi-activities Company Ltd
MIK	Multiple Indicator Kriging
Min	minutes
Mm^3	million cubic metres
MBTU	Million British Thermal Units
MoM	Ministry of Minerals
MSMCL	Meyas Sand Mineral Company Ltd
mS/m	milli Siemens per metre
Mtpa	million tonne per annum
Mt	million tonne
NEZ	North East Zone
NPV	Net Present Value
NQ	Exploration drill size (75. 5mm OD / 47.6 mm ID)
oz	31.10348 grams
PDS	Particle Size Distribution
PFS	Pre-Feasibility Study
PIR	Passive Infra-red
ppm	parts per million
PQ	Exploration drill core size (122.6 mm OD / 85 mm 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
RFS	Revised Feasibility Study
RMR	Rock Mass Rating
ROM	Run-of-Mine
RQD	Rock Quality Designation
SAG mill	Semi Autogenous Grinding mill
SMCL	Sand Metals Company Ltd
SMRC	Sudan Mineral Resource Company
SD	Standard Deviation
SG	Specific Gravity
t	metric tonne (1,000 kg)
TDS	Total Dissolved Solids
TOR	Terms of Reference
TSF	Tailings Storage Facility
VMS	Volcanogenic Massive Sulphide
WD	Wadi Doum
uS/cm	micro Siemens per centimetre
µm	micron

3. RELIANCE ON OTHER EXPERTS

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

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

General: the Author has relied on information provided by Orca Gold Inc. for Sections 1.3, 1.4, 1.5, 4, 5 and 6.

Geology: the Author has relied on information provided by Orca Gold Inc for Sections 1.6, 1.7, 1.8, 1.9, 1.10, 1.11, 7, 8, 9, 10, 11 and 12.

Mineral Resources: the Author has relied on information provided by MPR Geological Consultants for Sections 1.13 and 14.

Mining: the Author has relied on information provided by Deswik Europe Limited for Sections 1.14, 1.15, 15, 16, 21.3, 21.5, 25.1, 25.6.1, 25.6.2 and 26.2.2.

Metallurgical Testwork: the Author has relied on information provided by MPH Minerals Consultancy Ltd for Sections 1.12, 13, 25.6.2 and 26.2.3. Lycopodium has reviewed the metallurgical testwork results and concurs with their interpretation.

Hydrogeology: the Author has relied on information provided by GCS Water and Environmental Consultants for Sections 1.17.1, 18.1 and 26.2.4.

Environment and Social: the Author has relied on information provided by Mineesia Limited for Sections 1.19, 20, 25.4 and 26.2.1.

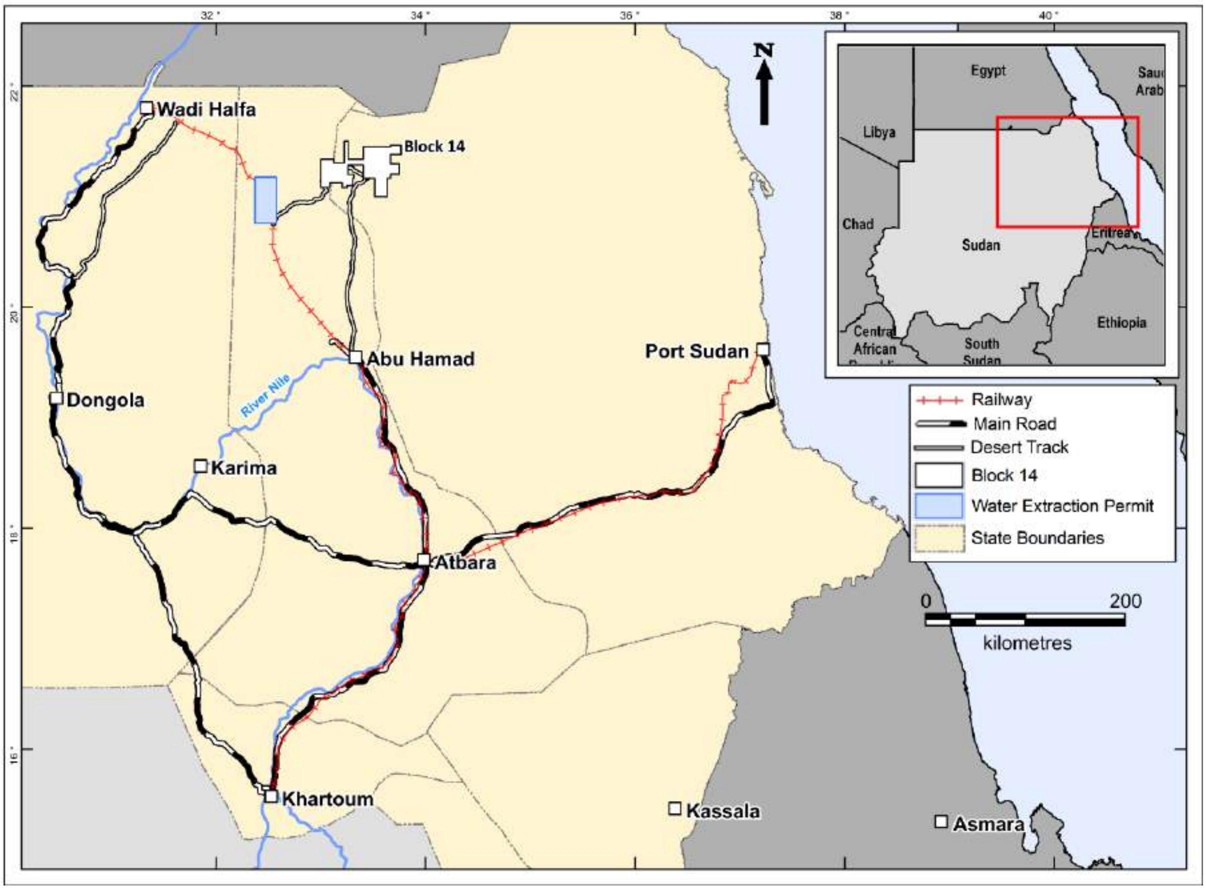
Financial: the Author has relied upon the financial analysis by Orca Gold in Sections 1.21 and 22 of this report. Lycopodium has reviewed the inputs and basis for the financial analysis.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Block 14 mineral exploration project covers an area of 1,000 km^2. It is located in the Nubian Desert in the north of the Republic of the Sudan, close to the international border with Egypt (Figure 4.1). The property is situated 700 km north of the national capital Khartoum and straddles the boundary between the Red Sea and Nile States. The nearest population centre is the town of Abu Hamad, located 180 km due south of the property.

Figure 4.1 Block 14 Current Tenement Boundary



Source Orca Gold Inc.

4.2 Sudan Mineral Tenure and Fiscal Frame Work

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

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

4.2.1 Exclusive Prospecting Licence (EPL)

An EPL is granted by the Minister of Minerals and gives the holder the exclusive right to explore for specified minerals, both on the surface and at depth, within a certain parcel of land described by a set of co-ordinates. The EPL also gives the holder the exclusive right, at any time, to convert the licence into a Mining Lease in accordance with the Concession Agreement.

4.2.2 Mining Lease

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

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

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

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

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

4.2.3 Ownership

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

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

Under the terms of the Purchase Agreement, SMCL have paid Meyas Nub US$9.5M in three instalments in exchange for an increasing ownership interest. All payments have been made to Meyas Nub, who now retain the remaining 30% interest in MSMCL.

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

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 4.1.

Table 4.1 Fiscal Elements

Tenement Name	Block 14
Current Area	1,000 km^2
Concession Type	Gold and associated minerals
Annual Surface Rental (US$ /km^2)	10
Annual training Fund Contribution	EUR36,000
Second Extension Period	1 year
Royalty	7%
Corporate Tax	15%
Government Free Carry (%)	20%
Partner Free Carry (%)	10%
Mining Lease Period	30 years
Mining Lease Extension Period	10 years

4.3 Mineral Tenure of the Block 14 Project

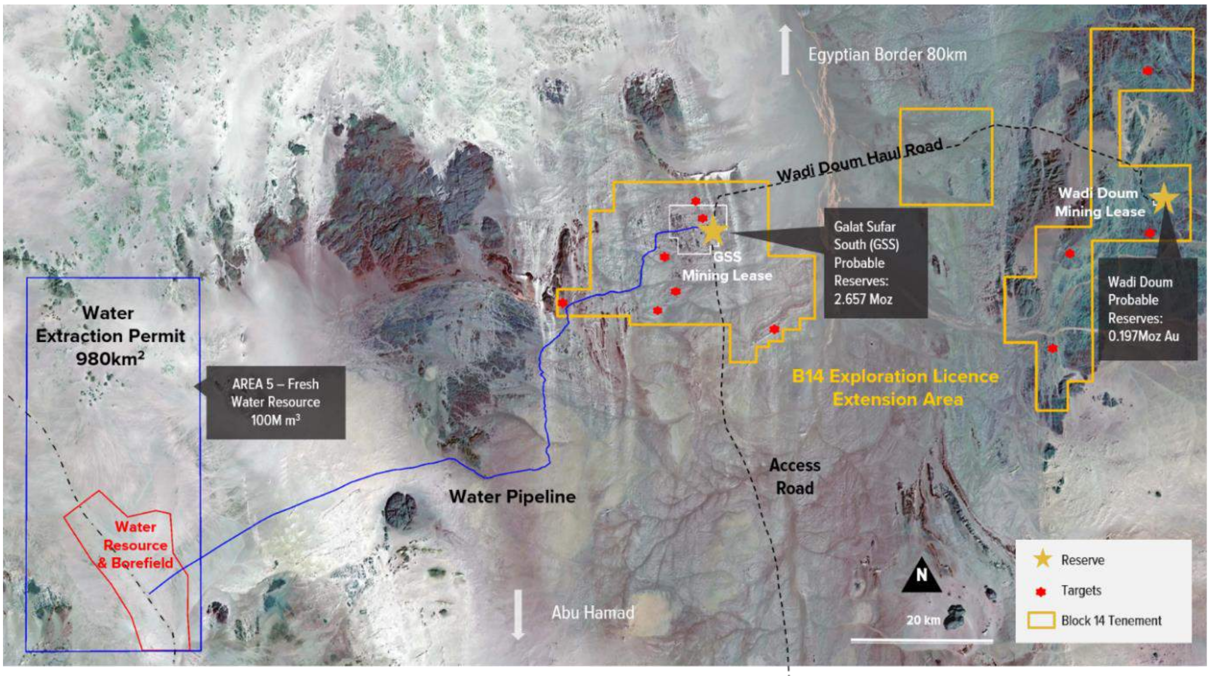
The Exclusive Prospecting Licence is granted under terms of the concession agreement for the exploration and exploitation of gold and associated metals and minerals in Lower Gabgaba Area ("Block 14"). The initial exploration period was for four years, however during 2014, the joint venture company MSMCL, was granted an additional year extension to the Initial Exploration Period. Details of key dates and terms are detailed in Table 4.2.

Table 4.2 Summary of Concession Agreement Obligations and Key Dates

Start Date	Expiry Date	Area (km^2)	Terms
19 May 2010	19 May 2014	7,046	Initial exploration period.
19 May 2014	19 May 2015	7,046	Additional year compensation period granted, extending the expiry date of the initial exploration period.
19 May 2015	19 May 2017	3,747	Formal notification of Mineral Discovery Areas, relinquishment of 50% of the Initial Exploration area.
19 May 2017	19 May 2018	2,170	Relinquishment of 50% of remaining area (excluding mineral discovery areas).
19 May 2018	19 November 2018	2,170	Extension of the final period of the permit.
18 December 2020	18 November 2022	1,000	Initial term of an exploration permit over 1,000km^2.

On 27 February 2015, MSMCL submitted a declaration of Mineral Discovery Areasand requested the first two-year extension period in line with the concession agreement. On 19 February 2017, MSMCL formally requested the second extension period (May 2017 to May 2018) and relinquished 50% of the permit area (less thae declared Mineral Discovery Areas). The permit extension was granted. On 10 May 2018, MSMCL formally requested a six-month extension to submit an application for a Mining Licence. This extension was granted on 20 May 2018 and the permit is valid to 19 November 2018. Following the submission of the Mining Licencne Application, a new exploration permit was granted covering 1,000km^2 (Figure 4.2) with an inntial expiry date of 18 November 2022.

Figure 4.2 Map Showing Block 14 EPL



Source: Orca Gold Inc.

4.4 Environmental Responsibilities

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

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

4.5 Other Factors and Risks

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

Under the terms of the Concession Agreement, the company has the right to access all areas for the purpose of mineral exploration. The area is uninhabited and there are no areas that are held by individuals. Historically, a large number of artisanal miners have operated within Block 14. In 2016, the artisanal miners were moved off the Wadi Doum site and in April 2018, the artisanal miners were removed from the GSS site. A number of artisinal miners have recently returned to GSS The Mining Police are scheduled to remove them from site in Q3 2020.

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 and neighbouring countries provide daily flights from Khartoum. The airport at Port Sudan also has international flight handling capability, although services are limited to neighbouring countries. Daily flights between Khartoum and Port Sudan are available.

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, where a well-used desert road capable of handling moderate to large loads connects to the project area (Figure 4.1 and Figure 5.1). In general, vehicular access to the Project is good and is not affected by seasonal variations.

Figure 5.1 Transporting 20 t Reverse Circulation Drill Rig

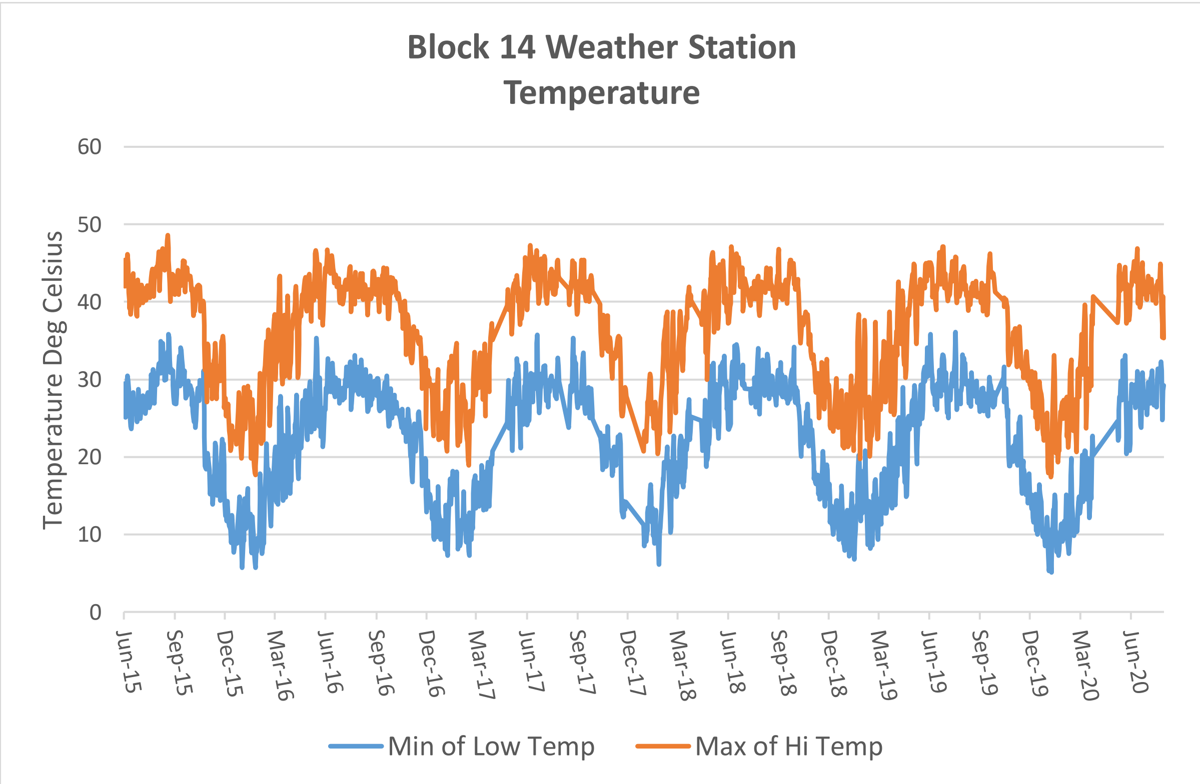
Source: Orca Gold Inc.

5.2 Climate

Since June 2015, a weather station has been recording data in Block 14 (Figures 5.2 to 5.4). The climate is arid with a hot season from June to September, during which the maximum temperatures range from 45° - 49°C and minimum of 23° - 35 °C (Figure 5.2). 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 season, the project area is subject to strong winds, predominantly from the north (Figure 5.3).

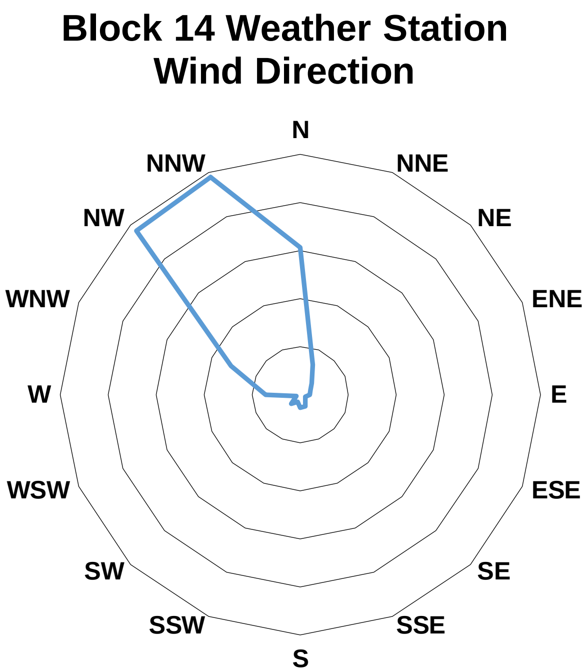
The northern desert of the Republic of the Sudan has infrequent precipitation and regional records indicate average annual precipitation to be nil. However, infrequent rainstorms do occur, with three events of rainfall >5 mm (Figure 5.4).

Figure 5.2 Site Temperature Records from Block 14



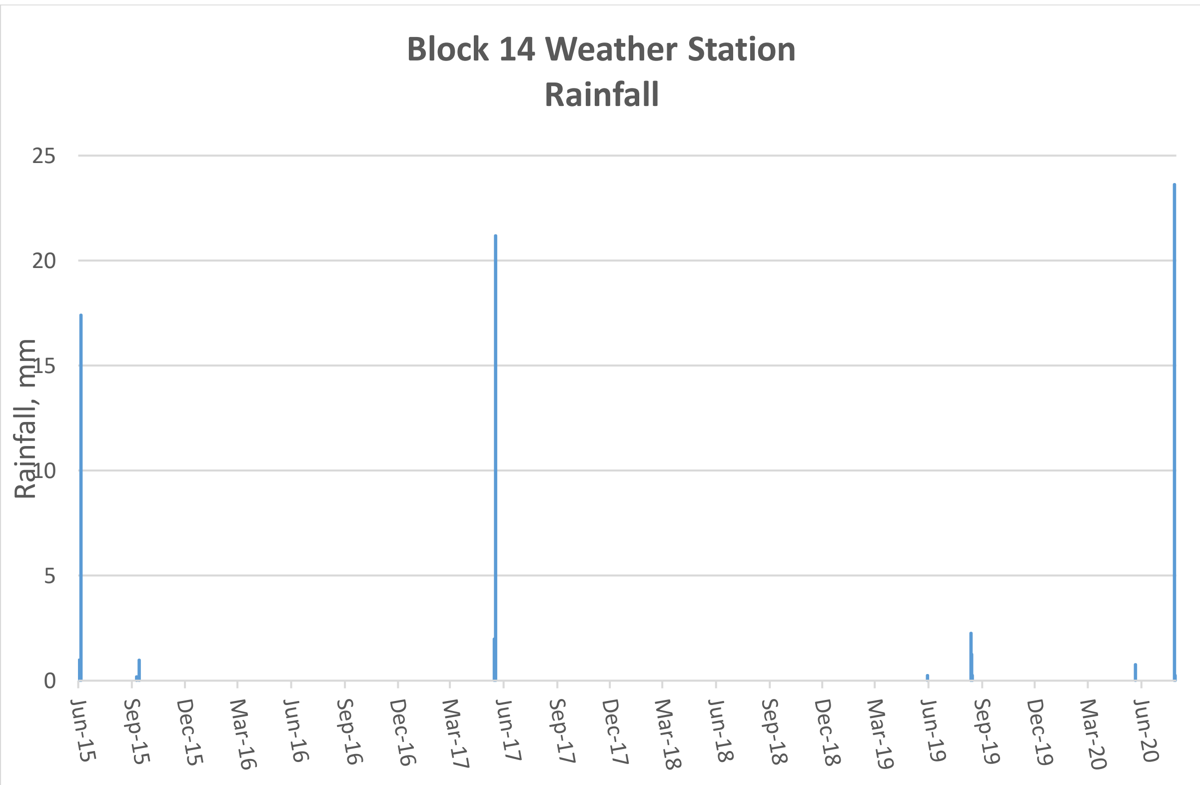
Source: Orca Gold Inc.

Figure 5.3 Wind Direction (Origin) Recorded at Block 14



Source: Orca Gold Inc.

Figure 5.4 Rainfall Recorded at Block 14



Source: Orca Gold Inc.

5.3 Local Resources

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

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

5.4 Infrastructure

The Company operates a 100-person exploration camp in Block 14, comprising converted sea containers and brick built buildings that house exploration and support staff (Figure 5.5). In Q3 2020, it is planned to establish a camp at GSS to enable surface works to commence. Power is self-generated from fuel delivered as needed in 25,000 L tankers. Fresh water is supplied by tanker from Abu Hamad.

Sudan has an established paved road network which is generally of good quality.

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.

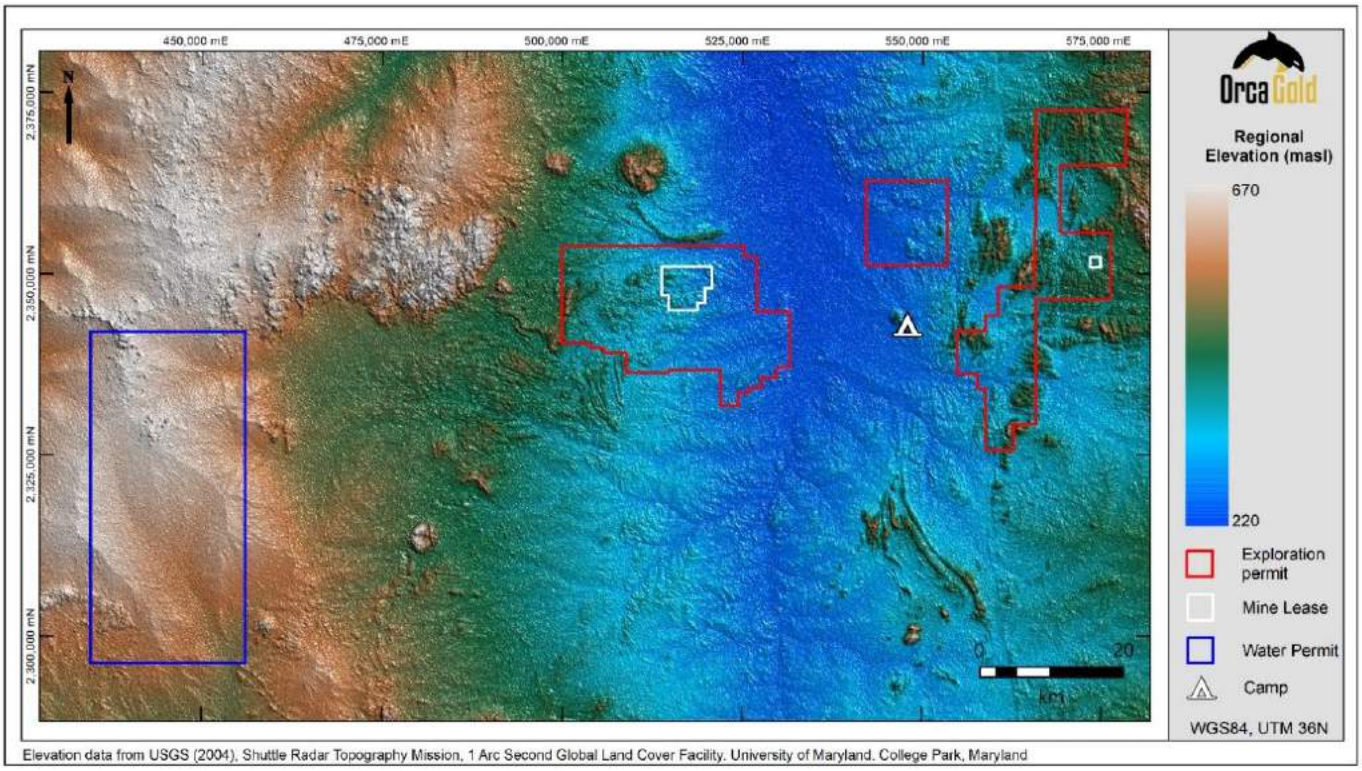
Figure 5.5 MSMCL Exploration Camp

Source: Orca Gold Inc.

5.5 Physiography

Elevation varies between 225 and 775 masl with the bulk of the higher ground in the east of the project area (Figure 5.6). Wadi Gabgaba, which runs north-south through the centre of the project area, is the main drainage system in the area. The western part of Block 14 contains more subdued topography.

Figure 5.6 Physiography of Block 14 Tenement



Source: Orca Gold Inc.

Figure 5.7 Photograph of Wadi Gabgaba

Source: Orca Gold Inc.

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

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

6. HISTORY

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

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

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

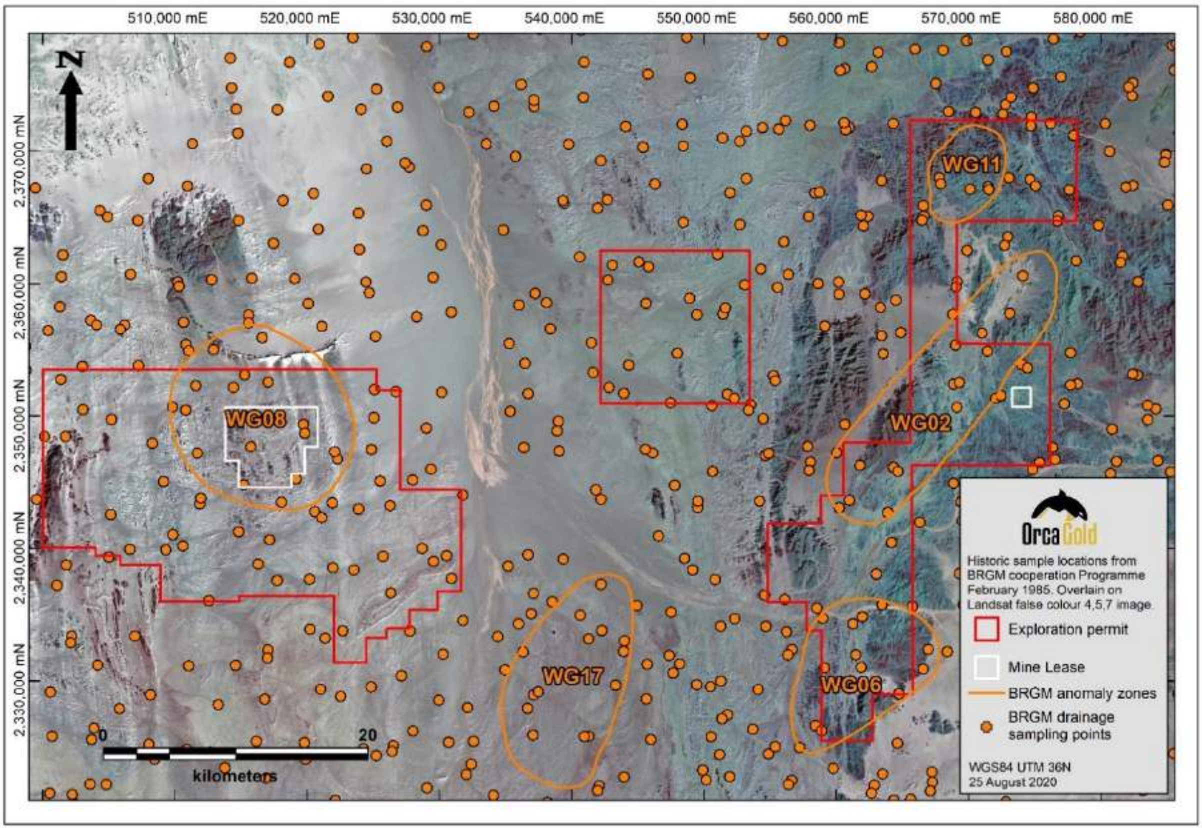
In January 1996, an agreement was signed between the Government of the Republic of Sudan and Cominor to explore for gold in the Nubian Desert and in 1997 the BRGM affiliate La Source acquired 90% of Cominor's 90% in the Nubian Desert Gold Project. In 1996, the Nubian Desert Gold Study initiated grass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

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

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

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

Figure 6.1 BRGM Nubian Gold Project Drainage Anomalies and Sampling Points



Source: Orca Gold.

Gold exploration in the Sudan has attracted significant international interest over recent times, encouraged by the rapid rise in artisanal gold mining in the Nubian Desert and Red Sea Hills over the last four years. Artisanal mining is concentrated in Block 14 and in the Block 15 EPL adjacent to the south (held by Managem). The Government estimates that more than 1,000,000 people are now involved in artisanal mining in the Sudan. Reported gold production was 103 t in 2017, with plans to increase production to 110 t in 2018, making Sudan the second largest gold producer in Africa (Arab News, 28 Dec 2017).

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

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 January 2012. 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. REGIONAL GEOLOGY

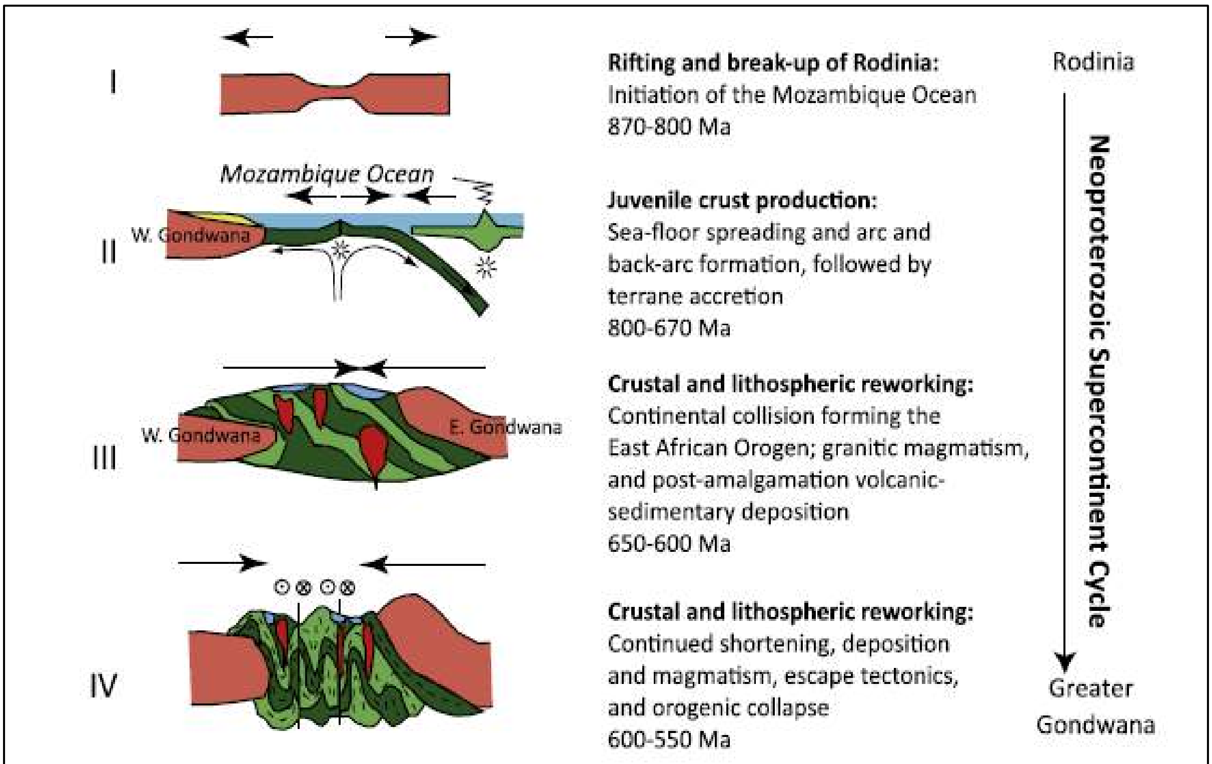
7.1 Regional Geological Setting of the Nubian Desert

7.1.1 Tectonic History of the Arabian Nubian Desert

The Arabian-Nubian Shield (ANS) is an assemblage of Neoproterozoic accreted intra-oceanic island arc and back-arc basin complexes that formed during the rifting and break-up of the Rodinia supercontinent and the creation of the Mozambique ocean between 870 and 670 Ma (Ahmed et al, 2000, Johnson et al, 2013) (Figure 7.1).

Closure of the Mozambique ocean resulted in the accretion of multiple island arcs terranes which formed the Arabian Nubian Shield, which later collided with the Saharan Craton during the East African Orogeny and the formation of Gondwana (650 - 542 Ma).

Figure 7.1 Schematic Diagram Leading to Formation of the ANS



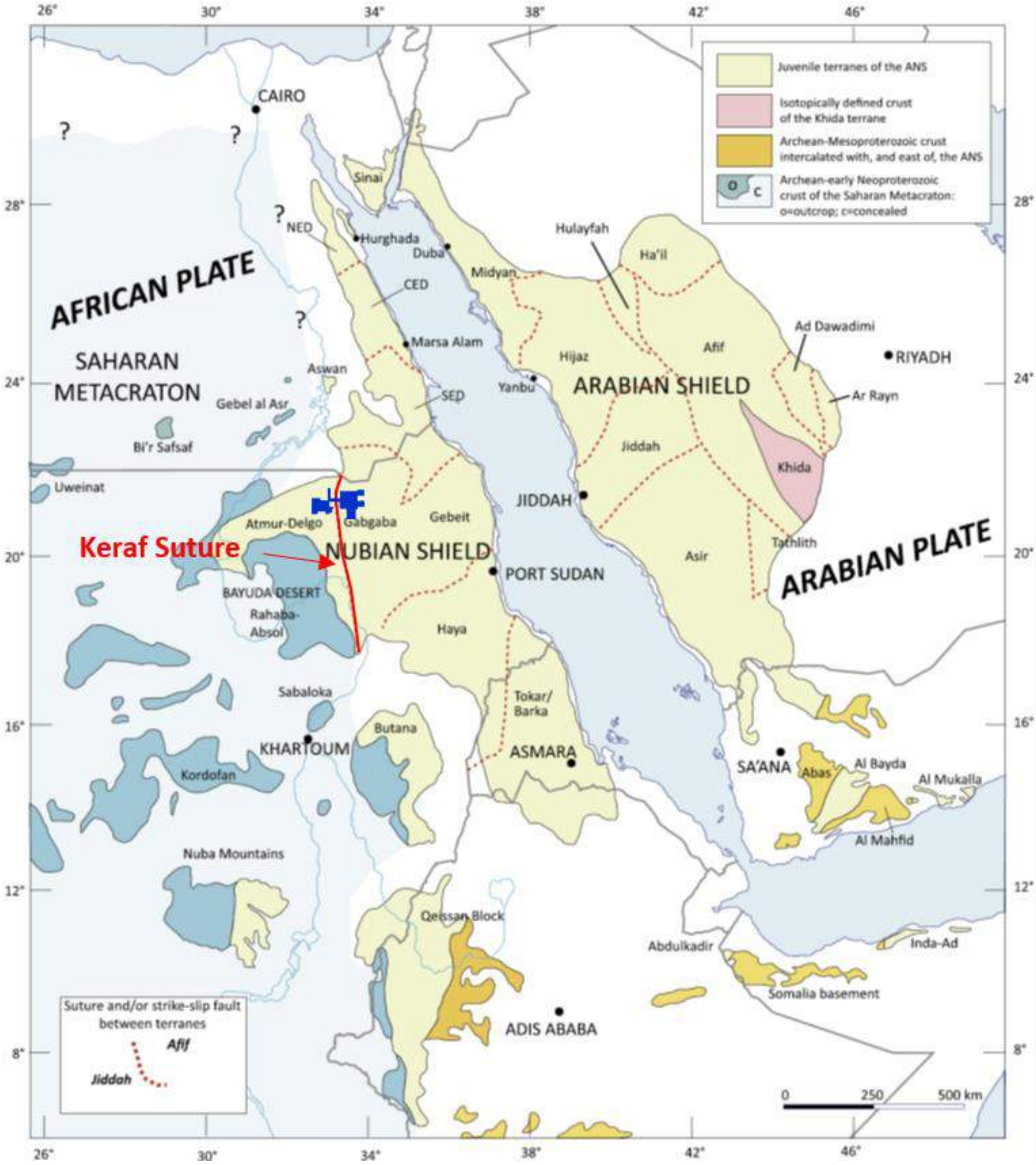
Source: Johnson et al, 2013

7.1.2 Geology of the Arabian Nubian Shield and Saharan Meta-craton in Northern Sudan

In Northern Sudan, the Precambrian geology is dominated by two significant crustal domains separated by the N-S trending Keraf Suture (Figure 7.2). To the west of this domain boundary is the Sahara Craton (which refers to the Nile / Sahara Craton and the African Craton combined), whilst to the east of the Keraf Suture lie the accreted terrains of the Arabian Nubian Shield.

The Sahara Craton is geologically distinct from the Arabian Nubian Shield. It is Paleo-Proterozoic in age and is described in the literature as metamorphosed to upper amphibolite facies and therefore characterised by high grade gneisses, high grade supra crustal sediments and ophiolitic melanges (Johnson et al, 2013, Ahmed et al 2000).

Figure 7.2 Present Day Location of Arabian-Nubian Shield



Source: Johnson, 2013

The ANS domain in Sudan is composed of three Neoproterozoic juvenile volcanic terranes that formed in an island arc / back-arc basin setting and were subsequently metamorphosed to greenschist facies during accretion / East African Orogeny (Figure 7.3). These terranes are referred to as:

1. The Barka – Nafka Terrane (870 – 840 Ma).
2. The Haya Terrane (870 - 790 Ma).
3. The Gebeit / Gabgaba Terrane (830 - 720 Ma).

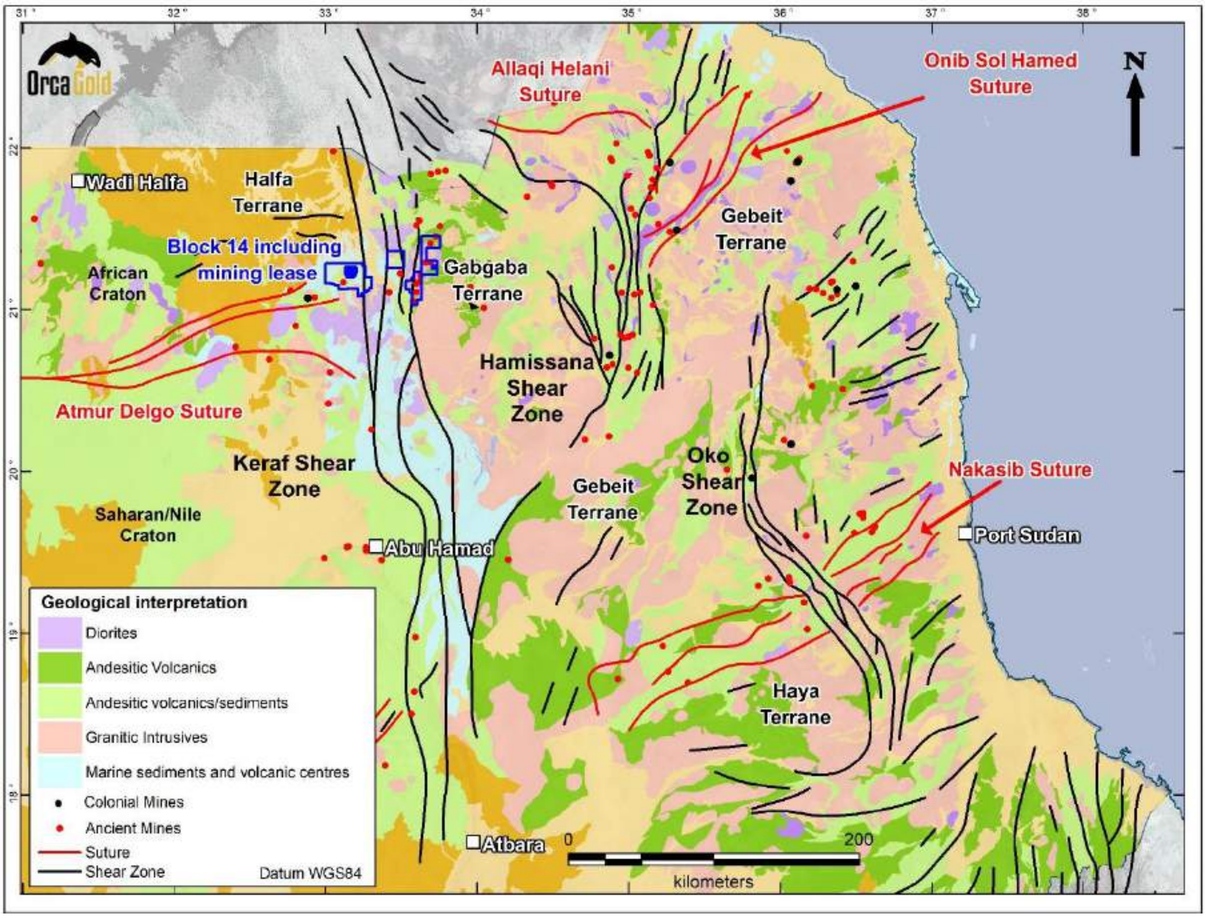
The earliest major tectonic features of the Arabian Nubian Shield in Sudan are the arc-arc accretionary sutures that formed during the collision and accretion of these terrains and now form the boundaries between them:

1. Nakasib.
2. Sol Hamid.

3. Barka.
4. Atmur Delgo.

The Atmur Delgo Suture is a greenschist facies Neoproterozic oceanic / island arc formed during the collision of the Palaeoproterozoic Bayuda and Halfa terranes during the accretion of the ANS to the Sahara Craton.

Figure 7.3 Regional Schematic Geology



Source: Orca Gold Inc.

7.1.3 The Keraf Suture

During the East Africa Orogeny (650 – 600 Ma, terminal collision 580 Ma), the largely formed Arabian Nubian Shield collided with the Sahara Craton along a N-S trending suture zone, extending from Southern Kenya to Southern Egypt, (Johnson et al, 2013, Abdel Rahman et al., 1993, Burke and Sengor 1986).

In Sudan, the exposure of this suture zone is known as the Keraf Suture; a 500 km long, N-S trending, arc-continental suture with a complex, polyphase deformation history. It is a fault bound wedge of metamorphosed marine sediments varying in width from 30 km in the south (Berber area) to 80 km in the north.

Lithologies in the Keraf comprise siliciclastic and carbonate-rich meta-sediments which intercalate with sills of meta-basalt and micro-diorite and post-tectonic granitoids and diorites.

Sedimentary structures and textures (cross and graded bedding and oolitic textures) in the low grade meta-sediments indicate deposition in a shallow-passive marine environment.

Geochemistry of the post-tectonic intrusions within the Keraf Suture indicates that the older plutons are calc-alkaline, medium-K, meta-luminous and collisional related I-type diorite and granodiorite.

They have trace element signature of subduction zone emplacement. The younger plutons are high-K and post-collisional A-type granites. REE patterns favour derivation of the two types of plutons from a common magmatic source (Ahmed, 2000).

Ahmed et Al (2013) suggests a four-phased (D1-D4) transpressive deformation for the Keraf which is characterised by N-S sinistral faulting (simple shear). Shearing was associated with E-W horizontal compression (crustal shortening) and vertical extension (crustal thickening). The tectonic history of the shear zone manifests a scenario of ocean basin closure (the Mozambique Ocean) followed by collision of the Arabian Nubian Shield with the Sahara Craton.

7.1.4 Magmatism in the ANS

Throughout the accretionary period, significant magmatism occurred and abundant, plutons and batholiths, granitice to dioritice in composition, intrude the Arabian Nubian Shield in Sudan.

7.1.5 Post East African Orogeny Geological Events

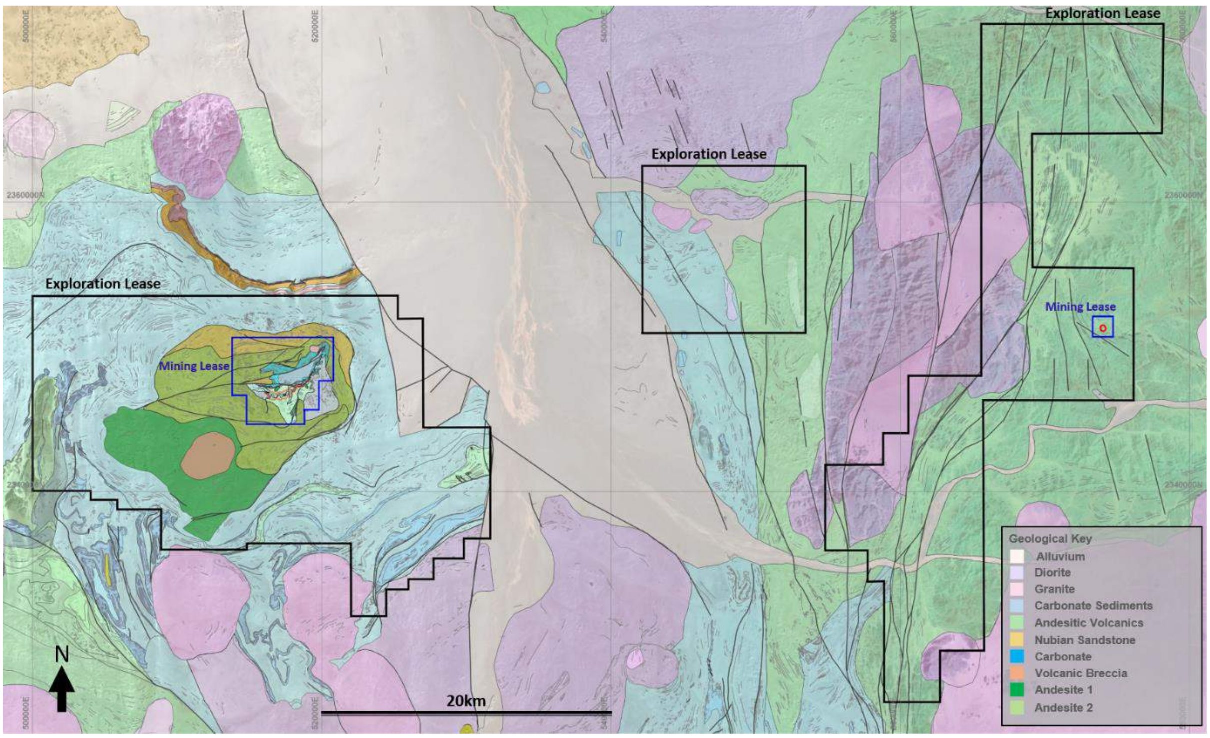
During the lower Palaeozoic / Cretaceous period, the Nubian Sandstone formation was unconformably deposited onto portions of the Arabian Nubian Shield. This formation is of particular hydrogeological significance and forms the host rocks for the Area 5 aquifer that will be used during production at the Galat Sufar South Deposit, Block 14.

Tertiary rifting and volcanism associated with the opening of the Red Sea is the most recent tectonic activity and is expressed as broadly N-S, Red Sea parallel, extensional faults, grabens, half grabens, mafic dyke intrusion and localised volcanism.

7.2 Block 14 Project Scale Geology

The geological framework of Block 14 is dominated by two distinct geological domains - the andesite dominated Gabgaba terrain of the ANS to the east and the marine sediments of the Keraf Suture in Western Gabgaba; these are separated by the Eastern Gabgaba Fault System (EGFS) (Figure 7.4).

Figure 7.4 Block 14 Geology



Source: Orca Gold Inc.

The EGFS is a major polyphase brittle-ductile fault system which divides the two terrains and exhibits reverse faulting during the collisional phase. A sinistral strike slip displacement is evident along the EGFS. Related faulting (splays) are seen to the east within the Gabgaba Terrane with similar displacement.

The central part of the licence is dominated by the Northern Gabgaba Graben a downthrown portion of the Keraf that has been infilled by clastic sediments.

Western Gabgaba and Galat Sufar South are located at the junction between two main structural belts: the Keraf Suture (trending N-S) and the Atmur Delgo belt (trending E-W).

The structural geology of Western Gabgaba is dominated by complex interference folding within the sedimentary rocks of the Keraf Suture.

Importantly, these interference folds are unique to the junction between the Keraf Belt and the Atmur Delgo Belt and have not been observed in other locations along the Keraf. Furthermore, when examined on satellite imagery, the complexly folding limestone ridges from the Keraf sediments can be traced into the Atmur Delgo suture; there is no hard structural boundary between the two belts which may suggest that both the Atmur Delgo belt and Keraf belt were experiencing ocean closure and shortening simultaneously.

Generally speaking, the geological framework of the Gabgaba terrane is relatively simple; the andesitic sequence is un-foliated and preserves its original mineralogy and textures and does not display evidence of folding. Where recognised, the primary layering in the volcano-sedimentary rocks is east-west. The Western margin of the Gabgaba Terrane is bound by the brittle-ductile EGFS which cuts and displaces late tectonic granites with a sinistral sense of movement.

The Keraf sediments are a thick sequence of folded / thrusted marine sediments dominated by pelites, marls and limestone units, with localised occurrences of coarser grained siliciclastic sediments. This is a significant geological formation in the region occurring along the length of the Keraf Suture.

Metamorphic grade is in the lower greenschist facies but the lithologies preserve their original mineralogy and textures. Sedimentary structures such as grading, trough cross bedding and truncations are rare and therefore, younging directions are therefore largely unknown (Davies and McDonnell, 2017). The dominant characteristic of the Keraf Sediments is the extensive and complex polyphase interference folding.

The Galat Sufar Andesite domain is an anomalous exposure of andesitic volcanic rocks within the thick sequence of Keraf sediments. It is interpreted as a doubly plunging antiformal fold created by complex inference folding which exposes the andesitic volcanic-sedimentary underlying the shallow marine sediments (the exact relationship between the andesites and sediments is unclear and is potentially structural). It is within this andesite domain that the Galat Sufar South deposit is located.

7.3 Geology of the Galat Sufar South Deposit

7.3.1 Introduction

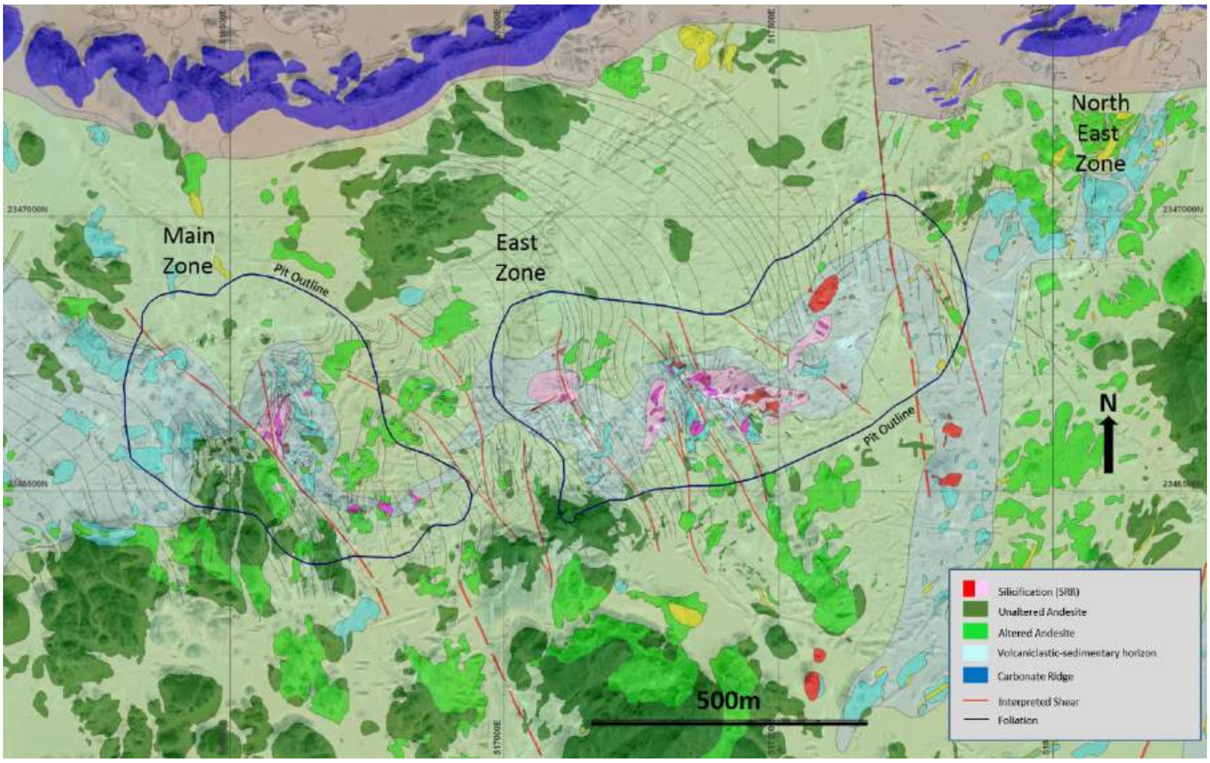
The GSS deposit is located in the central portion of the Galat Sufar Andesite Domain, an antiformal dome complex. Within the andesitic dome is a core of marine sedimentary rocks (limestones, marls and pelites), which are thought to be a remnant of the Keraf sediments that have been structurally interleaved and folded with the andesites.

The GSS deposit is located just south of the contact between marine sediments to the north, marked by a distinct carbonate dominated ridge line and an andesitic volcanic sequence to the south. The andesitic sequence is heterogeneous comprising lava flows, pyroclastic deposits and primary volcanic breccias.

Of particular importance to deposit formation, the andesite sequence contains a discrete, 80 – 200 m wide, volcaniclastic-sedimentary horizon which contains dioritic sills / dykes (shown in blue in Figure 7.5). Mineralisation and alteration are concentrated in this unit which is bordered to the north and south by increasingly unaltered andesitic flows and further volcaniclastics which display distinct and different geochemical signatures.

The host unit has been sequentially and intensely altered by the addition of albite, sericite, silica and lastly carbonate. Alteration grades from largely unaltered andesitic lavas and volcaniclastic host rocks to strongly altered and foliated silica – sericite schists in which the protolith cannot be identified.

Figure 7.5 Geology of the Galat Sufar South Deposit Area



Source: Orca Gold Inc.

Within the GSS deposit area, recognition of bedding (S0) is difficult due to the massive nature of andesite host rocks and strong alteration overprint but is oriented roughly ENE-WSW. Mineralisation is thought to have formed within the volcaniclastic / sedimentary horizon parallel to S0.

The dominant foliation at the prospect scale (S1) is pervasively developed throughout the GSS deposit area. It is sub-vertical and strikes towards the NW (330˚ - 340˚) at moderate to high angles to the orientation of the mineralised unit.

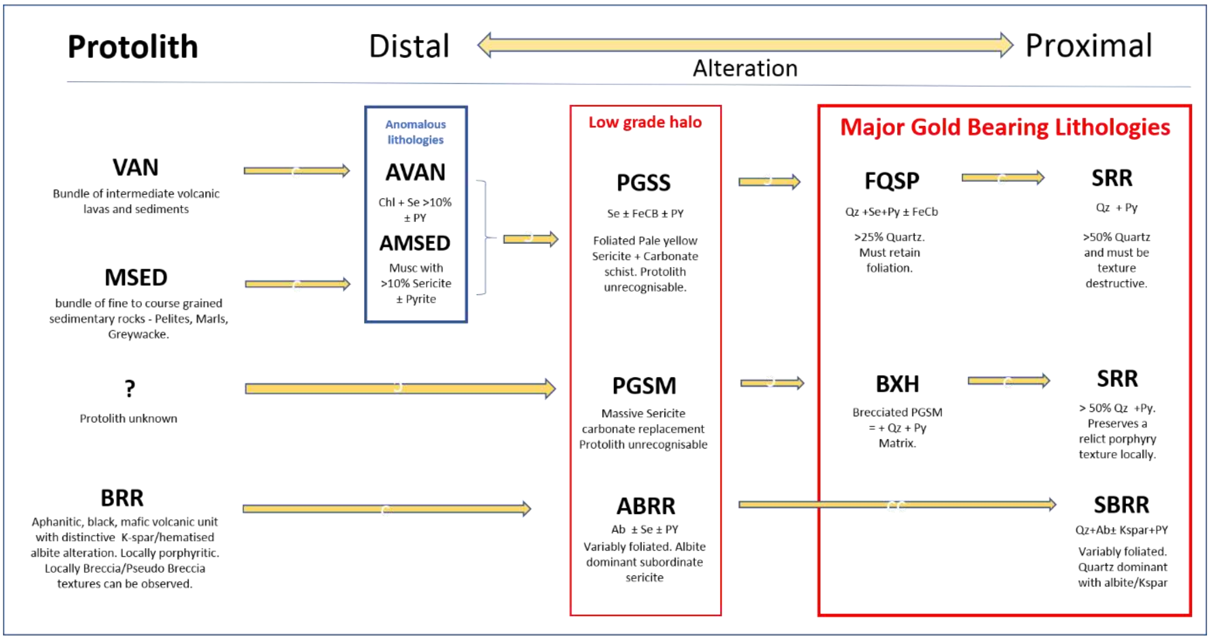
Across the GSS deposit the mapped foliation (S1) is parallel to the axial plane of these folds indicating that they formed during the same deformation event – D1 and hence the mapped foliation is S1 which overprints the deposit. The folding is open within the East Zone of the deposit but tighter folding is seen in the Main and 050 sections of GSS.

Asymmetric fabrics and strain markers indicative of simple shear are not observed at GSS suggesting that the ore forming event has not involved displacement across a shear zone (Mason, October 2017). The strain environment was characterised by pure shear or flattening and the distribution of alteration and mineralisation is lithologically controlled as opposed to a classic "shear zone" (Davies, 2016).

7.3.2 Deposit Scale Geological Units at Galat Sufar South

The classification system for the deposit scale geological units at Galat Sufar South is illustrated in Figure 7.6 and lithological descriptions are included in Table 7.1. In the more distal alteration facies, where primary mineralogy and texture can be observed, the units are classified according to their primary geological characteristics. Proximal alteration is pervasive and primary mineralogy and textures have been almost completely destroyed and therefore most units are defined by their alteration assemblages.

Figure 7.6 Deposit Scale Lithological Units



Source: Orca Gold Inc.

Due to the pervasive and texture destructive character of proximal alteration, geological units are defined by their alteration characteristic in the proximal facies and their primary mineralogy and texture in the distal alteration facies.

Table 7.1 Lithological Codes and Descriptions for the GSS Deposit, Block 14

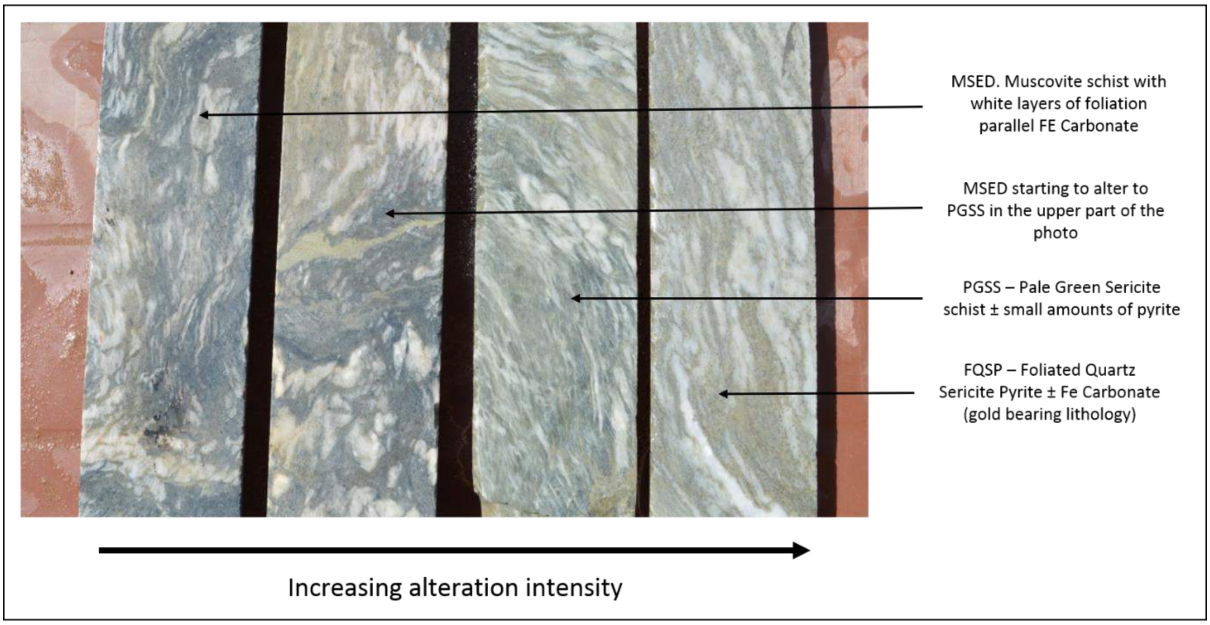
Assemblage	Logging Code	Description
Host Rocks	VAN	Andesite: Intermediate volcanics comprising chlorite altered lavas, volcanic sediments and breccias.
	MSED	Volcanic sediment: Fine grained muscovite schist interpreted as being a volcaniclastic lithology. Clastic matrix common (varied grain size).
	BRR	Vetric (?), massive, phyric basaltic andesite. Commonly subjected to early hematised albite, giving characteristic black red colouration.
Distal	AVAN AMSED	Sericite altered versions of their respective protoliths, where the sericite is not mineralogically destructive and the protolith can be identified.
Intermediate	PGSS	Variably altered, fine grained, foliated, schist. Protolith not identifiable. Green and mustard yellow sericite with Fe carbonate streaks / lamina. Can form from MSED, VAN, or BRR. Alteration varies from low to intense (close to FQSP).

Assemblage	Logging Code	Description
Proximal	PGSM	Extremely fine grained, massive, pale green altered rock with occasional relict phenocrysts. Early massive albite flooding in places overprinted by sericite and Fe carbonate (anomalous to low grade). Protolith unknown. Brittle deformation common with multiple stages of fracturing and brecciation which is veined / cemented with silica (moderate to high greade – transition to BXH).
	ABRR	Albite / Kspar and Sericite altered BRR. Often contains fine sulphide disseminations. Weakly foliated and locally massive.
	FQSP	Major gold bearing lithology. Fine grained, strongly foliated, intensely altered schist. Alteration comprises early albite, sericite / silica and late Fe carbonate. Typically 5 – 10% pyrite occurs as fine dissemination and clusters.
	SRR	Major gold bearing lithology. Massive, smokey grey silicification with 5 – 10% disseminated sulphide. Occasional relict porphyry texture. Seen in distinct bodies East Zone. Interpreted as dioritic sill(s) that have undergone brittle fracturing and intense pervasive silicifcation.
	BXH	Gold bearing lithology. Advanced brecciation and alteration by silica of PGSM with 5 – 10% disseminated pyrite.
	SBRR	Silifictaed BRR. Composed of >50% quartz and visible remnants of albite / Kspar preserved. No foliation.
Unknown	BXK	Clasts of K-feldspar / albite altered material within a dull siliceous matrix containing significant pyrite. Anomalous to low grade. Irregular plug type body within fold closure in west of East Zone.

Mineralisation at GSS is associated with an intensely altered and silicified core represented by the SRR, BXH, SBRR and FQSP lithologies. Outwards of the silicified core is a broad, laterally extensive low grade alteration halo dominated by PGSS. MSED occurs throughout the PGSS alteration halo as scattered and discontinuous pockets and in close association with the PGSS, which suggests the PGSS is dominantly a hydrothermal overprint of MSED. Due to its patchy, discontinuous nature MSED is not mapped or modelled as a coherent unit, but documenting its occurrence has been key in developing the understanding the deposit scale geology.

The alteration transition from VAN / MSED to PGSS to FQSP with the associated increase in alteration intensity is frequently observed in core (Figure 7.7).

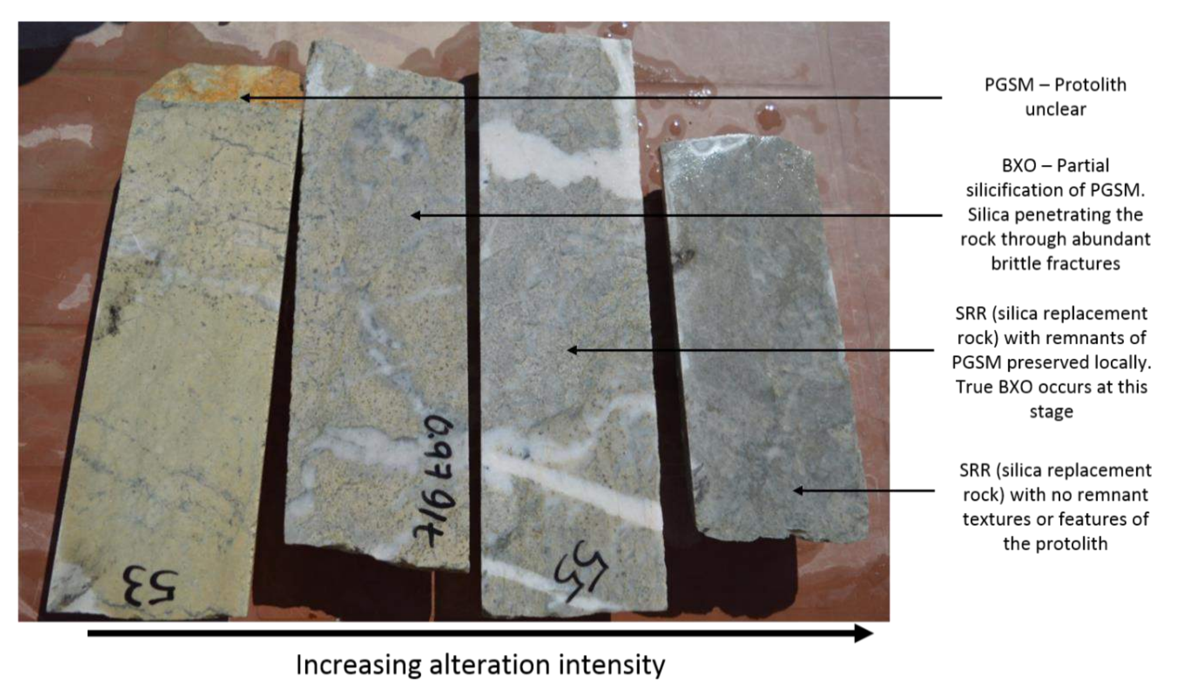
Figure 7.7 Alteration Progression of MSED Lithology to FQSP



Source: Orca Gold Inc.

The protolith of PGSM is unknown due to the massive albite flooding, although relict plagioclase phenocrysts are often seen. Progressive alteration from PGSM, through BXH, to SRR is readily observed in the Far East Zone, with relict phenocrysts preserved across the transition (Figure 7.8).

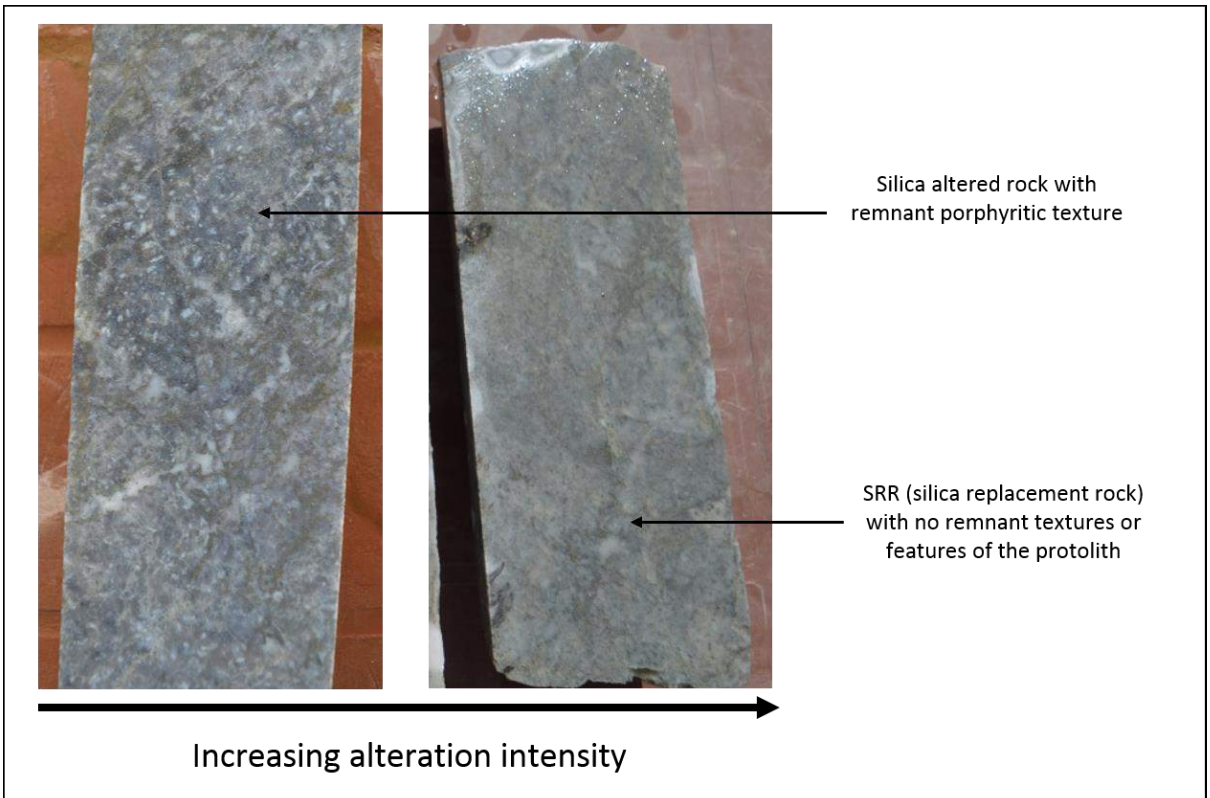
Figure 7.8 Progressive Alteration Overprint of the SRR Unit



Source: Orca Gold Inc.

Within the East Zone at GSS, two areas show the presence of what is believed to be one or more dioritic sills within the volcaniclastic-sedimentary horizon. These rocks have deformed in a brittle manner and been flooded and largely replaced by silica and pyrite (Figure 7.9).

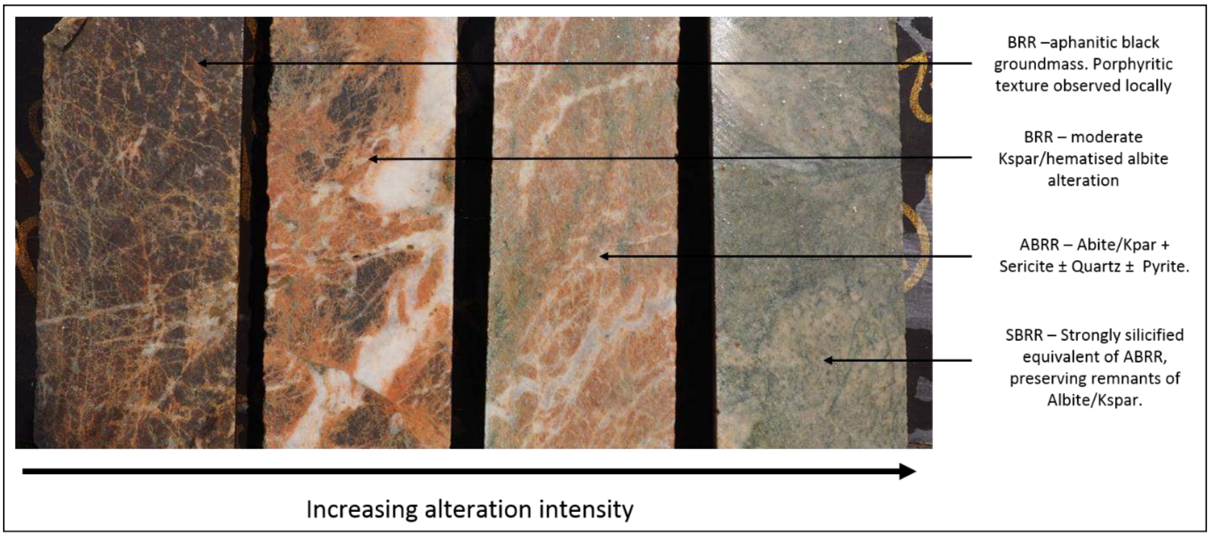
Figure 7.9 Alteration Overprint of an Interpreted Porphyrytic Diorite



Source: Orca Gold Inc.

Figure 7.10 shows the alteration progression of the k-feldspar / hematised albite bearing BRR host rock.

Figure 7.10 Alteration Sequence Showing Progressive Alteration to SBRR



Source: Orca Gold Inc.

BXK is the exception within GSS, intense silicification and pyrite mineralisation do not correlate with gold grades. Importantly, significant gold mineralisation occurs at the contacts of the BXK unit, indicating that it predates the main gold event and the gold precipitated at its margins due to the rheological contrast between the silicified unit and adjacent sericite schists. The BXK silicification event is interpreted to have occurred early in the alteration paragenetic sequence.

7.3.3 Sulphides and Gold Deportment

Pyrite is by far the most dominant sulphide with chalcopyrite, sphalerite, galena and tennantite / tetrahedrite occasionally seen in core and confirmed in petrological investigation.

Gold is fine grained, typically <40 µm. 95% of the gold occurs as free gold with the remainder as Petzite (Ag_3AuTe_3). The gold contains ± 20% silver.

7.3.4 Alteration Paragenesis: Distribution of Alteration and Vein Foliation Relationships

Alteration and mineralisation is synchronous with the development of the S1 foliation, as alteration products (silicification, sericite alteration and quartz veins) are parallel to the S1 fabric. This is evidenced by vein / S1 relationships seen in core:

1. Sub-parallelism of veins and foliation (Figure 7.11).
2. Boudinage where veins follow the S1 foliation.
3. Folded veins with axial planar S1 foliation (Figure 7.12).
4. Transposition.
5. Overprinting of deformed veins by less deformed veins.

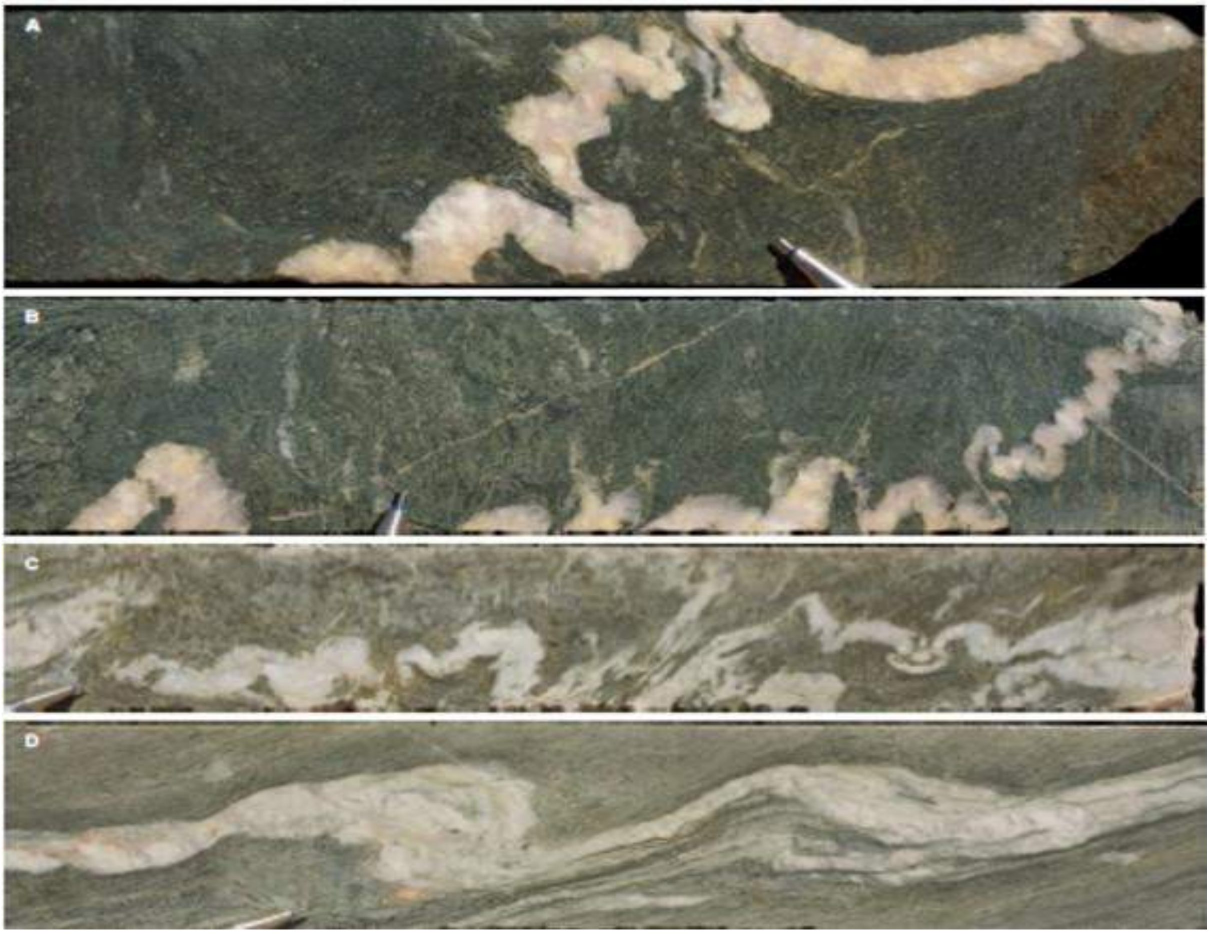
The overall trend of alteration and gold mineralisation however is at a high angle / perpendicular to the S1 fabric.

It is interpreted that rock failure, fluid emplacement and ultimately gold distribution is controlled by rheological contrasts within the primary stratigraphy and, additionally, the presence of intrusives within the MSED horizon as evidenced by remnant porphyritic textures in the Far East Zone SRR unit.

Figure 7.11 Examples of the FQSP Lithology

Source: Orca Gold Inc.

Figure 7.12 Example of Buckled and Folded Vein at GSS



Source: Davies 2016.

7.3.5 Galat Sufar South Structural Geology

Within the GSS deposit area, recognition of Bedding (S0) is difficult due to the massive nature of Andesite host rocks and pervasive alteration overprint (Figure 7.13). S0 measurements are widespread and tenuous and therefore geometric analysis of D1 structure is presented tentatively. However, measurements of recorded S0 shows a great circle distribution (Figure 7.14) to which the S1 fabric is axial planar so, although S0 is tenuously identified, the measurements still make sense when reviewed in context of S1 data, a more confidently identifiable structural feature (Mason, November 2017).

S1 foliation is pervasively developed throughout the GSS deposit area. It is sub-vertical and striking towards the NW (330˚ - 340˚). To the North of the GSS deposit area, the S1 foliation can be observed to swing / fold (see D2 section) to an ENE orientation where it is observed to be roughly axial planar to an F1 fold of S0 with axial plane striking ENE to EW.

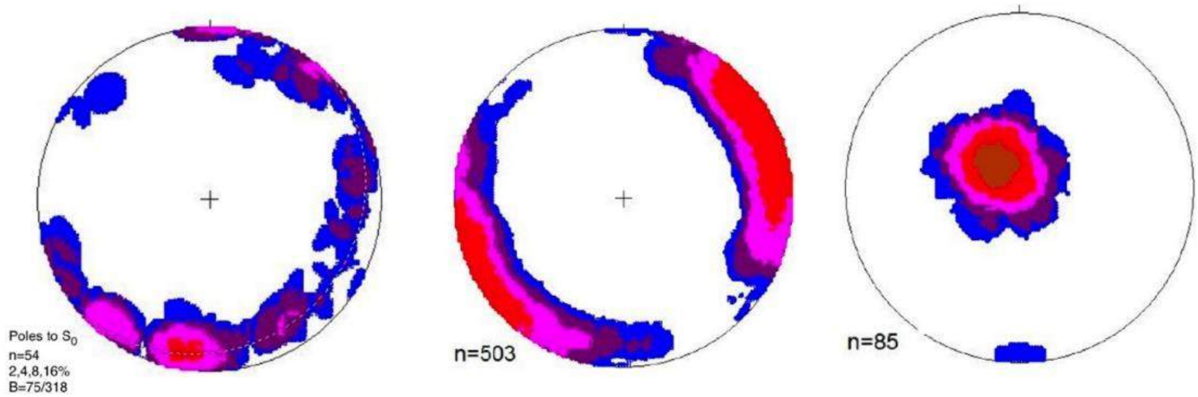
Discernible folds in the in the East zone of the GSS deposit are folds of the VAN–PGSS contact. Given that PGSS is interpreted to preferentially form after the muscovite schist (MSED), these are folds of the pseudo-bedding and thus F1 folds. These folds do not fold a foliation and therefore are not recognised as F2 folds. Across the GSS deposit the mapped foliation is parallel to the axial plane of these folds indicting that they formed during the same deformation event – D1 and hence the mapped foliation is S1.

S1 Foliation is striking NW throughout the GSS deposit but is folded into an E-W orientation to the north of the deposit area, by a macroscale D2 fold (Figure 7.15). Geological mapping scale 1:25,000 identified the same fold pattern in the distribution of the andesites and meta-sediments and airborne magnetic survey data displays the same fold pattern on a regional scale. Outcrop scale D2 folds were not identified in the mapped area and no significant, widespread, penetrative fabric development appears to have occurred during this event.

Figure 7.13 Examples of GSS Bedding (Left) and Foliation (Right)

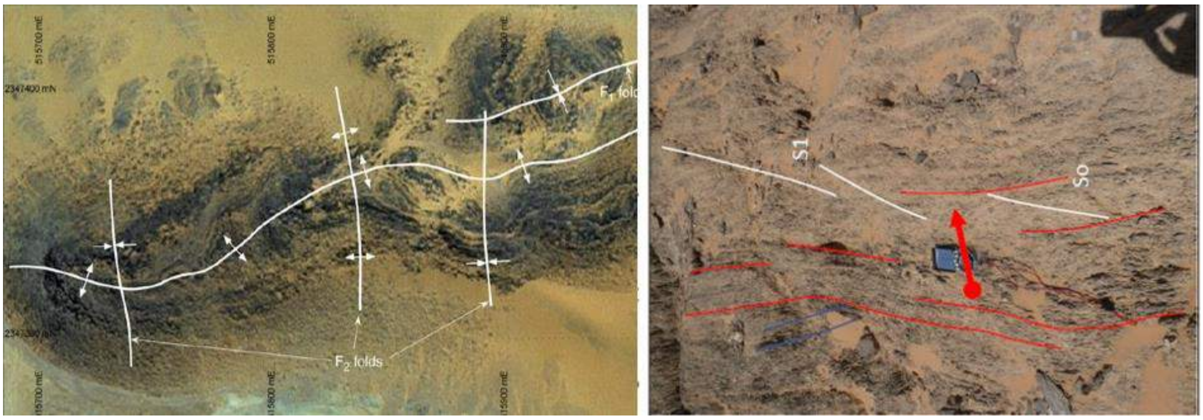
Source: Orca Gold Inc.

Figure 7.14 Stereonet F1 Fold (Left), S1 Foliation (Middle) and S1/S0 Intersection (Right)



Source: Mason, November 2017

Figure 7.15 F1 Fold Trend (Left) and S1 Foliation (Right)

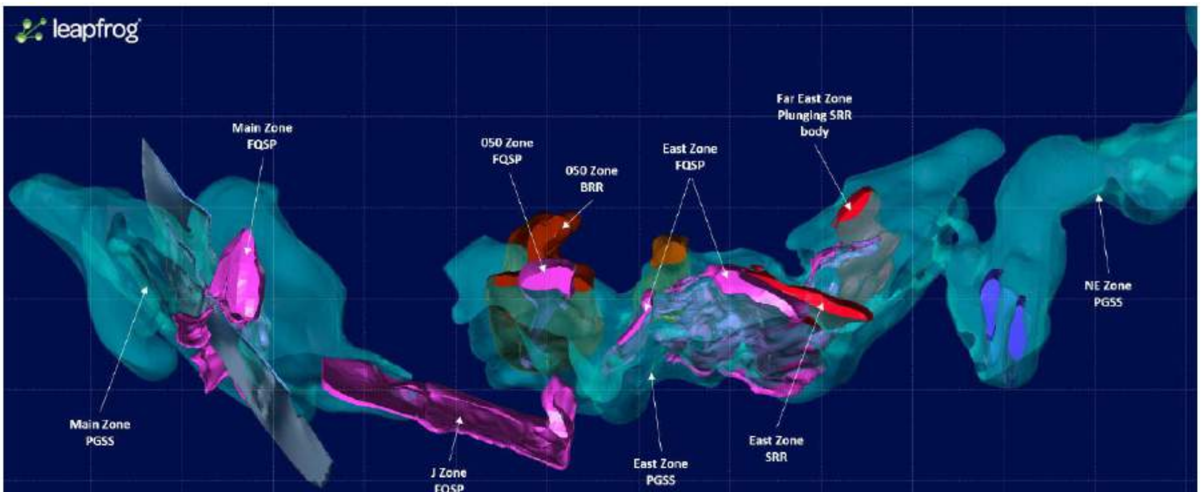


Source: (Left) Mason, November 2017 (Right) Seago, November 2017

7.3.6 Geological Model

Using the revised lithological definitions, a 3D geological model was built using Leapfrog Software (Figure 7.16). This model was then used as the basis for the assignment of densities in the Mineral Resource estimate and for the selection and classification of metallurgical samples.

Figure 7.16 Galat Sufar South 3D Lithological Model



Source: Orca Gold Inc

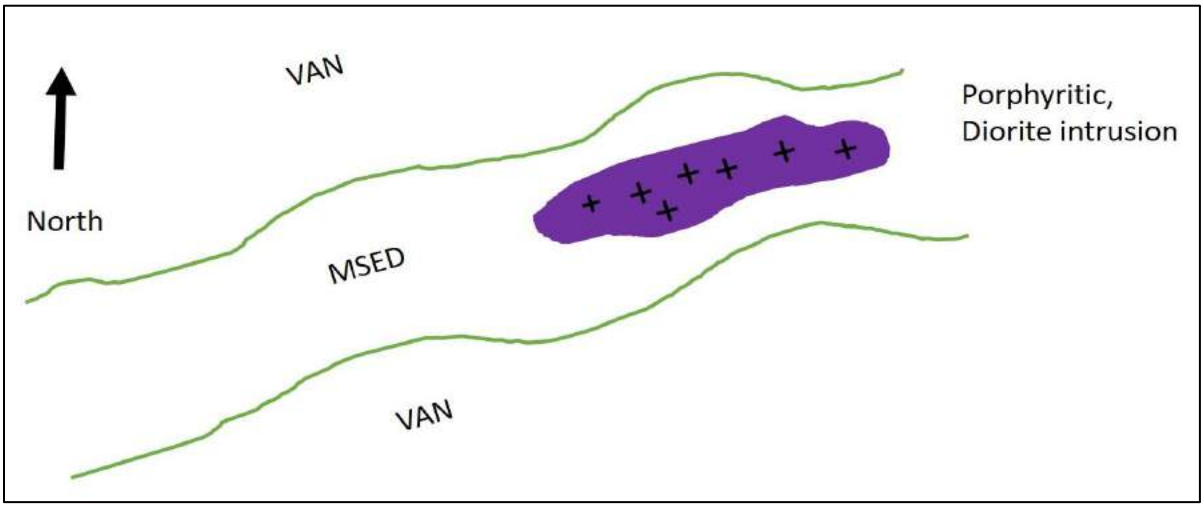
7.3.7 Summary

The current interpretation of setting of GSS is shown in Figure 7.17 a) to c) (for the East Zone of the deposit) and described below:

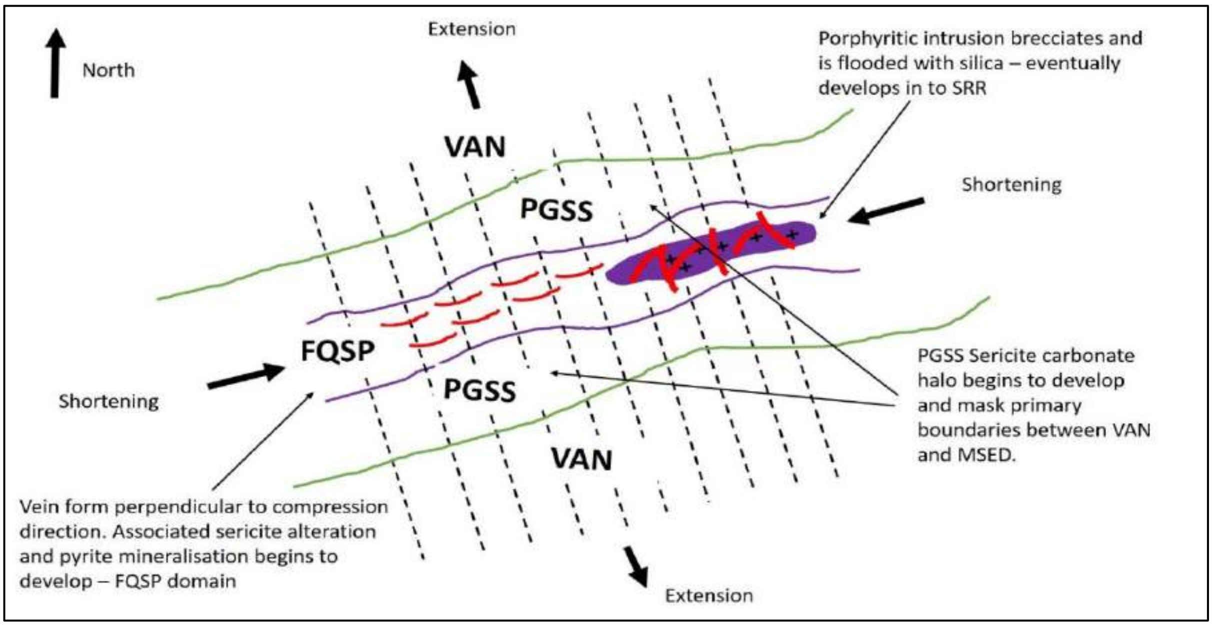
- The pre-mineralisation geological environment consisted of an ENE striking volcanclastic - horizon (MSED) containing porphyritic intrusions sitting between two units of andesitic volcanic rocks.
- Shortening on an ENE-WSW axis produced a strong NW trending S1 foliation.
- Early syn-tectonic emplacement of intense quartz veining with associated sericite and pyrite alteration focussed within a weak volcaniclastic / sedimentary horizon containing porphyritic subvolcanic intrusions.
- As deformation progresses, veins were rotated into parallelism with the S1 foliation which resulting in an ore body whose overall geometry is broadly perpendicular to NW strike of the foliation. The vein bearing horizon (FQSP) begins to fold and late stage shears develop and begin to disjoint the ore body locally.

Figure 7.17 Summary Schematic Illustrating the Deformation and Formation of the GSS Deposit

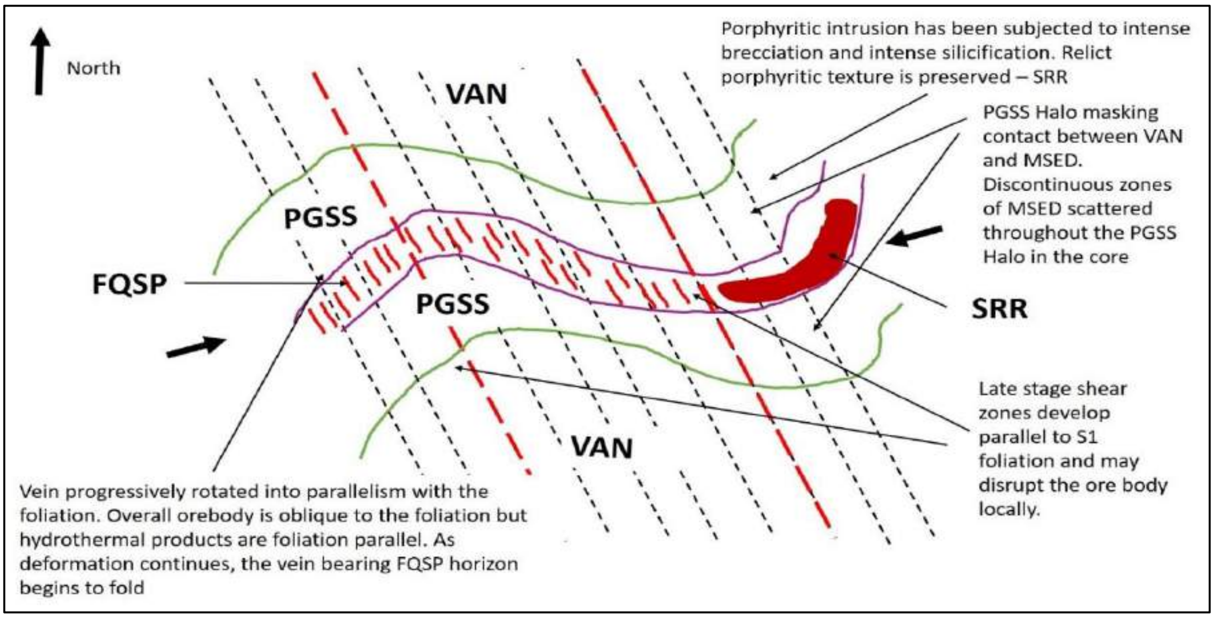
a) Pre-deformation Setting



b) Onset of ENE-WNW Shortening



c) Post Deformation



Source: Orca Gold Inc.

7.4 Geology of the Wadi Doum Deposit

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.

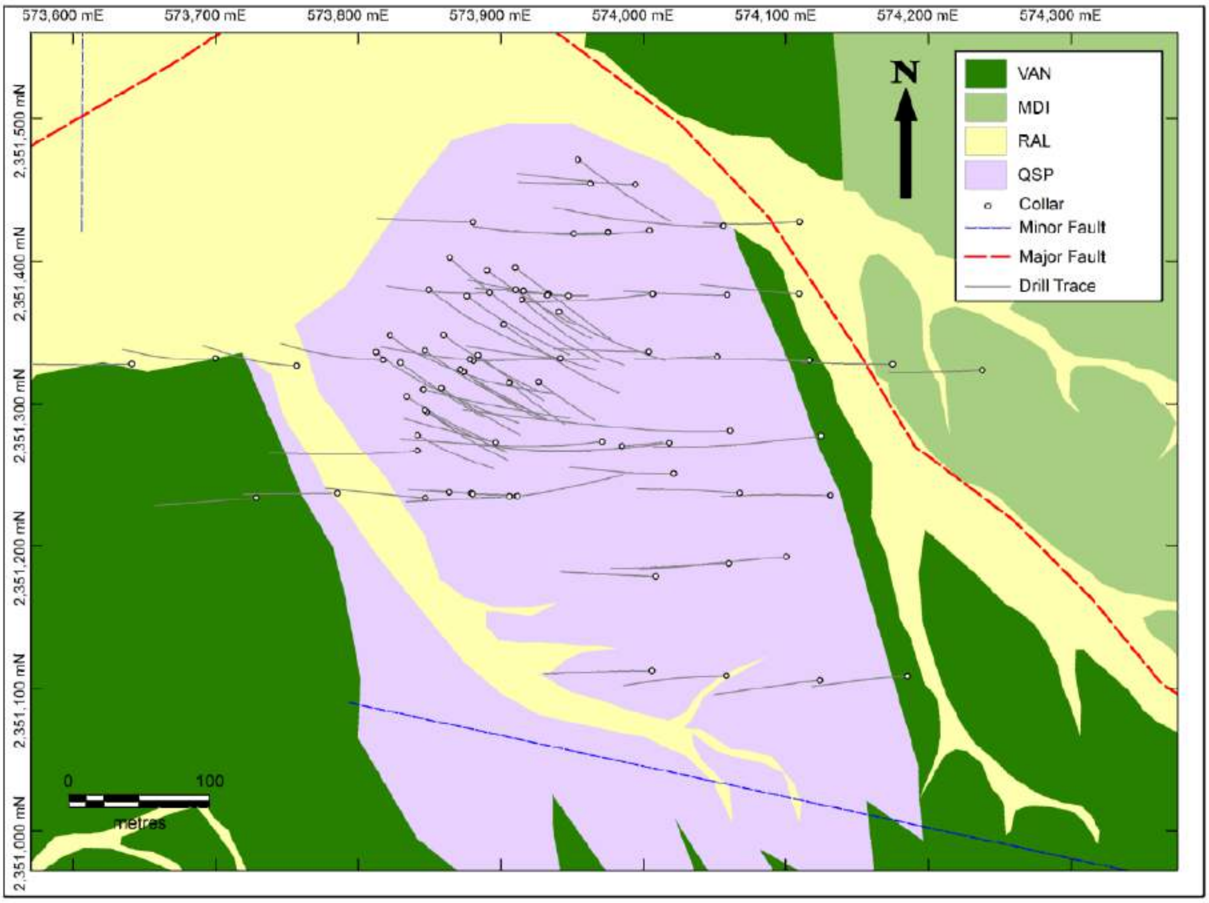
The high grade mineralisation is hosted within the volcanoclastic units which are confined by late felsic and mafic dykes. The mineralization is divided into a western volcaniclastic unit characterized by a dark colour caused by very fine grained sulphides (>10 - 15%), which contains some of the best 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.

Figure 7.18 Summary Geology of the Wadi Doum Deposit



Source: Orca Gold Inc.

8. MINERAL DEPOSIT TYPES

This Arabian Nubian Shield is known to be prospective for multi-commodity Volcanogenic Massive Sulphide (VMS) deposits, which formed during the opening of the Mozambique Ocean and mesothermal gold deposits, which developed during the East African Orogeny. Most recently, porphyry style mineralisation has been speculated and is thought to derive from felsic intrusions during peak or post peak metamorphism.

Examples of Arabian-Nubian hosted deposits include:

- Jabal Sayid, a copper rich VMS deposit in Saudi Arabia - 2017 reserves: 23.6 Mt at 2.10%. Source: Barrick Gold Corp website.
- Bisha Mine, a multi-commodity VMS deposit in Eritrea - 2016 Reserves: 9.58 Mt at 1.05% Cu, 6.16% Zn, 0.69 g/t Au and 45 g/t Ag. Source: Nevsun Resources Ltd website.
- Sukari Gold Mine, an mesothermal type gold deposit in Southern Egypt - 2015 Reserves: 239 Mt at 0.9 g/t Au (open pit) and 5.4 Mt at 4.5 g/t (underground). Source: Centamin PLC website.
- Jebel Oheir, a porphyry copper deposit in Sudan. Qatar Mining Sudan Ltd. 2018 Resource: Inferred Resource of 246.3Mt grading 0.44% copper and 0.08 g/t gold.

9. EXPLORATION

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

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

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

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

Table 9.1 Summary of Exploration Work Completed on Block 14, 2012 - 2018

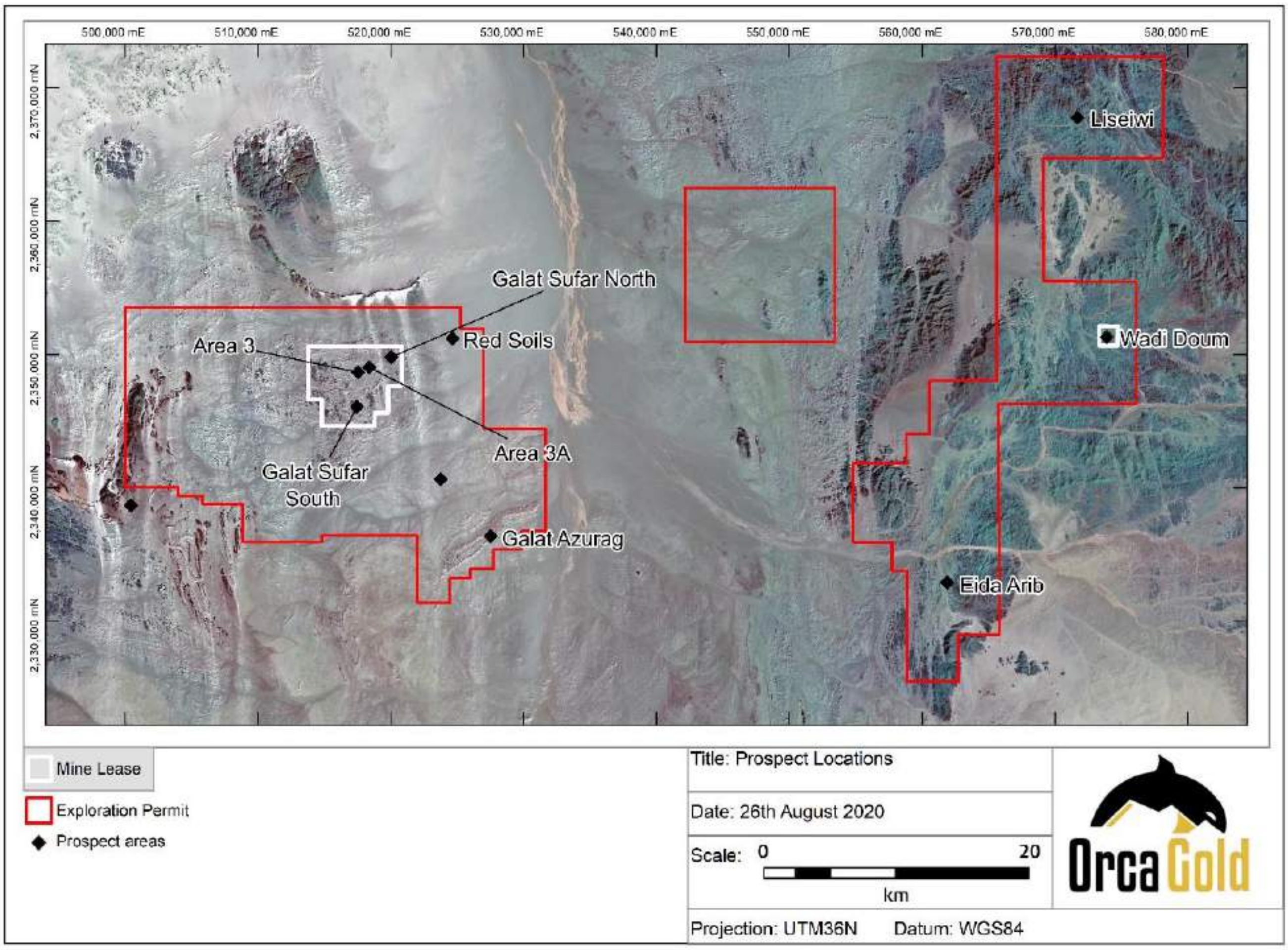
Surface Sampling	
Rock Chip samples	4,753
Soil Samples	2,682
Chip Channels (m)	58,427
Trenching (m)	40,954
Airborne Geophysics	
SKYTEM Electromagnetic survey (2017) (km^2)	2,415
VTEM Electromagnetic Survey (km^2)	835
Magnetic and radiometric Survey (km^2)	3,407
Satellite Imagery Acquired	
Landsat TM/Aster satellite Imagery (km^2)	7,046
SPOT Imagery	3,250
Worldview Imagery	1,095
Drilling	
Reverse circulation drilling (m)	101,397
Diamond core drilling (m)	25,704
Water Borehole Drilling (m)	6,721
Ground Geophysics	
Ground Magnetics (km^2)	7.01
Ground Radiometrics (km^2)	7.01
Time domain electromagnetic (TEM) profiling (km)	261.6

Exploration in Block 14 is currently focussed on geological mapping in the wider resource area and on target generation within the 1,000km^2 EPL.

10. DRILLING

Reverse Circulation (RC) and diamond core drilling in the Block 14 project area commenced in November 2012. During the period to June 2013, drilling was focussed at GSS however several other prospects were also drilled (Figure 10.1). From July to December 2013 all drilling was focussed at GSS. In 2014 several other targets were drilled including the discovery holes at WD with subsequent infill drilling. In 2015 drilling was undertaken at a number of prospects and included two diamond core holes at WD. Between 2016 and 2018, further resource drilling was carried out at GSS and Wadi Doum.

Figure 10.1 Drilled Prospect Areas



Source: Orca Gold Inc.

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

Table 10.1 Drilling Completed in the Block 14 Project Area

Prospect	Core Drilling		RC Drilling	
	Holes	Metres	Holes	Metres
GSS	100	24,584	591	77,088
WD	8	1,120	112	13,630
Galat Azurag	-	-	36	3,944
Mussieye	-	-	4	447
Area 3	-	-	6	750
Area 3A	-	-	5	640
GSS North	-	-	4	354
Red Soils	-	-	14	1,840
Target P	-	-	3	114
FE Ox 07	-	-	3	160
Liseiwi	-	-	18	2,055
EG2.2	-	-	9	375
Total	**108**	**25,704**	**804**	**101,397**

Figure 10.2 RC Drilling at East Zone, GSS

Source: Orca Gold Inc.

10.1 Drilling and Sampling Procedures

All drilling up to October 2017 has been undertaken by General Exploration Drilling (GED) using multipurpose KL400, KL500 and GED 850 rigs (Figure 10.1). In October 2017 Dal Mining Services commenced work with one Hanjin multi-purpose rig to which a second was added in early 2018.

10.1.1 Drilling and Sampling Method

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

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

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

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

10.1.2 Drill Sample Quality

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

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

Table 10.2 Samples with Poor RC Recoveries

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

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

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

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

Table 10.3 Wet Sample Statistics

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

Through the use of PQ sized core in the upper part of the hole, reducing to HQ in fresh rock and NQ in deeper holes (>4,500 m), core recovery for diamond drilling was generally good with an average of 95.6%.

Of a total 24,199 Diamond Drilling samples with recovery data, 630 (2.60%) were within runs with <80% recovery, 187 samples (0.77%) were within runs of <60% recovery. Of 7,561 Diamond Drilling samples with grades >0.5 g/t, 238 (3.15%) were within runs with <80% recovery, 66 (0.87%) were within runs of <60% recovery).

10.1.3 Drillhole Surveying

Collar Location Surveying

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

Figure 10.3 Drillhole Collar

Source: Orca Gold Inc.

Azimuth and Dip Surveying

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

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

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

10.1.4 Geological Logging

RC Drilling

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

Figure 10.4 Example of an RC Geological Log

RC Drilling Log Sheet

Hole ID:GSRC 181 Licence:B14 Project:MEYAS SAND Target: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%	VD %	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	MSG	G	W1	Fo	2	SE	1	Cl	3			Py	2									
92	93	16	100	D	L	MSG	G	W1	Fo	3	SE	1	Cl	3			Py	2									
93	94	20.5	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	3					20				
94	95	21.5	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	3					20				
95	96	24	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	3					20				
96	97	10.5	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	2									16/06/13
97	98	21	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	2									
98	99	22	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	2									
99	100	24	100	D	L	MSC	G	W1	Fo	3	SE	1	Cl	3			Py	2									

Source: Orca Gold Inc.

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

Figure 10.5 1 m Interval Sieved RC chip Sample

Source: Orca Gold Inc.

Diamond Drilling

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

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

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

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

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

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

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

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

10.1.5 Density Measurements

All core holes are systematically sampled for the purpose of density measurements, 88 holes from GSS and six holes from WD. A total of 1,855 samples were selected for density measurements based on lithological domains and oxidation state, with the aim of selecting a representative suite of samples for each lithology / oxidation domain.

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

The samples are dried in an oven for 24 hr (at 100˚C) and weighed. The dried samples are then waxed and weighed, before being submerged in water to record the volume of water displaced (hence density through the appropriate formula, applying a wax density of 0.9, to correct for the volume of wax). The bulk density is then calculated as: Bulk density = [Mass core] / [(Mass in air – Mass in water) – (Mass wax / 0.9)]. Table 10.4 below shows a summary of the density measurements made.

Table 10.4 Density Sampling

Lith. Domain	Oxcat	N	Min	Max	Average
MZ PGSS	Oxide	47	2.22	2.80	**2.56**
	Trans	90	2.04	3.04	**2.67**
	Fresh	296	2.44	3.14	**2.80**
MZ FQSP	Oxide	9	2.32	2.61	**2.46**
	Trans	8	2.36	2.85	**2.64**
	Fresh	100	2.52	3.06	**2.82**
MZ PGSS	Oxide	47	2.22	2.80	**2.56**
	Trans	90	2.04	3.04	**2.67**
	Fresh	296	2.44	3.14	**2.80**
MZ FQSP	Oxide	9	2.32	2.61	**2.46**
	Trans	8	2.36	2.85	**2.64**
	Fresh	10	2.52	3.06	**2.82**
ALL VAN	Oxide	33	2.19	2.79	**2.55**
	Trans	57	2.17	2.90	**2.64**
	Fresh	324	2.33	3.06	**2.78**
050 SBRR	Oxide	NA	-	-	**2.52***
	Trans	NA	-	-	**2.66***
	Fresh	19	2.47	2.98	**2.67**
EZ FQSP	Oxide	22	2.28	2.83	**2.50**
	Trans	13	2.47	2.83	**2.63**
	Fresh	90	2.43	3.09	**2.77**
EZ PGSS	Oxide	100	2.16	2.87	**2.52**
	Trans	72	2.13	2.98	**2.67**
	Fresh	436	2.29	3.09	**2.75**
EZ SBRR	Oxide	NA	-	-	**2.52***
	Trans	NA	-	-	**2.66***
	Fresh	27	2.34	2.86	**2.69**
EZ SRR	Oxide	9	2.30	2.50	**2.42**
	Trans	3	2.41	2.61	**2.54**
	Fresh	20	2.41	2.82	**2.69**
FEZ SRR	Oxide	3	2.38	2.44	**2.41**
	Trans	4	2.37	2.70	**2.55**
	Fresh	73	2.50	3.02	**2.79**

***Domain average used**

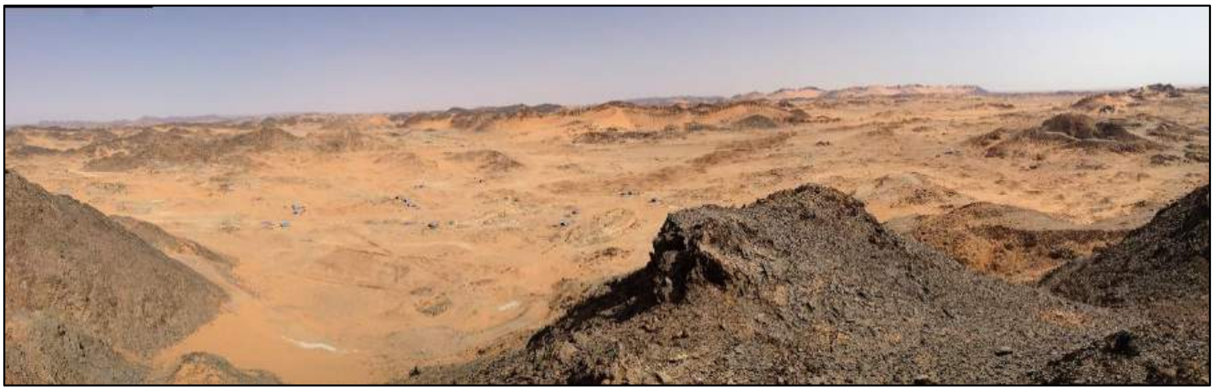
10.2 Drilling Completed and Significant Results

Since the discovery of GSS, it has been the focus of drilling completed on the project with 63,267 m completed (88% of project total) comprising 66,819 m of RC drilling in 520 holes and 21,531 m of core drilling in 88 holes. A further 10,312 m has been completed at WD (12% of project total), comprising 10,062 m of RC and 250 m of core drilling.

10.2.1 Galat Sufar South

Drilling and other exploration work at GSS (Figure 10.6) has identified a significant mineralized system and Mineral Resource over an area of 2 x 1 km (Figure 10.7). Mineralization is hosted within variably sheared intermediate intrusive and is associated with intense quartz-sericite-carbonate alteration and pyrite. Drilling at GSS has identified significant mineralization, with true widths in excess of 80 m intersected (Figure 10.8 and Figure 10.9).

Figure 10.6 View over GSS Resource Area



Source: Orca Gold Inc.

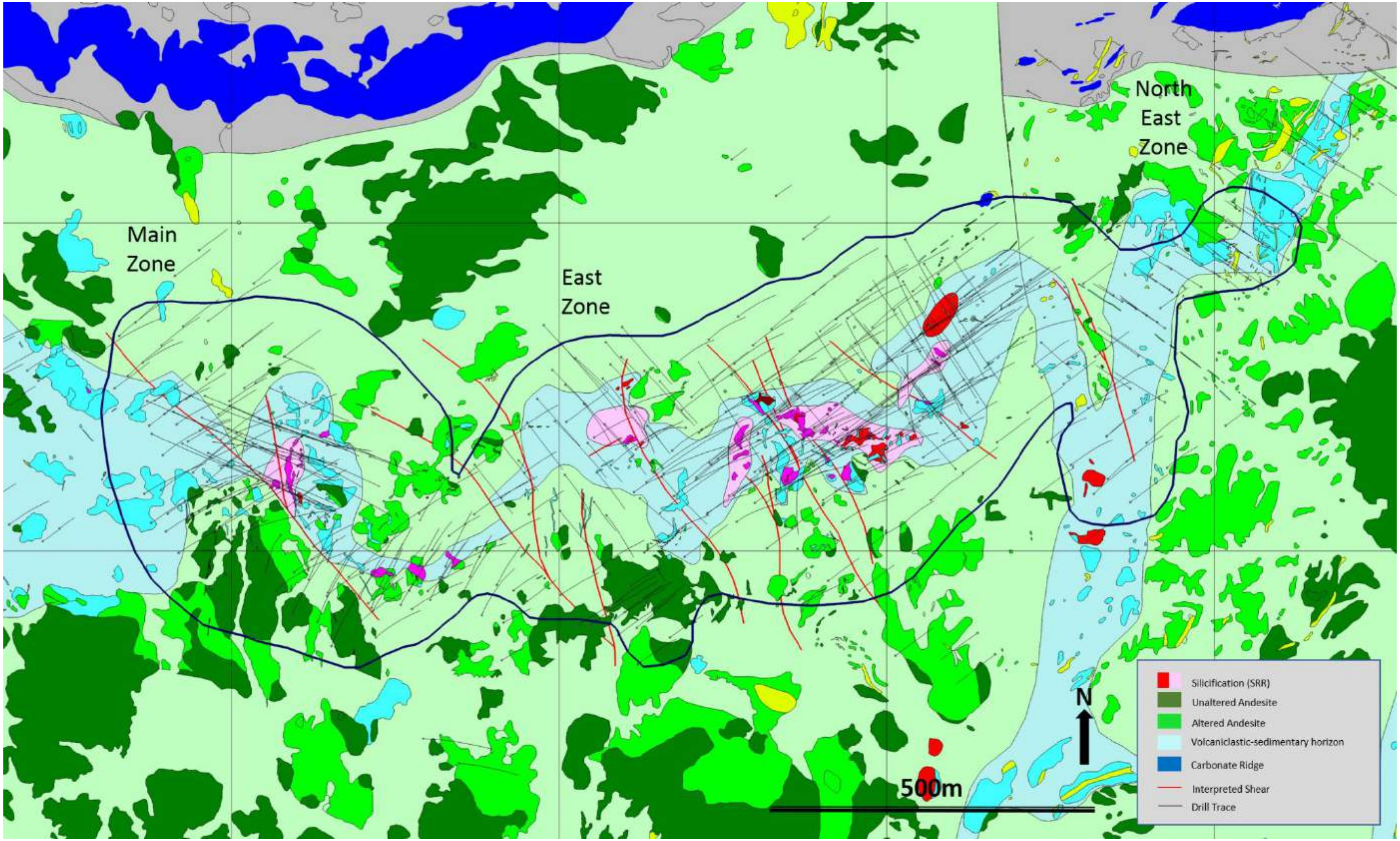
A selection of drill intercepts from the most recent drill programme is shown below in Table 10.5 below.

Table 10.5 Selected Intercepts from GSS

Hole	From	To	Metres	Au g/t Uncut	Ag
GSDD016	67.0	163.0	96.0	2.68	2.48
	245.6	266.0	20.4	3.80	0.67
GSDD016	214.0	276.0	62.0	3.06	3.44
	297.2	340.0	42.8	1.98	5.48
GSDD018	71.0	96.6	25.6	1.43	2.62
	276.2	303.0	26.8	1.15	3.02
GSDD019	29.6	41.0	11.4	1.41	1.01
	45.0	97.0	52.0	1.53	2.02
GSDD020A	35.3	56.0	20.7	2.01	1.94
	124.8	140.0	15.2	4.95	4.91
	171.0	214.0	43.0	1.26	1.37
	354.6	372.0	17.4	7.92	5.68
GSDD021	116.0	16.8	44.8	2.87	0.99
GSDD022	308.3	402.0	93.7	1.72	4.21
GSDD023	181.0	226.4	45.4	2.34	1.92
	234.5	265.0	30.5	1.42	1.44
	370.0	453.0	83.0	1.20	1.74
GSDD024	0.0	26.8	26.8	2.25	1.57
	302.7	375.0	72.3	1.67	2.28
	400.0	411.0	11.0	3.68	1.36
GSDD025	114.0	125.2	11.2	1.37	1.60
GSDD026	7.0	32.0	25.0	2.89	4.20
	47.0	74.8	27.8	1.62	1.92
	105.0	159.0	54.0	1.40	2.02
GSDD027	172.2	215.0	42.8	1.31	0.98
	218.0	270.0	52.0	1.43	2.97

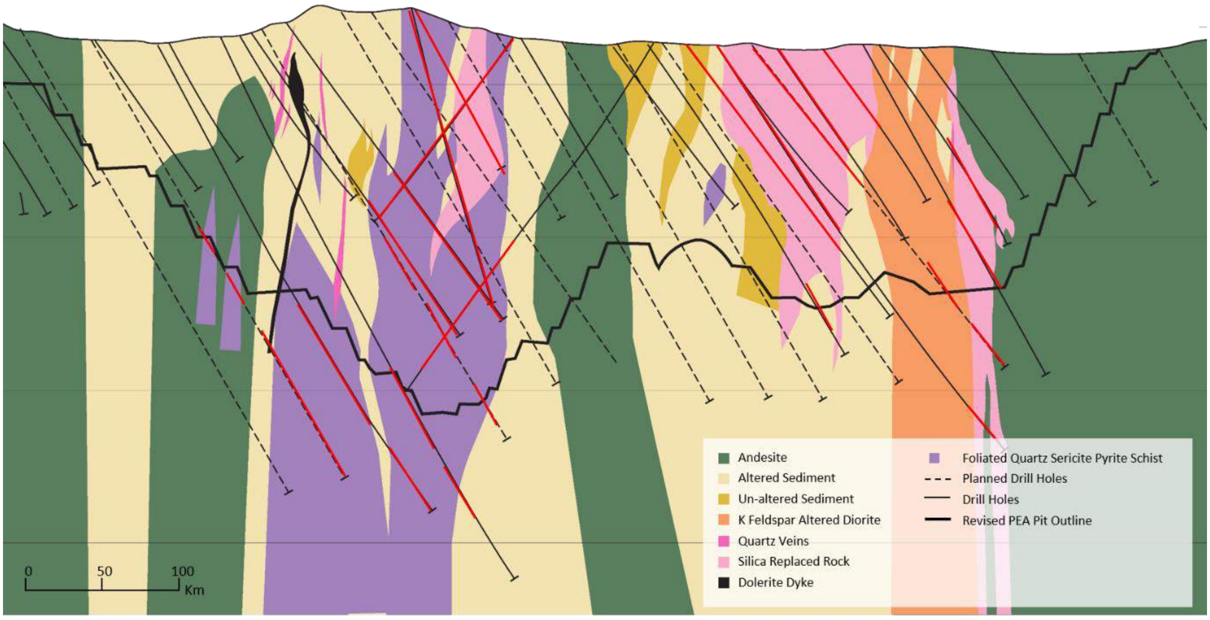
True widths are 65 - 75% of intercept widths.

Figure 10.7 GSS: Geology, Mineralization and Drill Traces



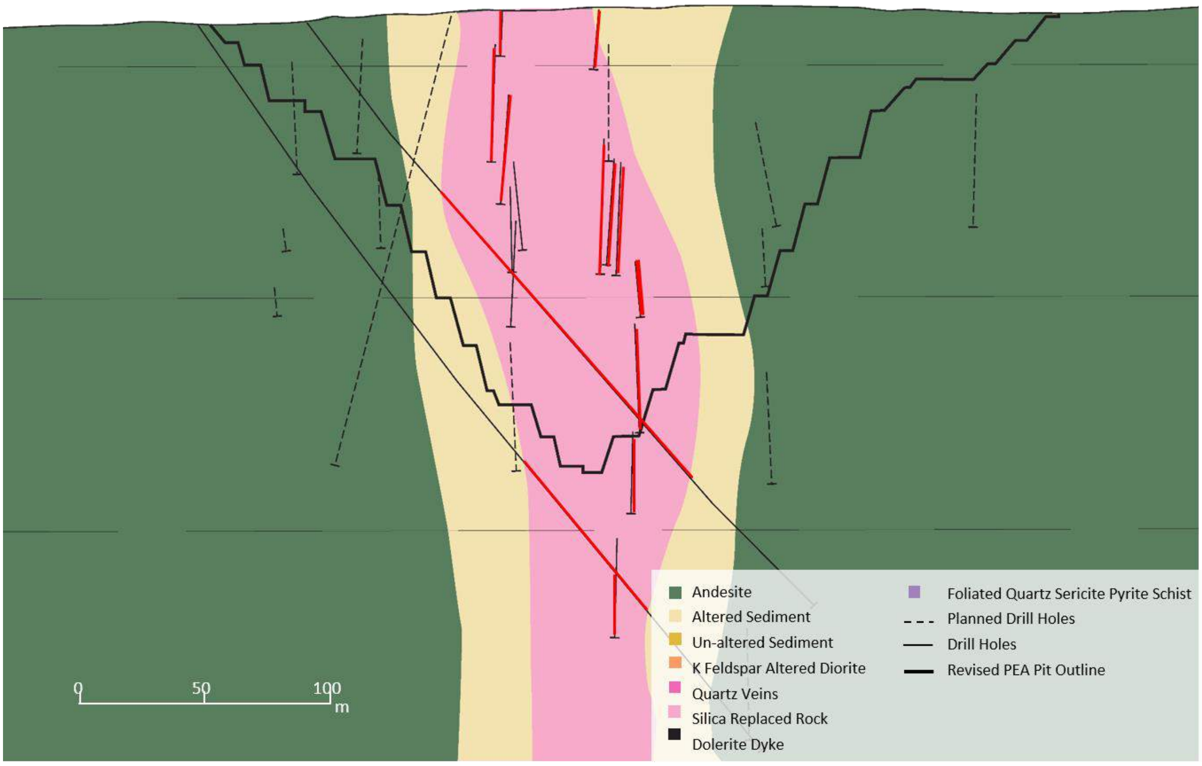
Source: Orca Gold Inc.

Figure 10.8 East Zone Drill Section 1



Source: Orca Gold Inc.

Figure 10.9 East Zone Drill Section 2



Source: Orca Gold Inc.

10.2.2 WD

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

Figure 10.10 View over WD Resource Area

Source: Orca Gold Inc.

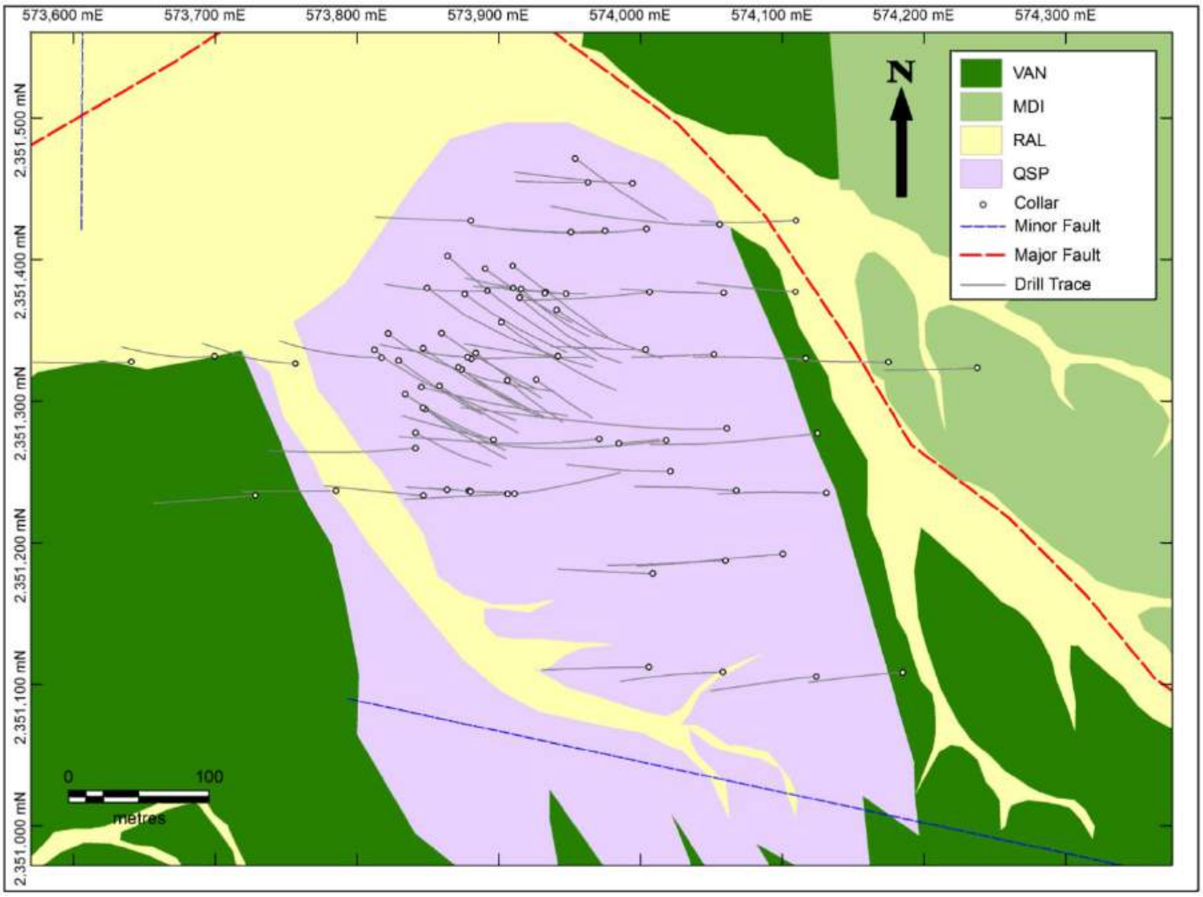
High grade mineralization is hosted by volcaniclastic units outcropping at the base of the hill and is characterised by significant levels of fine grained pyrite and a lack of silicification. Further mineralization is located below the main part of the hill with wide areas of anomalous to low grade containing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

Table 10.6 WD Drill Intercepts

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

True widths are 60 - 75% of intercept.

Figure 10.11 Location of Targets Drilling within the Block 14 Project

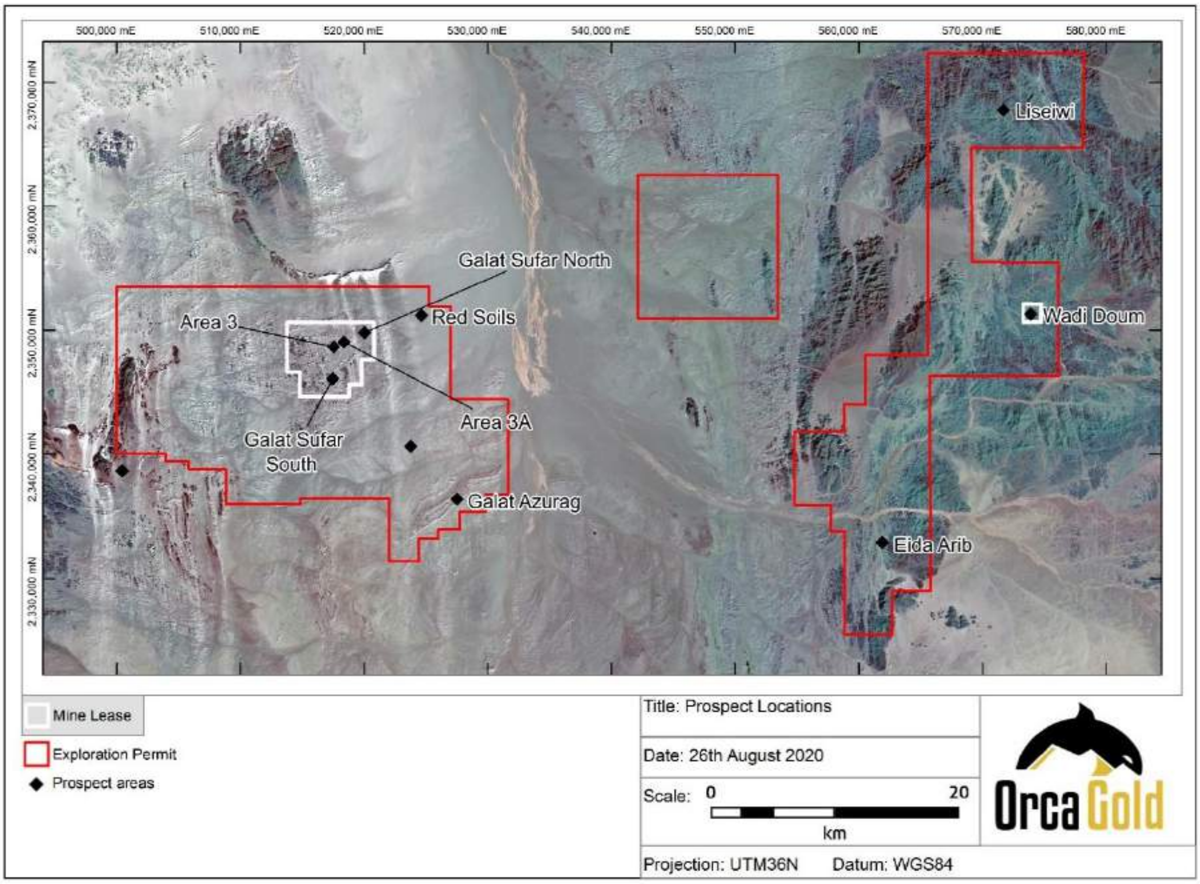


Source: Orca Gold Inc.

10.2.3 Other Prospects Drilled

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

Figure 10.12 Location of Targets Drilling within the Block 14 Project



Source: Orca Gold Inc.

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

Table 10.7 Significant Intercepts from Other Prospects

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

True widths are 60 - 75% of intercept

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

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

In October 2011 Orca commissioned a 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. Some silver assays have been completed at the ALS facility in Loughrea, Ireland.

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

11.1 Sample Submission Procedures

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

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

11.2 Sample Preparation and Analysis

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

Up to March 2013

- All drill core, trench, channel, 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 µ, 5% sieve tests.
- A split of 250g pulp is shipped to the Rosia Montana laboratory and a second 500 g sample is taken for archive (Figure 11.5).
- Pulps are shipped in batches in sealed boxes via the Orca office in Khartoum and international airfreight to Bucharest in Romania where they are collected by ALS staff and delivered to the laboratory at Rosia Montana.
- All drill core, RC chip, trench and rock chip sample are analysed for gold by 50 g fire assay with lead collection, solvent extraction and AAS finish (Au-AA26). From August 2017 onwards samples are now routinely assayed for Ag (AA45) at ALS alongside the Au assays.

- During the 2017 drilling program, 51,414 sample pulps were re-submitted from the Atbara prep lab to ALS Romania for Ag Assay (AA45 and MEICP41). At the same time, 12,458 pulp samples that were already in storage at ALS Romania were also assayed for Ag. This total of 63,872 samples consisted of drill intervals located within the 2017 PEA resource wireframes for GSS and WD which had previously not been assayed for Ag.
- 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.

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 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

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

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

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

Table 11.1 Quality Control Regime

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

In addition, the laboratory, ALS Chemex, have their own internal quality performance processes. These follow best practice guidelines required for qualification under International Organisation for Standardisation ("ISO") standards. The standard QA/QC protocols for the laboratories includes the insertion of CRMs, blank, duplicates and repeat assaying to monitor the quality of the preparation and analytical processes of the laboratory. The results of the internal laboratory quality control are reported regularly to Orca on a batch-by-batch basis and the results are closely monitored by Orca personnel.

11.4.1 Certified Reference Materials

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

For pulp samples re-submitted for Ag analysis in 2017 new CRM samples were submitted into the sample sequence which contained certified concentrations of Ag for QA/QC of the Ag analyses.

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

11.4.4 Field Duplicates

Field duplicates are one specific type of duplicate consisting of a sample, sampled by the same primary sampling method as the original. In the case of drainage and rock chip samples (surface samples), a second sample is collected from the same immediate locality as the original. For trenching, a second channel is cut along the line of the first sample. For RC drilling field duplicates the sample is split through a riffle splitter to produce an original sample and a duplicate sample. For diamond drilling, a quarter core is submitted as the duplicate sample.

11.4.5 Umpire Lab, Pulp Duplicates

Umpire Lab

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

A batch of 782 pulp duplicates was submitted to Intertek, Perth in November of 2017 for umpire assay by 50 g fire assay with ICP-OES finish. These pulps samples consist of a representative range of intervals across mineralised areas of the resource. The splits were 125 g pulp reject splits taken from reference samples stored at the Atbara preparation facility. New standard material was added in sequence to this batch.

Bottle Roll

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

11.5 Monitoring of QC Samples

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

11.5.1 Blank Material

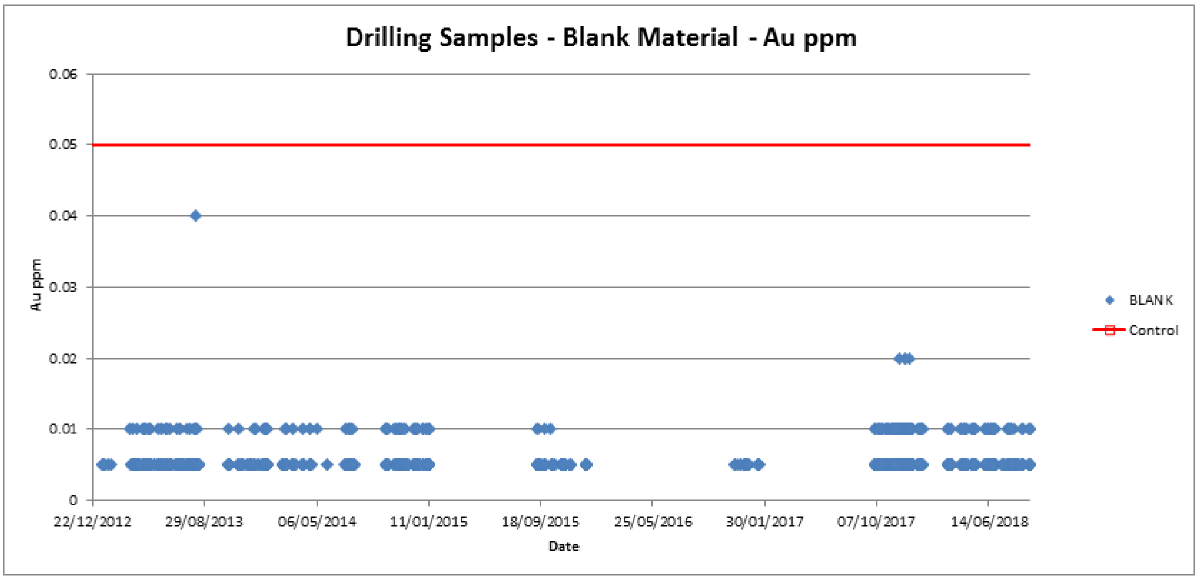
Blanks are assessed by graphical representation of the assay value and the maximum control value. Blank samples for drill sampling are displayed in Figure 11.6. A summary of all blank samples is reported in Table 11.2 for both Au and Ag analyses.

Results show that for drilling samples and surface samples, all blank Au 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 Au blank material assays return grades within control limits (<=0.05 ppm). 100% of Ag blank assays report within limits (<=0.5 ppm – Ag detection limit is 0.2 ppm). Ag results are only available from September 2017 onwards; these represent samples submitted as part of the 2017 -

2018 infill drilling program as well as Ag assaying of sample splits from previously sampled holes within the resource areas.

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, Au and Ag



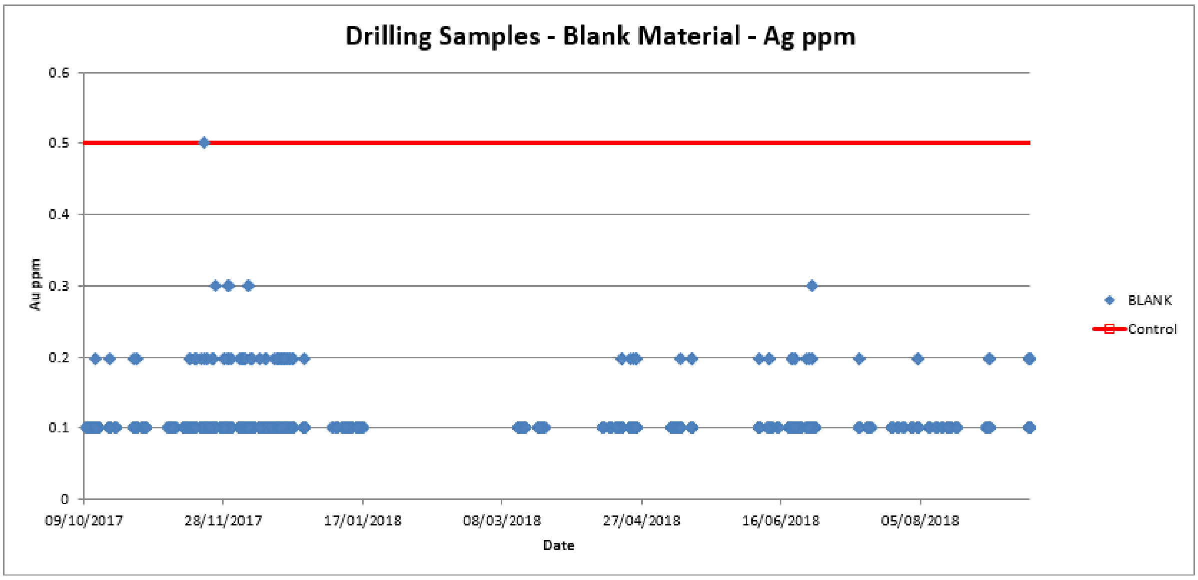


Table 11.2 Summary of Blank Samples for all Sample Methods

Sampling	STD	Count	Control Value	Result Average	STDEV of Results	% Fail
Driling (DD, RC) Au	Blank	6,230	0.05	0.006	0.002	0%
Drilling (DD, RC) Ag	Blank	4,363	0.5	0.103	0.019	0%
Surface (RCHIP, Drainage)	Blank	89	0.05	0.005	0.005	0%
Trench (TR, HC)	Blank	936	0.05	0.005	0.004	0.1%

Note: all samples assaying below detection limit of 0.01 ppm Au or 0.2 ppm Ag (DL) are rounded to ½ DL or 0.005 ppm Au, 0.1 ppm Ag.

11.5.2 Field Duplicates

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

Table 11.3 Quantity of Duplicate Pairs

Type	Pairs with Au Results	Pairs with Ag Results
Diamond Drill Core	1,139	1,090
Reverse Circulation Drilling	4,811	3,089
Trench	1,108	0

X-Y Scatter Plots

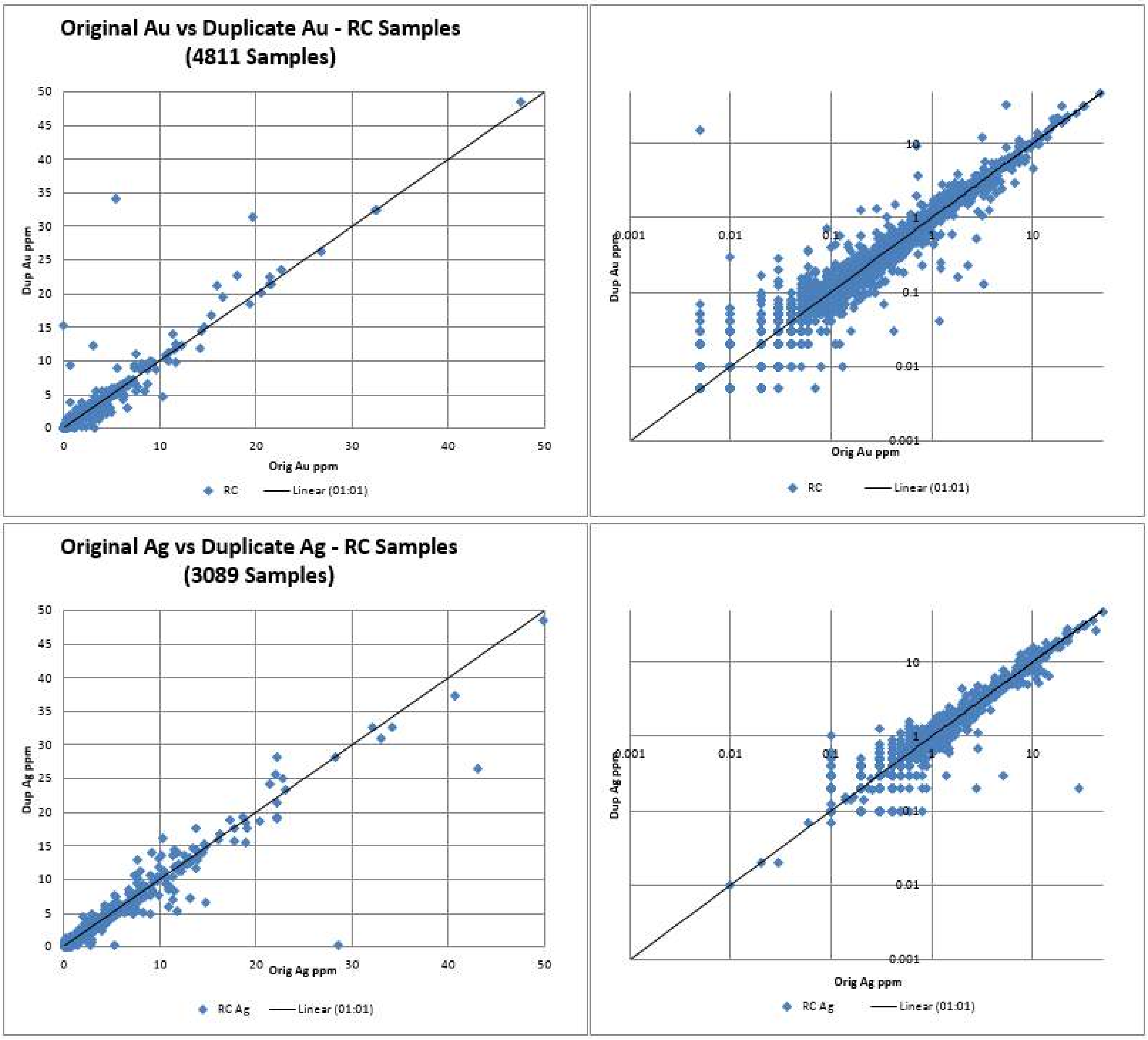
The simplest initial analysis is accomplished using an X-Y scatter plot to gain a general view of the repeatability of results and to identify obvious errors in samples; these are generated both with normal axis and log axis. The normal scatter plot aptly demonstrates correlation above 1 ppm; however,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

One outlier has not been displayed, field duplicate (Original B14324674) returned an Au duplicate value 116.65 ppm higher than its original value (9.85 ppm Orig, 126.5 ppm Dup). The sampled interval was logged as quartz carbonate vein and so therefore this extreme difference is accredited to coarse 'nuggety' gold within this vein. This vein mineralisation is not characteristic of the majority of the resource which is generally finely disseminated mineralisation within the host rock. Visible gold in core has only ever been identified in two specimens.

Scatter plots have been generated for both Au and Ag 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 Au fire assay, 0.2 ppm for Ag). See Figure 11.7 for an example of the normal and log scatter plots for the RC field duplicates for both Au and Ag. 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 – Au and Ag



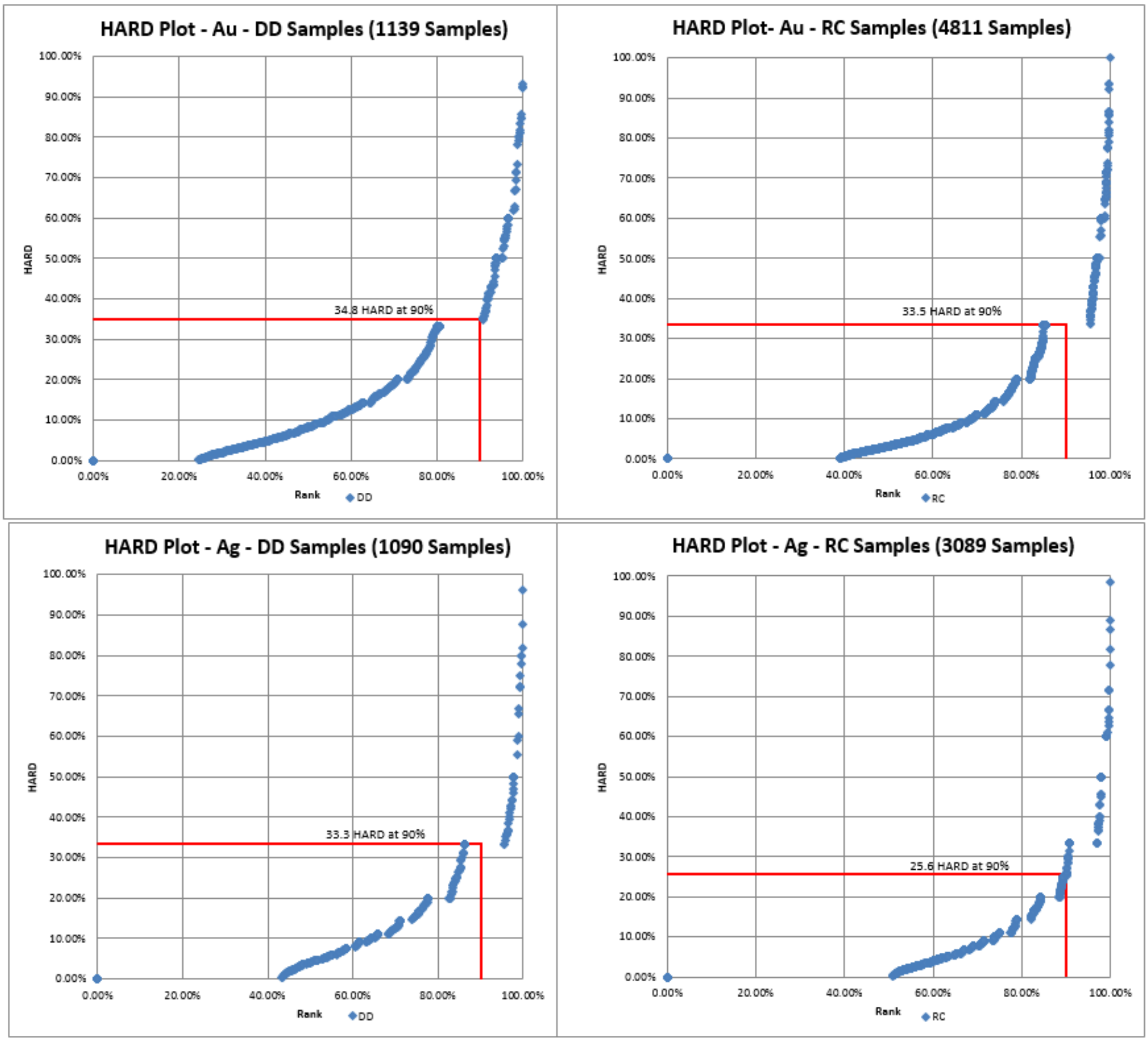
Ranked Half Absolute Relative Difference (HARD)

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

Ranked HARD plots have been generated for DD duplicate samples and RC duplicate samples for both Au and Ag (Figure 11.8). These plots demonstrate that RC duplicates have a better precision at the 90% proportion (33.5 HARD at 90%, note that the HARD only increases to 33.7 at 95%) compared to the DD duplicates (34.8% 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 over four times as many duplicate pairs. The 2017 and 2018 infill programs have added a further 871 pairs with Au results for DD sampling. This has seen the HARD value at 90% of the population decrease from 41.4 to 34.8. This suggests that duplicates have been performing better and is likely due to the majority of this drilling being HQ core (larger sample size) and also that the majority of this drilling has been targeted at ore grade areas of the resource therefore reducing the number of potential waste / near detection duplicates being sampled.

Ag duplicates perform better than Au for both sampling methods with DD sampling giving a HARD of 33.3 at 90% and RC giving 25.6 HARD at 90%. This lower HARD for the Ag compared to the Au likely indicates that the Ag mineralisation is more homogenous than the Au mineralisation, which is supported by visual assessment of the assay intercepts throughout the deposit.

Figure 11.8 Ranked Hard Plots for DD and RC Duplicate Samples

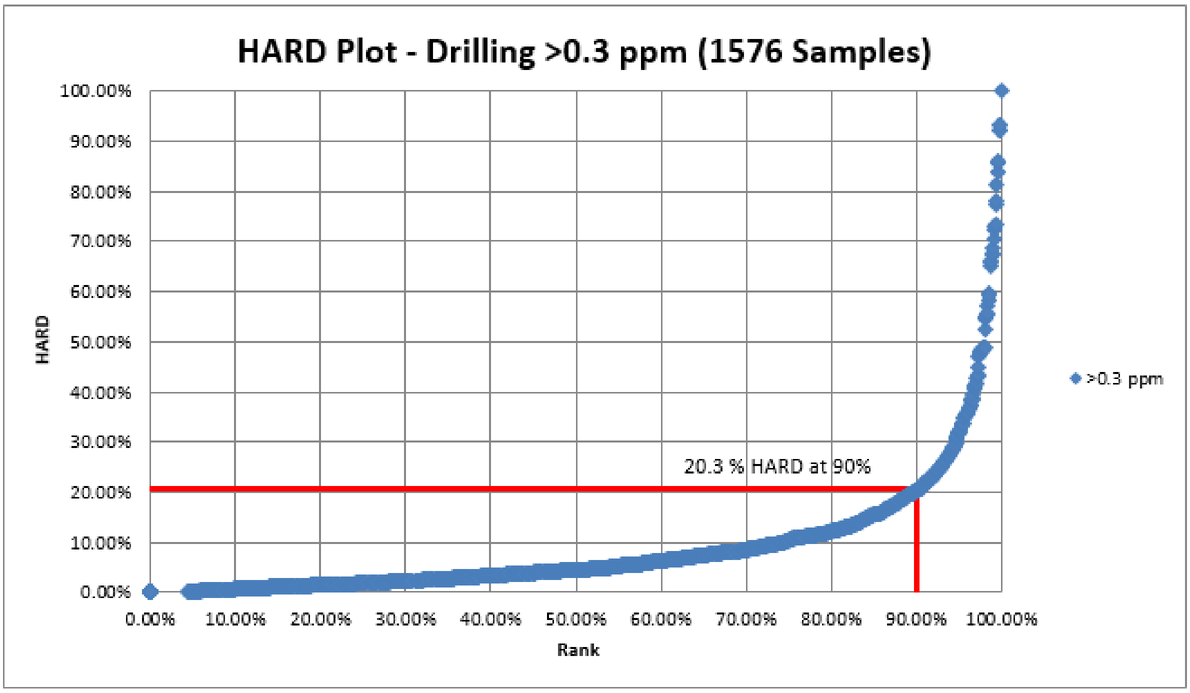


Waste / near detection limit assays compose a significant proportion of both data sets. When assessing relative differences, these near detection results can have significant influence. For example, an original assay of 0.03 ppm Au and a duplicate of 0.01 ppm Au with a pair mean of 0.02 ppm Au have a %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 a pair mean of 0.3 ppm, only for Au (Figure 11.9).

This data set consists of only 1,576 duplicate pairs compared to the total of 5,950 for the previous Au data sets. Therefore ~3/4 of the previous data set had a pair mean <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 Au 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. This value has increased by 0.7 prior to the 2017 infill program which again provides confidence in the repeatability of ore grade assays within this deposit, considering that the majority of this program targeted ore grades within the deposit.

Figure 11.9 Ranked Hard Plot for all Drilling Duplicates (Au >0.3 ppm)



Relative Difference (%RD)

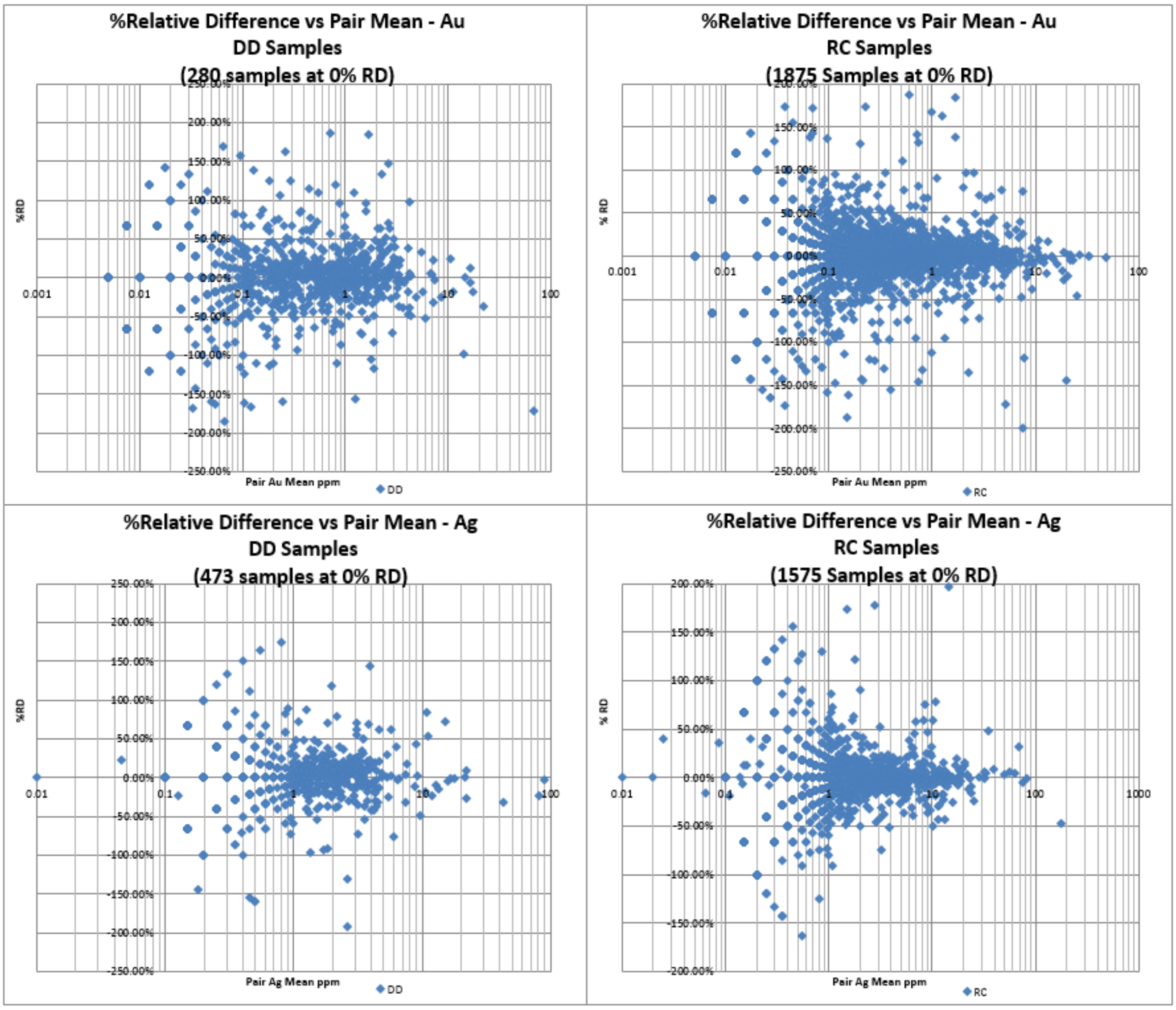
As discussed, the Ranked HARD plot is not able to be used to assess bias therefore a Relative Difference plot is used. If there is no bias the %RD plot is expected to be near symmetrical, forming a funnel shape with larger %RD at lower grades approaching the detection limit, due to the lower precision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

For both Au and Ag duplicated of diamond and RC sampling, this funnel shape is evident (Figure 11.10). Au 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 in early Diamond drilling (comparing a quarter core sample to a half core original sample). In the 2017 - 2018 program, true quarter core duplicates were taken where both the original and the duplicate are of quarter core size.

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

Figure 11.10 %Relative Difference Plot for DD and RC Duplicates



11.5.3 Pulp Duplicates (Umpire Assay)

1,069 pulp duplicates were sent to an umpire lab in 2013 and the results of this umpire assessment has been discussed in previous reports. The result of this analysis indicated good quality precision and increased confidence in the results of the primary laboratory analyses.

A further batch of 782 pulp duplicates was submitted to Intertek, Perth in November of 2017 for umpire assay by 50 g fire assay with ICP-OES finish. These pulps samples consist of a representative range of intervals across mineralised areas of the resource. The splits were 125 g pulp reject splits taken from reference samples stored at the Atbara preparation facility. New standard material was added in sequence to this batch which already contained pulp samples of blank material. These standards and blanks have been assessed separately and have passed QA/QC thresholds. In total, 68 blanks and standards are included within the 782 sample batch; the remaining 714 are original samples.

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 three points are evident below the 1:1 distribution of the majority of the population. On the log scale plot it can be seen that duplicates perform worse at near detection limit levels, as expected.

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 (715). This batch of umpire samples performs better than the previous batch, with the 90% population HARD of all original values giving 11.7%, compared to 25%. This is further improved by removing all <=0.3 ppm pairs, giving a HARD of 8.47% (593 pairs). This result indicates good quality precision of umpire lab duplicates. A few noted outliers would suggest potential 'nuggety' coarse gold in those samples, but in general this is not a trend seen often throughout the deposit.

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

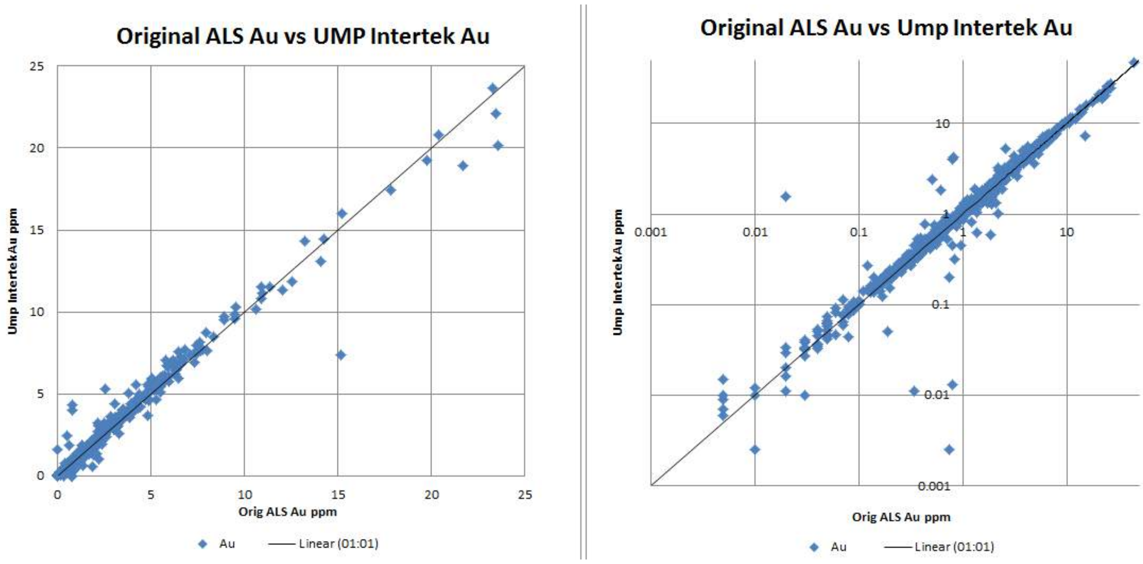
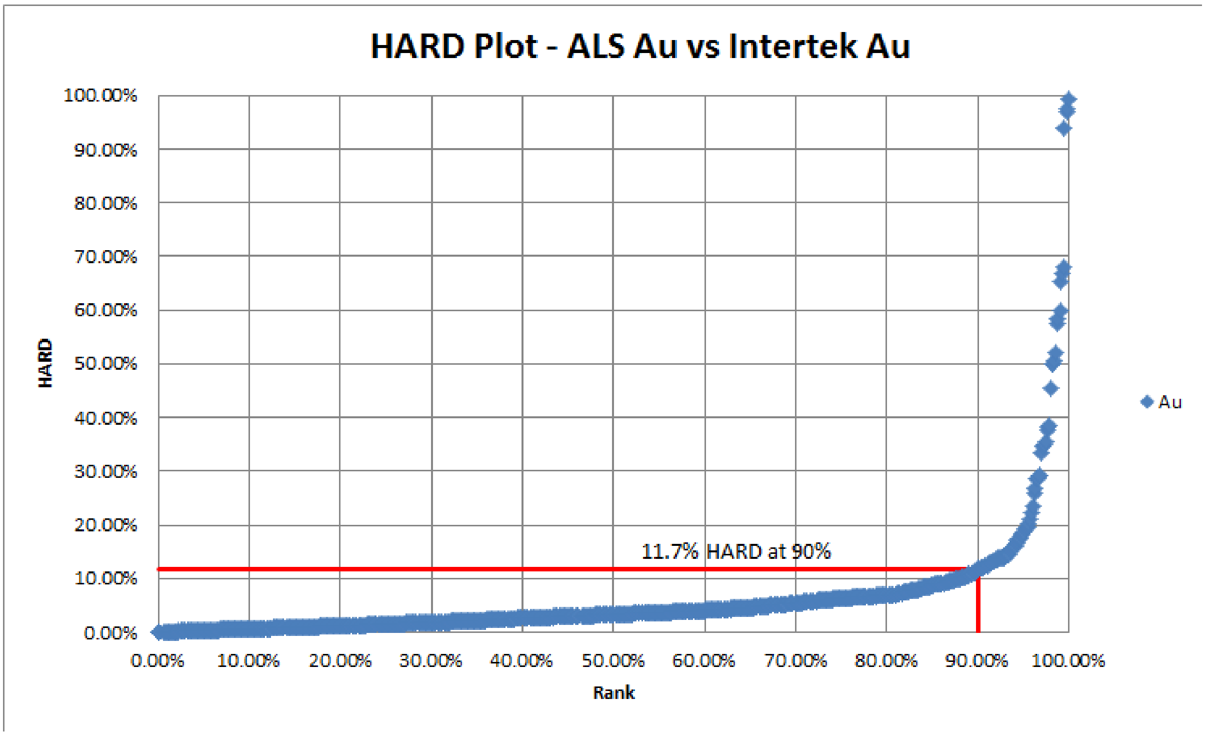


Figure 11.12 Ranked HARD Plot of all Umpire Duplicates



11.5.4 Bottle Roll Pulp Duplicate

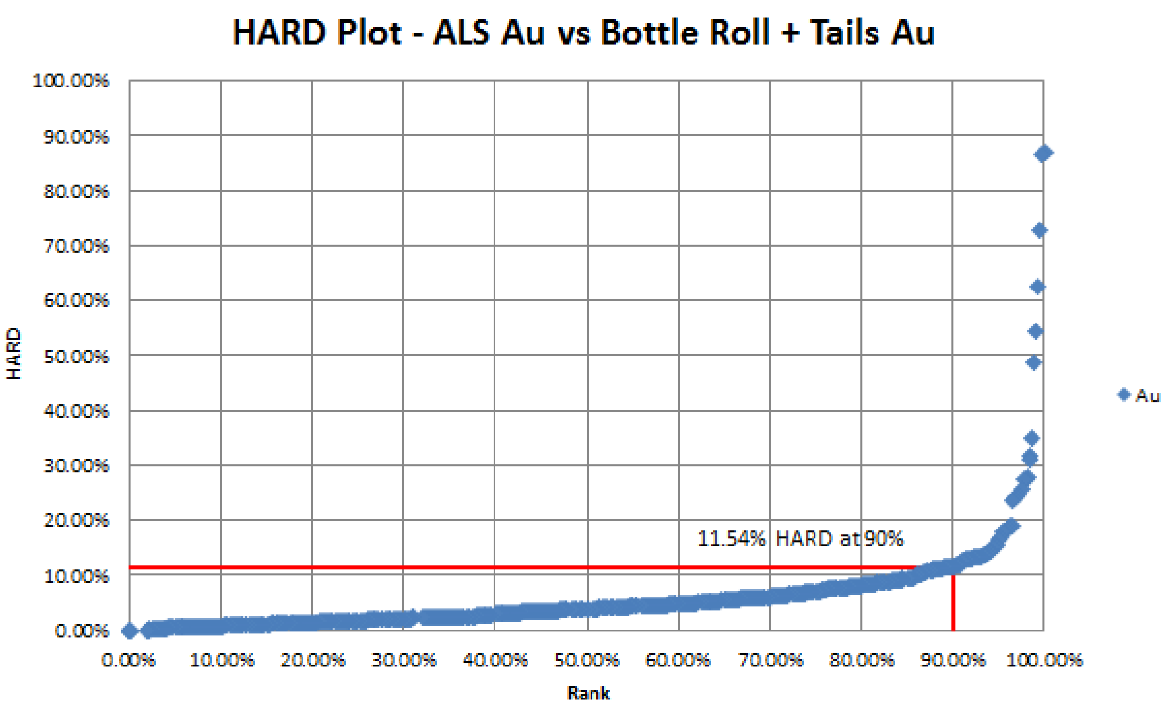
481 samples of 500 g were assayed by bottle roll. For the QA/QC assessment the leach assay and the fire assay of the leach tails have been summed and compared to the original fire assay result.

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

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

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

Figure 11.13 Ranked HARD Plot of Bottle Roll Duplicate Pairs



11.5.5 Certified Reference Material (CRM)

CRM Assessment Overview

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

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

For simplicity the CRM assay results have been tabulated and summarised for Drilling bellow in Table 11.4.

Drilling CRM

A total of 5,716 Au CRM samples have been included in drilling sample sequences as summarised below in Table 11.4.

Table 11.4 Summary Table of all Drilling CRM Results

STD	Count	Cert Value	Result Average	STDEV of Results	% 3SD Fail (#)	% >±2 SD (#)	% >±1 SD (#)
G311-7	1404	0.4	0.39	0.024	0.1 % (2)	0 %	0.4 % (5)
G907-1	617	0.79	0.77	0.018	0 %	0 %	0 %
G300-9	606	1.53	1.51	0.028	0 %	0 %	1 % (6)
G308-3	560	2.47	2.51	0.115	0.2 % (1)	0 %	3.8 % (21)
G303-2	486	4.11	4.24	0.106	0 %	0 %	5.3 % (26)
G307-7	370	7.87	7.90	0.167	0 %	0 %	10.8 % (40)
G900-7	359	3.22	3.23	0.081	0 %	0 %	2.8 % (10)
GLG303-1	281	0.164	0.16	0.008	0 %	0 %	4.3 % (12)
G910-1	237	1.43	1.45	0.039	0 %	1.3 % (3)	8 % (19)
G908-3	229	1.03	1.03	0.023	0 %	0 %	2.2 % (5)
G312-5	184	1.6	1.61	0.054	0 %	0.5 % (1)	10.9 % (20)
G913-2	134	2.4	2.44	0.051	0 %	2.2 % (3)	16.4 % (22)
G312-4	77	5.3	5.48	0.171	1.3 % (1)	6.5 % (5)	26 % (20)
G913-8	76	4.87	5.02	0.109	1.3 % (1)	3.9 % (3)	35.5 % (27)
GLG310-5	44	0.07983	0.08	0.011	2.3 % (1)	11.4 % (5)	45.5 % (20)
G901-7	22	1.53	1.48	0.030	0 %	0 %	0 %
G909-6	15	0.56	0.54	0.013	0 %	0 %	6.7 % (1)
GLG901-2	11	0.00992	0.01	0.002	0 %	0 %	63.6 % (7)
GBMS304-5	4	1.62	1.52	0.022	0 %	0 %	75 % (3)
TOTAL	**5,716**			**Number**	**6**	**20**	**264**
				%	**0.10%**	**0.35%**	**4.62%**

Some of 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.

A total of 2,637 Ag CRM samples were included in the 2017 Ag re-assay batches and are summarised in Table 11.5

Table 11.5 Summary Table of Drilling CRM Results for Ag

STD	Count	Cert Value	Result Average	STDEV of Results	% 3SD Fail (#)	% >±2 SD (#)	% >±1 SD (#)
GBM315-3	886	2.6	2.52	0.288	0.1% (1)	0%	4.5% (40)
GBM315-6	879	1.3	1.29	0.120	0%	0.3% (3)	5% (44)
GBM915-2	872	9.9	9.88	0.303	0%	0%	7.6% (66)
TOTAL	**2,637**			**Number**	**1**	**3**	**150**
				%	**0.04%**	**0.11%**	**5.69%**

12. DATA VERIFICATION

Nic Johnson of MPR Geological Consultants visited the Block 14 project area in 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 Western Gabgaba domain.

The visit included discussions in Khartoum and on site with the exploration personnel who have managed and supervised the exploration programmes to date.

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

- Geology and exploration history of the permit areas.
- Exploration model and strategy.
- Current exploration data and exploration procedures.
- Geochemistry and geophysics results.
- QA/QC procedures and control data.
- Data and database management systems.
- Sample handling and storage.
- The sample preparation facility in Atbara.
- Future work programmes and budgets for the next 12 months.
- The author has reviewed procedures, drill data and QA/QC sampling data undertaken as part of the exploration programmes on the Block 14 project.

12.1 Data Validation

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

Orca provided exports of the data required and has supplied all relevant data to the Author to undertake various validation checks of the stored information.

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

13. MINERAL PROCESSING AND METALLURGICAL TESTING

Various phases of metallurgical testwork have been completed since 2014 which are summarized below. Further detail can be found in the Preliminary Economic Assessment NI 43-101 Report (30 August 2016) and Revised Preliminary Assessment NI 43-101 Report (20 May 2017).

13.1 Metallurgical Sampling

Following the development of a lithological model, samples were selected to approximately represent the proportion of each lithology within the potential pit for Main, East, North East and Wadi Resource.

The total number of samples tested that were used to provide data for the Feasibility Study are tabulated in Table 13.1.

Table 13.1 Summary of Metallurgical Samples

Pits	Number of Samples
Leach Variability	112
SMC	78
BWI	125
CWI	12
AWI	34

13.2 Metallurgical Testing

13.2.1 ALS Metallurgical testwork 2014

ALS conducted scoping level testwork on seven composite samples and conclusions were as follows:

- Gravity Recovery and free gold not observed due to gold being extremely fine grained.
- Gold readily leaches in cyanide at fixed d_{80} = 75 µm:
  - gold recoveries of 78 – 87% observed
  - 16 – 18% gold locked in sulphides and 2 - 4% in host rocks
  - QEMScan mineralogy confirmed Gold is mainly pyrite hosted
  - average gold grain size is between 1 - 10 µm.

13.2.2 SGS Mineral Services UK Testwork 2015

SGS Minerals Services UK conducted several phases of scoping testwork for Orca as follows:

- Heap Leach was tested on East Oxides and gold recoveries of 73% at 12 mm crush size down to 6 mm crush size were obtained.
- Cyanide leaching with and without carbon was performed on East Fresh 3 and Main Fresh 3 composite samples to identify any “Preg-Robbing” characteristics at d_{100} 106, 75 and 53 µm:
  - no preg-robbing characteristics were observed
  - gravity testing again showed no free gold and due to the extremely fine grained gold, gravity recovery was rejected as a potential feature in the flowsheet
  - gold recoveries were found to increase generally with decreasing grind size from 70% at d_{100} 150 µm to 78 - 83% at d_{100} 53 µm
  - gold recoveries were found to increase generally on East Fresh 3 with decreasing grind size from 73% at d_{100} 150 µm to 83% at d_{100} 53 µm

- cyanide consumptions were found to be low at 0.13 - 0.37 kg/t.

- Due to the fine nature of the gold and its tendency to associate with sulphides, flotation was subsequently tested on new samples (Main Fresh 4 and East Fresh 4):

  - 93 – 94% recovery of gold and 98% recovery of sulphur was achieved into a rougher flotation concentrate consisting 12 - 15% of the feed weight
  - good flotation recoveries were observed at d_{100} 125 µm
  - cleaning of this rougher concentrate lost 5 - 6% of the gold into cleaner tailings but reduced the weight recovery to cleaner concentrate to 7 – 9%
  - regrinding and flotation of the rougher concentrate to 20 µm did not improve gold recovery, undoubtedly due to the very fine nature of the gold
  - overall gold recovery was 78% for East Fresh 4 and 83% for Main Fresh 4
  - attempts were made to float East Oxide 2 composite using sulphidising conditions but gold recoveries were consistently low at 47 – 58%.

- A final additional phase of work was conducted on Wadi Doum Fresh 1, 2 and 3 composite samples:

  - gold recoveries at d_{100} 75 µm grind size ranged from 77 – 93% depending upon the head assay. The lower grade ore giving the lower gold recovery whilst the higher grade ore giving much higher recoveries
  - rougher flotation testing gave similar results as East and Main Fresh 3 samples at 91 – 97% gold and 97 – 99% sulphur with 10 – 32% weight yield to rougher concentrate.

13.2.3 SGS Mineral Services South Africa Testwork 2016

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

- Comminution Testwork:

  - highlighted that the oxide materials were the softest of all the various domains and transitional and fresh were harder, East Zone Fresh being the hardest
  - a similar trend was observed with Abrasion Index
  - this data has been used as part of the overall data base for the DFS study.

- Gold Mineralogy at d_{80} 75 µm:

  - the majority (>95%) of gold present is native gold (Au-Ag), the remaining minor quantity of gold is present as Petzite (Ag_3AuTe_2) which has much slower leach kinetics than native gold
  - petzite was slightly more prevalent in the East Zone Fresh 4 sample when compared with Main Zone Fresh 4 sample
  - the typical quantity of Ag in the native gold was 20% in East and Main Zone Fresh 4 samples, but was found to be as high as 35% in Wadi Doum sample tested
  - the majority of gold particles were associated with sulphides but Wadi Doum appears to contain more liberated gold particles
  - the gold grain sizes are extremely fine (<40 µm) and Wadi Doum gold grain size is slightly coarser when compared with East and Main Zone Fresh 4 samples

- gold is either present as free grains or more predominantly as fine (1 - 10 µm) associations and/or inclusions in pyrite
- the predominant gold carrier in East and Main Fresh 4 samples is pyrite whereas in Wadi Doum, Arsenopyrite, Spahlerite and Galena were also observed carrying gold.

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

13.2.4 SGS Mineral Services South Africa 2016 and 2017

The hardness testing that was conducted by SGS RSA has been included in the DFS data base.

13.2.5 SGS South Africa – Flash Flotation Investigation

This testwork optimised Flash Flotation on two samples (East Fresh 6 and Main Fresh 6) and 32 variability samples were tested to ascertain the variability of flash flotation on all East and Main Fresh and Transition domains and Wadi Doum Fresh domain only.

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

The culmination of the above scoping level testing provided the following flowsheet development direction for the following stages of flowsheet development:

- Gravity Concentration not feasible due to the lack of coarse free gold grains.
- Flotation and/or Flash Flotation were rejected following Trade Off Studies.
- Whole Ore Leaching selected as the optimum flowsheet for the project.

13.3 SGS Vancouver – Whole Material Leach Optimization by Size (2017)

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

13.3.1 Cyanide Leach Variability Bottle Roll Testwork

This testwork focused on Primary Grind Optimization using Whole Rock Cyanide Leach Variability Testing on 23 samples coming from East Zone Fresh, Transition and Oxide domains, Main Zone Fresh, Transition and Oxide domains, North East Oxide and Wadi Doum Zone Fresh domains.

Samples were generally tested at four different grind sizes (P_{80} 53, 75, 106 and 125 µm).

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

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

A total of 87 whole rock cyanide leach variability tests were conducted as follows:

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

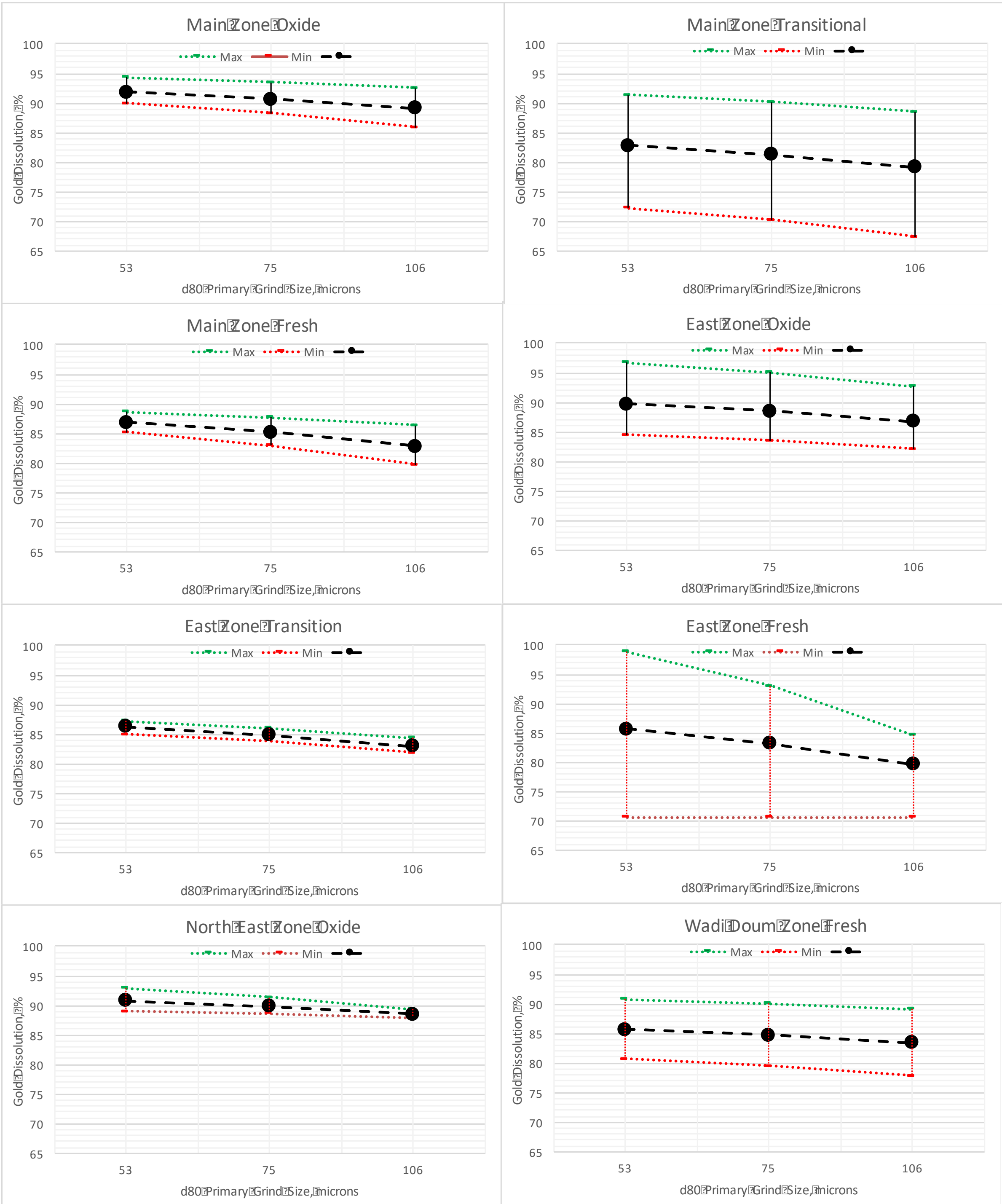
The output data from this study was utilised in the grind optimization study.

13.3.2 Gold Dissolution Trend with Grind Size and Variability

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

In two domains, East Zone Transition and North East Zone Oxide, the gold dissolution variability between the three samples tested was found to be minimal and because this data provided better r^2 data, the trend line between overall samples was taken as opposed to the average as was the case with all other six domains. This strategy ensured that the strongest trend relationships were employed at all times.

Figure 13.1 Gold Dissolution Trends and Variability



Conclusions - Gold Dissolution Trend with Grind Size and Variability

- All 23 samples from the eight domains tested showed a generic trend for gold dissolution to increase with decreasing grind size.
- The fresh domains in all three zones appear on average to be the most sensitive to grind size (typically 5% gold dissolution loss between 106 and 53 µm) and some samples demonstrated quite extreme sensitivity. For example, in East Zone Fresh 13 sample, 14% gold dissolution was observed:
  - this phenomenon is likely to be due to inherent more tightly bound fresh fine gold mineralogy when compared with the more oxidized gold in oxides, which will increase fine gold grain exposure.

- The highest gold dissolutions (typically > 90%) were obtained on the oxide domains.
- East Zone Transition and North East Zone Oxide vary the least, whereas Main Zone Transition and East Zone Fresh vary the most.

13.4 SGS Vancouver Feasibility Testwork (2017 – 2018)

The work was carried out in two phases, the initial phase of work detailed results are available in SGS Report Number 16135-002 April 2018 and the second phase of work in SGS Report Number 16135-03 October 2018.

The focus of the feasibility study focus had the following objectives:

- Provide comminution variability data in addition to the work described above to provide a robust input data base for lithology structured comminution circuit design.
- Using d_{80} 53 µm (Oxides and Wadi Doum Fresh), 75 µm for all transitional domains and 90 µm for East and Main Fresh domains, conduct further leach variability testing to optimise residence time whilst also providing more specific data on each lithological code within each domain to develop a lithological coded metal recovery and reagent cost data base for the financial modelling.
- Confirmatory thickener design testwork using the Area 5 fresh water.
- Confirmation of any affects the new fresh water had on metal recovery and/or reagent consumption rates.
- Confirmation of the benefits of introducing both higher pulp density (55% solids w/w) in conjunction with either pre-aeration and/or pre-oxygenation on reagent consumption rates and metal recovery rates.
- Confirm the effect of simulating recycled cyanide solutions from tailings thickener overflow to the grinding circuit.

13.4.1 Comminution Testing

The FS has focused upon further strengthening and understanding of physical and mechanical properties of the four pits and major domains and lithologies that supplements the initial work conducted by SGS South Africa in 2016.

Table 13.2 summarises the comminution tests conducted as part of PFS and DFS studies.

Table 13.2 Summary of Comminution Tests

Study	Crusher Wi	Abrasion Index	SMC	Bond Ballmill Wi
PFS	0	6	12	62
DFS	12	28	66	63
Total	**12**	**34**	**78**	**125**

The sample selection was designed based upon lithology driven comminution design and allowed for greater definition of anticipated milling equipment sizing and power requirements to enhance the accuracy of the capital and operating costs associated with comminution.

Table 13.3 summarises the overall comminution results from both PFS and DFS studies.

Table 13.3 Summary of all Comminution Results

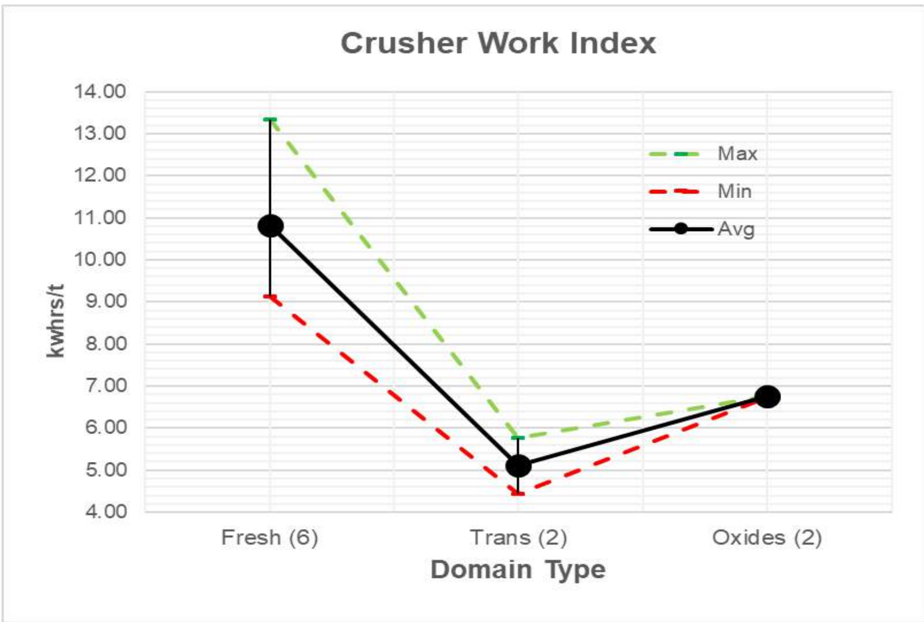
Lab	Pit	Domain	Lithology	Sample ID	A x b	t_a	BWI (kWh/t)
	Pit	Domai	Litholo	Sample ID	A x b	t_a	BWI
SGS Canada	Main	Oxide	FQSP	MO8	87.7	0.93	11.0
SGS RSA	Main	Oxide	PGSS	MO5	87.0	0.93	10.9
SGS Canada	Main	Oxide	PGSS	MW1	175.7	1.70	8.9
SGS Canada	Main	Oxide	VAN	MO20	122.1	1.26	
SGS Canada	Main	Oxide	VAN	MO21			9.0
SGS RSA	Main	Trans	FQSP	MT1	68.1	0.75	8.9
SGS Canada	Main	Trans	FQSP	MT8	59.4	0.57	10.8
SGS Canada	Main	Trans	VAN	MT20	60.7	0.62	
SGS Canada	Main	Trans	VAN	MW2	64.3	0.60	9.6
SGS RSA	Main	Fresh	FQSP	MF5	54.7	0.54	10.1
SGS RSA	Main	Fresh	FQSP	MF16	65.3	0.59	11.3
SGS Canada	Main	Fresh	FQSP	MF17	52.3	0.48	11.5
SGS Canada	Main	Fresh	FQSP	MF18	30.0	0.28	14.7
SGS Canada	Main	Fresh	FQSP	MF23	38.9	0.37	13.2
SGS Canada	Main	Fresh	FQSP	MF25	35.8	0.32	11.3
SGS Canada	Main	Fresh	FQSP	MF27	40.0	0.36	12.0
SGS Canada	Main	Fresh	PGSS	MF24	40.1	0.36	11.8
SGS Canada	Main	Fresh	PGSS	MF26	45.6	0.41	11.2
SGS Canada	Main	Fresh	PGSS	MF37	49.8	0.46	11.3
SGS Canada	Main	Fresh	PGSS	MF38	49.3	0.45	11.1
SGS Canada	Main	Fresh	PGSS	MF39	57.7	0.52	10.7
SGS Canada	Main	Fresh	PGSS	MW4	65.7	0.60	11.2
SGS Canada	Main	Fresh	VAN	MF33	41.8	0.38	13.1
SGS Canada	Main	Fresh	VAN	MF34	51.4	0.47	11.2
SGS Canada	Main	Fresh	VAN	MF35	55.6	0.51	10.7
SGS Canada	Main	Fresh	VAN	MF36	40.8	0.38	11.2
SGS Canada	Main	Fresh	VAN	MW3	54.4	0.50	10.8
SGS Canada	Main	Waste	PGSS	GW4	72.4	0.65	10.4
SGS Canada	Main	Waste	VAN	GW3	51.0	0.47	11.2
SGS RSA	East	Oxide	FQSP	EO3	119.2	1.27	8.7
SGS Canada	East	Oxide	FQSP	EO21			9.0
SGS Canada	East	Oxide	PGSS	EW1	86.3	0.88	8.2
SGS RSA	East	Trans	FQSP	EO10/11	105.0	1.08	
SGS Canada	East	Trans	FQSP	EO13	139.3	1.44	10.1
SGS RSA	East	Trans	FQSP	ET2	66.5	0.70	10.7
SGS Canada	East	Trans	FQSP	ET9	63.7	0.63	13.5
SGS Canada	East	Trans	PGSS	EO12	78.6	0.80	9.7
SGS RSA	East	Trans	PGSS	ET8	45.2	0.43	11.1
SGS Canada	East	Trans	VAN	ET20	79.7	0.78	
SGS Canada	East	Trans	VAN	EW2	65.3	0.63	10.3
SGS Canada	East	Fresh	FQSP	EF20	52.8	0.48	13.7
SGS Canada	East	Fresh	FQSP	EF21	32.4	0.30	15.8
SGS Canada	East	Fresh	FQSP	EF60	38.3	0.36	14.3
SGS Canada	East	Fresh	FQSP	EF62	45.2	0.43	12.5
SGS Canada	East	Fresh	PGSS	EF61	44.3	0.41	10.0
SGS Canada	East	Fresh	PGSS	EF63	47.6	0.44	11.9
SGS Canada	East	Fresh	PGSS	EF64	47.8	0.43	10.7
SGS Canada	East	Fresh	PGSS	EF65	39.5	0.36	12.2
SGS Canada	East	Fresh	PGSS	EF66	42.0	0.39	12.2
SGS Canada	East	Fresh	PGSS	EF66	42.0	0.39	12.2
SGS Canada	East	Fresh	PGSS	EF67	36.9	0.34	12.8
SGS Canada	East	Fresh	PGSS	EF72	51.8	0.50	15.4
SGS Canada	East	Fresh	PGSS	EW4	77.2	0.70	10.1
SGS RSA	East	Fresh	SRR	EF5	45.3	0.43	12.0
SGS Canada	East	Fresh	SRR	EF17	32.0	0.30	19.8
SGS Canada	East	Fresh	SRR	EF18	36.6	0.35	20.6
SGS Canada	East	Fresh	SRR	EF18	36.6	0.35	20.6
SGS Canada	East	Fresh	SRR	EF19	30.0	0.28	17.5
SGS Canada	East	Fresh	SRR	EF68	34.6	0.33	17.6
SGS Canada	East	Fresh	SRR	EF69	32.2	0.30	17.5
SGS Canada	East	Fresh	SRR	EF70	38.4	0.36	12.0
SGS Canada	East	Fresh	SRR	EF71	34.3	0.33	18.5
SGS Canada	East	Fresh	SRR	EF73	40.6	0.40	17.1
SGS Canada	East	Fresh	VAN	EF22	43.7	0.42	19.3
SGS Canada	East	Fresh	VAN	EF85	46.0	0.42	12.2
SGS Canada	East	Fresh	VAN	EF86	35.9	0.32	14.1
SGS Canada	East	Fresh	VAN	EF87	49.8	0.45	12.4
SGS Canada	East	Fresh	VAN	EF88	44.6	0.42	11.0
SGS Canada	East	Fresh	VAN	EW3	38.7	0.35	12.1
SGS Canada	East	Waste	FQSP	GW2	33.9	0.32	17.7
SGS Canada	East	Waste	PGSS	GW1	40.2	0.38	18.1
SGS Canada	NEZ	Oxide	PGSS	NEO6	113.6	1.16	9.5
SGS Canada	NEZ	Trans	PGSS	NET1	128.0	1.17	10.6
SGS RSA	Wadi	Fresh	QPR	WF11	50.9	0.47	12.4
SGS RSA	Wadi	Fresh	VC3	WF4	50.6	0.48	10.6
SGS Canada	Wadi	Fresh	VC3	WF19	42.5	0.38	12.1
SGS Canada	Wadi	Fresh	VC3	WF20	40.1	0.36	12.7
SGS Canada	Wadi	Waste	VAN	WW1	34.6	0.32	12.6
SGS Canada	Wadi	Waste	VAN	WW2	34.5	0.32	13.0

The above data has formed the basis for the input data for comminution design.

The data has allowed the following trend graphs to be developed as follows:

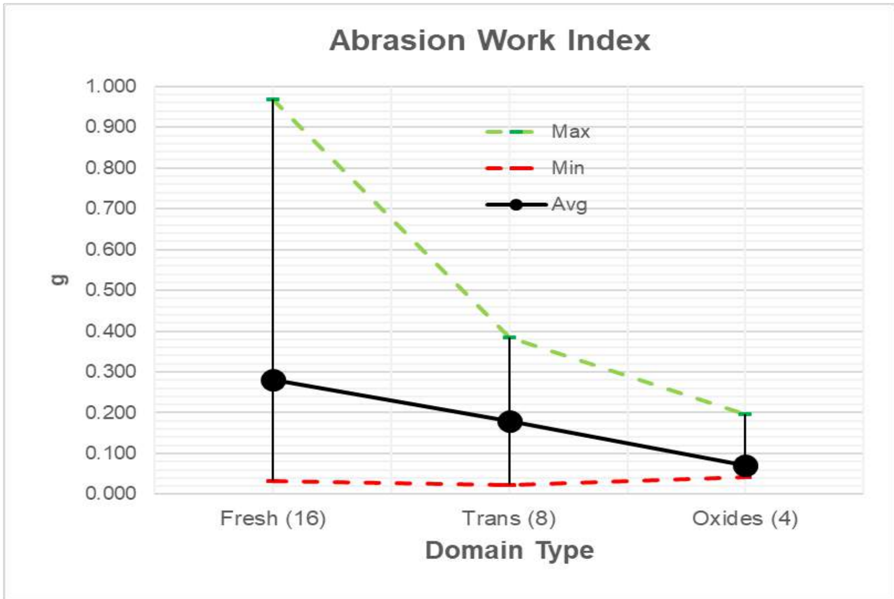
- Referring to Figure 13.2, the crusher work index trend of the fresh domains varies the most and are generally ranked "Easy-Medium".
- The crusher work index of the transition and oxide domains are ranked as "Very Easy" and vary much lower in value, albeit only two samples were tested in these domains.

Figure 13.2 Crusher Work Index Trends by Domain Type



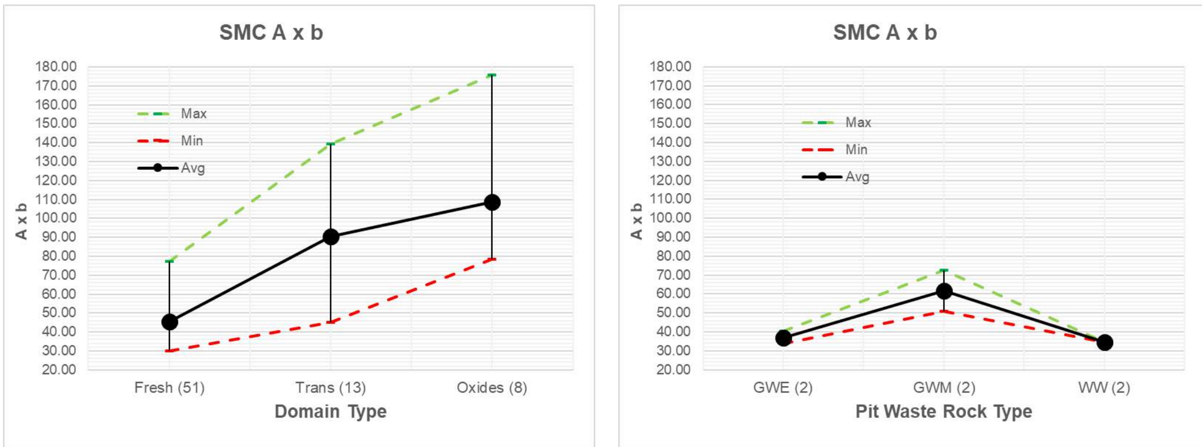
- Referring to Figure 13.3, the Fresh Domains were again seen to be highest and vary quite significantly from "Non Abrasive" to "Very Abrasive".
- The Abrasion Index reduces in transition and oxides domains and are both ranked as "Slightly Abrasive".

Figure 13.3 Abrasion Work Index by Domain Type



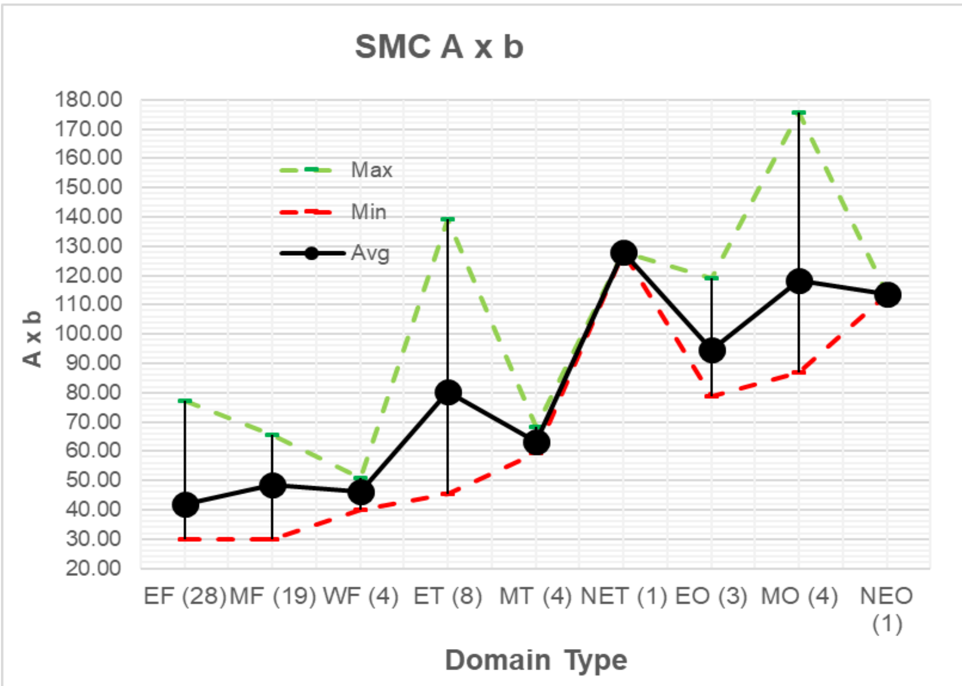
- Referring to Figure 13.4, the A x b SAG mill value follows the same trends as the other comminution testing and is clearly hardest in the fresh domains, but differs in that it varies across all domains.
- The A x b SAG mill value again indicates a reduction in hardness for the transition and oxide domains.
- The waste rock samples tested in all three pits are more consistent and all harder in terms of resistance to SAG milling.

Figure 13.4 SMC (A x b) by Domain (Mineralised Rocks and Waste Rocks)



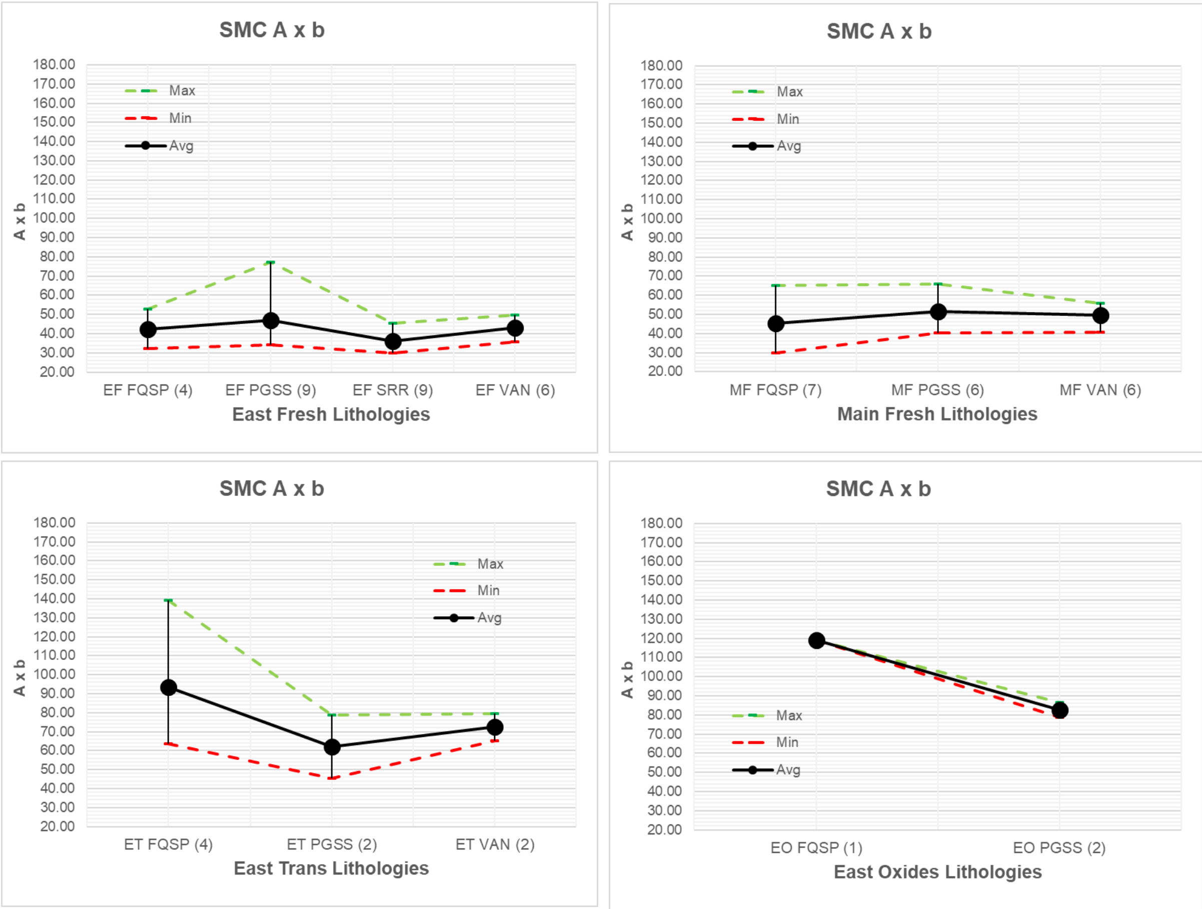
- Referring to Figure 13.5, the Fresh Domains are the hardest and were seen to vary from “Hard” to “Soft”. The East Fresh varies the most of all three fresh domains.
- The East Transition domain varies the most of all domains tested.
- Main Oxide was seen to vary considerably but between “very soft” and “soft”.

Figure 13.5 SMC (A x b) Variability by Domain Type



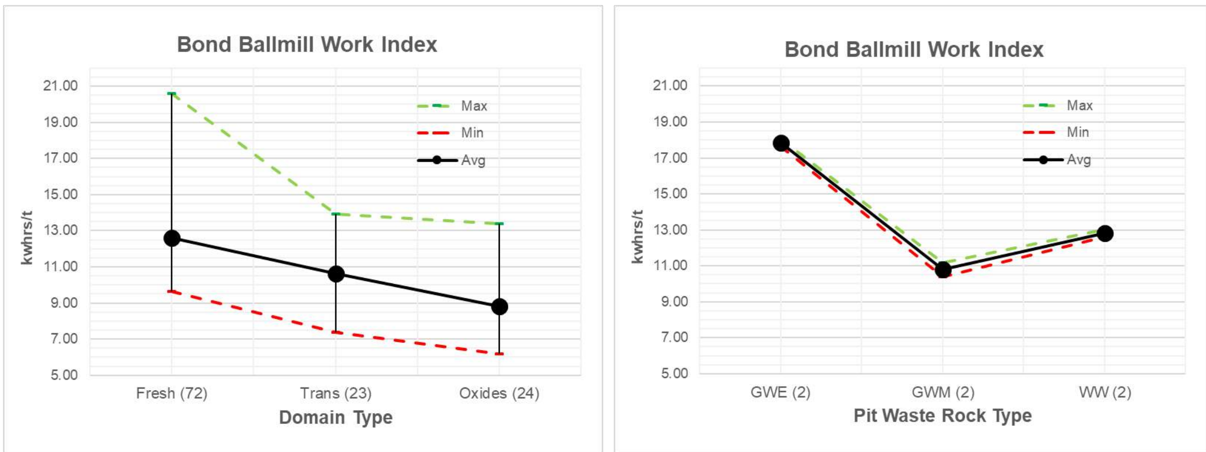
- Referring to Figure 13.6, the variability trends are broken down by Lithology for each of the major (>10% LOM Tonnage) domains which comprise 80% of the LOM tonnage. (EF=38%, MF=17%, ET=14%, EO=11%).
- East Trans FQSP and PGSS along with East Fresh PGSS varies the most when compared with the other major domains and lithologies.

Figure 13.6 SMC (A x b) Variability by Lithology



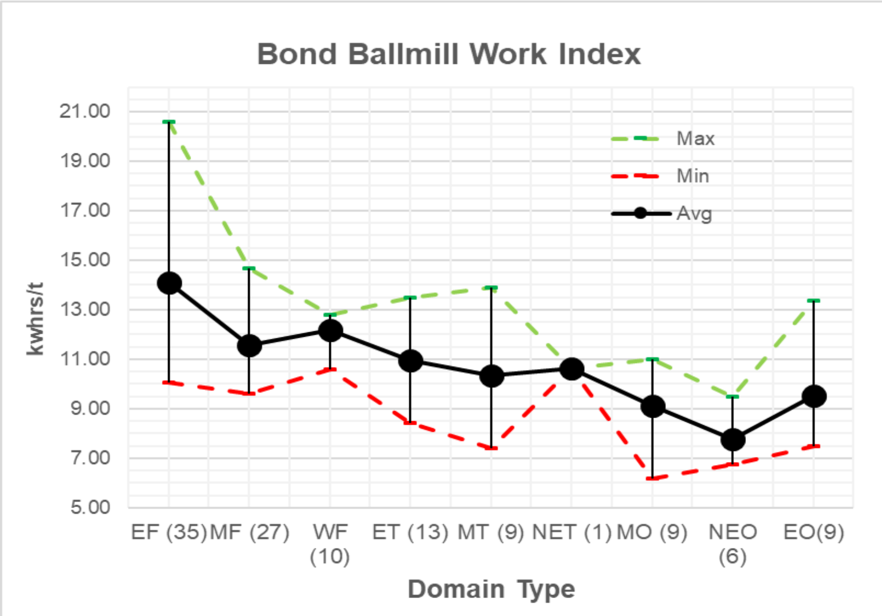
- Referring to Figure 13.7, the Bond Ballmill Work Index follows the same trends as the other comminution testing and is clearly hardest and varies the most in the fresh domains.
- The Bond Ballmill Work Index again reduces in hardness in transition and oxide domains.
- The waste rock samples tested in East pit are much harder than the waste rock samples in the Main or Wadi Doum Pits.

Figure 13.7 Ballmill Work Index by Domain (Mineralised Rocks and Waste Rocks)



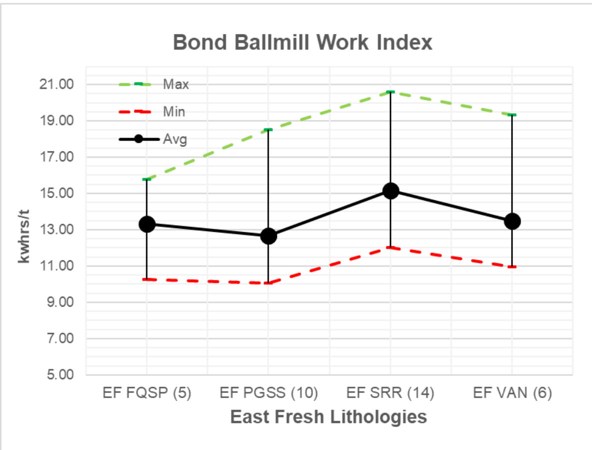
- Referring to Figure 13.8, the East Fresh Domain is the hardest and varies the most.
- The general trend is for ore hardness to be “Medium to Very Hard” for the fresh domains, “Medium” for transition domains and “Soft to Medium” in oxide domains.
- Wadi Doum Fresh and North East Oxide Domains very the least.

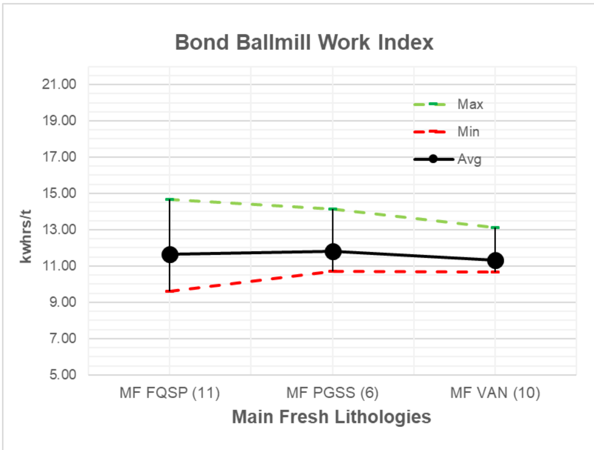
Figure 13.8 Bondmill Work Index Variability by Domain Type

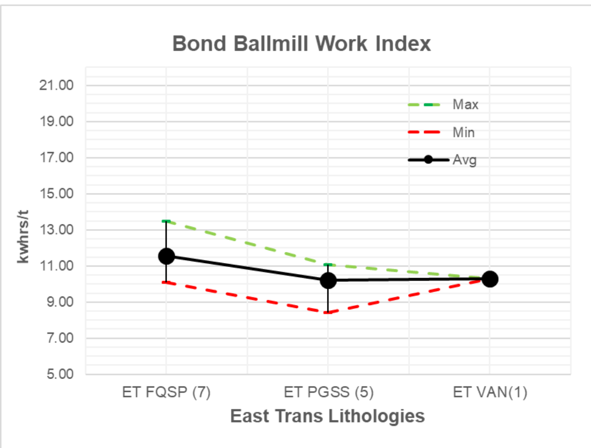


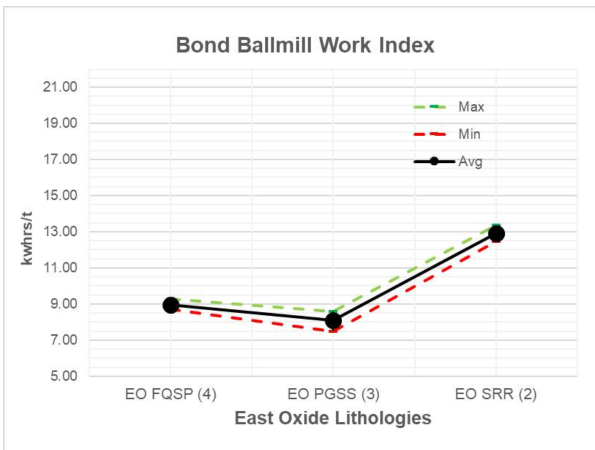
- Referring to Figure 13.9, the variability trends are broken down by Lithology for each of the major (>10% LOM Tonnage) domains which comprise 80% of the LOM tonnage. (EF=38%, MF=17%, ET=14%, EO=11%).
- East Fresh domain varies the most and whilst the average values for each East Fresh domain lithology are relatively similar, East Fresh SRR showed marginally greater resistance to ballmilling and greatest variability of all four lithologies.

Figure 13.9 Bond Mill Work Index Variability by Lithology









Conclusions

- The fresh domains and specifically East Fresh shows the most competent rock types and will drive comminution design capacity.
- East Fresh is by far the most variable domain.
- The SAG mill capacity will be sensitive to mining dilution.
- The Ballmill capacity will be sensitive to specifically the harder East Pit Mining dilution.

13.4.2 Cyanide Leach Variability Testing

The PFS Study cyanide leach variability data was designed to provide data to determine the overall economic optimum grind size for the Project, with consideration of milling costs, gold recovery and reagent consumption rates.

The optimal grinds are as follows:

- D_{80} 53 µm for all oxide domains and Wadi Doum Fresh.
- D_{80} 75 µm for all transition domains.
- D_{80} 90 µm for East and Main Fresh domains.

The above grind sizes were used in the follow-on Feasibility study using standard parameters as follows:

- pH Minimum 10.5.
- 48 hr Leach Residence time.
- Natural aeration, no additional pre-aeration step prior to cyanidation[1].
- 50% solids w/w.
- 0.5 g/l CN Concentration maintained throughout the leach.
- Site Water.

A total of 150 cyanide bottle roll tests have been performed to support the current feasibility study using the above conditions. A further 50 tests have been performed at elevated (55%) solids.

Ten tests were terminated after 32 hr to check on the terminal residue assays.

Table 13.4 summarises the number of samples tested in each domain and lithology and includes the tests that were used from the PFS to supplement the DFS data.

Table 13.4 Number of Samples Tested by Domain and Lithology

Pit	Domain	Phase	PEA		DFS		Total Study	
		% Solids	50 %	55%	50%	55%	50%	55%
		Lithology	# Sample	# Sample	# Sample	# Sample	# Sample	# Sample
East	Oxide	FQSP	-	-	5	2	5	2
East	Oxide	PGSS	-	-	5	3	5	3
East	Oxide	SRR	-	-	4	2	4	2
East	Oxide	VAN	-	-	4	2	4	2
East	**Oxide**	**Sub Total**	**-**	**-**	**18**	**9**	**18**	**9**
East	Trans	FQSP	-	-	10	3	10	3
East	Trans	PGSS	-	-	5	2	5	2
East	Trans	SRR	-	-	3	2	3	2

[1] *Note that the flowsheet development work described below shows that a pre-aeration step is beneficial. However, the standard bottle roll regime was NOT adjusted to allow for this so that all data in PFS and DFS is consistent.*

Pit	Domain	Phase	PEA		DFS		Total Study	
		% Solids	50 %	55%	50%	55%	50%	55%
		Lithology	# Sample	# Sample	# Sample	# Sample	# Sample	# Sample
East	Trans	VAN	-	-	5	1	5	1
East	**Trans**	**Sub Total**	**-**	**-**	**23**	**8**	**23**	**8**
East	Fresh	FQSP	1	-	7	3	8	3
East	Fresh	PGSS	-	-	15	7	15	7
East	Fresh	SRR	-	-	14	3	14	3
East	Fresh	VAN	-	-	7	2	7	2
East	**Fresh**	**Sub Total**	**1**	**-**	**43**	**15**	**44**	**15**
Main	Oxide	FQSP	-	-	4	-	4	-
Main	Oxide	PGSS	2	-	2	-	4	-
Main	Oxide	VAN	-	-	3	-	3	-
Main	**Oxide**	**Sub Total**	**2**	**-**	**9**	**-**	**11**	**-**
Main	Trans	FQSP	1	-	5	6	6	6
Main	Trans	PGSS	-	-	4	-	4	-
Main	Trans	VAN	-	-	4	-	4	-
Main	**Trans**	**Sub Total**	**1**	**-**	**13**	**6**	**14**	**6**
Main	Fresh	FQSP	-	-	12	7	12	7
Main	Fresh	PGSS	1	-	8	-	9	-
Main	Fresh	VAN	-	-	4	-	4	-
Main	**Fresh**	**Sub Total**	**1**	**-**	**24**	**7**	**25**	**7**
NEZ	Oxide	FQSP	-	-	1	-	1	-
NEZ	Oxide	PGSS	1	-	2	-	3	-
NEZ	Oxide	VAN	-	-	-	-	-	-
NEZ	**Oxide**	**Sub Total**	**1**	**-**	**3**	**-**	**4**	**-**
NEZ	Trans	FQSP	-	-	-	-	-	-
NEZ	Trans	PGSS	-	-	1	-	1	-
NEZ	Trans	VAN	-	-	-	-	-	-
NEZ	**Trans**	**Sub Total**	**-**	**-**	**1**	**-**	**1**	**-**
Wadi	Oxide	QPR	-	-	-	-	-	-
Wadi	Oxide	VAN	-	-	-	-	-	-
Wadi	Oxide	VC1	-	-	2	-	2	-
Wadi	Oxide	VC3	-	-	-	-	-	-
Wadi	**Oxide**	**Sub Total**	**-**	**-**	**2**	**-**	**2**	**-**
Wadi	Trans	QPR	-	-	-	-	-	-
Wadi	Trans	VAN	-	-	-	-	-	-
Wadi	Trans	VC1	-	-	2	-	2	-
Wadi	Trans	VC3	-	-	-	-	-	-
Wadi	**Trans**	**Sub Total**	**-**	**-**	**2**	**-**	**2**	**-**
Wadi	Fresh	QPR	-	-	2	-	2	-
Wadi	Fresh	VAN	-	-	-	-	-	-
Wadi	Fresh	VC1	-	-	3	1	3	1
Wadi	Fresh	VC3	1	-	-	4	1	4
Wadi	**Fresh**	**Sub Total**	**1**	**-**	**5**	**5**	**6**	**5**
	Total		**7**	**-**	**143**	**50**	**150**	**50**

The main objectives were as follows:

- Optimise the Leach Residence time.
- Increase the understanding of the whole ore cyanide leach flowsheet metal extraction rates (Au, Ag, Cu and Hg) and reagent consumption rates (Cyanide and Lime) to provide greater accuracy on the overall metal revenue projections as well as the process consumable operating costs.
- Verify any potential savings in cyanide consumption rates using the pre-aeration stage.

Table 13.5 summarises the carbon stage recoveries that were measured during carbon modelling and the assumed losses to carbon fines, and Taable 13.6 summarises the metal recovery rates and reagent consumptions after 48 hr leach residence time. The gold recoveries are adjusted using regression analysis to 45 hr since all bottle roll testing employed gold kinetic data whereas Ag, Cu and Hg used terminal 48 hr leach data.

The data below has been adjusted to allow for the carbon adsorption stage recoveries measured, gold and silver losses to carbon fines were assumed to be 0.15%.

Table 13.5 Adsorption Stage Recoveries and Carbon Fines Losses

Losses	%	Comments
Carbon Fines Metal Loss, %	0.15	MPH Assumed
ADR Au Efficiency, %	99.6	SGS Carbon Modelling – All Ore Types
ADR Ag Oxide Efficiency, %	27.1	Yr2 HO – SGS Carbon Isotherm
ADR AgTrans Efficiency, %	51.0	Yr6 HT – SGS Carbon Isotherm
ADR Ag Fresh Efficiency, %	67.3	Yr10 HF – SGS Carbon Isotherm
ADR Cu Oxide Efficiency, %	2.3	Y2HO-CM-CN3R2
ADR Cu Trans Efficiency, %	1.3	Y10 HF-CM-CN3R
ADR Cu Fresh Efficiency, %	10.0	Y10 HF-CM-CN3R
ADR Hg Oxide Efficiency, %	8.9	Y2 HO-CM-CN3R
ADR Hg Trans Efficiency, %	8.1	Y6 HT-CM-CN3R
ADR Hg Fresh Efficiency, %	11.5	Y10 HF-CM-CN3R

Table 13.6 Summary of Metal Recoveries by Domain and Lithology

Pit	Domain	Lithology	Tonnes (Millions)	Ozs Au (000's)	45 hr Au Recovery, %	48 hr Ag Recovery, %
Main	Oxide	FQSP	1.1	72	89.3	10.5
		PGSS	1.4	49	91.8	7.4
		VAN	1.8	56	85.1	9.4
	Trans	FQSP	0.9	50	81.2	22.2
		PGSS	2.1	74	82.7	26.3
		VAN	2.0	71	83.9	11.9
	Fresh	FQSP	3.0	174	71.2	43.1
		PGSS	6.4	230	79.9	43.0
		VAN	4.1	167	81.7	47.8
East	Oxide	FQSP	0.8	38	86.5	9.4
		PGSS	5.3	136	93.3	7.3
		SRR	0.5	22	90.0	10.5
		VAN	1.8	43	82.9	9.3
	Trans	FQSP	0.9	44	82.9	29.2
		PGSS	5.3	148	81.9	26.9
		SRR	0.5	21	85.6	30.2
		VAN	4.5	110	90.0	24.1
	Fresh	FQSP	5.2	222	76.6	43.6
		PGSS	16.3	497	83.8	35.1
		SRR	2.6	127	77.0	43.9
		VAN	6.7	194	81.6	34.4
NEZ	Oxide	FQSP	0.1	4	95.9	7.0
		PGSS	1.4	35	91.0	5.8
		VAN	0.2	4	82.9	9.3
	Trans	FQSP	0.1	4	82.9	34.7
		PGSS	1.6	43	81.6	34.7
		VAN	0.5	12	90.0	34.7
	Fresh	FQSP	0.0	0	76.6	43.6
		PGSS	0.2	4	83.8	35.1
		VAN	0.2	6	81.6	34.4

Pit	Domain	Lithology	Tonnes (Millions)	Ozs Au (000's)	45 hr Au Recovery, %	48 hr Ag Recovery, %
Wadi Doum	Oxide	QPR	0.3	12	87.2	1.3
		VAN	0.1	4	87.2	1.3
		VC1	0.0	5	87.2	1.3
		VC3	0.1	11	87.2	1.3
	Trans	QPR	0.0	3	84.7	22.0
		VAN	0.0	1	84.7	22.0
		VC1	0.0	2	84.7	22.0
		VC3	0.0	3	84.7	22.0
	Fresh	QPR	0.8	46	77.8	32.0
		VAN	0.3	27	80.8	37.6
		VC1	0.2	14	83.7	41.3
		VC3	0.7	69	65.8	39.4
LOM Weighted Average			**79.9**	**2,853**	**82.0**	**32.2**

Note that the highlighted cells were not tested due to their insiginificance tonnage.

The following assumptions were used wherever actual data was not available:

- **North East Lithological data was the same as the same East Lithological data**
- **Wadi Oxides and Transition QPR and VC3 data was the same as VC1 data**
- **Wadi Fresh VAN lithology was the same as the arithmetic average of QPR and VC3.**

- The LOM weighted (ozs Au) Gold Dissolution is 82.0%:
  - North East Oxide FQSP and East Oxide PGSS have the highest gold dissolution rates at 95.9% and 93.3% respectively
  - Wadi Doum Fresh VC3 and Main Fresh FQSP have the lowest gold dissolution rates at 65.8% and 71.2% respectively
  - further discussion on gold dissolution variance is detailed below.
- The LOM weighted average (tonnes) Silver Recovery is 32.2% and further discussion on silver dissolution variance is detailed below.
- The LOM weighted average (tonnes) Copper Recovery is 1.1%:
  - the copper dissolution in Main Transition PGSS was the highest copper dissolution observed and this is supported by elevated slightly elevated copper in the pregnant solution and slightly elevated cyanide consumption
  - excluding Main Transition PGSS, Copper dissolution varied significantly the very low adsorption onto carbon reduces the overall recovery.
- The LOM weighted average (tonnes) Mercury Recovery is 3.0%:
  - mercury dissolution varied significantly but are generally low (2 - 49%).

Gold and Silver Metal Dissolution Variability Trends

Figure 13.10 to Figure 13.15 summarises the 48 hr gold and silver dissolution variability (Max., Min. and Average) for each Lithology in each domain. The number in brackets adjacent to the Lithological code on the y axis refers to the number of samples that were tested.

Figure 13.10 Summary of East Pit 48 hr Gold Leach Dissolution by Domain and Lithology

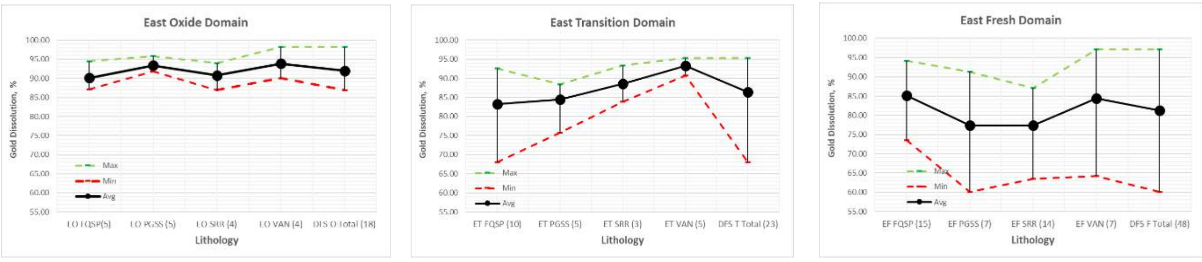


Figure 13.11 Summary of Main Pit 48 hr Gold Leach Dissolution by Domain and Lithology

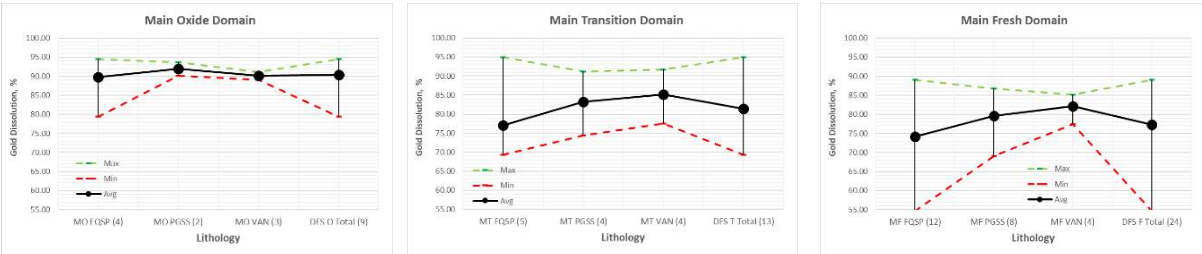
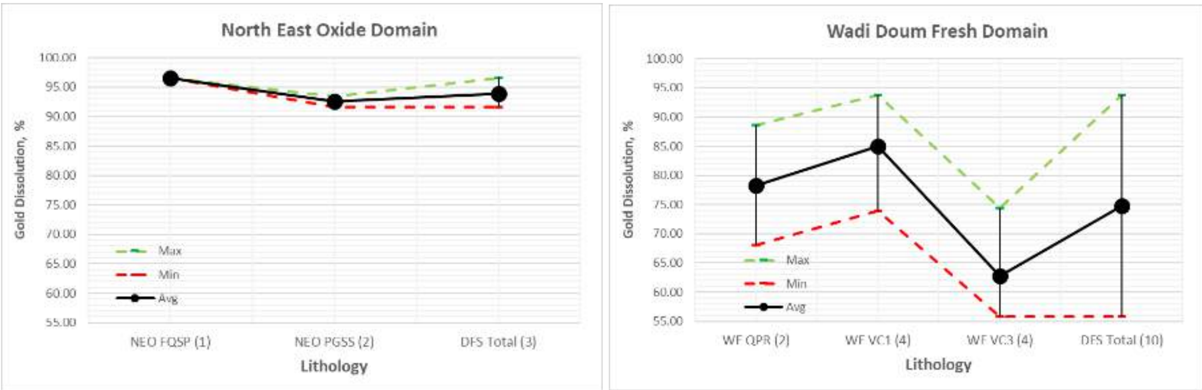


Figure 13.12 Summary of Wadi Doum & North East Pit 48 hr Gold Leach Dissolution by Domain and Lithology



Referring to Figure 13.10 to Figure 13.12:

- The gold dissolution trend graphs show clearly how the oxide domains all leach well and all are relatively low in variability with exception of Main Oxide FQSP which has medium variability.
- Main and East Transition FQSP both exhibit high variability in gold recovery.
- All Fresh domains exhibit consistent high variability across all lithologies tested with exception of Main Fresh VAN.

Figure 13.13 Summary of East Pit 48 hr Silver Leach Dissolution by Domain and Lithology

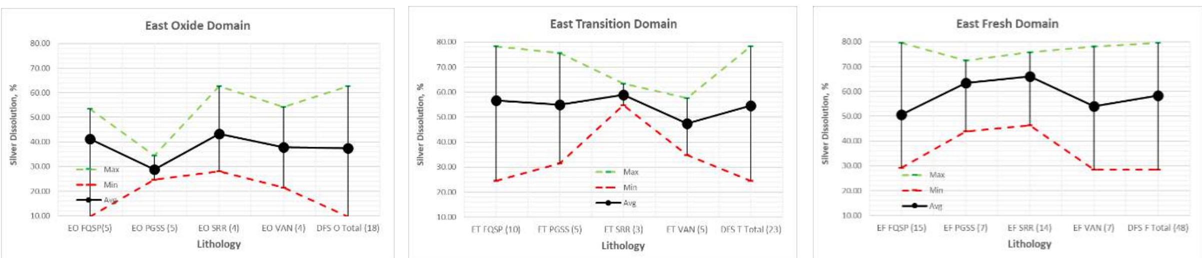


Figure 13.14 Summary of Main Pit 48 hr Silver Leach Dissolution by Domain and Lithology

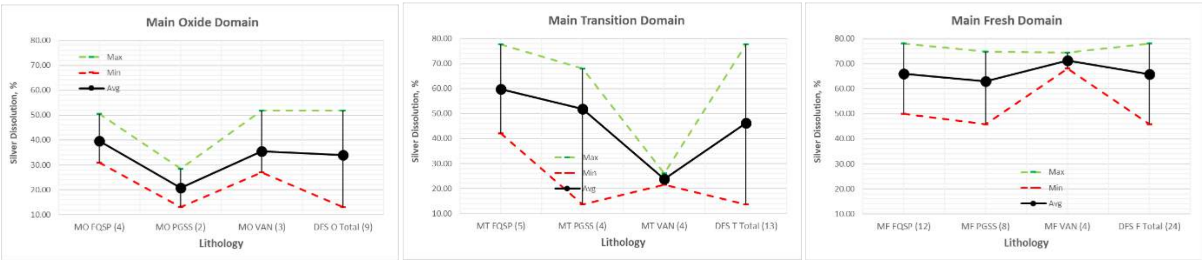
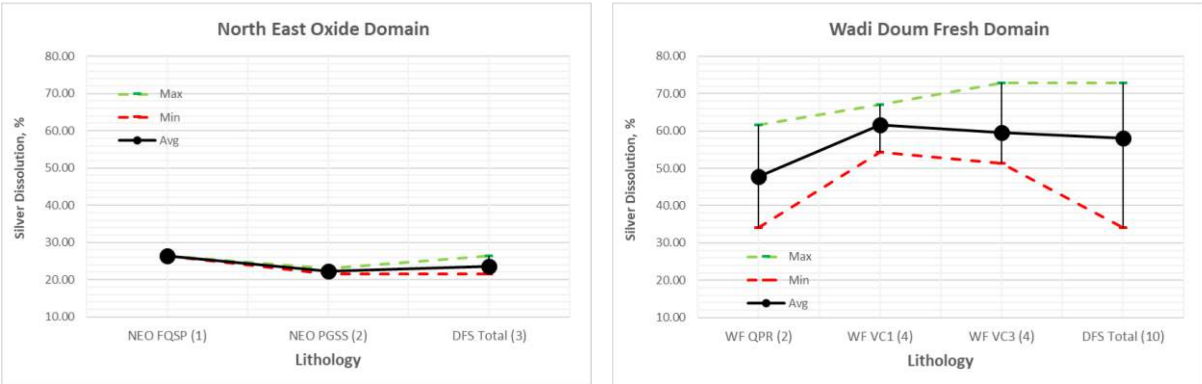


Figure 13.15 Summary of Wadi Doum & North East Pit 48 hr Silver Leach Dissolution by Domain and Lithology



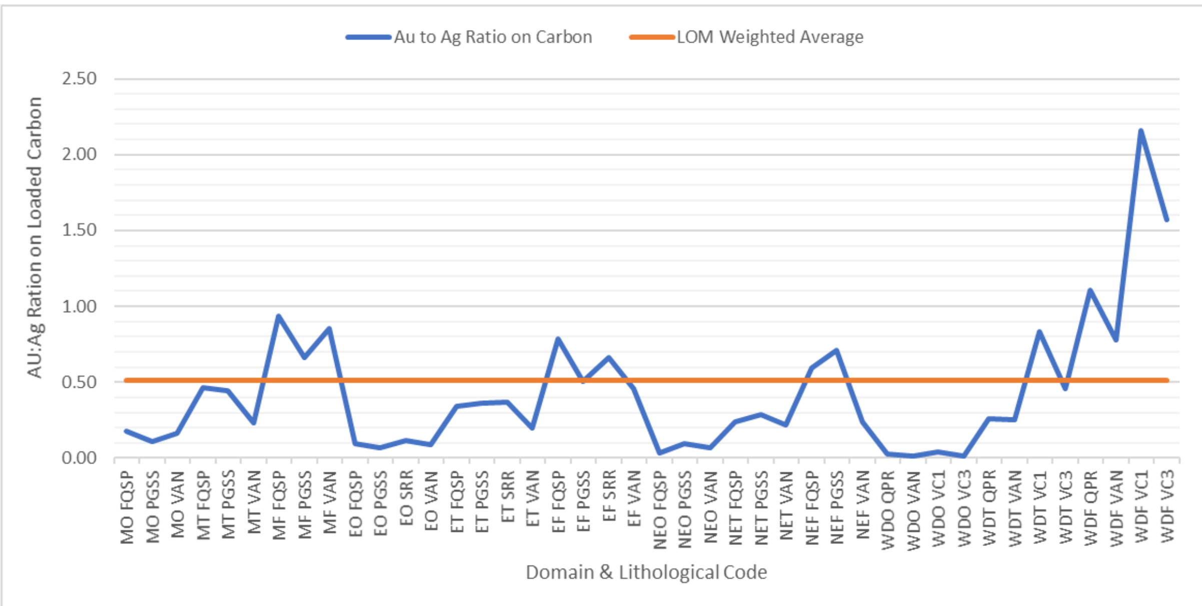
Referring to Figure 13.13 to Figure 13.15:

- The silver dissolution rates are all much lower than gold and tend to vary much more than the gold dissolution variance.
- North East Oxide is by far the most consistent silver leach dissolution of all those tested.

Silver to Gold Ratio

Figure 13.16 below summarises the Au:Ag ratio of the loaded carbon reporting to the elution circuit.

Figure 13.16 Summary of Gold to Silver Reporting to Elution Circuit by Lithology



Referring to Figure 13.16 the following conclusions can be made:

- A general trend for oxide domains to have the lowest ratio due to the poor silver stage adsorption efficiency. Transition domains have a higher ratio and fresh domains have the highest ratio.
- Wadi Doum Fresh and Transition VC1 domains have the highest ratio due to the much higher silver grades in these two domains.

Diagnostic Leach Tests

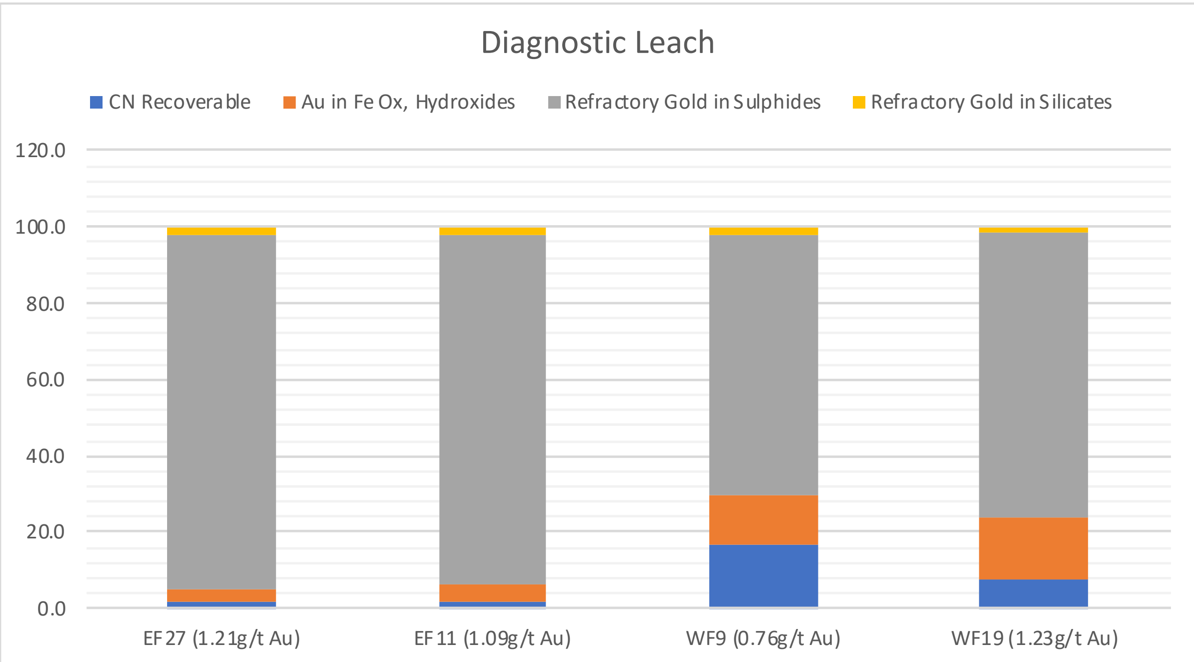
Table 13.7 summarises the diagnostic leach testing conducted on the four lowest leach dissolution bottle roll tests, focused mainly in East and Wadi Doum Fresh domains.

Figure 13.17 summarises the diagnostic leach trends.

Table 13.7 Summary of Diagnostic Leach Tests

Description	EF27 (1.21 g/t Au)		EF11 (1.09 g/t Au)		WF9 (0.76 g/t Au)		WF19 (1.23 g/t Au)	
	Overall	Diag. Leach	Overall	Diag. Leach	Overall	Diag. Leach	Overall	Diag. Leach
Bottle Roll Recovery	57.4	-	62.6	-	57.9	-	55.8	-
Bottle Roll Solids Residue	42.6	-	37.4	-	42.1	-	44.2	-
CN Recoverable	0.7	1.6	0.7	2.0	7.1	16.8	3.4	7.8
Au in Fe Ox, Hydroxides	1.4	3.4	1.5	4.1	5.4	12.9	7.1	16.0
Refractory Gold in Sulphides	39.7	93.2	34.3	91.8	28.7	68.1	33.1	74.8
Refractory Gold in Silicates	0.8	1.8	0.8	2.1	0.9	2.2	0.6	1.4
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

Figure 13.17 Diagnostic Leach Trends



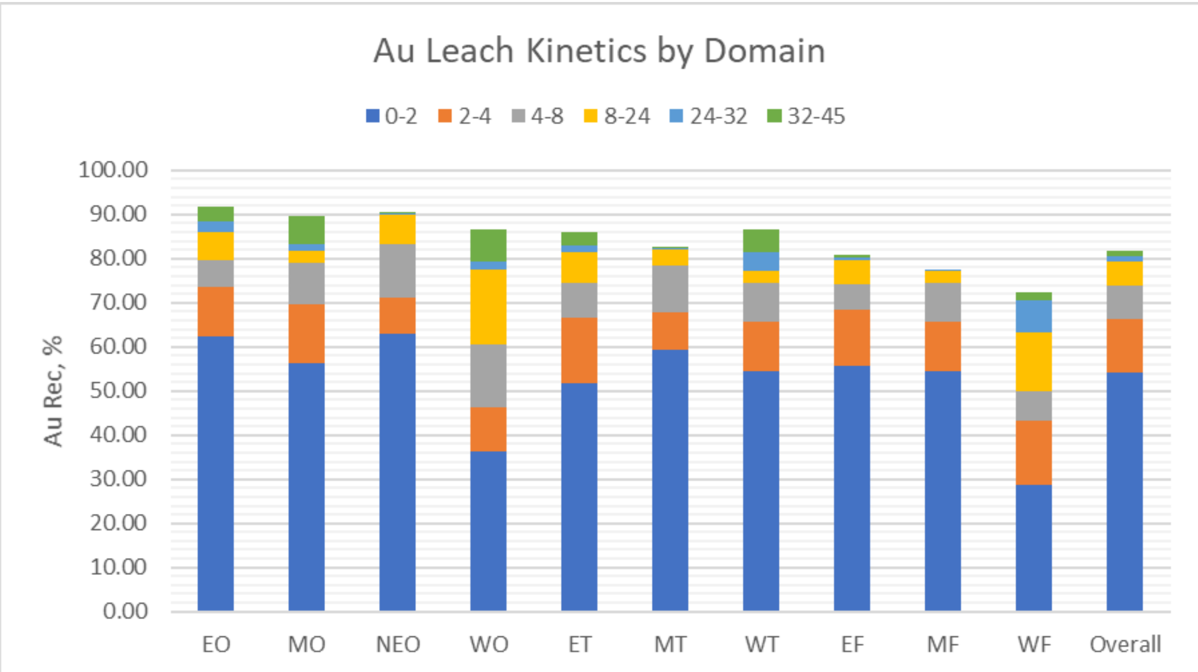
Referring to Table 13.7 and Figure 13.17, the following conclusions can be made:

- The low gold dissolutions observed in the two samples tested from East Fresh are almost all associated with gold encapsulation with sulphides:
  - 34 - 40% of the gold in these samples is refractory due to sulphide encapsulation.
- The low gold dissolutions observed in the two samples tested from Wadi Doum Fresh are mainly associated with gold encapsulation with sulphides, but these samples differ from East Fresh in that more gold is also encapsulated by Iron Oxides / Hydroxides. There is also more gold that leaches in intensive cyanide conditions:
  - 8 – 17% of the gold was still available for cyanide leaching
  - 20 - 33% of the gold in these samples is refractory due to sulphide encapsulation
  - 5 - 7% of the gold in these samples is refractory due to sulphide encapsulation.

Cyanide Leach Residence Time Optimisation

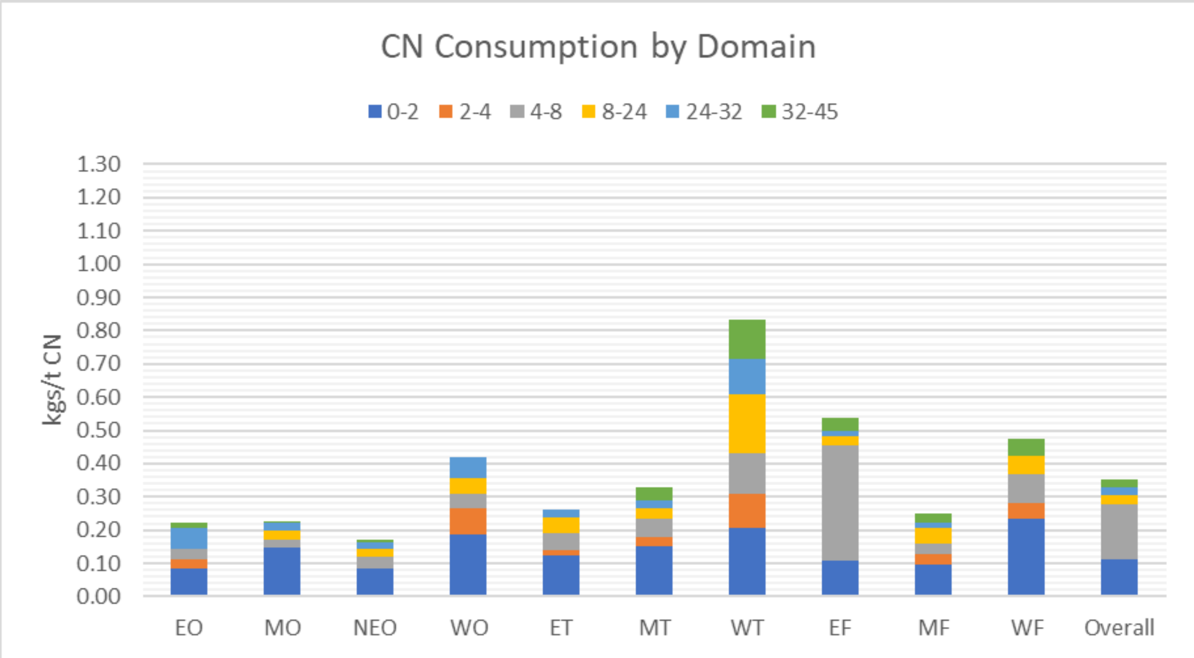
Figure 13.18 summarises the leach kinetic profiles for each domain, initially in absolute gold dissolution terms, secondly as a relative percent of total recovery and thirdly as a function of the overall mine sequence.

Figure 13.18 Summary of Au Leach Kinetics by Domain



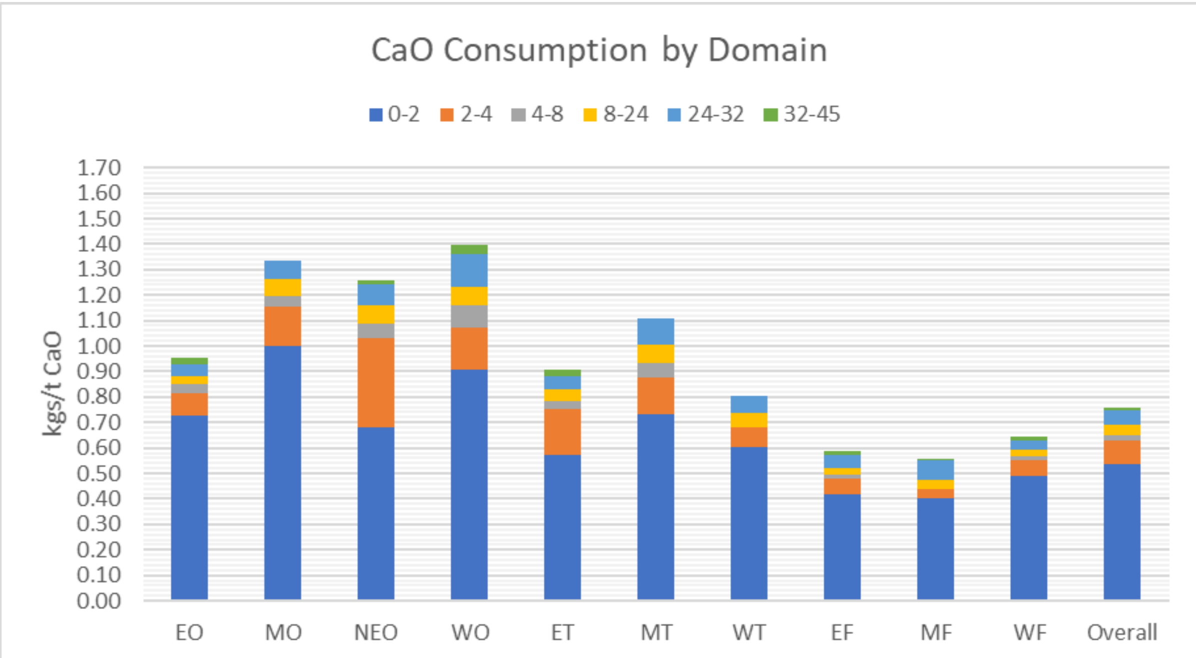
- The majority of gold dissolution occurs in the first 24 hr (specifically the first 2 hr) but some domains are slower leaching.
- Wadi Doum Fresh (WDF) is the slowest leaching domain of all the samples tested and this is potentially a function of the inherently higher head grades in this domain.
- North East Oxide is the fastest leaching domain.

Figure 13.19 Summary of Cyanide Consumption Rates vs. Leach Time



- The majority of cyanide consumption occurs in the first 24 hr (specifically the first 2 hr) but some domains continue to leach and consume cyanide after 24 hr.
- North East Oxide is the lowest cyanide consumer at 0.19 kg/t.

Figure 13.20 Summary of Lime Consumption Rates vs. Leach Time



- The majority of lime consumption occurs in the first 24 hr (specifically the first 2 hr) but some domains continue to leach and consume lime after 24 hr.
- As a general trend, the lime consumption is highest in the oxide domains, lower in the transition domains and lowest in the fresh domains.

Using the data generated, the incremental revenues and operating costs were estimated to evaluate the optimal leach residence time using the following assumptions:

- Gold Price US$1,200 /oz.
- Cyanide Price delivered site US$2,210 /t.
- Incremental Operating Cost per tank US$0.06 /t.
- Incremental Capital Cost per tank US$1.8M.

The plant design is based upon 45 hr leaching time followed by carbon adsorption where leaching will continue. The gold recovery and reagent consumptions are based upon 45 hr extrapolated kinetic data. The silver metal recoveries utilise 48 hr data as kinetic data is not available.

Cyanide Leach Reagent Consumptions

Table 13.8 summarises the cyanide and lime reagent consumptions after 48 hr residence time.

Table 13.8 Summary of Reagent Consumptions by Domain and Lithology

Pit	Domain	Lithology	Tonnes (Millions)	45 hr CN (kg/t)	45 hr CaO (kg/t)
Main	Oxide	FQSP	1.1	0.24	1.35
		PGSS	1.4	0.24	1.30
		VAN	1.8	0.20	1.40
	Trans	FQSP	0.9	0.40	0.75
		PGSS	2.1	0.39	1.08
		VAN	2.0	0.15	1.11
	Fresh	FQSP	3.0	0.21	0.78
		PGSS	6.4	0.18	0.76
		VAN	4.1	0.16	0.60
East	Oxide	FQSP	0.8	0.21	0.85
		PGSS	5.3	0.18	0.98
		SRR	0.5	0.22	0.73
		VAN	1.8	0.21	0.93
	Trans	FQSP	0.9	0.29	1.04
		PGSS	5.3	0.26	0.83
		SRR	0.5	0.19	1.06
		VAN	4.5	0.20	0.95
	Fresh	FQSP	5.2	0.23	0.62
		PGSS	16.3	0.21	0.66
		SRR	2.6	0.26	0.62
		VAN	6.7	0.25	0.66
NEZ	Oxide	FQSP	0.1	0.16	1.45
		PGSS	1.4	0.17	1.25
		VAN	0.2	0.2	0.93
	Trans	FQSP	0.1	0.3	0.94
		PGSS	1.6	0.3	0.94
		VAN	0.5	0.2	0.94
	Fresh	FQSP	0.0	0.3	0.56
		PGSS	0.2	0.2	0.60
		VAN	0.2	0.3	0.60

Pit	Domain	Lithology	Tonnes (Millions)	45 hr CN (kg/t)	45 hr CaO (kg/t)
Wadi Doum	Oxide	QPR	0.3	0.6	1.57
		VAN	0.1	0.6	1.57
		VC1	0.0	0.6	1.57
		VC3	0.1	0.6	1.57
	Trans	QPR	0.0	1.2	0.75
		VAN	0.0	1.2	0.75
		VC1	0.0	1.2	0.75
		VC3	0.0	1.2	0.75
	Fresh	QPR	0.8	0.4	0.58
		VAN	0.3	0.4	0.69
		VC1	0.2	0.49	0.83
		VC3	0.7	0.50	0.66
LOM Weighted Average			**79.9**	**0.23**	**0.81**

Note that the salmon coloured consumption rates have been adjusted by a derating factor caused by the pre-aeration stage. Refer to table 13.9 below. Note that the mustard coloured highlighted cells were not tested due to their insiginificance tonnage.

The following assumptions were used wherever actual data was not available:

- **North East Lithological data was the same as the same East Lithological data**
- **Wadi Oxides and Transition QPR and VC3 data was the same as VC1 data**
- **Wadi Fresh VAN lithology was the same as the arithmetic average of QPR and VC3.**

- The LOM Weighted Average (tonnes) Cyanide Consumption is 0.23 kg/t, which is relatively low:
  - North East Oxide FQSP, PGSS, East Oxide PGSS and Main Transition VAN all had very low cyanide consumption rates at <0.20 kg/t cyanide consumption rates
  - Wadi Doum Transition VC1 had the highest cyanide consumption of all domains and was relatively unique at 1.22 kg/t. The LOM Weighted Average (tonnes) lime consumption is 0.81 kg/t.

Cyanide and Lime Consumption Variability Trends

The 48 hr lime and cyanide consumption rates have been determined using extrapolation of the average reagent consumption cumulative test data by leach time for each lithology.

Figure 13.21 to Figure 13.26 summarises the 48 hr cyanide and lime consumption variability (Max., Min. and Average) for each lithology in each domain. The number in brackets adjacent to the lithological code on the y axis refers to the number of samples that were tested.

Figure 13.21 Summary of East Pit 48 hr Cyanide Consumption by Domain and Lithology

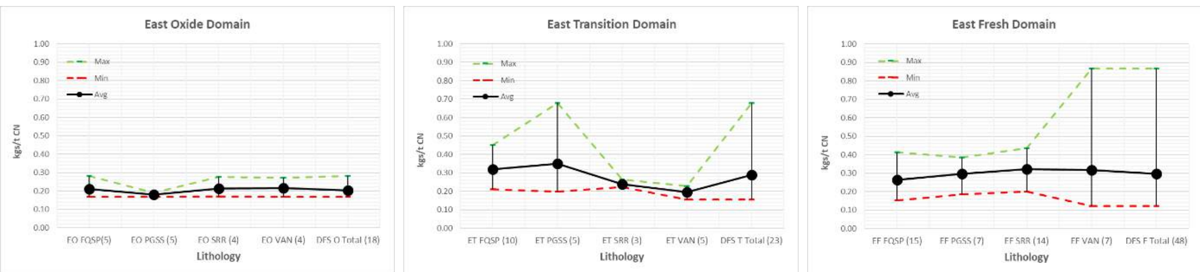


Figure 13.22 Summary of Main Pit 48 hr Cyanide Consumption by Domain and Lithology

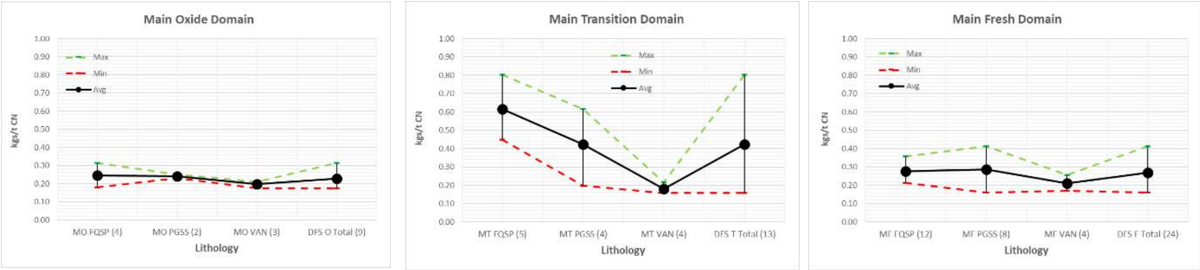
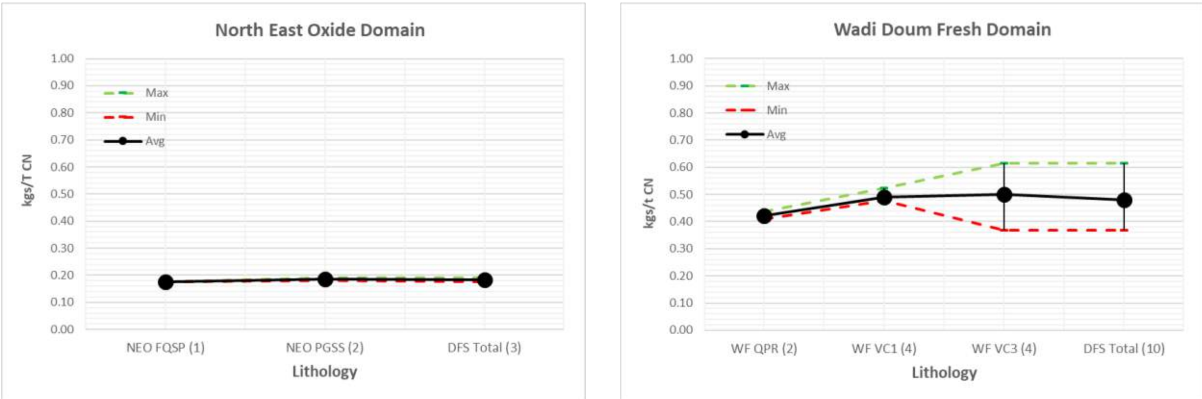


Figure 13.23 Summary of Wadi Doum and North East Pit 48 hr Cyanide Consumption by Domain and Lithology



Referring to Figure 13.21 to Figure 13.23 the following trends are observed:

- The cyanide consumptions for all oxide domains tends to be low and very consistent with low variability.
- East Transition domain has lower cyanide consumption rates than Main transition domain and both tend to show more variance in the FQSP and PGSS lithologies.

Figure 13.24 Summary of East Pit 48 hr Lime Consumption by Domain and Lithology

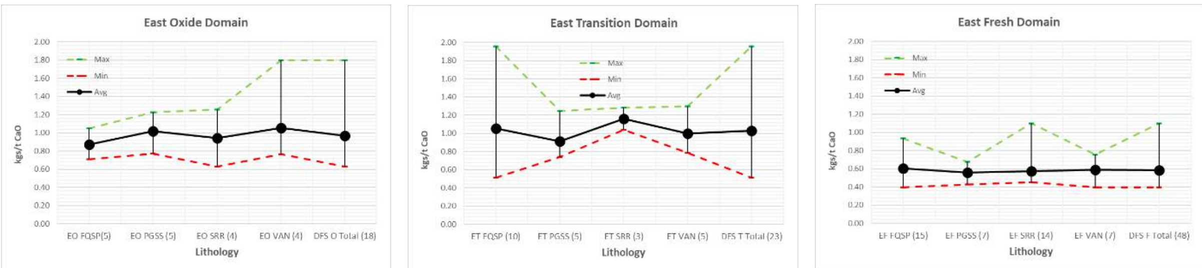


Figure 13.25 Summary of Main Pit 48 hr Lime Consumption by Domain and Lithology

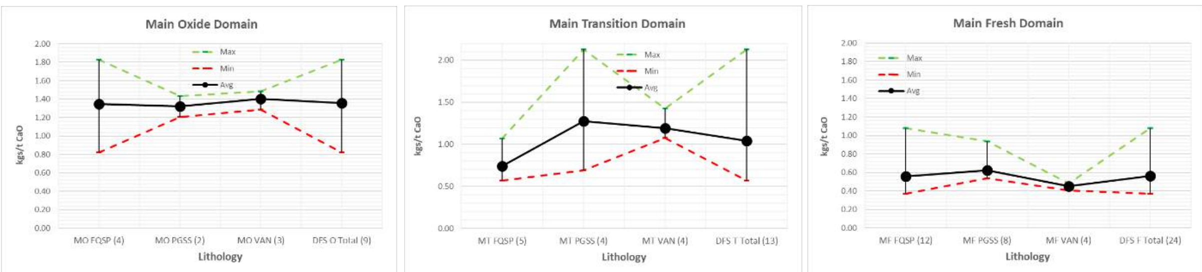
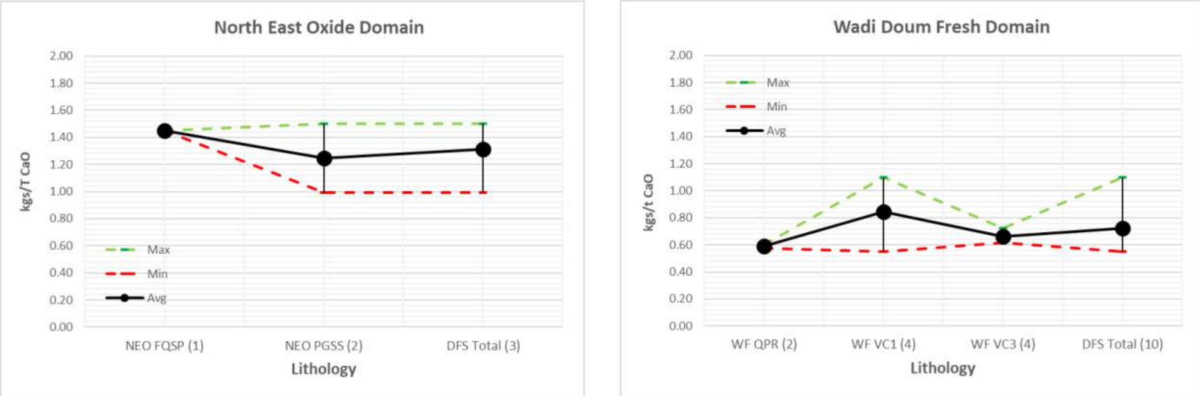


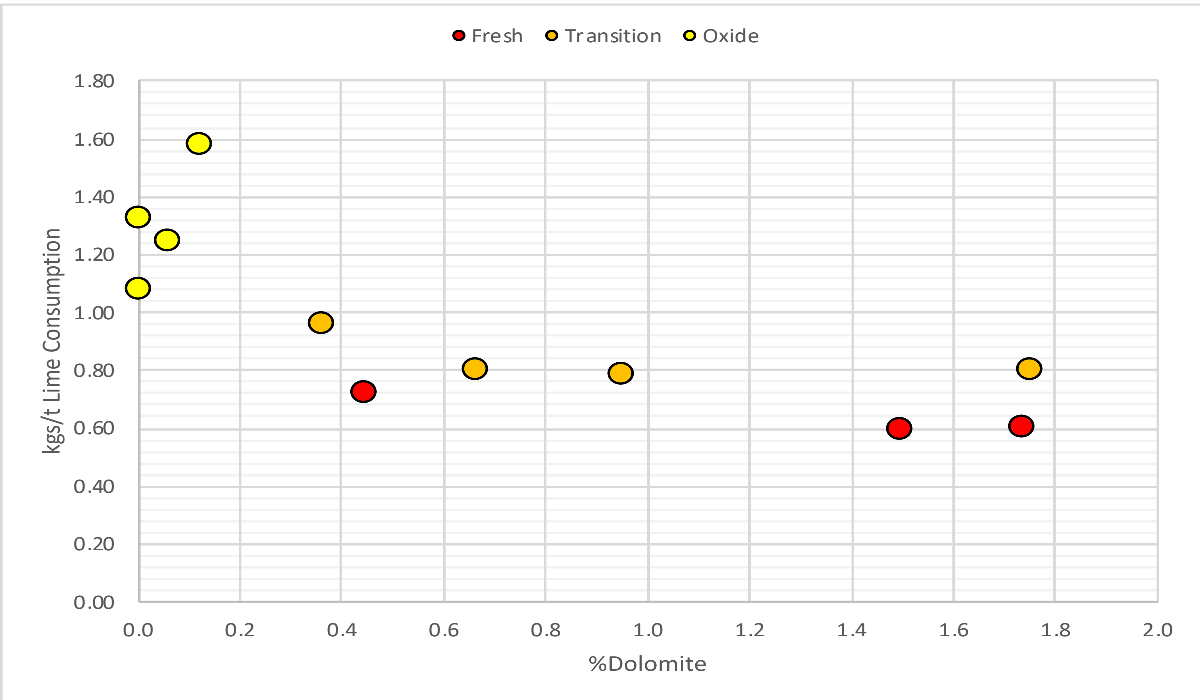
Figure 13.26 Summary of Wadi Doum and North East Pit 45 hr Lime Consumption by Domain and Lithology



Referring to Figure 13.24 to Figure 13.26, the following conclusions can be made:

- The general trend is for lime consumptions to be highest in oxide and transition domains and are lowest in fresh.
- Referring to Figure 13.27, the reason lime consumption is higher in oxide domains could well be due to much lower dolomite content in these domains.

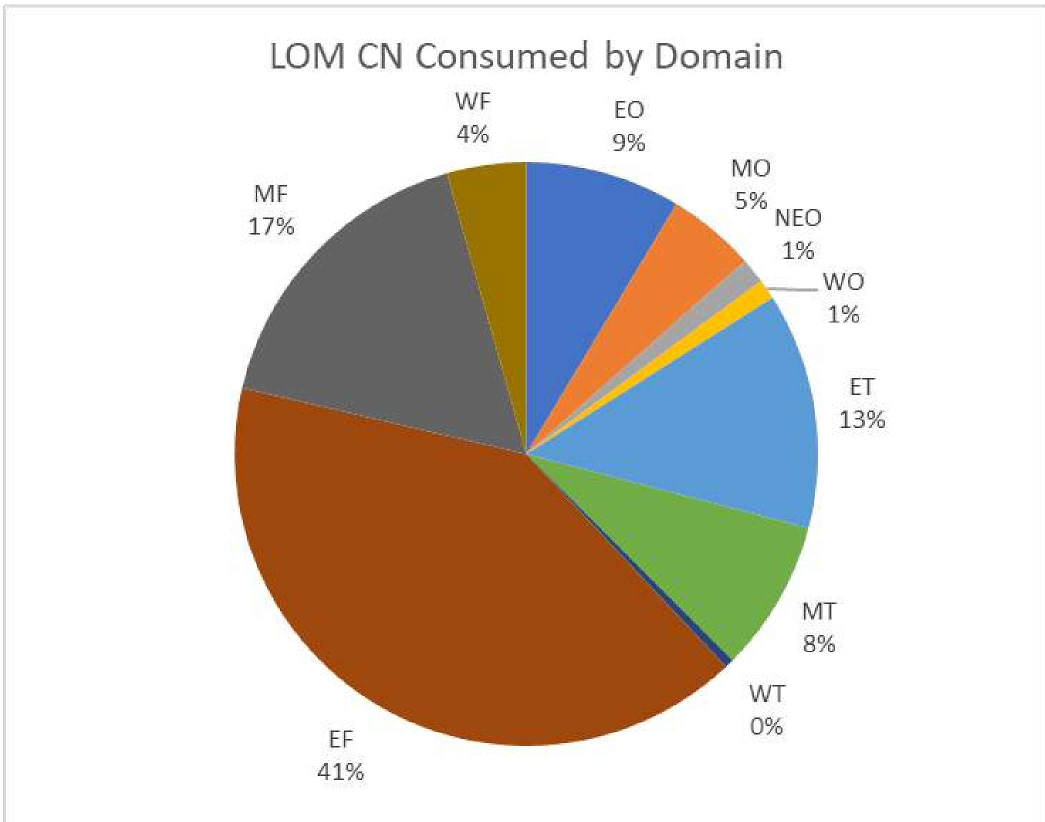
Figure 13.27 Lime Consumption vs. % Dolomite



Cyanide Consumption Analysis

Figure 13.28 summarises the breakdown of cyanide consumption by domain.

Figure 13.28 Breakdown of Cyanide Consumption by Domain

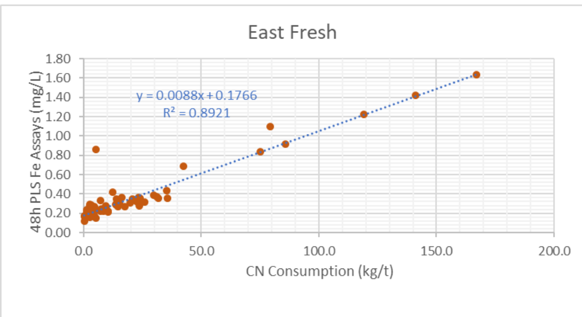


Referring to Figure 13.28 the main cyanide consuming domains over the life of mine are

- East Fresh 41%.
- Main Fresh 17%.
- East Transition 13%.
- East Oxide 9%.
- Main Transition 8%.

Figures 13.29 to Figure 13.35 inclusive show the relationship between 48 hr pregnant solution mg/l Fe, Cu and Fe + Cu versus Cyanide Consumption rates.

Figure 13.29 Cyanide Consumption Analysis of East Fresh Domain



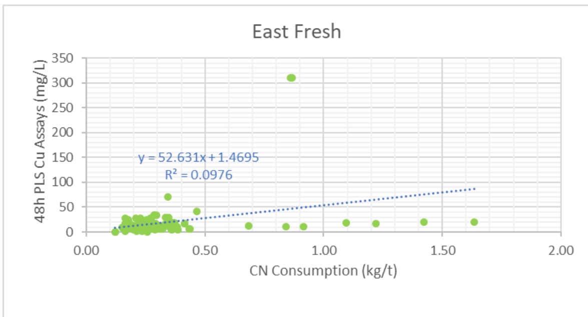
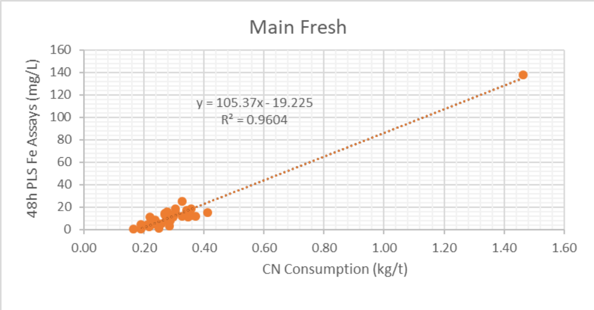


Figure 13.30 Cyanide Consumption Analysis of Main Fresh Domain



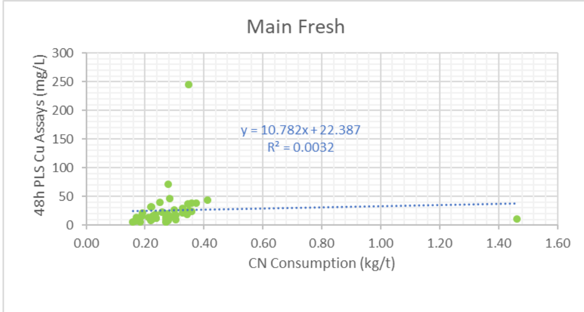
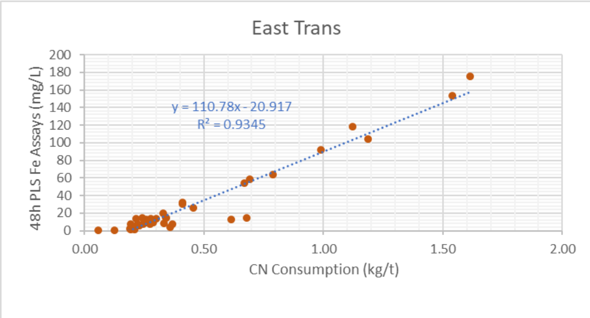


Figure 13.31 Cyanide Consumption Analysis of East Transition Domain



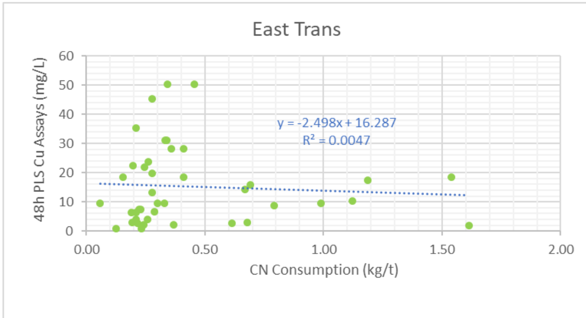
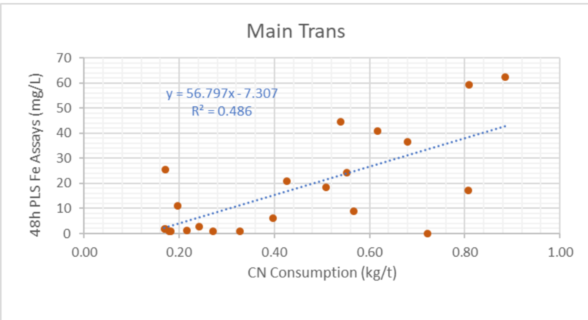


Figure 13.32 Cyanide Consumption Analysis of Main Transition Domain



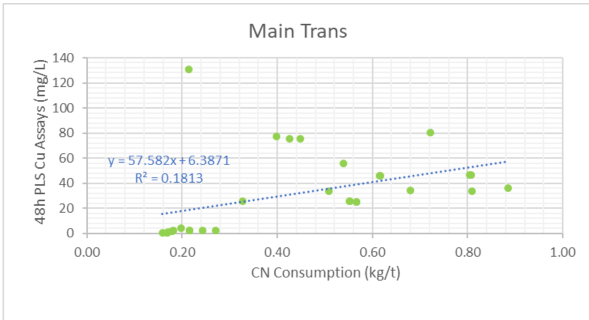
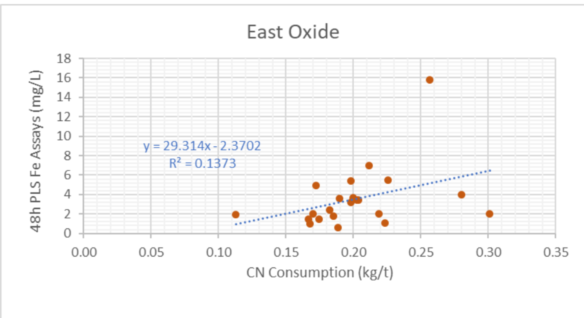


Figure 13.33 Cyanide Consumption Analysis of East Oxide Domain



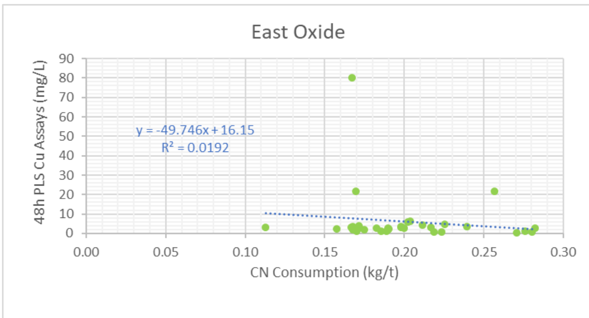


Figure 13.34 Cyanide Consumption Analysis of Wadi Doum Domain

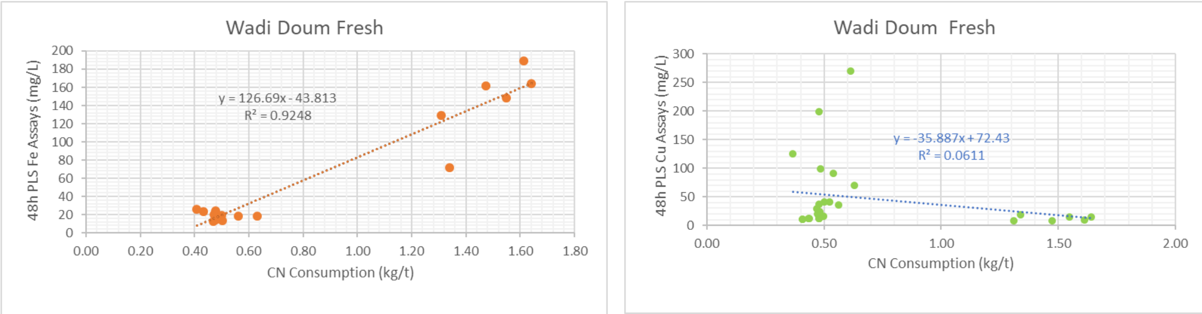
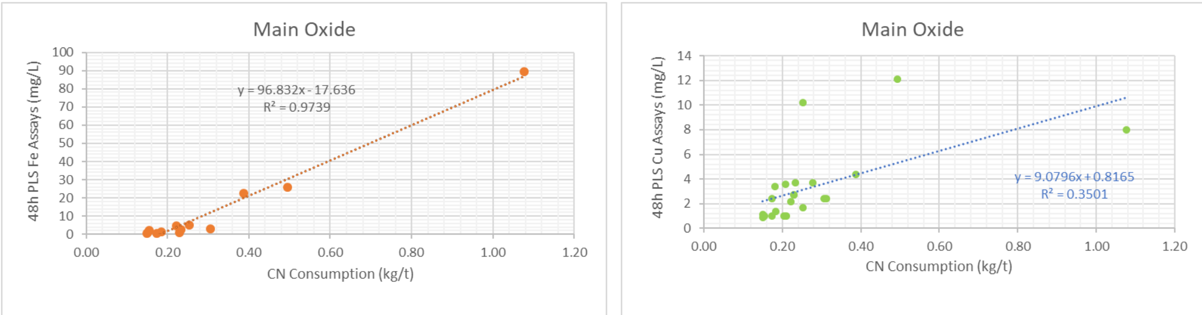


Figure 13.35 Cyanide Consumption Analysis of Main Oxide Domain



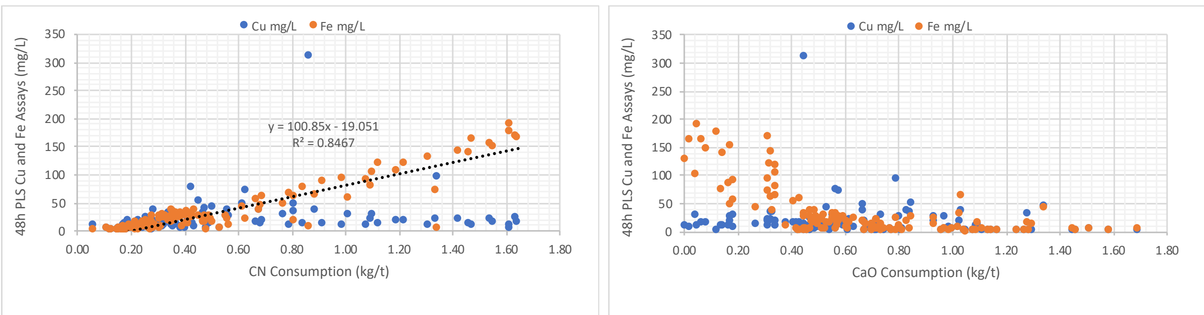
Referring to Figure 13.29 to Figure 13.35, the following conclusions can be made:

- Soluble Iron is the key driver on cyanide consumption across all ore types.
- It is clear that it is important to maintain >10.5 pH and the reason why the pre-aeration stage was observed to reduce cyanide consumption rates by oxidising cyanide soluble Fe2+ to non cyanide soluble Fe3+.

13.4.3 Effect of pH on Cyanide and Lime Consumption

Figure 13.36 below shows the relationship between mg/l Fe and Cu in solution and Cyanide and Lime Consumption Rates.

Figure 13.36 Relationships between Cyanide & Lime Consumption and mg/l Fe and Cu in Solution



Referring to Figure 13.36 above, the following conclusions can be made:

- The mg/l Fe going into solution is very much dependent upon lime consumption and therefore leach pH, specifically at the start of leaching.
- The solubility of copper is not driven by the pH variance observed during testing.

13.4.4 Effect of Pre-Aeration and Addition of Oxidants

Further testwork was conducted to investigate further the potential benefits of leaching at 55% solids w/w versus 50% solids and at 55% solids with:

- 4 hr pre-aeration prior to adding cyanide.
- 8 hr pre-aeration prior to adding cyanide.
- 4 hr pre-oxygenation prior to adding cyanide.

Figure 13.37 and Figure 13.38 summarise the relative differences in reagent consumptions and metal recoveries respectively when using different pre-aeration, oxygenation at 55% solids.

Figure 13.37 Effect of Higher % Solids and Oxidants on Reagent Consumptions

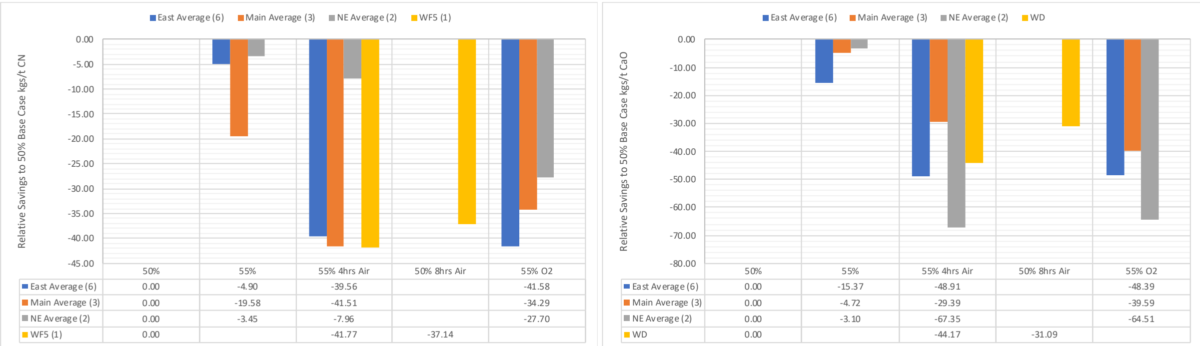
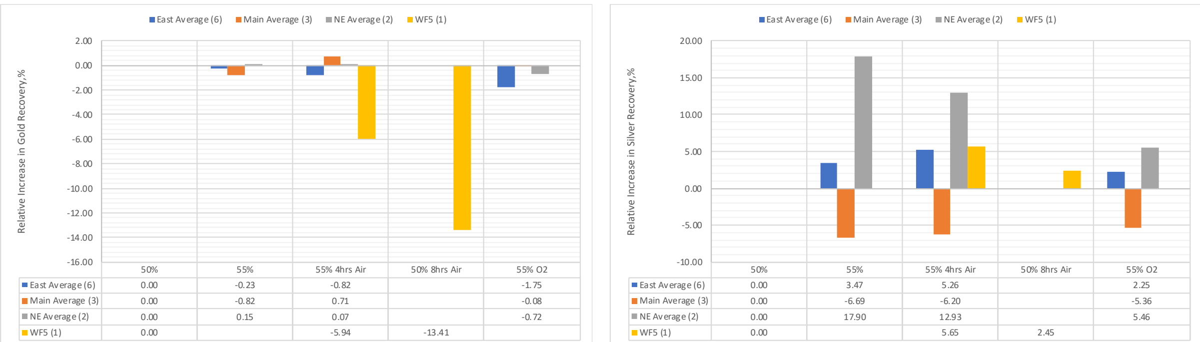


Figure 13.38 Effect of Higher % Solids and Oxidants on Metal Recovery Rates



Referring to Figure 13.37, the following conclusions can be made:

- 5 - 10% relative decrease in cyanide and lime consumptions can be achieved by leaching at 55% solids.
- The testing of pure oxygen injection for 4 hr did not provide any additional benefits when compared with 4 hr pre-aeration.
- Up to 40% relative decrease in cyanide and lime consumptions can be achieved by introducing a 4 hr pre-aeration step. (i.e. one additional tank).
- The pre-aeration step will encourage any Fe2+ ions that are in solution to be oxidised to Fe3+ ions which in turn will reduce cyanide consumption due to the much reduced affinity of Fe3+ to complex with cyanide.

Referring to Figure 13.38, the following conclusions can be made:

- No effect observed on gold or silver metal dissolution when leaching at 55% solids.
- With exception of WF5 sample, the eleven other samples tested did not tests show any adverse bias in metal recovery rates. It is logical to assume that the pre-aeration stage should not have any effect on metal dissolution rates.
- The pre-aeration step will encourage any Fe2+ ions that are in solution to be oxidised to Fe3+ ions which in turn will reduce cyanide consumption due to the much reduced affinity of Fe3+ to complex with cyanide.

As a consequence of the above preliminary results, a further phase of testing was performed on variability samples selected from the five most predominant cyanide consuming domains in the deposit. This testwork was designed to compare 50% solids with no pre-aeration with 55% solids with 4 hrs pre-aeration.

The summary of the effects on reagent consumptions is shown in Table 13.9 below, the numbers in brackets denote the number of test performed.

Table 13.9 Summary of Effect on Cyanide and Lime Consumption Rates

Domain (No Samples)	Relavite % Change in kg/t CN				
	24	**32**	**48**	***45***	***Used***
EF (14)	-33.32	-23.61	-15.78	*-17.75*	***-15***
ET (8)	-12.11	-0.97	-0.70	*-4.59*	***0***
EO (8)	-35.86	18.86	-4.07	*-7.02*	***0***
MF (5)	10.71	-22.41	-23.89	*-23.15*	***-20***
MT (6)	-17.20	-17.61	-6.90	*-9.11*	***-8***

Domain (No Samples)	Relavite % Change in kg/t CaO				
	24	**32**	**48**	***45***	***Used***
EF (14)	11.91	8.16	11.14	*9.65*	***10***
ET (8)	-16.29	-12.72	-16.82	*-14.77*	***0***
EO (8)	-12.34	-9.71	-12.07	*-11.37*	***0***
MF (5)	45.44	35.49	35.66	*35.57*	***36***
MT (6)	-2.04	-2.52	0.48	*-1.02*	***0***

Referring to table 13.9, East Fresh (-15% CN, +10% Lime), Main Fresh (-20% CN, +36% Lime) and Main Transition (-8% CN, 0% Lime) cyanide and lime consumptions show consistent trends and so have been utilised to adjust the reagent consumptions in these three domains only.

No allowance has been made for:

- Operating at 55% solids compared with 50% solids will naturally tend to reduce reagent costs by 9%.
- Tailings Thickener overflow will be recycled to the milling circuit.

As a consequence, the reagent consumptions are conservative.

One pre-aeration tank with nominal 4.5 hr residence time is included in the design.

13.4.5 Composite Sample Testing

Effect of Site Water

The previous studies were all executed using the local laboratory tap water. However, in order to ensure that the effect of site water was fully understood, the DFS leaching and flocculation and thickening studies were all conducted using a bulk sample of site water from the Area 5 aquifer.

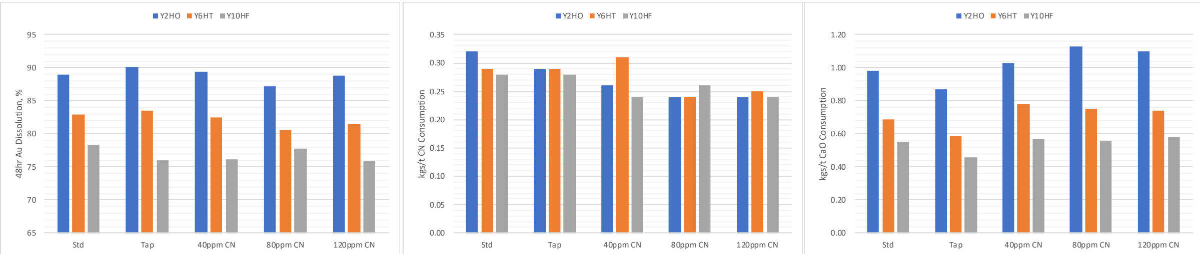
The site water has been fully analysed and reported by SGS and has extremely low salinity and almost potable quality.

Three composite samples were prepared to represent Year 2 High Oxide (Y2HO), Year 6 High Transition (Y6HT) and Year 10 High Fresh (Y10HF). These three samples were ground to d_{80} = 53 µm (Y2HO), 75 µm (Y6HT) and 90 µm Y10HF as per design grinds.

Figure 13.39 summarises the 48 hr Au Dissolution, kg/t CN and kg/t CaO using:

- Standard Leach Conditions with Site Water.
- Standard Conditions with Vancouver Tap Water.
- Recycling 40 ppm CN solution to Grinding Circuit.
- Recycling 80 ppm CN solution to Grinding Circuit.
- Recycling 120 ppm CN solution to Grinding Circuit.

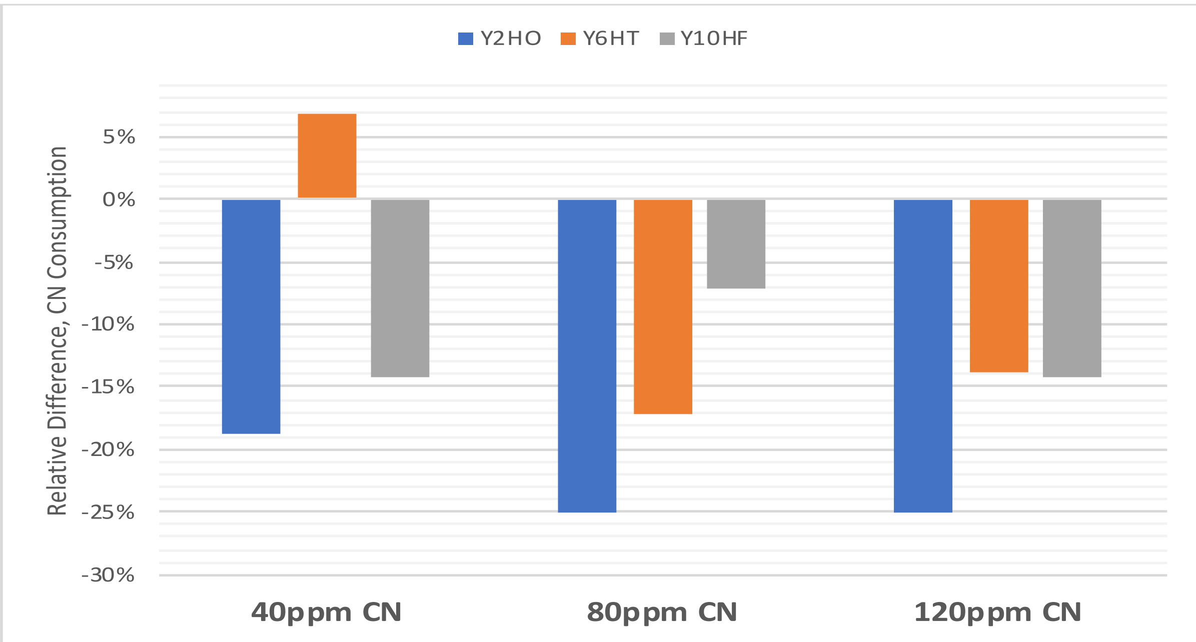
Figure 13.39 Effect of Site Water vs. Tap Water and Recycling Cyanide Solution



Referring to Figure 13.39 the following conclusions can be made:

- The site water has no detrimental effect on either metal recovery or reagent consumption rates.
- Recycling of Cyanide solutions to simulate the tailings thickener overflow returning to the process water tank, has beneficial effects on reducing cyanide consumption rates.

Figure 13.40 Relative Savings in Cyanide Consumption kg/t



Referring to Figure 13.40, a 10% saving in cyanide consumption rates can be realised by recycling cyanide.

Carbon Modelling

The three composite samples were all individually subjected to carbon modelling testwork to provide carbon adsorption isotherm data and Kk constants so as to optimise overall leach circuit configuration.

Table 13.9 summarises the constants that were developed from this testwork, all three samples had Kk constants of over 100, which means that adsorption kinetics are fast for all three samples.

The Y10HF had the fastest adsorption kinetics, followed by Y6HT and the slowest was Y2HO.

Table 13.10 Carbon Modelling Adsorption Kinetic and Equiplibrium Data

Constant	Year 2	Year 6	Year 10
Kinetic Constant (k), h^{-1}	0.017	0.013	0.020
Equilibrium Constant (K), g/t	9,280	14,124	15,180
Product of Equilibrium and Kinetic Constants (kK)	153	190	300

Table 13.11 Carbon Isotherm Data

Year 2	Au in Solution mg/l	Au on Carbon g/t	Ag in Solution mg/l	Ag on Carbon g/t
A	0.05	479	0.08	556
B	0.05	954	0.21	1,051
C	0.05	1,677	0.55	1,556
D	0.10	2,498	0.87	2,176
E	0.29	3,679	1.28	2,673
F	0.47	4,213	1.49	2,442
G	0.59	4,598	1.61	2,418

Year 6	Au in Solution mg/l	Au on Carbon g/t	Ag in Solution mg/l	Ag on Carbon g/t
A	0.05	466	0.08	722
B	0.05	733	0.18	1,129
C	0.05	1,132	0.37	1,594
D	0.05	1,977	0.82	2,188
E	0.17	2,880	1.12	2,725
F	0.32	3,407	1.35	2,851
G	0.42	3,603	1.51	2,835

Year 10	Au in Solution mg/l	Au on Carbon g/t	Ag in Solution mg/l	Ag on Carbon g/t
A	0.05	451	NA	NA
B	0.05	711	NA	NA
C	0.05	1,213	NA	NA
D	0.05	2,072	NA	NA
E	0.18	3,011	NA	NA
F	0.38	3,303	NA	NA
G	0.54	3,493	NA	NA
C-R	0.12	1,820	0.53	2,837
D-R	0.09	1,134	0.86	2,112

NA = not assayed

Referring to Table 13.10, the carbon isotherm data in above, the production of low 0.05 mg/l gold solutions is achievable depending upon gold loading levels on carbon.

Generally, the Y2HO gave adsorption tailings solution losses of 0.010 mg/l and <0.010 mg/l for Y6HT and Y10HF.

Gold Adsorption efficiencies of 99.6% were achieved for all ore types and this has been used as the stage recovery for gold in the generation of overall gold recovery for financial modelling.

The silver stage recoveries are very dependent upon the level of gold adsorbed onto carbon and therefore carbon movement. However, using the carbon isotherm data, the stage recoveries were calculated based upon expected leach discharge mg/l Ag and final stage adsorption mg/l Ag.

0.15%[2] Gold and Silver Loss has been assumed for carbon fines losses in the financial modelling.

SGS conducted additional tests to define the Copper and Mercury adsorption stage efficiencies.

Table 13.11 summarises the adsorption stage recoveries.

Table 13.12 ADR Stage Efficiencies

Samples	Au	Ag	Hg	Cu
Y2HO	99.6	27.1	8.9	2.3
Y6HT	99.6	51.0	8.1	1.3
Y10HF	99.6	67.3	11.5	

Multiple scenarios were run by SGS Lakefield and the recommended optimum configuration was as follows:

- CIP Mode Operation due to lower carbon elution, acid washing and regeneration costs and greater operational flexibility.
- Minimum 32 hr leach residence time.
- Six stages adsorption.
- 1,100 m^3 tanks containing 25 g/l carbon.
- 12 t/day carbon elution, acid washing and regeneration circuit based upon:
  - 50 g/t Au in eluted carbon
  - 90% regeneration efficiency when compared with fresh carbon.

The use of Kemix pump cell technology is recommended due to the increased carbon concentrations possible and resultant lower carbon movements required.

Using the above data, overall metal recoveries have been calculated based upon:

- Average 45 hr lithological leach dissolutions from the variability testwork.
- Application of Y2HO adsorption stage recoveries to all oxide domains.
- Application of Y6HT adsorption stage recoveries to all transition domains.
- Application of Y10HF adsorption stage recoveries to all fresh domains.
- 0.15% gold and silver loss as carbon fines.

Table 13.12 provides a summary of overall precious metal recoveries.

Table 13.13 summarises the relative quantities of Cu and Hg that will report to the elution circuit.

[2] *MPH Estimate*

Table 13.13 Estimated Cu and Hg Recovery to Elution

Pit	Domain	Lithology	Tonnes (Millions)	g/t Cu	g/t Hg	48 hr Cu Recovery (%)	48 Hg Recovery (%)	kg Cu / day	kgs Hg / day
Main	Oxide	FQSP	1.1	75	0.88	0.1	1.4	1.6	0.22
		PGSS	1.4	111	1.61	0.2	1.3	3.6	0.39
		VAN	1.8	133	0.36	0.0	3.6	0.7	0.23
	Trans	FQSP	0.9	158	3.13	0.2	1.1	6.2	0.59
		PGSS	2.1	126	4.78	0.5	0.7	10.2	0.62
		VAN	2.0	111	0.37	0.0	2.6	0.4	0.18
	Fresh	FQSP	3.0	114	1.23	2.0	2.2	41.8	0.49
		PGSS	6.4	125	1.14	1.7	1.7	37.4	0.34
		VAN	4.1	122	0.25	1.6	5.5	34.3	0.25
East	Oxide	FQSP	0.8	131	0.92	0.1	1.1	1.8	0.18
		PGSS	5.3	86	0.59	0.1	2.4	0.9	0.26
		SRR	0.5	127	1.18	0.1	1.1	2.4	0.24
		VAN	1.8	78	0.62	0.1	2.8	0.9	0.32
	Trans	FQSP	0.9	96	1.41	0.3	1.2	6.0	0.29
		PGSS	5.3	145	1.59	0.3	1.4	8.3	0.40
		SRR	0.5	63	0.68	0.2	1.5	1.8	0.19
		VAN	4.5	124	0.43	0.1	3.4	1.3	0.26
	Fresh	FQSP	5.2	93	1.76	1.8	1.2	30.5	0.39
		PGSS	16.3	90	0.75	1.7	4.6	28.1	0.63
		SRR	2.6	77	1.37	1.4	1.4	19.1	0.33
		VAN	6.7	125	0.65	2.2	5.1	48.5	0.59
NEZ	Oxide	FQSP	0.1	114	0.57	0.1	1.7	1.0	0.18
		PGSS	1.4	123	0.67	0.2	1.8	3.4	0.22
		VAN	0.2	118	0.62	0.1	1.8	2.2	0.20
	Trans	FQSP	0.1	110	2.39	0.3	0.4	5.9	0.17
		PGSS	1.6	110	2.39	0.3	0.4	5.9	0.17
		VAN	0.5	110	2.39	0.3	0.4	5.9	0.17
	Fresh	FQSP	0.0	114	1.50	1.8	1.2	37.4	0.33
		PGSS	0.2	114	1.50	1.7	4.6	35.5	1.26
		VAN	0.2	114	1.50	1.4	1.4	28.2	0.37
Wadi Doum	Oxide	QPR	0.3	94	0.37	0.1	4.4	2..4	0.29
		VAN	0.1	94	0.37	0.1	4.4	2.4	0.29
		VC1	0.0	94	0.37	0.1	4.4	2.4	0.29
		VC3	0.1	94	0.37	0.1	4.4	2.4	0.29
	Trans	QPR	0.0	305	0.70	0.2	3.4	12.3	0.42
		VAN	0.0	305	0.70	0.2	3.4	12.3	0.42
		VC1	0.0	305	0.70	0.2	3.4	12.3	0.42
		VC3	0.0	305	0.70	0.2	3.4	12.3	0.42
	Fresh	QPR	0.8	142	0.95	0.9	2.7	22.1	0.27
		VAN	0.3	141	0.51	1.7	4.4	41.8	0.40
		VC1	0.2	137	0.31	2.1	5.6	52.0	0.31
		VC3	0.7	143	0.28	2.0	4.8	50.9	0.24
LOM Weighted Average			**79.9**	**110**	**1.06**	**1.1**	**3.0**	**21.3**	**0.41**

Note that the highlighted cells were not tested due to their insiginificance tonnage and the same assumptions were made as described above.

Referring to Table 13.13, the following conclusions can be made:

- Copper and Mercury recoveries are low at 1.1% and 3.0% respectively. The low adsorption stage recoveries result in the majority of the soluble Cu and Hg reporting to adsorption tailings:
  - the fresh domains attract the highest recovery of copper to the elution circuit
    - the average LOM recovery of copper to the elution circuit is 21.3 kg/day.
    - the worst lithological zones are Wadi Doum Fresh VC1, VC3 and East Fresh VAN where circa 50 kg/day of copper will report to elution circuit based upon 18,000 t/day ore treatment rate.
  - Main Transition PGSS, FQSP, Main Fresh FQSP, East Fresh PGSS, VAN and North East Fresh PGSS all have higher than average mercury/day reporting to elution.
    - The average LOM recovery of mercury to the elution circuit is 0.41 kg/day.
    - The highest mercury reporting to the elution circuit is North East Fresh PGSS at a rate of 1.26 kg/day at 18,000 t/day ore treatment rate. It should be noted that North East Fresh was not tested due to its insignificant tonnage. East Fresh data was used to model North East Fresh data.

Flocculant and Thickener Design Testwork

FLSmidth conducted Flocculant Screening and optimisation testwork and then used this optimised flocculant parameters to conduct dynamic thickening testwork to provide High Rate thickener sizing data.

This work is reported in detail in FLSmidth Report written by Kyle Nowicki, FL Smidth Minerals Testing and Research Center, Utah, USA dated December 2017.

Three samples were tested as part of the revised PEA study on:

- Master Composite Fresh Leach Tailings (FrMC Tailings).
- Master Composite Transition Leach Tailings (TrMC Tailings).
- Master Composite Oxide Leach Tailings (OxMC Tailings).

This thickener sizing was verified further on Y2HO (Year 2 High Oxide) master composite as part of the FS study. The oxide ore is the most onerous sample of the three main domain types in terms of thickener unit capacity and will be used for sizing.

Table 13.14 summarises the flocculation and thickening testwork results.

Table 13.14 Summary of Flocculation and Thickening Sizing Parameters

High Rate Thickener Testing	FrMC Tailings	TrMC Tailings	OxMC Tailings	Y2HO	Y2HO
Testing Stage	PEA	PEA	PEA	DFS	DFS
Test Material d80	52	68	43	41	41
Flocculant Type	AN-905 SH				AN-923VHM
Feed Suspended Solids, Wt%	12	12	10	10	10
Flocculant Dosage, g/t	20	20	30	30	30
Design Unit Area, m^2/tpd	0.04	0.04	0.05	0.05	0.04
Min. Mud Residue time required, hrs	2.0	1.5	1.5	2.0	2.0
Design Underflow Solids, Wt%	65	65	63	63	62
Recommended Yield Stress for Rake Design, Pa	50	50	50	50	50
Design Overflow Clarity, ppm	<50	<50	<50	<50	<50

Referring to Table 13.14, FLSmidth conducted Flocculant Screening and optimisation testwork and then used this optimised flocculant parameters to conduct dynamic thickening testwork to provide High Rate thickener sizing data.

This work is reported in detail in FLSmidth Report written by Kyle Nowicki, FLSmidth Minerals Testing and Research Centre, Utah, USA dated December 2017.

Three samples were tested as part of the revised PEA study on:

- Master Composite Fresh Leach Tailings (FrMC Tailings).
- Master Composite Transition Leach Tailings (TrMC Tailings).
- Master Composite Oxide Leach Tailings (OxMC Tailings).

This thickener sizing was verified further on Y2HO (Year 2 High Oxide) master composite as part of the FS study. The oxide ore is the most onerous sample of the three main domain types in terms of thickener unit capacity and will be used for sizing.

The following conclusions can be made:

- The DFS testwork verified the work conducted previously in the PEA study.
- 20 – 30 g/t of an anionic polyacrylamide flocculant is optimal flocculant dosage.
- The thickener feedwell % solids should be 10 - 12% solids w/w.
- The thickener unit area of 0.04 m^2/tpd should be used for design purposes.
- The typical underflow density achieved was 65% solids for fresh and transition and 63% for oxide ores.
- The overflow clarity is <50 ppm.

14. MINERAL RESOURCE ESTIMATES

14.1 Introduction

In September 2018, MPR Geological Consultants Pty Ltd (MPR) estimated gold and silver Mineral Resources for the Galat Sufar South and Wadi Doum deposits. The estimates incorporate results from additional RC and diamond drilling compiled since MPR last estimated resources for these deposits in January 2018.

As with previous estimates for Galat Sufar South and Wadi Doum, for the current study MPR estimated gold recoverable resources by Multiple Indicator Kriging (MIK) with block support correction to reflect open pit mining selectivity at gold cut-off grades, a method that has been demonstrated to provide reliable estimates of resources recoverable by open pit mining for a wide range of mineralization styles. Silver E-type estimates by MIK were assigned to the recoverable estimates at gold cut-off grades. This approach reflects the moderate correlation between gold and silver grades and generally comparatively low silver grades.

The current estimates are based RC and diamond drilling data supplied by Orca in September 2018. Details of this sampling and assay are described in previous sections of this report.

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

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

The Qualified Person responsible for the Mineral Resources is 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 16 and 22 January, 2014.

14.2 Estimation of GSS Resources

14.2.1 Resource Dataset

Relative to the January 2018 dataset, the GSS sampling database available for the current review incudes an additional 74 RC holes and 13 diamond cored holes drilled during 2018.

The 2018 RC and diamond holes included in-fill and extensional drilling. The infill drilling targeted several areas previously tested by comparatively broad-spaced drilling, increasing confidence in estimated resources for these areas. The extensional drilling primarily targeted potential mineralisation beneath previously sampled areas, allowing expansion of the volume included in resource estimates.

The current estimates are based on two metre down-hole composited gold grades from RC and diamond drilling. Rare un-assayed intervals were excluded from the estimation dataset. Surface rock chip and trench samples were excluded from the resource dataset, along with peripheral drill holes not relevant to the estimates.

Silver assays are available for around 80% of the estimation dataset including 96% of mineralized domain composites. Composites without silver assays were assigned silver grades from a gold-silver function based on the moderate correlation between gold and silver assay grades. This approach was adopted to ensure local consistency between gold and silver estimates.

The resource dataset comprises 49,069 composites with gold grades ranging from 0.002 to 126 g/t and averaging 0.55 g/t, and silver grades ranging from 0.005 to 294 g/t and averaging 0.92 g/t. The dataset is dominated by composites from RC holes which represent 74% of mineralized domain composites. Additional data compiled since the January 2018 resource estimates provide around 15% of the combined dataset, including 17% of composites within mineralized domains.

14.2.2 Geological Interpretation and Domaining

Drilling to date has delineated several distinct bodies of gold mineralization at Galat Sufar South. In general, the transition from gold mineralization to barren host rock is characterized 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 14 mineralized domains interpreted by MPR on the basis of composited gold grades with reference wire-frames representing key rock units interpreted by Orca. Domain boundaries were digitized on cross-sections, snapped to drill hole traces where appropriate, then wire-framed into three-dimensional solids.

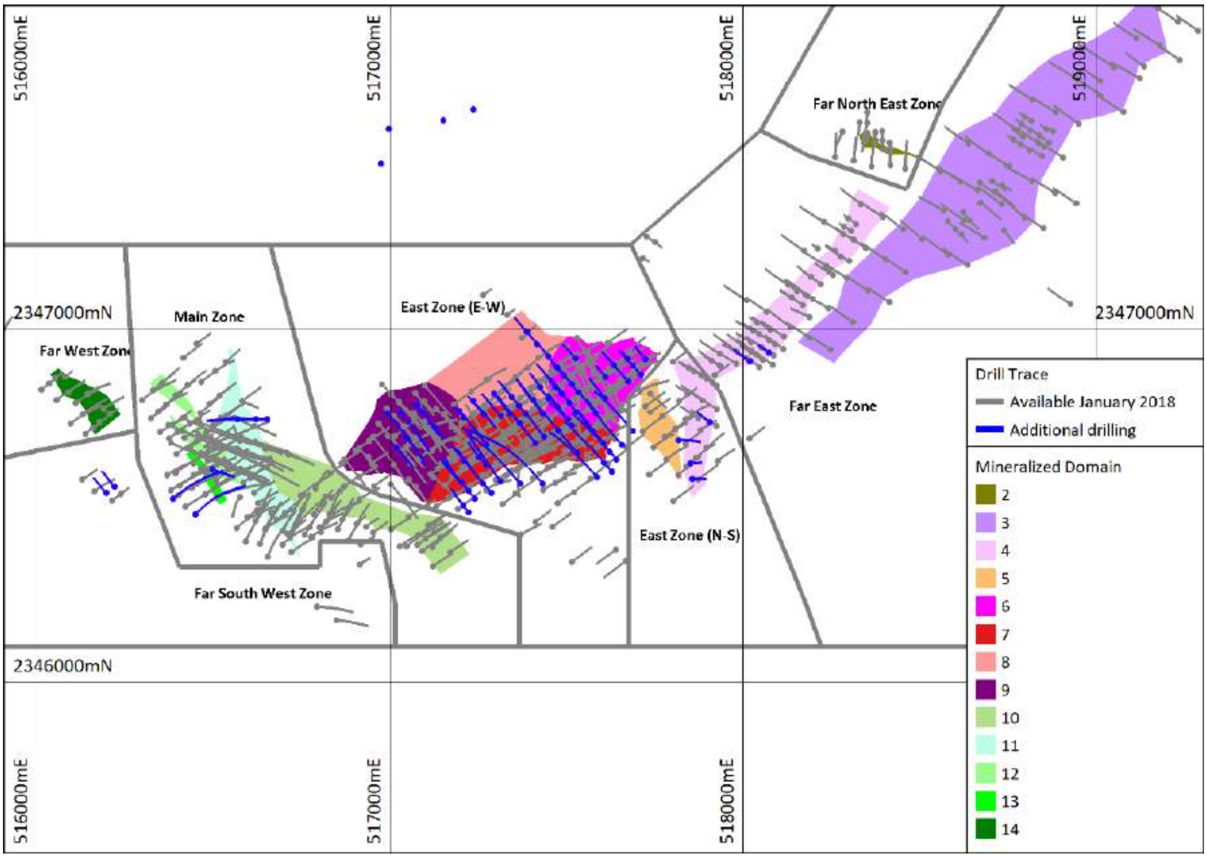
Figure 14.1 presents a plan-view of the surface expression of the mineralized domains relative to drill hole traces coloured by drilling type and phase.

The mineralized domains are designated as Domains 2 to 14, 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.

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 resources by oxidation type.

Depth to the interpreted base of complete oxidation ranges from locally zero where transitional material outcrops to around 75 m and averages approximately 24 m. Interpreted depth to fresh rock ranges from around 3 to 112 m depth and averages approximately 55 m.

Figure 14.1 GSS Mineralized Domains and Drill Traces



14.2.3 Gold / Silver Correlation

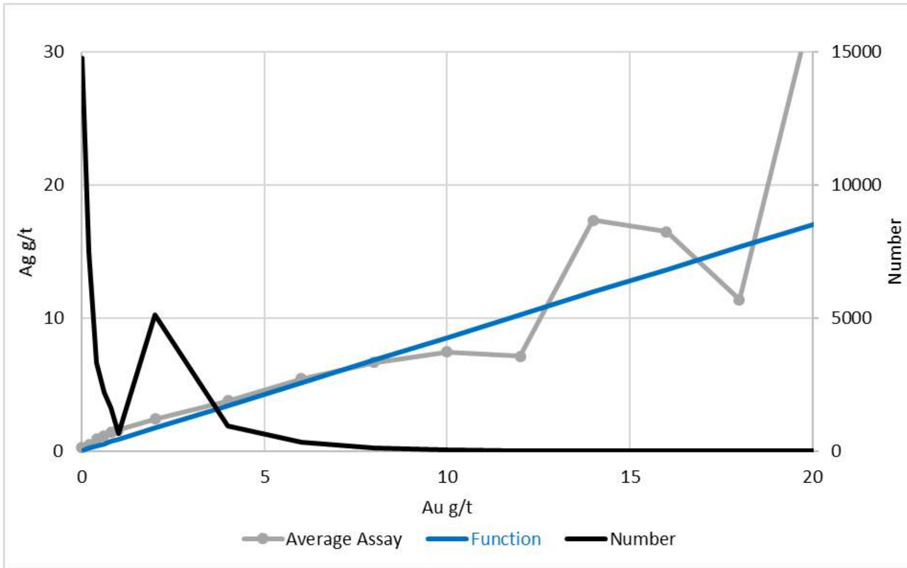
Table 14.1 and the trend plot in Figure 14.2 summarize correlation of gold and silver grades for GSS estimation composites with assays for both metals. This table and figure demonstrate that although there is considerable scatter for individual composites, there is a general association between higher silver grades with elevated gold grades.

Composites without silver assay grades were assigned silver grades from gold grades using the formula shown in Figure 14.2. The majority of mineralized domain samples have silver assays and detailed accuracy of the assigned silver grades do not significantly affect confidence in estimated resources.

Table 14.1 GSS Gold / Silver Correlation

	Domain 1		Domain 2 to 14		Combined	
	Au g/t	Ag g/t	Au g/t	Ag g/t	Au g/t	Ag g/t
Number	2,361		36,854		39,215	
Mean	0.15	0.55	0.70	1.03	0.66	1.00
Variance	0.52	8.06	4.74	13.3	4.51	13.0
Coef. Var.	4.70	5.14	3.13	3.53	3.20	3.59
Minimum	0.00	0.01	0.00	0.01	0.00	0.01
1st Quartile	0.01	0.10	0.04	0.10	0.03	0.10
Median	0.02	0.15	0.17	0.30	0.15	0.30
3rd Quartile	0.06	0.35	0.68	0.97	0.63	0.92
Maximum	13.7	92.3	126.1	294.0	126.1	294.0
Linear Correl.	0.74		0.40		0.40	
Rank Correl.	0.46		0.70		0.69	

Figure 14.2 GSS Gold / Silver Trend Plot



Function: Ag (g/t) = 0.85 x Au g/t + 0.05

14.2.4 Exploratory Data Analysis

Table 14.2 shows univariate statistics of composite gold and silver grades for the GSS resource dataset subdivided by mineralized domain. Notable features of these statistics include the following.

At 0.09 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 and silver grades show strong positive skewness with coefficients of variation of generally greater than three for mineralized domains, indicating that MIK is an appropriate estimation technique and that selective mining above elevated gold cut-off grades will be difficult.

Table 14.2 GSS Composite Statistics

Au g/t	Domain							
	1	2	3	4	5	6	7	8
Number	10,651	207	2,977	3,297	447	7,614	9,780	2,700
Mean	**0.09**	**1.83**	**0.32**	**0.50**	**0.20**	**0.69**	**0.77**	**0.48**
Variance	0.20	7.50	1.55	4.13	0.14	2.76	3.76	3.30
Coef. Var.	**5.19**	**1.50**	**3.85**	**4.06**	**1.86**	**2.40**	**2.52**	**3.79**
Minimum	0.00	0.01	0.01	0.01	0.00	0.01	0.00	0.01
1st Quartile	0.01	0.21	0.01	0.01	0.02	0.07	0.05	0.07
Median	0.01	0.90	0.03	0.07	0.07	0.22	0.26	0.15
3rd Quartile	0.04	2.25	0.18	0.41	0.23	0.76	0.91	0.39
Maximum	13.7	24.7	30.9	100.5	3.5	66.4	109.2	78.2
	Domain							
	9	**10**	**11**	**12**	**13**	**14**	**2-14**	**Total**
Number	3,685	2,197	3,387	1,431	333	363	38,418	49,069
Mean	**0.55**	**0.54**	**1.09**	**0.97**	**1.83**	**0.58**	**0.68**	**0.55**
Variance	3.11	9.06	11.1	5.40	29.12	5.52	4.61	3.72
Coef. Var.	**3.21**	**5.60**	**3.06**	**2.39**	**2.95**	**4.07**	**3.14**	**3.48**
Minimum	0.01	0.01	0.01	0.00	0.01	0.01	0.00	0.00
1st Quartile	0.04	0.02	0.04	0.06	0.07	0.01	0.04	0.02
Median	0.15	0.07	0.23	0.31	0.26	0.11	0.16	0.09
3rd Quartile	0.48	0.29	1.10	0.97	1.09	0.53	0.66	0.47
Maximum	61.2	122.3	126.1	41.7	48.7	33.3	126.1	126.1

Ag g/t	Domain							
	1	2	3	4	5	6	7	8
Number	10,651	207	2,977	3,297	447	7,614	9,780	2,700
Mean	**0.56**	**1.23**	**1.08**	**0.58**	**0.50**	**0.84**	**1.15**	**0.63**
Variance	1.85	1.84	48.7	1.81	0.33	2.43	7.51	1.97
Coef. Var.	**2.44**	**1.10**	**6.44**	**2.31**	**1.16**	**1.86**	**2.38**	**2.24**
Minimum	0.01	0.10	0.05	0.05	0.05	0.02	0.01	0.05
1st Quartile	0.50	0.45	0.10	0.10	0.10	0.10	0.20	0.10
Median	0.51	0.85	0.25	0.25	0.30	0.30	0.51	0.20
3rd Quartile	0.52	1.50	0.75	0.60	0.62	1.00	1.30	0.59
Maximum	92.3	11.1	294.0	44.6	4.1	33.4	95.3	42.4
	Domain							
	9	**10**	**11**	**12**	**13**	**14**	**2-14**	**Total**
Number	3,685	2,197	3,387	1,431	333	363	38,418	49,069
Mean	**0.59**	**0.72**	**1.66**	**2.45**	**3.37**	**1.32**	**1.03**	**0.92**
Variance	3.67	2.22	39.3	39.0	73.12	8.16	12.8	10.5
Coef. Var.	**3.23**	**2.08**	**3.79**	**2.55**	**2.54**	**2.17**	**3.49**	**3.50**
Minimum	0.05	0.05	0.04	0.03	0.10	0.05	0.01	0.01
1st Quartile	0.10	0.10	0.10	0.30	0.35	0.10	0.10	0.15
Median	0.20	0.25	0.40	0.70	0.80	0.20	0.35	0.50
3rd Quartile	0.55	0.65	1.30	2.05	2.30	1.05	0.95	0.75
Maximum	75.6	25.2	169.5	123.5	90.9	30.4	294.0	294.0

14.2.5 Indicator Thresholds and Bin Mean Grades

Composites from the larger, more populous domains were subdivided by oxidation subdomain for determination of indicator threshold and bin 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.

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 14.3 summarizes upper bin thresholds and bin mean grades for gold and silver and describes with the methodology used to determine upper bin grades.

Table 14.3 GSS Upper Bin Thresholds and Class Grades

Au g/t					
Domain	Subdomain	Upper Bin (>99%ile) Au g/t			Source of Bin Grade
		Threshold	Maximum	Bin grade	
1	Oxide	1.21	0.77	2.53	Mean exclud. comps. > 5.0 g/t
	Transition	1.35	0.82	2.46	Mean exclud. comps. > 5.0 g/t
	Fresh	1.34	0.86	1.97	Mean exclud. comps. > 5.0 g/t
2	All	8.65	7.96	9.26	Mean exclud. comps. > 10 g/t
3	Oxide	4.95	18.2	7.44	Median
	Transition	4.30	8.18	6.39	Mean exclud. comps. > 10 g/t
	Fresh	3.79	10.98	6.36	Mean exclud. comps. > 12 g/t
4	Oxide	4.89	7.51	5.10	Median exclud. comps. > 10 g/t
	Transition	5.18	9.71	7.11	Mean exclud. comps. > 10 g/t
	Fresh	4.69	11.53	6.51	Median
5	Ox. & Trans.	2.20	3.45	2.91	Mean
	Fresh	1.05	3.45	2.42	Mean
6	Oxide	3.66	6.42	4.793	Mean exclud. comps. > 30 g/t
	Transition	5.09	10.43	6.68	Mean exclud. comps. > 18 g/t
	Fresh	5.97	24.71	7.83	Median exclud. comps. > 30 g/t
7	Oxide	6.09	11.71	7.73	Median
	Transition	5.50	18.13	8.14	Mean exclud. comps. > 20 g/t
	Fresh	5.65	19.58	9.34	Mean exclud. comps. > 20 g/t
8	Oxide	3.58	3.85	3.75	Mean
	Transition	3.78	11.30	6.37	Mean
	Fresh	5.07	28.48	6.37	Median exclud. comps. > 60 g/t
9	Oxide	4.98	12.43	6.06	Median
	Transition	6.00	11.48	8.56	Mean exclud. comps. > 12 g/t
	Fresh	5.60	33.38	8.08	Median exclud. comps. > 35 g/t

Au g/t					
Domain	Subdomain	Upper Bin (>99%ile) Au g/t			Source of Bin Grade
		Threshold	Maximum	Bin grade	
10	Oxide	6.12	9.08	7.84	Mean exclud. comps. > 10 g/t
	Transition	7.02	11.23	8.91	Mean exclud. comps. > 12 g/t
	Fresh	5.81	14.18	9.01	Median
11	Oxide	11.34	14.47	13.98	Mean
	Transition	10.87	26.31	17.48	Median exclud. comps. > 35 g/t
	Fresh	8.90	24.55	15.63	Mean exclud. comps. > 40 g/t
12	Oxide	7.01	10.74	10.04	Mean
	Transition	5.00	7.14	5.84	Mean
	Fresh	12.36	19.48	16.33	Mean exclud. comps. >20 g/t
13	All	16.20	20.45	16.33	Mean exclud. comps. >20 g/t
14	Ox. & Trans.	2.94	3.49	3.37	Mean exclud. comps. >20 g/t
	Fresh	2.19	8.32	5.65	Mean

Ag g/t					
Domain	Subdomain	Upper Bin (>99%ile) Ag g/t			Source of Bin Grade
		Threshold	Maximum	Bin grade	
1	All	2.30	16.55	3.50	Median exclud. comps. > 30 g/t
2	All	5.65	7.35	6.63	Mean exclud. comps. > 10 g/t
3	All	9.45	54.45	15.15	Median exclud. comps. > 55 g/t
4	All	4.30	14.80	6.83	Mean exclud. comps. > 20 g/t
5	All	2.80	4.05	3.46	Mean
6	All	5.50	18.65	7.35	Median exclud. comps. > 20 g/t
7	All	8.30	57.55	11.95	Median exclud. comps. > 60 g/t
8	All	5.00	19.45	7.45	Median exclud. comps. > 20 g/t
9	All	4.65	17.35	8.63	Mean exclud. comps. > 20 g/t
10	All	7.50	18.55	10.63	Mean exclud. comps. > 20 g/t
11	All	16.07	58.85	24.95	Median exclud. comps. > 60 g/t
12	All	21.45	44.42	31.62	Mean exclud. comps. > 50 g/t
13	All	28.50	35.20	32.27	Mean exclud. comps. > 40 g/t
14	All	10.00	11.60	10.86	Mean exclud. comps. > 12 g/t

14.2.6 Variogram Models

For MIK modelling of gold and silver, some less populous domains were combined for variogram analysis. Table 14.4 summarizes domain grouping for modelling of the variograms used or the MIK estimation.

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

For estimation of Domain 4 within the East Zone (N-S) area where the domain strikes approximately north-south consistent with geological interpretations, the variogram models were rotated to reflect this orientation.

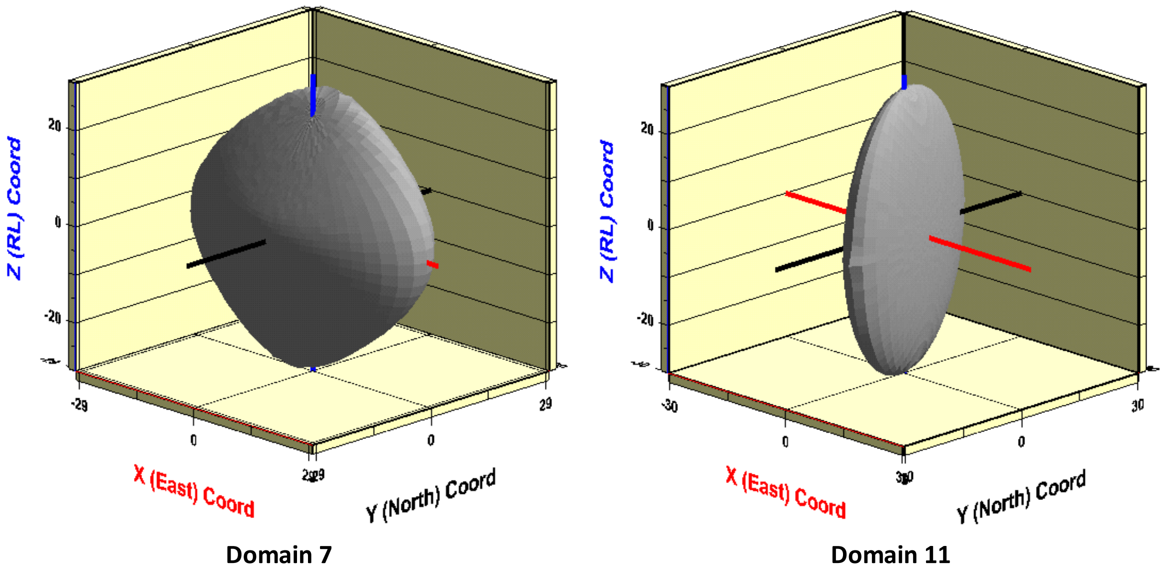
The spatial continuity observed in the variograms is consistent with geological interpretation and trends shown by resource composite gold grades. The fitted models generally have a fairly large short range structure and a smaller long range structure consistent with the strike and dip of dominant mineralized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

As examples of the variogram models, Figure 14.3 presents three dimensional variogram surface maps of the median indicator gold variogram model for selected domains at a variogram value of 0.7.

Table 14.4 GSS Variogram Models used for Estimation

Domain	Variogram Models
1	Use Dom 7
2	Use Dom 10,12,13,14
3,4	Combine for modelling
5	Use Dom11
6	Model
7	Model
8	Use Dom 7
9	Model
10,12,13,14	Combine for modelling
11	Model

Figure 14.3 GSS 3D Variogram Plots



14.2.7 Estimation Parameters

The block model frame work used for the MIK modelling covers the full extents of the composite dataset. It includes panels with dimensions of 15 m east-west by 15 m north-south by 5 m vertical. The plan-view panel dimensions reflect 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 14.5. For each domain the search ellipsoids were aligned with dominant domain orientation. Search pass 4 was used only for panels within the Far East and Far North East Zones reflecting the commonly broader spaced drilling in these areas.

Table 14.5 GSS Search Criteria

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

The gold resource estimates include a variance adjustment to give estimates of recoverable resources above gold cut-off grades for selective mining (SMU) dimensions of 5 m east by 5 m north by 2.5 m in elevation. The variance adjustments were applied using the direct lognormal method and the adjustment factors listed in Table 14.6.

For each block model panel, E-type silver estimates were assigned to the recoverable estimates at gold cut-off grades. This approach reflects the moderate correlation between gold and silver grades and generally comparatively low silver grades.

Table 14.6 GSS Variance Adjustment Factors

Domain	Block/ Panel	Information Effect	Total Adjustment
1,7,8	0.213	0.415	0.088
2,10,12,13,14	0.173	0.404	0.070
3,4	0.185	0.195	0.036
5,11	0.165	0.317	0.052
6	0.158	0.358	0.057
9	0.256	0.344	0.088

14.2.8 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 14.8.

All panels within the Far West, Main, East, and Far East zones informed by search passes 1 and 2 are classified as Indicated. For the Far North East zone, panels within the mineralized domain (Domain 2) informed by search pass 1 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 zone are assigned to the Inferred category.

A proportionally small number of near surface search pass 3 panels, for which the distribution of informing composites does not meet search pass 1 and 2 octant requirements were re-classified as Indicated. These panels lie within areas of generally Indicated estimates and represent a small proportion of estimated resources.

14.3 Estimation of WD Resources

14.3.1 Resource Dataset

Relative to the January 2018 dataset, the WD sampling database available for the current review incudes an additional eight RC holes and three diamond cored holes mostly drilled during 2018.

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

Silver assays are available for around 80 % of the WD estimation dataset including 93% of mineralized domain composites. Composites without silver assays were assigned silver grades from a gold-silver function based on the moderate correlation between gold and silver grades. This approach was adopted to ensure consistency between gold and silver estimates.

The resource dataset comprises 6,465 composites with gold grades ranging from 0.005 to 388.6 g/t and averaging 0.90 g/t and silver grades ranging from 0.00 to 122.6 g/t and averaging 3.37 g/t. The dataset is dominated by composites from RC holes which represent 94% of mineralized domain composites.

14.3.2 Geological Interpretation and Domaining

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

Mineralized domains used for the current estimates are updated from those used for the January 2018 estimates and comprise two mineralized domains interpreted by MPR on the basis of composited gold grades. Domain boundaries were digitized on cross-sections, snapped to drill hole traces where appropriate, then wire-framed into three-dimensional solids.

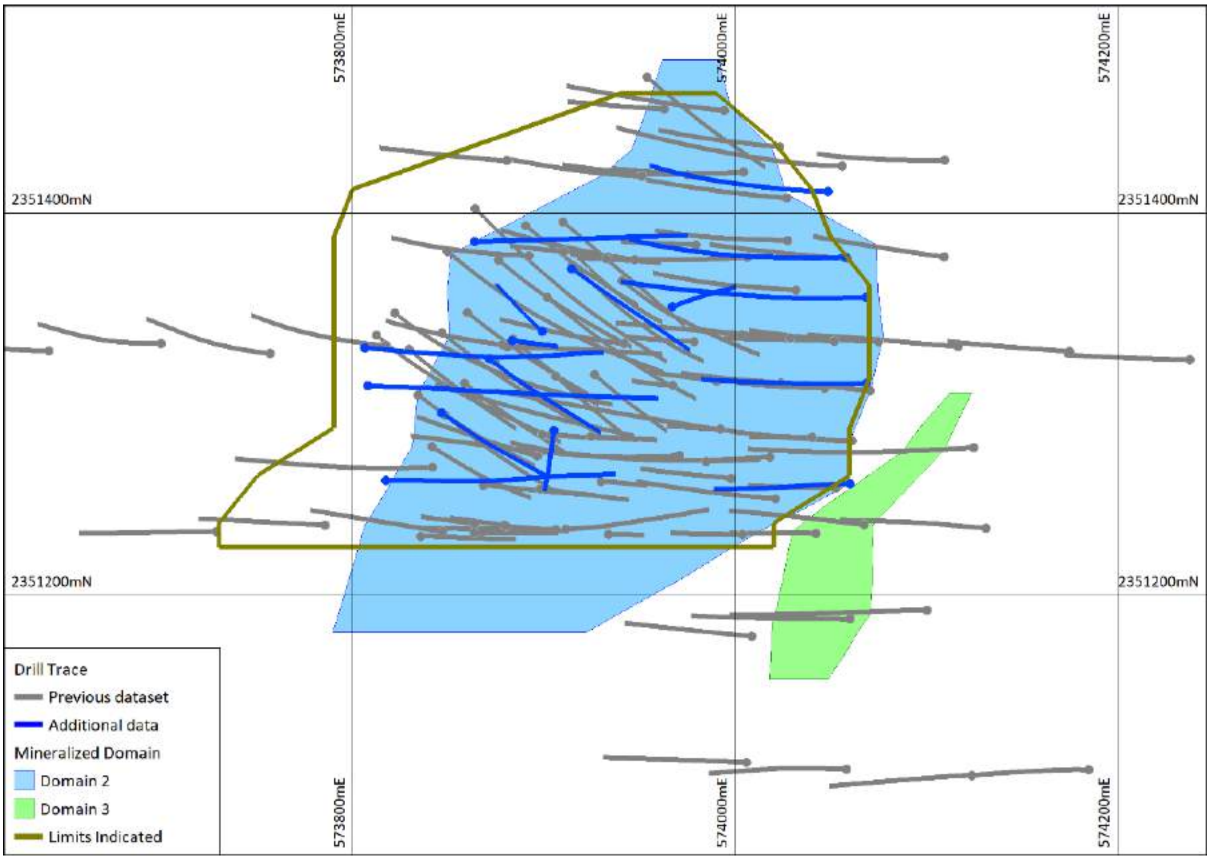
The mineralized domains are designated as Domains 2 and 3, with Domain 1 representing generally un-mineralized composites not captured by the mineralized domain wire-frames.

Figure 14.4 presents a plan-view of the surface expression of the mineralized domains relative to drill hole traces coloured by drilling type.

Oxidation domains used for the current estimates are unchanged from those used for the January 2018 estimates. They are derived from surfaces representing the base of oxidation and the top of fresh rock respectively interpreted from drill hole logging by Orca. These domains were used for flagging the resource composites into oxide, transition and fresh subdomains, density assignment and partitioning resources by oxidation type.

Depth to the interpreted base of complete oxidation ranges from around 7 m to approximately 50 m and averages approximately 15 m. Interpreted depth to fresh rock ranges from around 16 to 58 m depth and averages around 24 m.

Figure 14.4 WD Mineralized Domains and Drill Traces



14.3.3 Gold / Silver Correlation

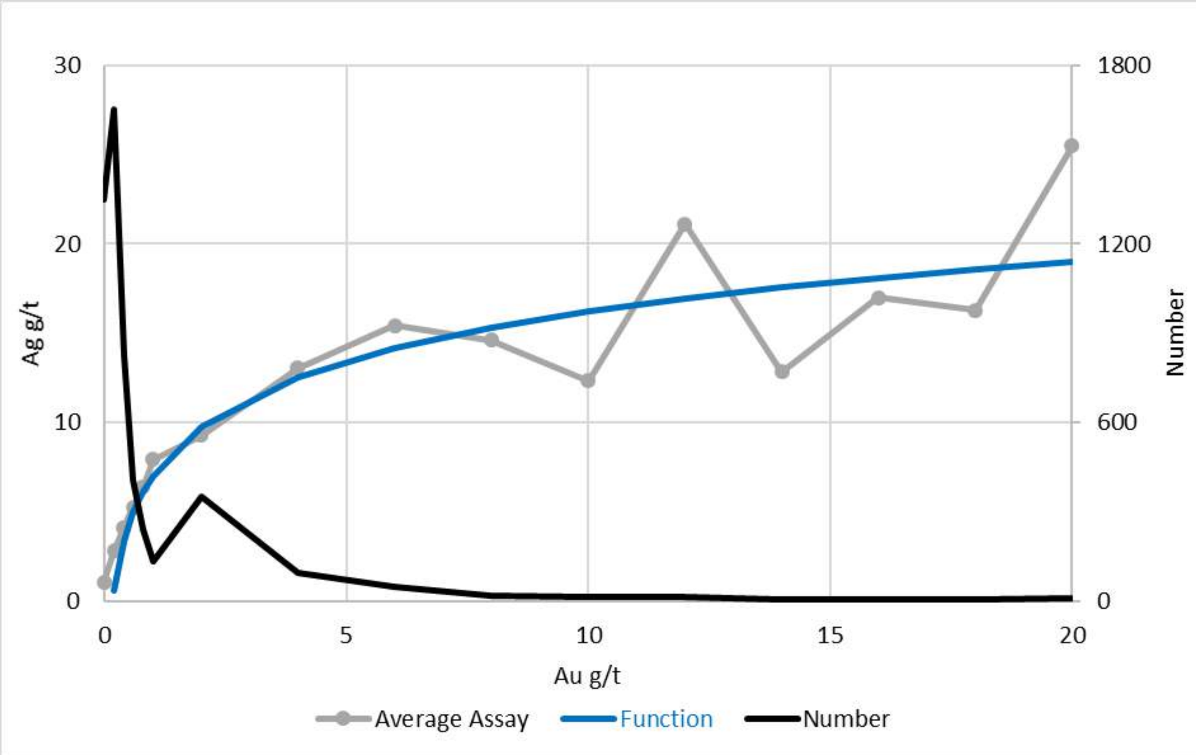
Table 14.7 and the trend plot in Figure 14.5 summarize correlation of gold and silver grades for WD composites with assays for both metals. This table and figure demonstrate that although there is considerable scatter for individual composites, there is a general association between higher silver grades with elevated gold grades.

Composites without silver assay grades were assigned silver grades from gold grade using the formula shown in Figure 14.5. Most mineralized domain samples have silver assays and detailed accuracy of the estimated silver grades does not significantly affect confidence in estimated resources.

Table 14.7 WD Gold / Silver Correlation

	Domain 2		Domain 3		Domain 2 and 3	
	Au g/t	Ag g/t	Au g/t	Ag g/t	Au g/t	Ag g/t
Number	4,956		144		5,100	
Mean	1.05	4.35	0.40	0.79	1.04	4.25
Variance	56.8	53.5	0.21	0.34	55.2	52.4
Coef. Var.	7.15	1.68	1.17	0.74	7.17	1.70
Minimum	0.01	0.05	0.01	0.10	0.01	0.05
1st Quartile	0.10	0.95	0.11	0.45	0.10	0.90
Median	0.24	2.05	0.26	0.60	0.24	2.00
3rd Quartile	0.54	5.00	0.50	0.95	0.54	4.90
Maximum	388.6	122.6	3.41	3.15	388.6	122.6
Linear Correl.	0.51		0.54		0.51	
Rank Correl.	0.65		0.58		0.64	

Figure 14.5 WD Gold / Silver Trend Plot



Function: Ag (g/t) = 4.0 x Ln(Au g/t) + 7

14.3.4 Exploratory Data Analysis

Table 14.8 shows univariate statistics of composite gold and silver grades for the resource dataset subdivided by mineralized domain. Notable features of these statistics include the following.

At 0.10 g/t, the mean gold grade for Domain 1 composites is notably lower than for the mineralized domains demonstrating that the domaining has been effective in assigning most mineralized composites into the mineralized domains.

Typical of many precious metal deposits, gold and silver composite grades show strong positive skewness with a coefficient of variation of around 6.9 and 1.8 respectively for the mineralized domains indicating that MIK is an appropriate estimation technique.

Table 14.8 WD Composit Statistics

Au g/t	Domain				
	1	2	3	2 and 3	Total
Number	1,004	5,310	151	5,461	6,465
Mean	**0.10**	**1.07**	**0.38**	**1.05**	**0.90**
Variance	0.05	54.2	0.21	52.7	44.6
Coef. Var.	**2.33**	**6.87**	**1.19**	**6.90**	**7.39**
Minimum	0.005	0.005	0.005	0.005	0.005
1st Quartile	0.02	0.09	0.1	0.095	0.07
Median	0.05	0.23	0.25	0.23	0.18
3rd Quartile	0.095	0.535	0.475	0.535	0.46
Maximum	5.27	388.6	3.41	388.6	388.6

Ag g/t	Domain				
	1	2	3	2 and 3	Total
Number	1,004	5,310	151	5,461	6,465
Mean	**0.13**	**4.06**	**0.76**	**3.97**	**3.37**
Variance	0.27	51.1	0.35	50.0	44.2
Coef. Var.	**4.13**	**1.76**	**0.79**	**1.78**	**1.97**
Minimum	0.00	0.00	0.00	0.00	0.00
1st Quartile	0.00	0.75	0.4	0.70	0.35
Median	0.00	1.85	0.55	1.80	1.35
3rd Quartile	0.00	4.75	0.90	4.60	3.8
Maximum	5.60	122.6	3.15	122.6	122.6

14.3.5 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 for gold and silver. 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 14.9 summarizes upper bin thresholds and bin grades for gold and silver and describes with the methodology used to determine upper bin grades.

Table 14.9 WD Upper Bin Thresholds and Bin Grades

Au g/t					
Domain	Subdomain	Upper bin (>99%ile) Au g/t			Source of bin grade
		Threshold	Maximum	Bin grade	
1	Combined	0.67	5.27	0.48	97th percentile
2	Combined	15.43	388.6	27.27	Bin median
3	Combined	1.64	3.41	2.77	Bin mean
Ag g/t					
Domain	**Subdomain**	**Upper bin (>99%ile) Ag g/t**			**Source of bin grade**
		Threshold	**Maximum**	**Bin grade**	
1	Combined	2.65	5.60	1.25	97th percentile
2	Combined	32.8	122.6	52.99	Bin median
3	Combined	2.85	3.15	3.10	Bin mean

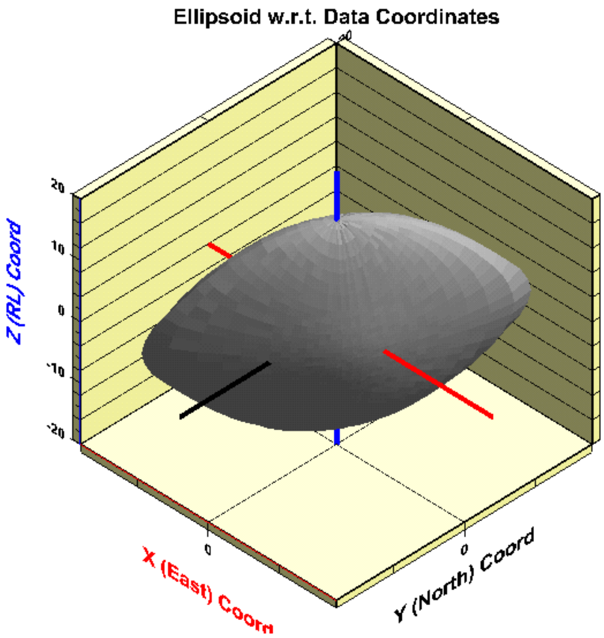
14.3.6 Variogram Models

Domains 1 and 3 contain too few mineralized composites for reliable variogram modelling and for gold and silver indicator variograms modelled from Domain 2 composites were 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 14.6 presents a three dimensional variogram surface map of the median indicator variogram model for gold at variogram value of 0.7.

Figure 14.6 WD Domain 2 - 3D Variogram Plot



14.3.7 Estimation Parameters

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

The three progressively more relaxed search criteria used for MIK estimation are presented in Table 14.10. Search ellipsoids were aligned with dominant domain mineralization orientation.

The resource estimates include a variance adjustment to give gold estimates of recoverable resources above gold cut off grades for selective mining (SMU) dimensions of 5 m east by 5 m north by 2.5 m in elevation. The variance adjustments were applied using the direct lognormal method and the adjustment factors listed in Table 14.11.

For each block model panel, E-type silver estimates were assigned to the recoverable estimates at gold cut-off grades. This approach reflects the moderate correlation between gold and silver grades and generally comparatively low silver grades.

Table 14.10 WD Search Criteria

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

Table 14.11 WD Variance Adjustment Factors

Domain	Block/ Panel	Information Effect	Total Adjustment
1	0.264	0.663	0.175
2	0.264	0.663	0.175
3	0.264	0.663	0.175

14.3.8 Resource Classification

Resource model panels were initially classified as Indicated or Inferred on the basis of search pass and a wire-frame outlining more closely drilled portions of the mineralisation. Figure 14.4 shows the surface projection of this wire-frame.

Panels within the classification wire-frame informed by search 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.

A proportionally small number of near surface search pass 3 panels, for which the distribution of informing composites does not meet search pass 1 and 2 octant requirements were re-classified as Indicated. These panels lie within areas of generally Indicated estimates and represent a small proportion of estimated resources.

14.3.9 Bulk Densities

Bulk densities were assigned to the GSS and WD models by rock type and oxidation type using the values shown in Table 14.12 and supplied geological wire-frames. These values reflect the average of available density measurements for each zone.

For GSS, densities were assigned by the individual rock type wire-frames supplied by Orca which are subdivided by rock type and deposit area. For WD, densities were assigned by rock type and oxidation.

Table 14.12 Bulk Densities

GSS Densities (t/bcm)			
Wire-Frame	**Oxide**	**Transition**	**Fresh**
EZ-FQSP	2.50	2.63	2.77
050-FQSP	2.49	2.64	2.80
NE-SBRR	2.52	2.66	2.68
EZ-SBRR	2.52	2.66	2.68
EZ-PGSS	2.52	2.67	2.75
050-SBRR	2.52	2.66	2.68
FEZ-SRR	2.41	2.55	2.79
EZ-SRR	2.42	2.54	2.69
TJ-FQSP	2.49	2.64	2.80
MZ-FQSP	2.46	2.64	2.82
MZ-PGSS	2.56	2.67	2.80
NE-FQSP	2.49	2.64	2.80
NE-PGSS	2.53	2.67	2.77
VAN	2.55	2.64	2.78
WD Densities (t/bcm)			
Rock Type	**Oxide**	**Transition**	**Fresh**
VC1	2.45	2.60	2.95
VC3	2.45	2.60	2.85
QPR	2.50	2.65	2.75
VAN	2.55	2.65	2.80

14.4 Model Reviews

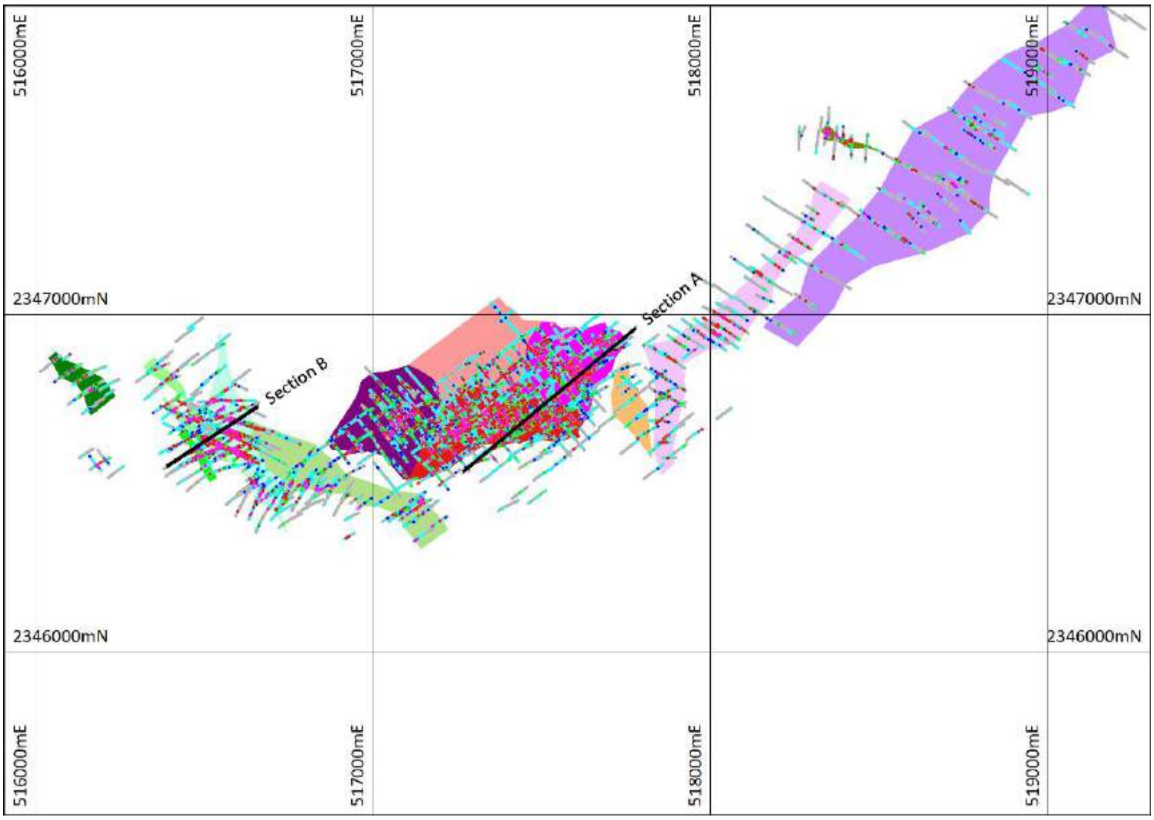
14.4.1 Plots of the Models

Figure 14.8 and Figure 14.9 show representative cross-sections of the GSS and WD resource models respectively. These plots show the resource model panels scaled by the estimated proportion above 0.6 g/t cut-off, and coloured by the estimated gold grade above this cut-off relative to the resource domains and drillholes traces coloured by two metre composited gold grades. Indicated panels are shown as solid colour and Inferred blocks are hatched. Figure 14.7 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 models have been presented. The model blocks plotted are only those that contain an estimated resource above 0.6 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 coordinate and therefore introduces the apparent miss-match between data and the resource model blocks.

Figure 14.7 Plan View of Section Traces

GSS



WD

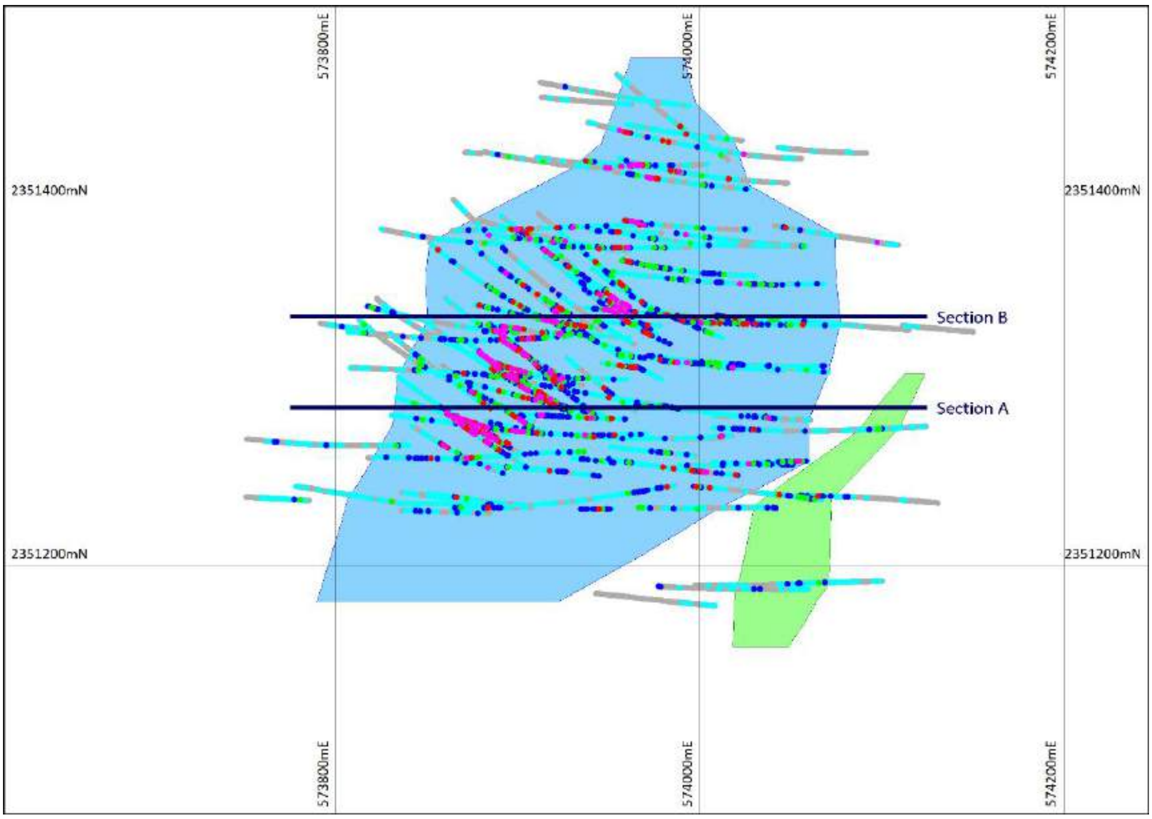
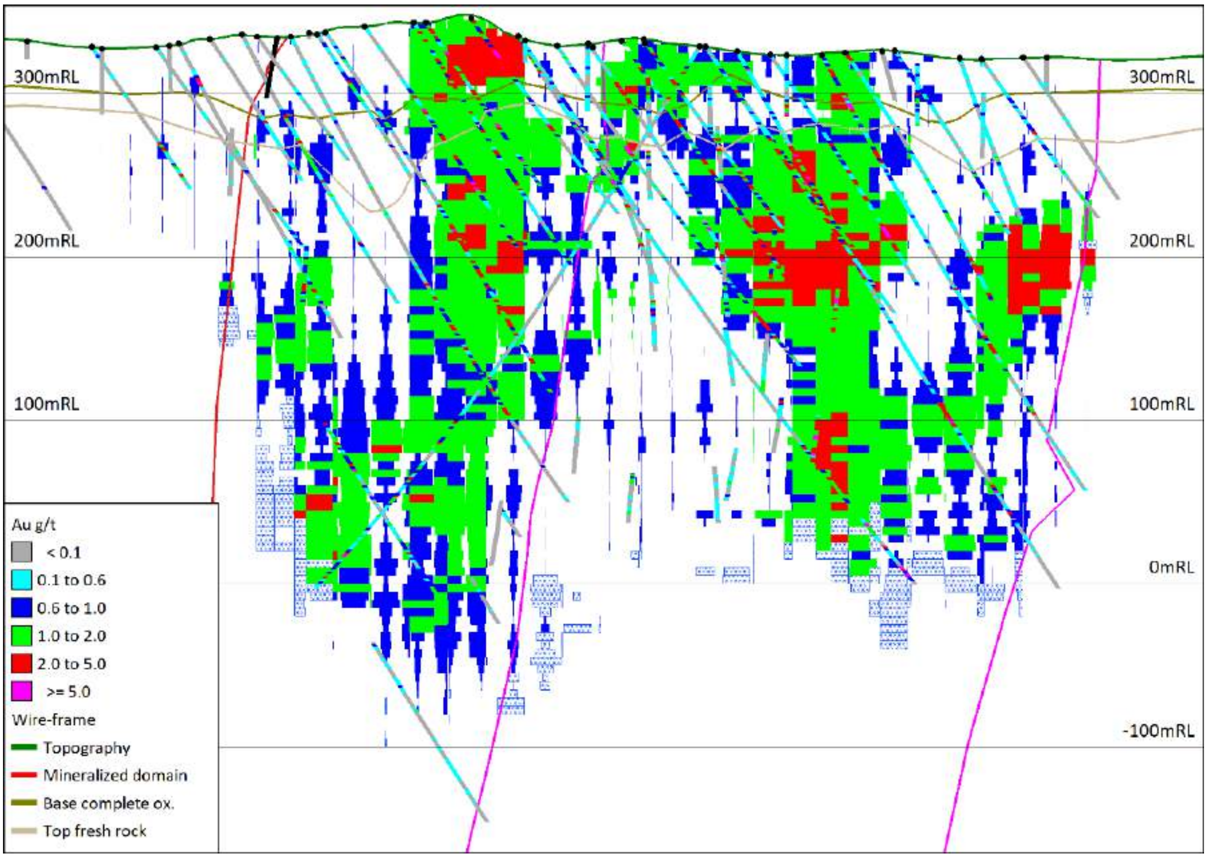


Figure 14.8 GSS Model Blocks at 0.5 g/t Cut-off

Section A



Section B

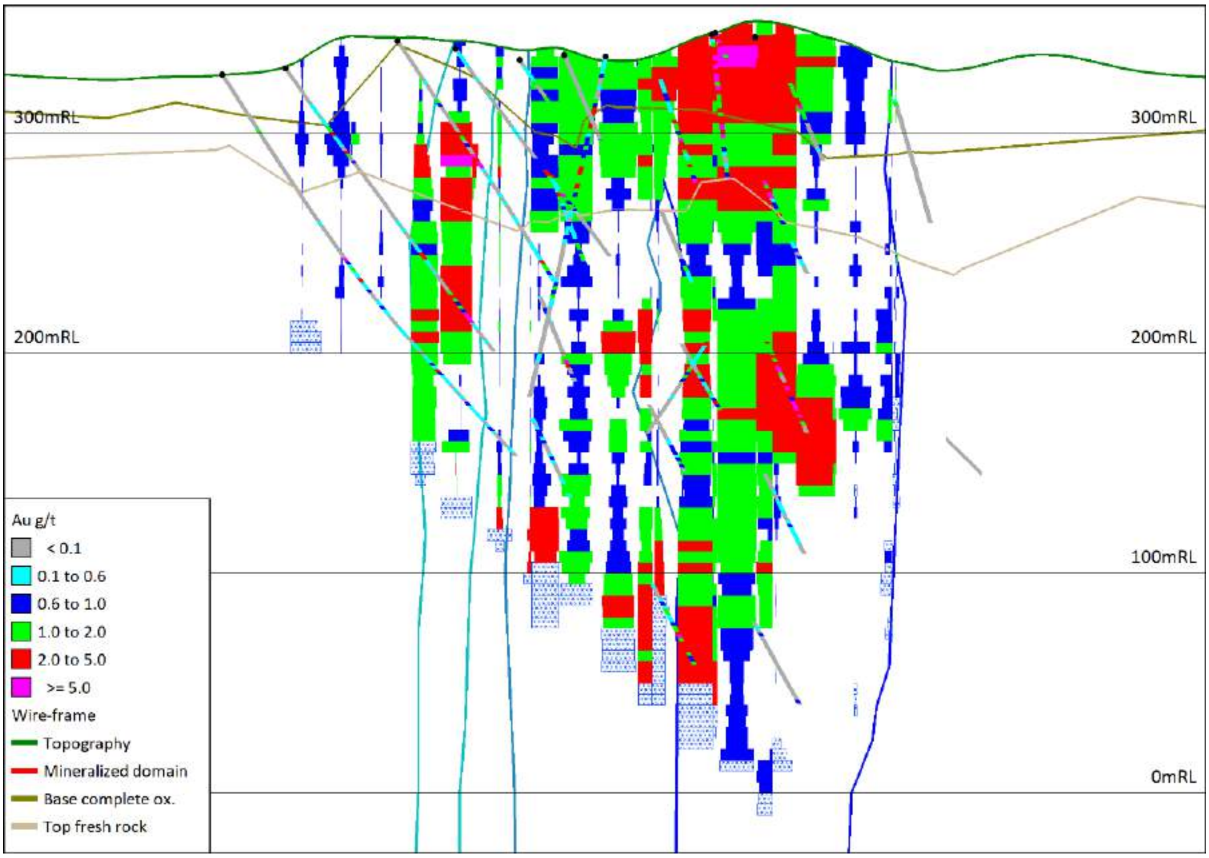
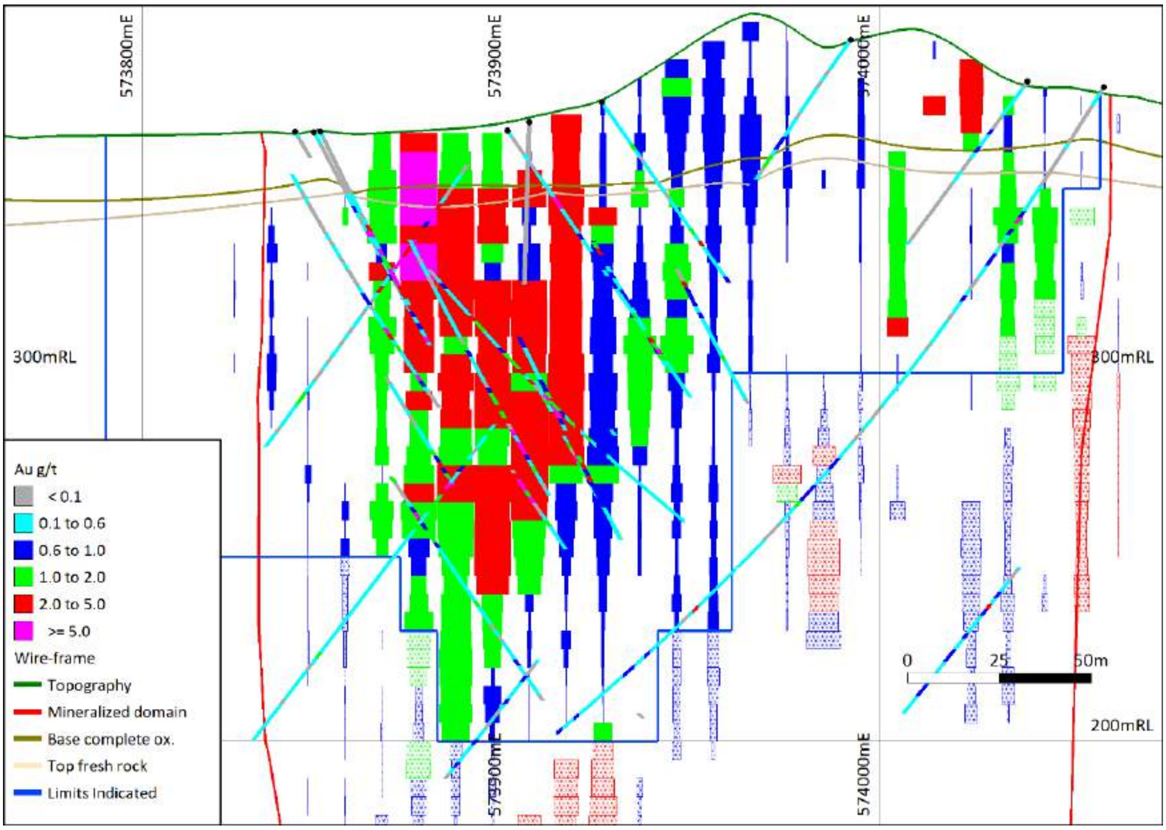
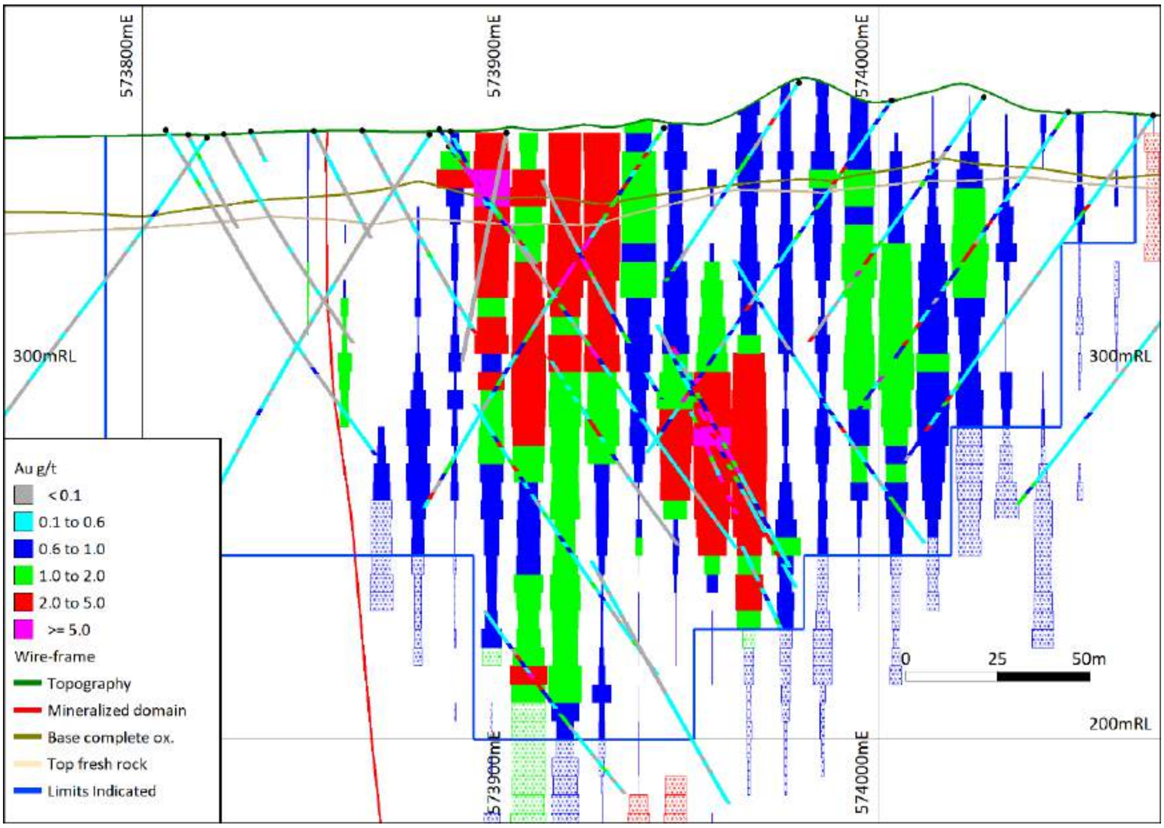


Figure 14.9 WD Model Blocks at 0.5 g/t Cut-off

Section A



Section B



14.4.2 Model Validation

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 14.10 and Figure 14.11 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 14.10 and Figure 14.11 show that although, as expected the average block grades estimated by MIK are smoothed compared to the average composite grades, they generally closely follow the trends shown by the composite mean grades with the exception of areas of variably spaced or limited sampling. There are minor local deviations between the model and composite trends seen on the plots and these are influenced the following features.

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

Figure 14.10 GSS Average Estimated Panel Grades vs. Compsite Grades

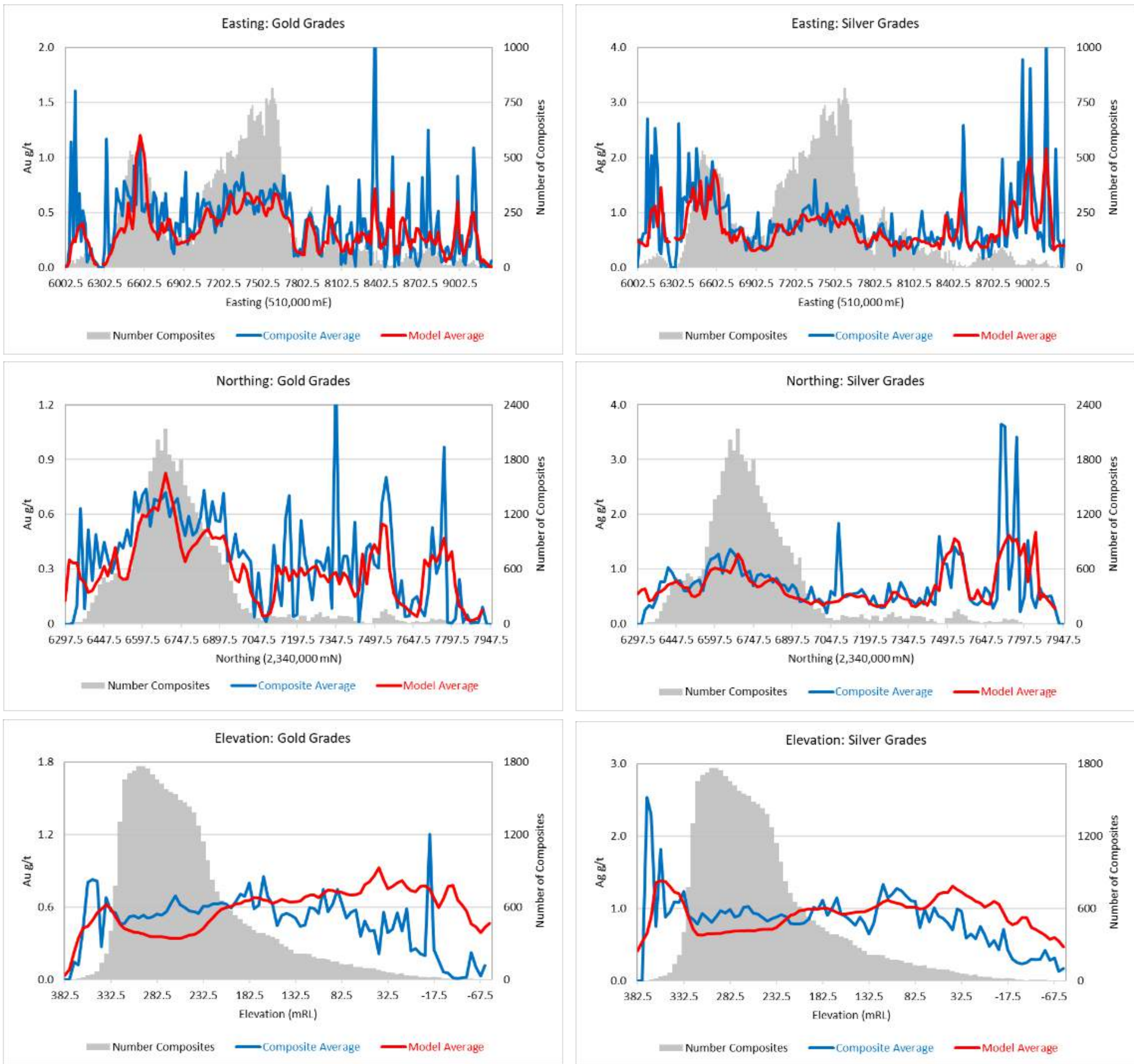
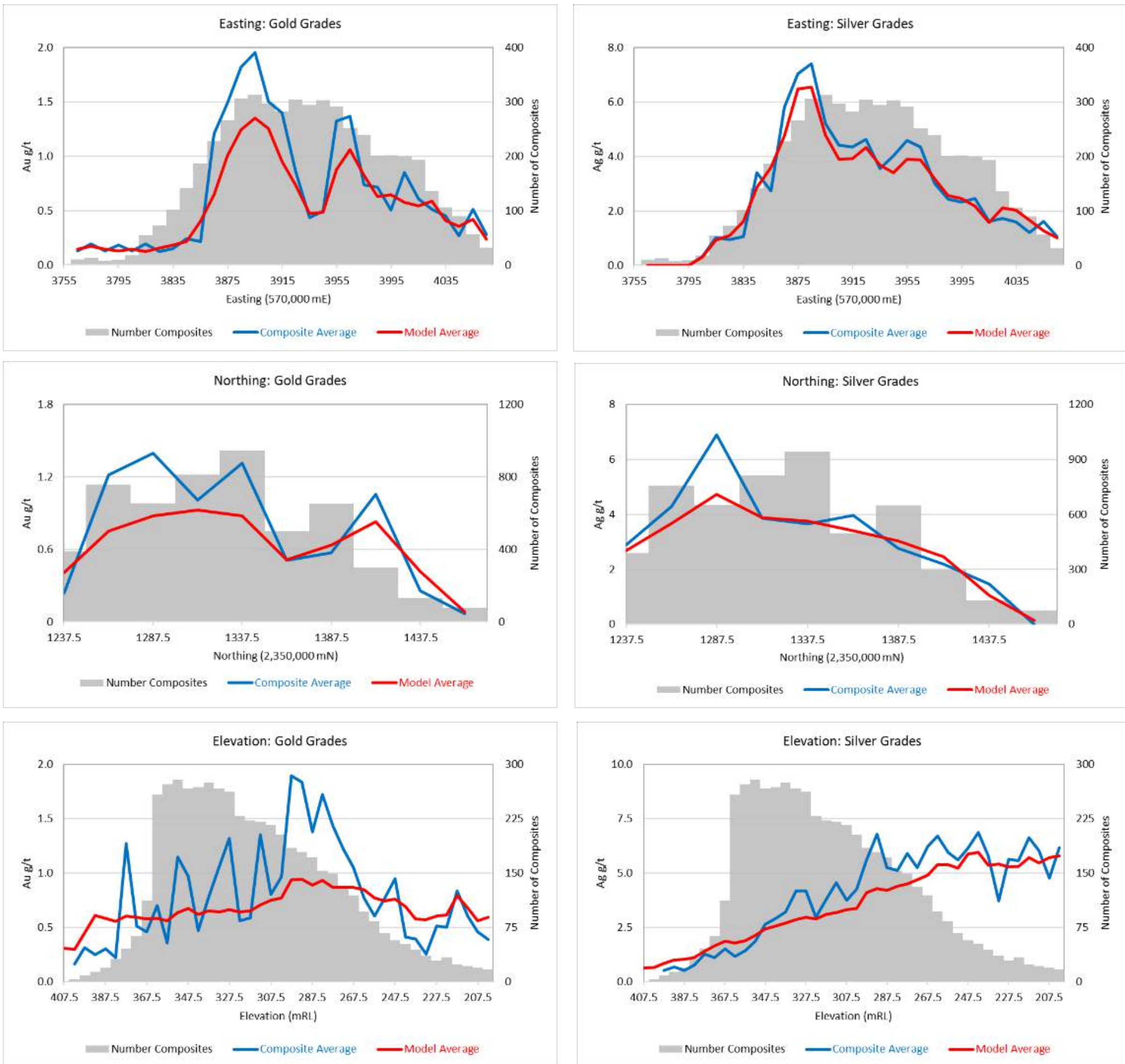


Figure 14.11 WD Average Estimated Panel Grades vs. Compsite Grades



14.5 Resource Estimates

Table 14.13 shows the current Mineral Resource Estimates for GSS and WD for a range of cut-off grades. Table 14.14 shows the estimates at 0.60 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 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.

GSS Mineral Resource estimates are truncated at 350 m depth with around 90% of the Indicated and Inferred resources occurring at depths of less than 240 and 300 m respectively. The WD estimates extend to around 220 m depth with around 90% of the Indicated and Inferred resources occurring at depths of less than 95 m and 150 m respectively.

For GSS combined oxidized and transitional material hosts around 35% and 15% of the Indicated and Inferred resources respectively with the remainder lying in fresh rock. For WD combined oxidized and transitional material hosts around 18% and 12% of the Indicated and Inferred resources respectively.

Table 14.13 January 2018 Mineral Resource Estimate

GSS										
Cut off Au g/t	Indicated					Inferred				
	Mt	Au g/t	Ag g/t	Au koz	Ag koz	Mt	Au g/t	Ag g/t	Au koz	Ag koz
0.3	129.0	0.92	1.19	3,823	4,954	35.4	0.8	1.1	901	1,289
0.4	106.9	1.04	1.29	3,575	4,448	27.0	0.9	1.2	807	1,058
0.5	89.5	1.16	1.39	3,326	4,002	21.1	1.1	1.3	722	873
0.6	**75.6**	**1.27**	**1.48**	**3,080**	**3,607**	**16.9**	**1.2**	**1.4**	**648**	**745**
0.7	64.0	1.38	1.58	2,839	3,249	13.7	1.3	1.5	582	640
0.8	54.5	1.49	1.67	2,610	2,924	11.3	1.4	1.5	524	558
0.9	46.5	1.60	1.76	2,392	2,629	9.4	1.6	1.6	471	487
1.0	39.8	1.71	1.84	2,187	2,359	7.8	1.7	1.7	424	427
1.1	34.1	1.82	1.93	1,994	2,114	6.6	1.8	1.8	382	375
1.2	29.3	1.93	2.01	1,815	1,892	5.6	1.9	1.9	344	331

WD										
Cut off Au g/t	Indicated					Inferred				
	Mt	Au g/t	Ag g/t	Au koz	Ag koz	Mt	Au g/t	Ag g/t	Au koz	Ag koz
0.3	7.8	1.23	4.70	311	1,184	4.9	0.7	3.1	106	477
0.4	6.3	1.46	4.98	293	1,003	3.3	0.8	3.2	89	337
0.5	5.1	1.68	5.22	277	862	2.3	1.0	3.4	75	250
0.6	**4.3**	**1.90**	**5.45**	**262**	**751**	**1.6**	**1.2**	**3.6**	**63**	**191**
0.7	3.7	2.12	5.65	249	664	1.2	1.4	3.8	54	149
0.8	3.2	2.32	5.84	237	595	0.9	1.6	4.1	47	119
0.9	2.8	2.52	6.00	227	540	0.7	1.8	4.3	41	97
1.0	2.5	2.71	6.15	218	494	0.6	2.1	4.5	37	81
1.1	2.3	2.89	6.27	210	456	0.5	2.3	4.7	33	69
1.2	2.1	3.05	6.39	202	423	0.4	2.5	4.9	31	59

Combined										
Cut off Au g/t	Indicated					Inferred				
	Mt	Au g/t	Ag g/t	Au koz	Ag koz	Mt	Au g/t	Ag g/t	Au koz	Ag koz
0.3	136.8	0.94	1.40	4,133	6,137	40.3	0.8	1.4	1,007	1,766
0.4	113.2	1.06	1.50	3,868	5,451	30.3	0.9	1.4	896	1,395
0.5	94.6	1.18	1.60	3,603	4,864	23.4	1.1	1.5	797	1,123
0.6	**79.9**	**1.30**	**1.70**	**3,342**	**4,358**	**18.5**	**1.2**	**1.6**	**711**	**936**
0.7	67.7	1.42	1.80	3,088	3,913	14.9	1.3	1.6	636	789
0.8	57.7	1.53	1.90	2,847	3,519	12.2	1.5	1.7	571	677
0.9	49.3	1.65	2.00	2,619	3,169	10.1	1.6	1.8	512	584
1.0	42.3	1.77	2.10	2,405	2,853	8.4	1.7	1.9	461	508
1.1	36.4	1.88	2.20	2,204	2,570	7.1	1.8	1.9	415	444
1.2	31.4	2.00	2.29	2,017	2,315	6.0	1.9	2.0	375	390

Table 14.14 January 2018 Mineral Resource Estimates at 0.6 g/t Cut-off by Oxidation Type

GSS										
Oxid.	**Indicated**					**Inferred**				
	Mt	**Au g/t**	**Ag g/t**	**Au koz**	**Ag koz**	**Mt**	**Au g/t**	**Ag g/t**	**Au koz**	**Ag koz**
Oxide	9.6	1.32	1.41	409	437	1.0	1.0	1.1	31	35
Trans.	13.2	1.22	1.31	517	557	1.5	1.0	1.1	49	55
Fresh	52.7	1.27	1.54	2,153	2,613	14.4	1.2	1.4	567	655
Total	**75.6**	**1.27**	**1.48**	**3,080**	**3,607**	**16.9**	**1.2**	**1.4**	**648**	**745**

WD										
Oxid.	**Indicated**					**Inferred**				
	Mt	**Au g/t**	**Ag g/t**	**Au koz**	**Ag koz**	**Mt**	**Au g/t**	**Ag g/t**	**Au koz**	**Ag koz**
Oxide	0.6	1.90	2.82	34	50	0.1	1.0	1.6	3	6
Trans.	0.2	2.02	3.69	10	18	0.04	0.9	1.6	1	2
Fresh	3.6	1.89	5.93	218	683	1.5	1.2	3.8	59	183
Total	**4.3**	**1.90**	**5.45**	**262**	**751**	**1.6**	**1.2**	**3.6**	**63**	**191**

Combined										
Oxid.	**Indicated**					**Inferred**				
	Mt	**Au g/t**	**Ag g/t**	**Au koz**	**Ag koz**	**Mt**	**Au g/t**	**Ag g/t**	**Au koz**	**Ag koz**
Oxide	10.2	1.35	1.49	443	487	1.1	1.0	1.2	34	41
Trans.	13.4	1.22	1.33	527	575	1.5	1.0	1.2	50	57
Fresh	56.3	1.31	1.82	2,371	3,296	15.9	1.2	1.6	626	838
Total	**79.9**	**1.30**	**1.70**	**3,342**	**4,358**	**18.5**	**1.2**	**1.6**	**711**	**936**

15. MINERAL RESERVE ESTIMATES

15.1 Statement of Reserves

The definition of a mineral reserve, as defined by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), is:

> *"A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include the application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.*
>
> *Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.*
>
> *The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported."*[3]

The mineral reserve classification is also based on the degree of certainty that can be attached to the estimate. This classification is broken into two categories; Probable and Proven. These are defined by the CIM as:

> *A "Probable Mineral Reserve" is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.*
>
> *A "Proven Mineral Reserve" is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.*

This section states the estimated Mineral Reserves for the Galat Sufar South and Wadi Doum deposits on Orca Gold's Block 14 project. Wadi Doum is treated as one deposit, while Galat Sufar South is treated as three separate areas: Main, East and North East. As both deposits are greenfield deposits, there are no existing stockpiles on the property which need to be considered.

Table 15.1 presents a summary of the Block 14 Mineral Reserve tonnages and grades, which have been estimated according to CIM standards. All units are in the metric system with material quantities specified in metric tonnes (or a derivative thereof) and all grades specified in grams per tonne. The

[3] *https://mrmr.cim.org/media/1068/cim_definition_standards_2014.pdf*

Mineral Resource for either deposit does not contain any Measured Mineral Resource material, therefore no Proven Mineral Reserves have been declared.

Table 15.1 November 2018 Mineral Reserve

	Classification	Oxide		Transitional		Fresh		Total	
		'000 tonnes	Au g/t	'000 tonnes	Au g/t	'000 tonnes	Au g/t	'000 tonnes	Au g/t
Main	Probable	4,347	1.27	5,088	1.19	13,488	1.31	22,923	1.28
East	Probable	8,302	0.89	11,236	0.89	30,729	1.05	50,267	0.99
North East	Probable	1,606	0.84	2,192	0.85	367	0.90	4,166	0.85
Total GSS	***Probable***	***14,255***	***1.00***	***18,516***	***0.97***	***44,584***	***1.13***	***77,356***	***1.07***
Wadi Doum	***Probable***	***527***	***1.90***	***119***	***2.37***	***1,941***	***2.49***	***2,588***	***2.36***
Block 14 Total	**Probable**	**14,783**	**1.03**	**18,635**	**0.98**	**46,525**	**1.19**	**79,943**	**1.11**

Cut-off grades vary for each deposit / area and are also dependent on the oxidation state and lithology. This is due to the changing processing costs and metallurgical recoveries associated with these different geological units. Cut-off grades range from 0.30 g/t to 0.55 g/t for GSS, and from 0.60 g/t to 0.85 g/t for Wadi Doum. The methodology for cut-off grade calculations is explained in Section 16 of this report.

The Mineral Reserve estimate was undertaken by Deswik Europe Ltd using the Deswik mine planning software (Version 2018.2) and demonstrated that mining of the deposit is practical and economically viable. More information can be found in Section 16.

15.2 Modifying Factors used in conversion to Mineral Reserve

15.2.1 Introduction

In order to convert a Mineral Resource to a Mineral Reserve a number of factors need to be considered. These are referred to as "Modifying Factors" or "Quantifying Elements". These cover aspects such as mining, processing, geotechnical, environmental and marketing elements, among others.

The CIM Standard states that while the appropriate level of details for any one of these items may vary and is left to the QP to determine suitability, overall the levels of detail and engineering must meet or exceed the criteria contained in the definition of a Preliminary Feasibility Study.

This section briefly discusses each of the key modifying factors and indicates where more information can be obtained.

15.2.2 Mining

Both the GSS and Wadi Doum deposits will be mined by conventional truck / shovel open pit operations. Pit optimisations based on the geological models provided and other factors discussed in this document were undertaken to provide optimal pit shells.

Engineered pit designs were constructed based on the RF=1.0 shell for each deposit and included geotechnical and practical aspects such as ramp access. Reconciliation of contained ounces and value within the designed pits to the shells were typically within 1% for Galat Sufar South and 7% for Wadi Doum. Mining costs were derived from first principles using information from equipment suppliers and then validated through a budgeting exercise with select international mining contractors.

A mining schedule was then developed which also utilised waste dump, stockpile and haul road designs, using an elevated cut-off grade strategy.

More information can be found in Section 16 of this report.

15.2.3 Processing

Processing parameters have been developed for the different deposits, oxidation states and lithologies. Metallurgical processing recoveries are based on extensive laboratory testwork of different lithological units, and processing costs have been derived from testwork and process plant design estimates.

Section 16 contains a description of the parameters used in the calculation of the Mineral Reserve, while Section 17 contains information on the development of recovery methods and process selection.

15.2.4 Geotechnical / Hydrological

SRK Consulting (UK) Ltd has conducted several geotechnical reviews of the project, with the latest being completed in late 2017.

Slope parameters from this report have been used in the optimisations and subsequent pit designs underpinning the Mineral Reserve statement.

Hydrological studies for the pit areas have been conducted by SRK as part of the geotechnical review and, given the location of the project, hydrogeology is not seen to pose a risk to the project. Investigations into a water bearing shear zone in GSS are underway, but the probable effects of this are minimal, if any.

Section 16 contains additional information on the geotechnical and hydrological aspects of the pit areas.

Regional hydrology has also been reviewed and water sources have been located that contain sufficient water for the planned operation of the mine. Section 18.1 contains additional information.

15.2.5 Environmental

The project is located in the Nubian Desert in northern Sudan. Summer maximum temperatures range from 45° – 49°C and precipitation is negligible. There are no permanent water courses and the desert contains few visible plants, with most areas being covered by bare soil or rock.

A tailings storage facility has been designed for the project along with waste rock dumps at both GSS and Wadi Doum. Rehabilitation and mine closure will consist of removal of infrastructure and minimal landform shaping. Section 20 contains more detailed information on the environmental aspects of the project.

15.2.6 Location and infrastructure

The project is located 180 km from the nearest town of Abu Hamad and there is no permanent habitation within 150 km of the project area. The project is accessible by a network of desert roads and tracks with sealed roads connecting Abu Hamad with both the capital Khartoum and the Port of Sudan.

Transportation costs of spares and consumables have been considered as part of the costing of the project. More information can be found in Section 4 and Section 18.

15.2.7 Marketing Elements or Factors

The Mineral Reserve estimation has assumed a gold price of US$1,100 /oz for the optimisation and pit design work and a price of US$1,250 /oz for financial analysis. A silver price of US$15.70 /oz was assumed.

Royalties have been set at 7% and a refining cost of US$4 /oz was applied to the gold.

All prices are quoted in US dollars.

15.2.8 Legal Elements

Applications for mining licences for GSS and Wadi Doum have been submitted to the Sudanese government and the granting of these is dependent on the submission of this Feasibility Study. Both deposits are covered by an EPL.

Section 4.2 has more information on the legal and ownership aspects of the project.

15.2.9 General Costs and Revenue Elements

Costs for the project have been obtained or calculated by the specialists for their respective areas. In most cases, costs have been derived from market enquiries and budget estimates received from suppliers.

Orca Gold has prepared a financial analysis of the project that further demonstrates that the project is economically viable.

Section 22 contains information on the economic analysis.

15.2.10 Social Elements

The project is in a harsh desert environment with no permanent surface water and therefore does not have a sustained population in place.

There are a large number of illegal artisanal miners working in the region. The mining activities are transient, moving from one prospective area to the next and the land shows extensive signs of disturbance from these activities. It is understood that the artisanal miners are not local and come from all regions of Sudan.

Current relationships with the Sudanese government and the mining groups themselves suggests that the presence of these miners near the operation will not cause issues.

Section 20 details the social situation in more detail.

16. MINING METHODS

16.1 Geotechnical Considerations

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

The pits will be mined in rock with a generally strong and competent rock mass. Pit slope stability will be controlled by the structural condition of the rock mass rather than inherent weaknesses.

Thirteen specific geotechnical boreholes (nine in 2016 and six in 2018) have been drilled at Galat Sufar South and Wadi Doum for the geotechnical study in order to gather rock mass and structural information to derive slope parameters. This data complimented the basic geotechnical data from 18 cored boreholes which was provided for the 2016 PEA study. A structural database was created from the logging of downhole geophysics OTV / ATV images, in addition to foliation and joint orientation measurements made in 2016. Structural domains were identified, presenting their own joint set orientations and attributes. Point load testing and laboratory strength testing of drill core was carried out to characterise the rock mass strength.

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

Table 16.1 Slope Design Recommendations

Pit	Domain	Slope Direction	Bench Height (m)	Bench Face Angle (°)	Bench Width (m)	Inter-ramp Angle (°)
Overburden		000 – 360	10	50	5	36.7
GSSW	Domain		20	75	6.5	59.3
GSSE – North Wall	4		20	75	6.5	59.3
	5 – 6		20	75	5	62.6
	7/8		20	75	6.5	59.3
GSSE – South Wall	4		20	70	5.5	57.4
	5		20	70	6.5	55.4
	7/8		20	70	6	56.4
GSS Far East	7/8	000 -360	20	75	6.5	59.3
Wadi Doum		000 - 360	20	75	4	64.9

16.2 Hydrogeology and Mine Dewatering

To date, there has been limited detailed hydrogeological fieldwork completed as indications from the initial drilling campaign showed that groundwater flow and occurrence will be limited.

Shear zones occur cross-cutting the Main Zone and East Zone of the GSS deposit. SRK observed that these zones are unlikely to carry significant quantities of groundwater and that flow is further limited due to the characteristics of these shear zones and local hydrogeology. This was based on available drilling observation data and personal communication.

SRK has concluded that a fairly poorly developed aquifer system exists within the proposed mining area. Influx of water may occur along geological contacts, but the static groundwater level does not indicate a saturated aquifer. Therefore, as the pit develops, dry rock will be encountered, and water will only occur in cracks, fractures, bedding planes and associated with structural geology.

It is expected that groundwater flow rates will be between 0.1 and 2.0 l/s and the flows will be limited to relatively short periods of time. Dewatering of this flow can be accomplished with portable pumps.

Investigations into a water bearing shear zone in GSS are underway, but the probable effects of this are minimal if any.

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. A risk analysis will need to be conducted to determine the impact on mining operations of these events and additional pumping capacity may be required to be on standby as the pits increase in depth.

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 Wadi Doum deposit will be exploited through a single pit approximately 130 m deep. The GSS deposit will be exploited by eight separate pits, six of which are shallow oxide / transitional pits with minimal fresh material. The other two pits (East and Main) are approximately 260 and 270 m deep respectively.

There is scope for larger pits under improved geotechnical or financial conditions, or an increase in defined mineralization below the base of the pits.

16.4 Mine Optimization

16.4.1 Cut-off Grade Determination

Both the GSS and Wadi Doum block models are partial percentage or proportional models. Grade bins were spaced at 0.05 g/t intervals from 0.3 g/t to 1.0 g/t, with additional bin divisions at 1.1 g/t, 1.2 g/t, 1.3g/t, 1.4g/t, and 1.5 g/t. In order to process these models, it was necessary to determine cut-off grades prior to running the optimizations.

Cut-off grades were calculated for oxide, transitional and fresh material and different lithologies for Wadi Doum, Main, East, and North East pits. This division was based on the availability of different processing costs and processing recoveries for these pits.

Processing recoveries for the optimisation were provided by Orca Gold and Lycopodium as fixed numbers for both gold and silver. Recoveries were provided for each oxidation domain and each lithology for each zone. The tables below (Table 16.2 to Table 16.5) show the metallurgical recoveries for GSS and Wadi Doum.

Table 16.2 Optimisation Processing Recoveries for Main

Pit	Lithology	Oxide State	Au Processing Recovery	Ag Processing Recovery
GSS Main	PGSS	Oxide	91.78%	7.44%
		Transitional	82.74%	26.26%
		Fresh	79.86%	42.97%
	FQSP	Oxide	89.26%	10.54%
		Transitional	80.53%	23.43%
		Fresh	70.74%	39.59%
	VAN	Oxide	85.11%	9.444%
		Transitional	83.86%	11.94%
		Fresh	81.65%	47.76%

Table 16.3 Optimisation Processing Recoveries for East

Pit	Lithology	Oxide State	Au Processing Recovery	Ag Processing Recovery
GSS East	PGSS	Oxide	93.81%	7.40%
		Transitional	83.92%	27.87%
		Fresh	86.61%	33.43%
	FQSP	Oxide	86.50%	11.00%
		Transitional	82.77%	28.73%
		Fresh	79.02%	43.27%
	SRR	Oxide	90.00%	11.55%
		Transitional	85.63%	29.94%
		Fresh	77.41%	43.85%
	VAN	Oxide	82.86%	10.07%
		Transitional	90.03%	24.07%
		Fresh	86.03%	35.95%

Table 16.4 Optimisation Processing Recoveries for North East

Pit	Lithology	Oxide State	Au Processing Recovery	Ag Processing Recovery
GSS Far East	PGSS	Oxide	91.04%	5.83%
		Transitional	81.59%	34.69%
		Fresh	86.61%	33.43%
	FQSP	Oxide	95.95%	6.99%
		Transitional	82.77%	34.69%
		Fresh	79.02%	43.27%
	VAN	Oxide	82.86%	10.07%
		Transitional	90.03%	34.69%
		Fresh	86.03%	35.95%

Table 16.5 Optimisation Processing Recoveries for Wadi Doum Oxide and Transitional Material

Pit	Lithology	Oxide State	Au Processing Recovery	Ag Processing Recovery
Wadi Doum	All	Oxide	87.21%	1.75%
		Transitional	83.53%	29.72%
	QPR	Fresh	77.85%	32.04%
	VAN	Fresh	80.80%	37.54%
	VC1	Fresh	83.74%	41.31%
	VC3	Fresh	67.96%	39.28%

Processing costs for both GSS and Wadi Doum were provided by Lycopodium based on recent test work. Fixed processing costs have been estimated at US$21.3M which equates to US$3.55 /t processed. Variable processing costs are based on lithology and oxidation state and are shown in Table 16.6.

Table 16.6 Optimisation Variable Processing Costs

Pit	Lithology	Oxide (US$)	Transitional (US$)	Fresh (US$)
GSS Main	PGSS	$10.44	$11.17	$10.16
	FQSP	$10.57	$11.29	$12.28
	VAN	$9.80	$10.44	$10.50
GSS East	PGSS	$9.90	$10.71	$11.31
	FQSP	$9.84	$11.48	$12.85
	SRR	$9.83	$10.76	$14.45
	VAN	$9.96	$10.38	$12.90
GSS Far East	PGSS	$9.97	$9.98	$11.31
	FQSP	$10.03	$9.98	$12.85
	VAN	$10.01	$9.73	$12.90
Wadi Doum	QPR	$7.33	$10.49	$8.27
	VAN	$7.33	$10.49	$8.61
	VC1	$7.33	$10.49	$8.73
	VC3	$7.33	$10.49	$8.85

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

As an elevated cut-off grade strategy will be used, lower grade material from GSS will be stockpiled near the ROM pad and then rehandled as required. An average rehandle cost of US$0.87 /t has been added to all material that is considered to be low grade, based on the grade of the block. The cut-off grades between low-grade and high-grade material are shown in Table 16.7.

Table 16.7 High-Grade Cut-off Grade for GSS Ore

Material	High-Grade Cut-Off Grade (g/t)
Oxide	0.6
Transitional	0.7
Fresh	0.9

The cut-off grades were calculated from first principles, using cost, recovery and revenue information. Tables 16.8 and Table 16.9 show the calculated cut-off grades and corresponding grade bins used in the optimisations.

Table 16.8 Cut-off Grades and MIK Grade Bins Used at Wadi Doum

Material	Calculated Cut-off (g/t)	MIK Grade Bin
Oxide	0.647	0.60
Transitional	0.791	0.75
Fresh QPR	0.761	0.75
Fresh VAN	0.746	0.7
Fresh VC1	0.724	0.7
Fresh VC3	0.898	0.85

Table 16.9 Cut-off Grades and MIK Grade Bins Used at GSS

	PGSS		FQSP		VAN		SRR	
	Calc.	MIK Bin	Calc.	MIK Bin	Calc.	MIK Bin	Calc.	MIK Bin
Main Oxide	0.347	0.30	0.362	0.35	0.352	0.35		
Main Trans.	0.412	0.40	0.428	0.40	0.380	0.35		
Main Fresh	0.388	0.35	0.530	0.50	0.393	0.35		
East Oxide	0.322	0.30	0.347	0.30	0.367	0.35	0.334	0.30
East Trans.	0.390	0.35	0.423	0.40	0.352	0.35	0.384	0.35
East Fresh	0.399	0.35	0.497	0.45	0.458	0.45	0.570	0.55
Far East Oxide	0.334	0.30	0.319	0.30	0.369	0.35		
Far East Trans.	0.373	0.35	0.368	0.35	0.330	0.30		
Far East Fresh	0.399	0.35	0.497	0.45	0.458	0.45		

16.4.2 Cost Data for Optimization

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

For cost calculations, the mining fleet for Wadi Doum and GSS was assumed to comprise of Caterpillar 777 rigid body haul trucks (91 t) with suitably sized loading unit.

Unit costs were determined for the following items:

- Loading.
- Fixed hauling component.
- Drill and Blast.
- Ancillary.
- Mine Admin.

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

Grade control costs were added to the fixed mining costs at a rate of US$0.26 /t ore.

Incremental haulage costs were also determined for the fleet and applied during the optimization process to account for vertical haulage.

16.4.3 Optimization Scenarios

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

GSS

A processing rate of 6.0 Mtpa was selected and only Indicated material was considered for processing. Run 58 represented the chosen Indicated optimisation results which included the geological model provided in September 2018.

Table 16.10 shows the comparison between the 2016 PEA, the 2017 Revised PEA and the 2018 FS update, considering the pit shell with revenue factor 1 for each scenario.

Table 16.10 Comparison of GSS (Main, East, and Far East) Optimization Results

	2016 PEA	2017 Revised PEA	2018 FS
Oxide tonnes (million)	10.5	11.1	15.0
Oxide grade	1.37	1.31	1.00
Transitional tonnes (million)	8.2	8.2	18.7
Transitional grade	1.33	1.35	0.97
Fresh tonnes (million)	12.2	23.2	45.2
Fresh grade	1.61	1.45	1.13
Total crusher feed tonnes (million)	30.8	42.5	78.8
Total crusher feed grade	1.46	1.40	1.07
Waste tonnes (million)	55.7	85.9	116.0
Total mined tonnes (million)	86.5	128.4	194.8
Strip Ratio	1.81	2.02	1.47
Total cost (US$ million)	**$851**	**$1,078**	**$1,471**
Revenue (US$ million)	**$1,261**	**$1,626**	**$2,326**
Value (US$ million)	**$409**	**$547**	**$855**
Recovered Ounces (thousand)	**1,233**	**1,596**	**2,267**

The 2018 Revision for GSS shows a significant increase in ore processed with a decrease in average grade and increase in ounces. This is due to a number of reasons including:

- A higher diesel price increasing the mining costs.

Offsett by:

- Reduced processing costs.
- More favourable processing recoveries.
- Lower mining costs due to increase from 3.4 Mtpa rate to 6 Mtpa rate.
- An expanded geological resource and conversion of Inferred material to Indicated material.

The updated model contained more transitional ore than had been previously identified. This is due in part to the decreasing cut-off grades and the transfer of transitional ore from the Inferred to the Indicated category.

Wadi Doum

The Wadi Doum optimization also considered the change to the pit shells based on changing economics around throughput rates. No consideration was given to treating material locally as this option had been discarded in earlier studies. The higher-grade Wadi Doum feed will be blended with the GSS feed for the first five years of the operation.

Table 16.10 provides a comparison of the Wadi Doum optimizations (examining the Revenue Factor = one shells).

Table 16.11 Comparison of Wadi Doum Optimization Results

	2016 PEA	**2017 Revised PEA**	**2018 FS**
Oxide tonnes ('000)	348.1	452.9	**553.0**
Oxide grade	2.72	2.23	**1.91**
Transitional tonnes ('000)	172.7	114.2	**118.3**
Transitional grade	2.59	2.60	**2.38**
Fresh tonnes ('000)	1530.8	2,038.7	**1,905.3**
Fresh grade	2.74	2.64	**2.51**
Total feed tonnes ('000)	2,051.7	2,605.9	**2,576.6**
Total crusher feed grade	2.73	2.56	**2.37**
Waste tonnes ('000)	7,796.4	8,807.4	**6,976.8**
Total mined tonnes ('000)	9,848.0	11,413.3	**9,553.4**
Strip Ratio	3.80	3.38	**2.71**
Total cost (US$ million)	**$85**	**$97**	**$97**
Revenue (US$ million)	**$152**	**$189**	**$157**
Value (US$ million)	**$67**	**$92**	**$60**
Recovered Ounces	**148,477**	**185,597**	**151,622**

The results of the optimisation for the Feasibility Study show a minor decrease in ore tonnes from the 2017 study, although it should be noted that the 2017 study included inferred material. This suggests that any changes to pricing or recovery have been offset by conversion of Inferred resources to Indicated resources.

In order to assist with the design process, a second optimisation was run on a block model that had reduced block sizes. This assisted with generating more accurate wall angles and aided the design process.

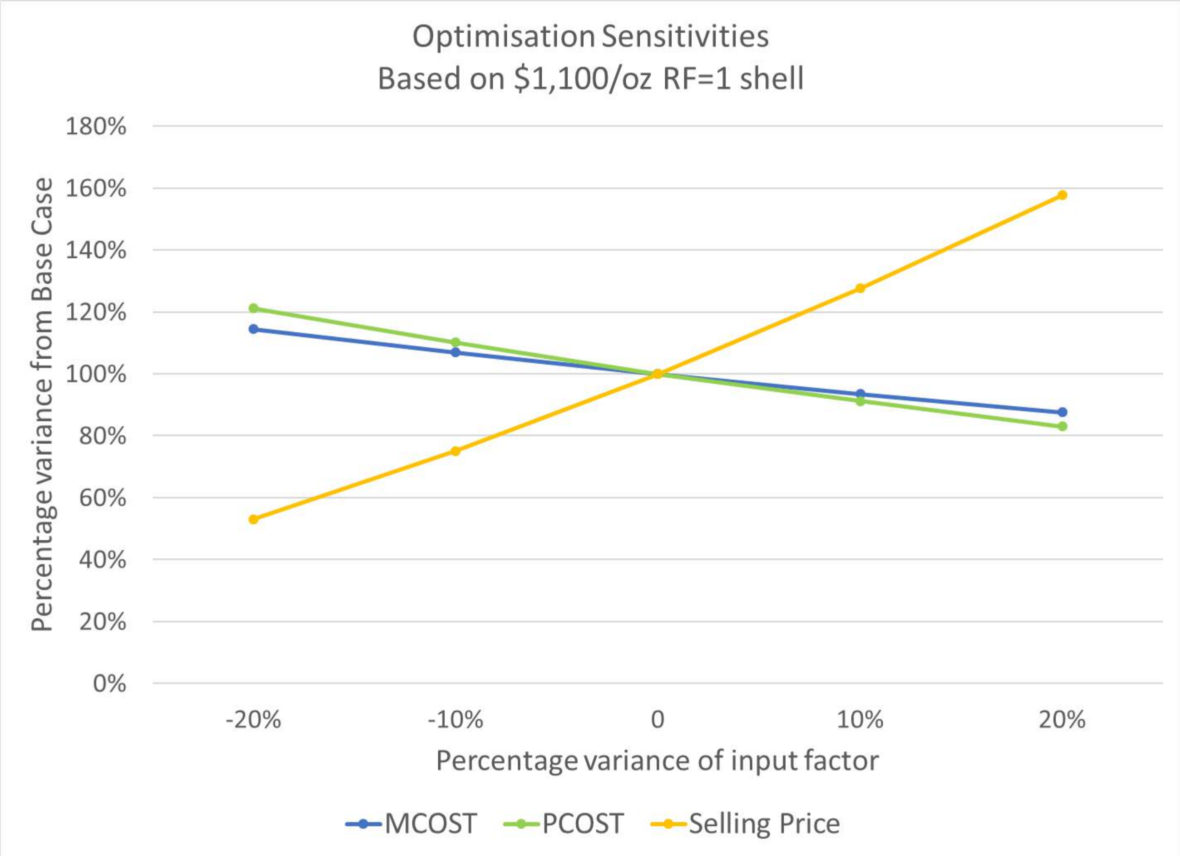
16.4.4 Sensitivity Analysis

A series of optimisations were run on the GSS optimisation with flexed input parameters to determine the sensitivity of the project. The input parameters examined are:

- Mining cost.
- Processing cost.
- Selling price (including royalties and selling costs).

Figure 16.1 shows that the processing cost has more of an impact on the value of the optimisation that mining cost does, however both are less than half of the magnitude of the change in value with change in selling price.

Figure 16.1 Optimisation sensitivity to processing cost, mining cost and selling price



16.5 Mine Design and Sequencing

16.5.1 Pit Shell Selection

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

- GSS: Run 58 – Indicated material at a 6.0 Mtpa Rate using variable cut-offs.
- Wadi Doum: Run 16 – Indicated material at a 6.0 Mtpa Rate using variable cut-offs.

Based on previous optimisations and subsequent pit designs, it has been shown that the slope angles used in the optimisations match very closely to the designed pits. Therefore, where it is normal to use a lower RF shell so as to account for ramps and other geotechnical requirements, it was deemed that in this instance the RF=1 shell could be reproduced accurately in an engineered pit design. Table 16.12 compares the pit shell to design which shows that this is the case, particularly with GSS.

Table 16.12 Comparison of Wadi Doum Optimization Results

	Contained Ounces	Value	Waste
GSS design	99%	99%	98%
Wadi Doum design	107%	92%	104%

16.5.2 Mine Design Parameters

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

Batter, berm and wall angle configurations varied for the different material types (Table 16.1) and ramp widths were varied between double lane and single lane to reflect likely mining practice. Where possible, a “goodbye cut” was designed at the base of each pit to maximise extraction of crusher feed.

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

16.5.3 Pit Statistics

GSS

The GSS pits contribute over 97% of the total crusher feed and 91% of the contained ounces. The 10 GSS pits will be exploited along a strike of over 3.2 km (Figure 16.2).

> ***The total in situ Probable Reserve for the GSS pits is* 77,356 *Mt at 1.07 g/t Au equating to 2.657 million contained ounces with 111.8 mt of waste***

The overall strip ratio for the GSS pits is 1.44:1. Table 16.12 shows the material contained within the GSS pits while Figure 16.3 shows the breakdown of material categories for the crusher feed.

Figure 16.2 GSS Pits

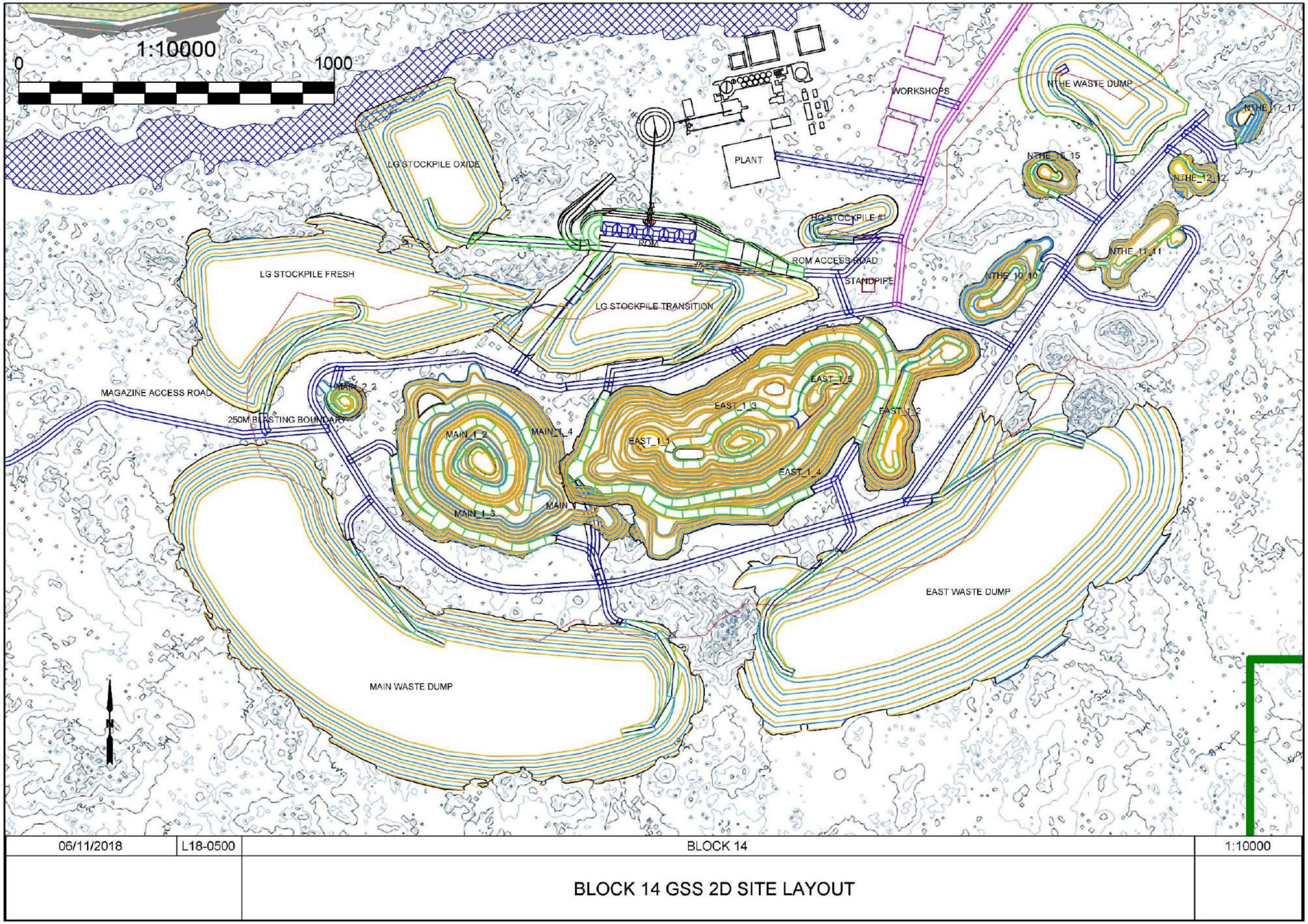
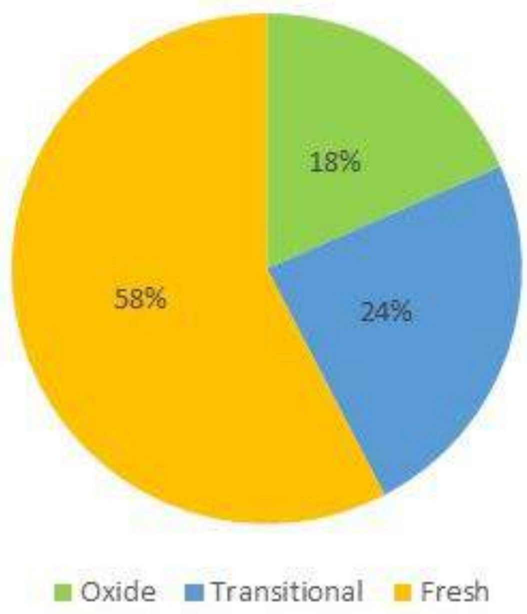


Table 16.13 GSS Pits Inventory

	Oxide		Transitional		Fresh		Total	
	'000 tonnes	Grade	'000 tonnes	Grade	'000 tonnes	Grade	'000 tonnes	Grade
Probable Ore	14,255	1.00	18,516	0.97	44,584	1.13	77,356	1.07
Waste	27,104	-	34,696	-	49,960	-	111,760	-
Total Material	**41,359**	**-**	**53,212**	**-**	**94,544**	**-**	**189,116**	**-**

Figure 16.3 GSS Resource and Material Categories for Crusher Feed



Wadi Doum

Although much smaller than the GSS pits to the west, the Wadi Doum deposit contains mineralization at over 2.2 times the grade. The deposit is exploited through a three stage 130 m deep pit (Figure 16.4). The overall strip ratio for the Wadi Doum pit is 2.89:1.

The total in situ Probable Reserve for Wadi Doum is 2.59 mt at 2.36 g/t producing 197 thousand contained ounces with 7.49 mt of waste

Table 16.14 shows the material contained within the Wadi Doum pit while Figure 16.5 shows the relative proportions of resource and material categories for the crusher feed.

Table 16.14 Wadi Doum Pit Inventory

	Oxide		Transitional		Fresh		Total	
	'000 Tonnes	Grade	'000 Tonnes	Grade	'000 Tonnes	Grade	'000 Tonnes	Grade
Ore	527	1.90	119	2.37	1,941	2.49	2,588	2.36
Waste	2,868	-	648	-	3,972	-	7,488	-
Total Material	**3,395**	**-**	**767**	**-**	**5,913**	**-**	**10,075**	**-**

Figure 16.4 Wadi Doum Pit

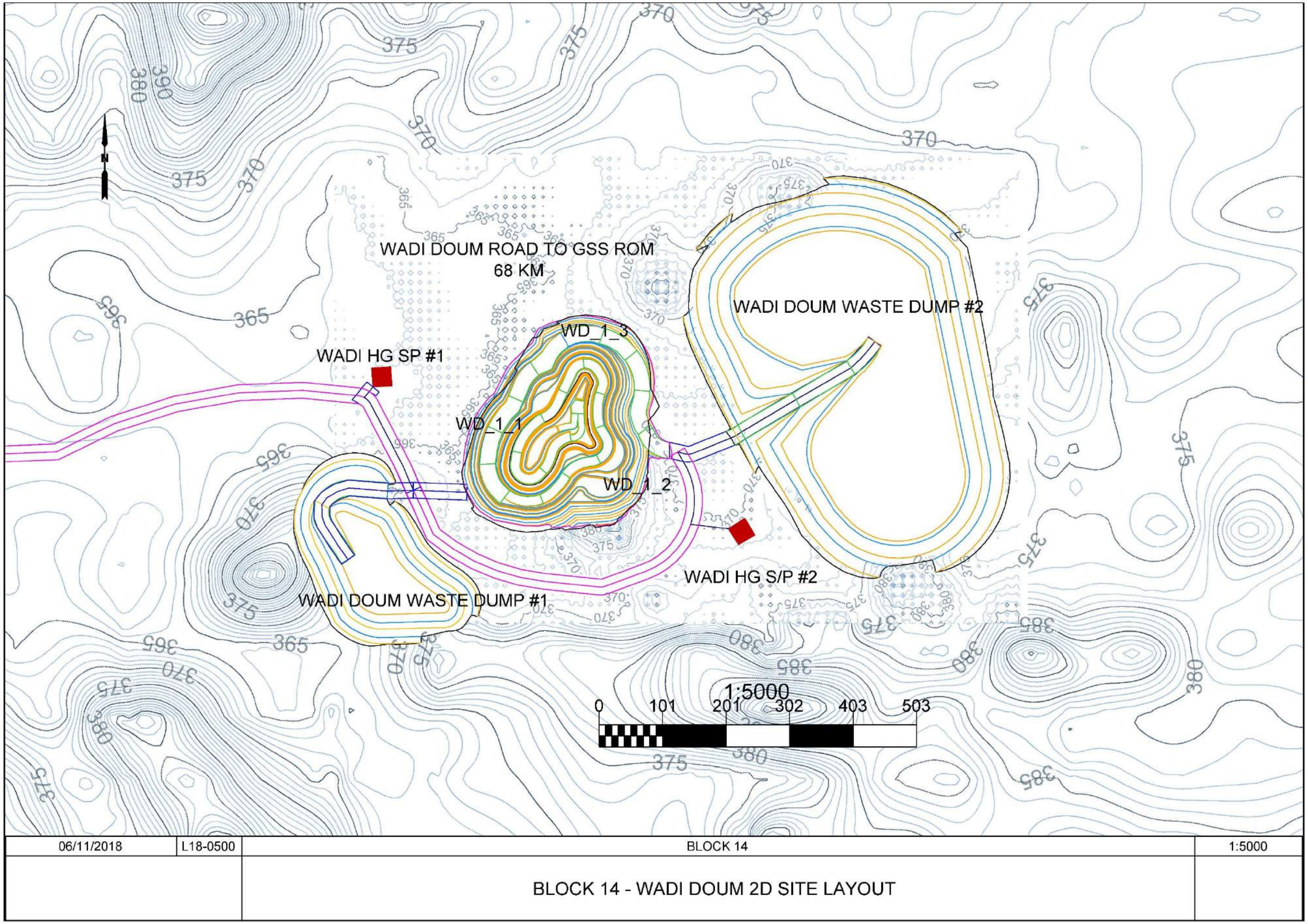
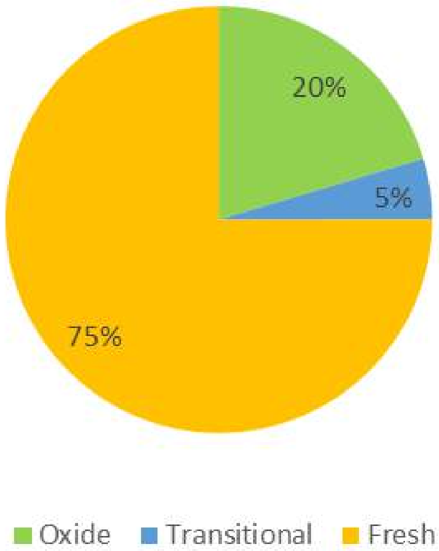


Figure 16.5 Wadi Doum Resource and Material Categories for Crusher Feed



Combined Summary

The two deposits produce a total Probable Reserve of 79.9 Mt @ 1.11 g/t Au amounting to 2.85 million contained ounces.

The previous Revised PEA study produced a total of 44.4 Mt at 1.47g/t Au of treatable material, therefore the results of the Feasibility Study represent a significant improvement.

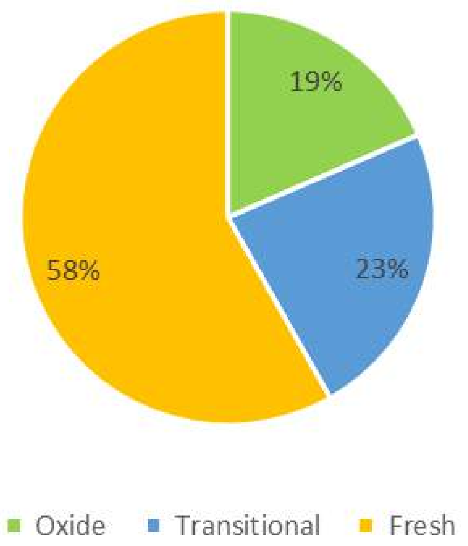
Waste material is 119.2 Mt (inclusive of the Inferred ore) with a strip ratio of 1.49:1. Total material movement is 199.2 Mt.

These results are summarized in Table 16.15 and Figure 16.6.

Table 16.15 Combined Project Inventory

	Oxide		Transitional		Fresh		Total	
	'000 tonnes	Grade	'000 tonnes	Grade	'000 tonnes	Grade	'000 tonnes	Grade
Probable Ore	14,783	1.03	18,635	0.98	46,525	1.19	79,943	1.11
Waste	29,972	-	35,345	-	53,932	-	119,249	-
Total Material	**44,755**	**-**	**53,980**	**-**	**100,457**	**-**	**199,192**	**-**

Figure 16.6 Project Resource and Material Categories for Crusher Feed



16.5.4 Waste Rock Dumps, Haul Roads and ROM Pads

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

A ROM pad for GSS was also designed and can cater for six separate ROM fingers, which will facilitate blending of the crusher feed. A skyway has been designed at the back of the ROM pad which facilitates the construction of the fingers at a height of 10 m, allowing approximately 20,000 t of rock to be stockpiled on each finger. It is envisioned that the crusher will be fed with two front end loaders working in unison. A rockbreaker will be required on a periodic basis to break up any oversize from the pits, although the crusher has been designed to accommodate single boulders up to 900 mm which should manage most ROM feed.

While space has been allocated for a ROM pad at Wadi Doum, no specific design has been completed. It is unlikely that ROM material will accumulate at Wadi Doum to such quantities as to require multiple lifts being constructed, however there is sufficient space at the location to do so.

The optimal processing strategy was determined to be a process of utilising an elevated cut-off grade during mining operations. This will result in higher grade material being fed to the processing plant in the earlier years of the operation and lower grade material being stockpiled on Low Grade stockpiles. The lower grade material is economic and reclamation of this has been included in the mine schedule as required to meet production targets, although primarily towards the end of the mine life.

The maximum stockpile size for the low-grade material is 34.7 Mt. This has been split between two stockpiles adjacent to the crusher, one of which can provide a rolling short haul low grade feed to the crusher when required. A majority of this material will require truck and excavator rehandle to the ROM fingers. As construction of the two main stockpiles nears completion, they are joined by a rock bridge which will allow for easier, quicker and more economic reclaim activities.

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

16.5.5 Analysis of Cut-off Grade and Blending Strategy

As mentioned above, an Elevated Cut-off Grade strategy was used in scheduling the feed to the processing plant. In order to determine the optimal cut-off grade for the high-grade material, several stockpiling scenarios were modelled.

Several processing and blending strategies were then examined. The first was to only process high grade material until it was all treated, then treat the LG material from the stockpiles. The second was to run with five years of high-grade material, then to a 50% feed split between low grade and high grade until the high grade was exhausted, the final remaining low-grade stockpiles would then be fed. The final option was to have no high-grade material at all and treat as the material was mined.

Several combinations of these cut-off grade and blending scenarios were run through the Deswik software on the design pits and the January 2018 block models. A discounted cash flow analysis was then completed for each scenario.

The results indicate that the best discounted cash flow is achieved using the highest cut-offs for the oxide, transition, and fresh material. To achieve the amount of high grade required by this schedule, it was necessary to operate at an unsustainable mining rate of 37.5 Mtpa. This meant an increase in the number of primary loading units to five, requiring five working faces to be active at any one time. This also required the most stockpiling capacity for the lower grade material (+30 Mt) and therefore increased rehandle.

In order to ensure a practical mining schedule was achieved, the maximum movement was restricted to the capacity of four loading units (under 30 Mtpa). This also reduced the requirements for rehandle and stockpile size for the low-grade material. The case that best met these criteria used mid-range cut-off grades and treated all high-grade material first followed but the reclamation of the low-grade stockpiles. This scenario then became the base case for comparisons for the current block model update and variable cost inputs. These elevated cut-off grades were then used in this phase of the study.

16.5.6 Analysis of Strategic Extraction Sequence

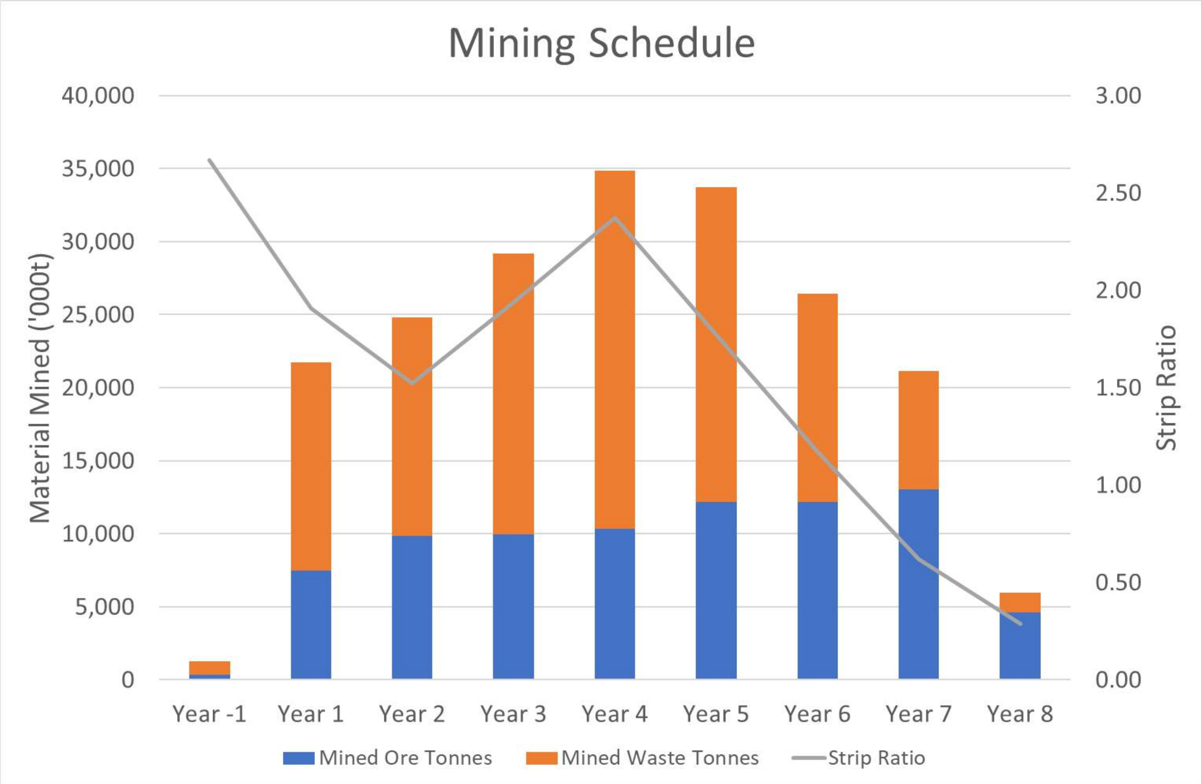
The Australian consulting firm, Cube Consulting, was commissioned to conduct a study to determine the optimal extraction sequence for the operation. The MineMax extraction sequence used by Cube verified that the Deswik schedule was valid and close to optimal.

16.5.7 Mining Schedule

The mining schedule demonstrated that the current reserves would be mined over a period of nine years. The first “pre-production” year, labelled Year-1, consisted of a six-month period where waste stripping was prioritised. It has been assumed that this will commence six months before the commissioning of the processing plant is complete. Given the nature and geometry of the deposit, some ore will be mined during this pre-production period. This ore will be stockpiled in the high-grade and low-grade stockpiles ready for reclaim. Mining occurs at both GSS and Wadi Doum during this time.

The remainder of the GSS and Wadi Doum deposits are mined over the subsequent eight years (Figure 16.7). Wadi Doum is exhausted after five years. The elevated cut-off grade strategy mentioned above was utilised for the entire mining period. This ensured that maximum grades, and therefore ounces, could be obtained in the early years of processing. Low grade ore is stockpiled in low-grade stockpiles and reclaimed as the high-grade or from the pits is exhausted.

Figure 16.7 Mining Schedule by Material



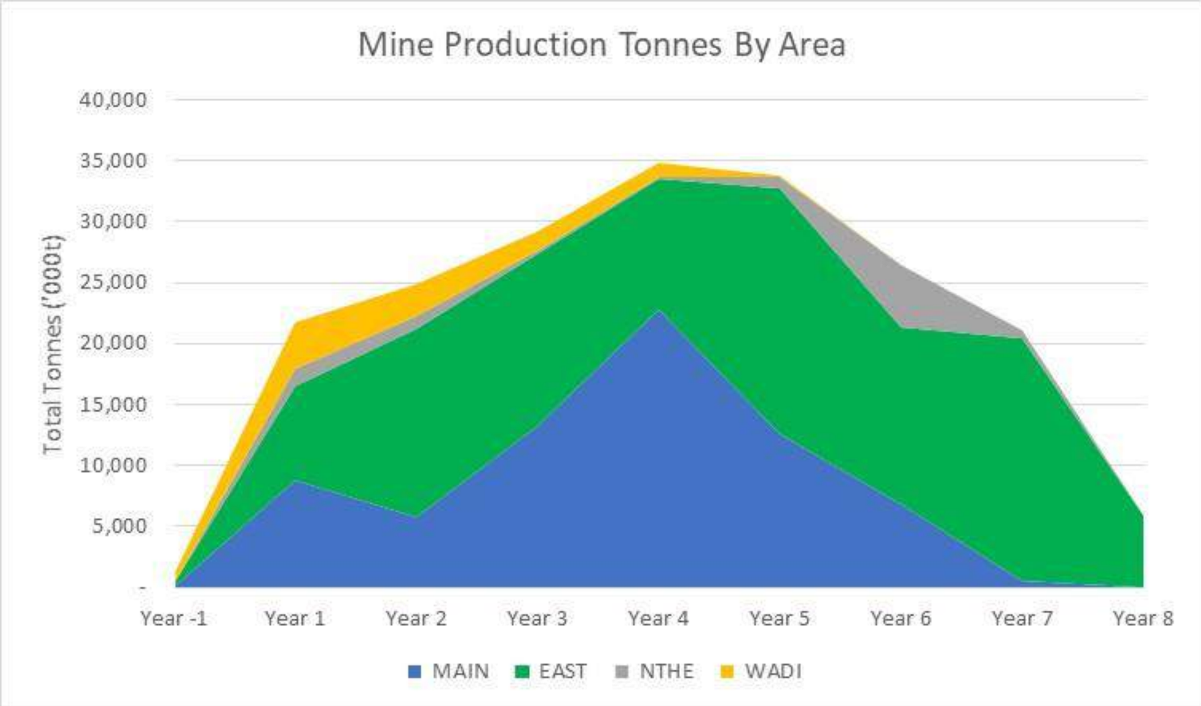
The philosophy of stripping waste as early as possible from a number of cutbacks, with several cutbacks designed for this purpose, has assisted with reducing the waste movement at the end of the mine life and allowed for a better scheduling of cutbacks from previous studies. It also allows for total movement to reduce towards the end of the mine in line with aging equipment, although in practice this would be managed by the mining contractor.

Mining of all areas occurs simultaneously and is driven by the strategic extraction sequence. The bulk of mining occurs in the East and Main pits for most of the mine life, with the North East pits being mined over the entire length of the project. Table 16.16 and Figure 16.8 show the total ex-pit movement for each of the mining areas over the life of the project (in '000 t).

Table 16.16 Ex-pit Movement ('000 t)

Period	Main	East	North East	Wadi Doum	Total
Year -1	121	456	11	699	1,287
Year 1	8,831	7,676	1,493	3,717	21,717
Year 2	5,746	15,511	982	2,584	24,823
Year 3	13,298	14,131	158	1,606	29,193
Year 4	22,748	10,799	53	1,248	34,848
Year 5	12,602	20,146	852	222	33,822
Year 6	6,770	14,601	5,040	0	26,411
Year 7	564	19,845	731	0	21,140
Year 8	0	5,951	0	0	5,951
Total	**70,680**	**109,116**	**9,320**	**10,076**	**199,192**

Figure 16.8 Total Ex-pit Movement by Period



The mine plan is presented in Table 16.17 and the processing schedule is presented in Table 16.18.

Table 16.17 Mining Schedule

	Units	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
GSS Main Ore Tonnes	'000 t	22,923.0	14.5	2,835.0	1,250.4	3,434.0	5,908.1	5,068.3	3,896.5	516.1	
GSS Main Ore Au Grade	g/t	1.28	0.68	1.24	1.52	1.20	1.13	1.35	1.37	1.87	
GSS Main Ore Ag Grade	g/t	1.84	0.72	1.54	2.16	2.02	1.81	1.78	1.81	2.56	
GSS Main Waste Tonnes	'000 t	47,756.7	106.9	5,996.0	4,495.3	9,863.9	16,839.9	7,533.3	2,873.6	47.8	
GSS Main Total Tonnes	'000 t	70,679.8	121.4	8,831.0	5,745.8	13,297.9	22,748.0	12,601.6	6,770.1	563.9	
GSS East Ore Tonnes	'000 t	50,266.9	205.8	3,303.4	7,454.8	5,902.8	3,818.2	6,769.8	6,018.2	12,164.9	4,628.8
GSS East Ore Au Grade	g/t	0.99	0.80	0.93	0.93	0.96	0.91	0.95	0.96	1.06	1.18
GSS East Ore Ag Grade	g/t	1.10	0.60	0.78	0.83	1.15	0.97	1.11	1.11	1.25	1.43
GSS East Waste Tonnes	'000 t	58,850.2	250.1	4,373.0	8,056.6	8,228.2	6,981.0	13,376.4	8,582.7	7,680.2	1,322.0
GSS East Total Tonnes	'000 t	109,117.1	455.9	7,676.4	15,511.4	14,131.1	10,799.3	20,146.2	14,600.9	19,845.2	5,950.8
GSS NEZ Ore Tonnes	'000 t	4,165.7	10.3	797.5	530.2	0	0.9	194.6	2,261.8	370.5	
GSS NEZ Ore Au Grade	g/t	0.85	0.97	0.85	0.75	0.00	0.40	1.87	0.80	0.78	
GSS NEZ Ore Ag Grade	g/t	0.76	0.51	0.54	0.57	0.00	0.28	0.98	0.81	1.38	
GSS NEZ Waste Tonnes	'000 t	5,152.9	0.6	695.1	451.5	158.1	51.8	657.7	2,778.1	360.1	
GSS NEZ Total Tonnes	'000 t	9,318.6	10.8	1,492.6	981.6	158.1	52.7	852.3	5,039.9	730.6	
Wadi Doum Ore Tonnes	'000 t	2,587.6	120.0	528.9	600.0	600.0	600.0	138.6			
Wadi Doum Ore Au Grade	g/t	2.36	2.51	1.85	2.38	2.37	2.73	2.55			
Wadi Doum Ore Ag Grade	g/t	5.69	3.89	2.69	5.60	6.63	7.15	8.62			
Wadi Doum Waste Tonnes	'000 t	7,488.3	578.7	3,188.2	1,984.0	1,006.2	647.8	83.3			
Wadi Doum Total Tonnes	'000 t	10,075.9	698.7	3,717.2	2,584.0	1,606.2	1,247.8	222.0			
Total Ore Tonnes	'000 t	79,943.2	350.6	7,464.9	9,835.4	9,936.9	10,327.2	12,171.3	12,176.5	13,051.6	4,628.8
Total Ore Au Grade	g/t	1.11	1.38	1.10	1.08	1.13	1.14	1.15	1.06	1.08	1.18
Total Ore Ag Grade	g/t	1.44	1.73	1.18	1.27	1.78	1.81	1.48	1.28	1.30	1.43
Total Waste Tonnes	'000 t	119,248.1	936.3	14,252.3	14,987.4	19,256.4	24,520.5	21,650.7	14,234.4	8,088.1	1,322.0
Total Tonnes	'000 t	199,191.3	1,286.9	21,717.2	24,822.8	29,193.3	34,847.8	33,822.0	26,410.8	21,139.7	5,950.8
Overall Strip Ratio		1.49:1	2.67:1	1.91:1	1.52:1	1.94:1	2.37:1	1.78:1	1.17:1	0.62:1	0.29:1

Table 16.18 Processing Schedule

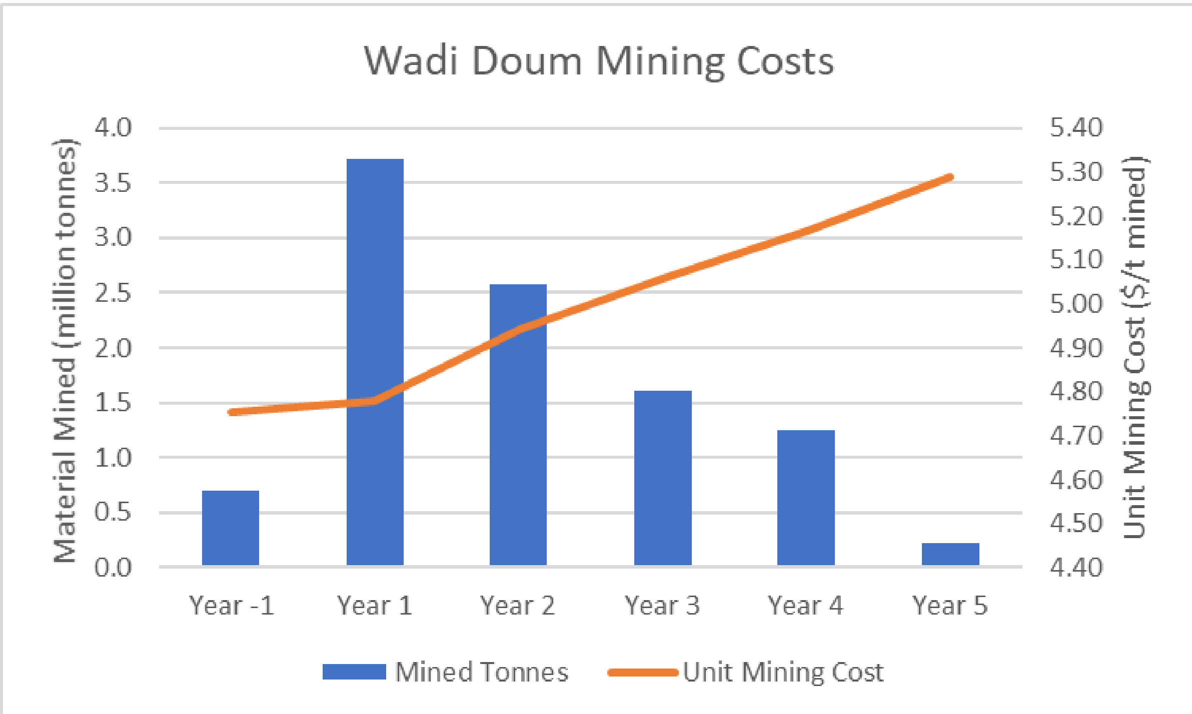
	Units	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
Galat Sufar South																	
GSS Oxide Processed Tonnes	'000 t	14,255	0	3,292	2,537	1,250	667	378	445	135	837	105	0	3	26	1,580	3,000
GSS Oxide Processed Grade	g/t	1.00	0.00	1.37	1.42	1.23	0.99	1.23	1.01	0.81	0.74	0.77	0.00	0.60	0.51	0.49	0.49
GSS Transition Processed Tonnes	'000 t	18,516	0	819	2,534	3,114	2,363	648	1,031	412	225	0	0	0	2,608	4,420	343
GSS Transition Processed Grade	g/t	0.97	0.00	1.23	1.26	1.39	1.23	1.26	1.02	0.86	0.82	0.00	0.00	0.00	0.55	0.57	0.55
GSS Fresh Processed Tonnes	'000 t	44,584	0	89	129	1,036	2,370	4,787	4,525	5,453	4,939	5,895	6,000	5,997	3,366	0	0
GSS Fresh Processed Grade	g/t	1.13	0.00	1.30	1.30	1.35	1.63	1.71	1.63	1.55	1.22	0.72	0.68	0.68	0.68	0.00	0.61
GSS Total Processed Tonnes	**'000 t**	**77,356**	**0**	**4,200**	**5,200**	**5,400**	**5,400**	**5,812**	**6,000**	**6,000**	**6,000**	**6,000**	**6,000**	**6,000**	**6,000**	**6,000**	**3,343**
GSS Total Processed Grade	**g/t**	**1.07**	**0.00**	**1.34**	**1.34**	**1.35**	**1.37**	**1.63**	**1.48**	**1.49**	**1.14**	**0.72**	**0.68**	**0.68**	**0.62**	**0.55**	**0.49**
Wadi Doum																	
WD Oxide Processed Tonnes	'000 t	527	0	486	41	0	0	0	0	0	0						
WD Oxide Processed Grade	g/t	1.90	0.00	1.92	1.68	0.00	0.00	0.76	0.00	0.00	0.00						
WD Transition Processed Tonnes	'000 t	119	0	59	42	0	0	18	0	0	0						
WD Transition Processed Grade	g/t	2.37	0.00	2.70	2.44	0.00	0.00	1.06	0.00	0.00	0.00						
WD Fresh Processed Tonnes	'000 t	1,941	0	54	517	600	600	170	0	0	0						
WD Fresh Processed Grade	g/t	2.49	0.00	2.40	2.42	2.34	2.74	2.35	0.00	0.00	0.00						
WD Total Processed Tonnes	**'000 t**	**2,588**	**0**	**600**	**600**	**600**	**600**	**188**	**0**	**0**	**0**						
WD Total Processed Grade	**g/t**	**2.36**	**0.00**	**2.04**	**2.37**	**2.34**	**2.74**	**2.23**	**0.00**	**0.00**	**0.00**						
	Units	**Total**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**	**Year 14**
Total Block 14																	
Oxide Processed Tonnes	'000 t	14,783	0	3,779	2,577	1,250	667	378	445	135	837	105	0	3	26	1,580	3,000
Oxide Processed Grade	g/t	1.03	0.00	1.44	1.42	1.23	0.99	1.23	1.01	0.81	0.74	0.77	0.00	0.60	0.51	0.49	0.49
Transition Processed Tonnes	'000 t	18,635	0	878	2,577	3,114	2,363	665	1,031	412	225	0	0	0	2,608	4,420	343
Transition Processed Grade	g/t	0.98	0.00	1.33	1.28	1.39	1.23	1.26	1.02	0.86	0.82	0.00	0.00	0.00	0.55	0.57	0.55
Fresh Processed Tonnes	'000 t	46,525	0	143	646	1,636	2,970	4,957	4,525	5,453	4,939	5,895	6,000	5,997	3,366	0	0
Fresh Processed Grade	g/t	1.18	0.00	1.72	2.20	1.71	1.85	1.73	1.63	1.55	1.22	0.72	0.68	0.68	0.68	0.00	0.61
Total Processed Tonnes	**'000 t**	**79,943**	**0**	**4,800**	**5,800**	**6,000**	**6,000**	**6,000**	**6,000**	**6,000**	**6,000**	**6,000**	**6,000**	**6,000**	**6,000**	**6,000**	**3,343**
Total Processed Grade	**g/t**	**1.11**	**0**	**1.43**	**1.44**	**1.45**	**1.51**	**1.64**	**1.48**	**1.49**	**1.14**	**0.72**	**0.68**	**0.68**	**0.62**	**0.55**	**0.49**

Ore mining costs for GSS were an average of US$2.83 /t with a range from US$2.09 /t in Year -1 to US$3.42 /t in Year 8. Waste mining costs were lower with an average of US$2.60 /t and a range from US$2.37 /t to US$3.20 /t (Figure 16.9).

Figure 16.9 GSS Mining Costs

Ore mining costs for Wadi Doum were an average of US$4.67 /t with a range from US$4.44 /tinYear -1 to US$4.88/t in Year 5. Waste mining costs were lower with an average of US$4.41/t and a range from US$4.30 /t to US$4.96 /t (Figure 16.10).

Figure 16.10 Wadi Doum Mining Costs



16.6 Mining Equipment and Haulage Analysis

While mining activities at GSS and Wadi Doum will be performed by mining contractor, it is necessary to select appropriate loading and hauling equipment so as to be able to provide reliable inputs into the haulage study. The make and model of equipment selected below is for indicative purposes only and is not a specific recommendation by Deswik.

In order to determine load and haul equipment numbers, operating cost estimates and capital purchase schedule a haulage analysis was performed on the selected mining schedule above. This was produced in the Deswik.LHS module. Deswik.LHS utilises a series of define haul paths, material handling rules and equipment specifications to determine the haulage / fleet requirements for a given period in a mining schedule.

The Deswik.LHS software was able to determine the number of loading and hauling units required for each part of the operation in each year. These are shown in Table 16.19. Stockpile rehandle requires less than one truck for the years before Year 9 so it will prove to be more efficient to utilise two to three trucks from pit operations on reclaim campaigns as required. Further optimisation of the mining schedule and waste dump strategy will allow for further smoothing of the fleet requirements. It may also be beneficial to consider smaller trucks in Wadi Doum, although current indications from the contractors are to use the same trucks at both mines for consistency and ease of fleet management.

Table 16.19 Load and Haul Fleet

	GSS Excavator	Stockpile Rehandle Excavator	Wadi Doum Excavator	Ore Haulage Excavator	GSS Haul Truck	Stockpile Rehandle Haul Truck	Wadi Doum Haul Truck	Ore Haulage Haul Truck
	CAT 6020	*CAT 6015*	*CAT 6015*	*CAT 6015*	*CAT 777G*	*CAT 777G*	*CAT 777G*	*Volvo FMX*
Year -1	1		1	1	3		1	5
Year 1	2		1	1	7	1	5	10
Year 2	2		1	1	9	1	4	10
Year 3	3		1	1	12	1	3	10
Year 4	3		1	1	18	1	1	10
Year 5	3	1	1	1	18	1	1	10
Year 6	3	1			18	1		
Year 7	2	1			18	1		
Year 8	1	1			12	1		
Year 9		1				2		
Year 10		1				2		
Year 11		1				2		
Year 12		1				2		
Year 13		1				2		
Year 14		1				2		

Equipment models are for illustrative purposes only.

Based on the mining schedule it is expected that the following ancillary equipment will be required (Table 16.20). Due to the distance between Wadi Doum and GSS, it is likely that Wadi Doum will be treated as a completely separate operation and will require its own fleet of equipment. Similarly, road maintenance of the Wadi Doum – GSS haul road will require a dedicated maintenance fleet given the length of road involved.

Once Wadi Doum is complete, the equipment can be repurposed to GSS and could conceivably be used in the smaller GSS pits. It will also be necessary to construct TSF lifts during the life of the project and equipment from Wadi Doum could be utilised for these operations.

Table 16.20 Ancillary Equipment Fleet

<table>
<tr><th></th><th>GSS</th><th>Wadi Doum</th><th>Haul Road</th></tr>
<tr><td>Caterpillar 777D Water Truck</td><td>2</td><td></td><td></td></tr>
<tr><td>Caterpillar 773D Water Truck</td><td></td><td>1</td><td></td></tr>
<tr><td>Volvo 40 t Water Truck</td><td></td><td></td><td>1</td></tr>
<tr><td>Caterpillar D10T Dozer</td><td>4</td><td></td><td></td></tr>
<tr><td>Catepillar D9T dozer</td><td></td><td>2</td><td></td></tr>
<tr><td>Caterpillar 16M Grader</td><td>2</td><td>1</td><td></td></tr>
<tr><td>Caterpillar 14H Grader</td><td></td><td>1</td><td>1</td></tr>
<tr><td>Caterpillar 988K FEL – Clean up and Ancillary</td><td>3</td><td></td><td></td></tr>
<tr><td>Caterpillar 966H FEL – Clean up and Ancillary</td><td>1</td><td>1</td><td></td></tr>
<tr><td>Caterpillar 930 FEL – Stemming</td><td>1</td><td>1</td><td></td></tr>
<tr><td>Caterpillar CS54 Roller</td><td colspan="2">1</td><td>1</td></tr>
<tr><td>Caterpillar 432F Backhoe</td><td>1</td><td>1</td><td></td></tr>
<tr><td>Caterpillar IT38 – Tool Carrier</td><td colspan="2">1</td><td></td></tr>
<tr><td>Atlas Copco T50 – Drill Rig</td><td>3</td><td>1</td><td></td></tr>
<tr><td>Atlas Copco 5060 CR – Drill Rig</td><td>1</td><td>1</td><td></td></tr>
<tr><td>Low Loader</td><td colspan="3">1</td></tr>
<tr><td>Grove RT800 – 80 t Crane</td><td colspan="2">1</td><td></td></tr>
<tr><td>6 x 4 Tyre Truck</td><td>1</td><td>1</td><td></td></tr>
<tr><td>6 x 4 Hiab Truck</td><td>1</td><td>1</td><td></td></tr>
<tr><td>6 x 4 Service truck</td><td>2</td><td>1</td><td></td></tr>
<tr><td>4 x 4 Spares Truck</td><td>2</td><td>1</td><td></td></tr>
<tr><td>Toyota 4WD</td><td>22</td><td>11</td><td>3</td></tr>
<tr><td>32-seat bus</td><td></td><td>1</td><td></td></tr>
<tr><td>Lighting Towers</td><td>8</td><td>4</td><td>3</td></tr>
<tr><td>Sykes XH150 Pump</td><td>1</td><td></td><td></td></tr>
<tr><td>Sykes CP150 Pump</td><td>1</td><td>1</td><td>1</td></tr>
<tr><td>Explosives Truck – MMU</td><td>1</td><td>1</td><td></td></tr>
<tr><td>Shotfirer’s Truck – Accessories</td><td colspan="2">1</td><td></td></tr>
</table>

16.7 Operating Strategy

The proposed approach for operating at both GSS and Wadi Doum is through the use of a mining contractor. The contractor will be responsible for all mining activities including:

- Ore and waste mining at GSS.
- Ore and waste mining at Wadi Doum.
- Ore Haulage from Wadi Doum to GSS including road maintenance.
- Stockpile rehandle to the ROM Pad.
- Wadi Doum – GSS haul road maintenance.

Additionally, it is likely that the contractor will also be utilised for:

- Civil earthworks for construction of processing plant, tailings storage facility, mine haul roads, and camp sites.
- Ongoing civil works associated with operational TSF lifts.

The contractor will be responsible for explosives supply and management including providing qualified shot firers.

Orca Gold will provide the contractor with:

- Fuel.
- Electricity where it is provided from the electrical distribution network.
- Water, both potable and raw.
- Accommodation at the main camp near GSS.

Given the distance between GSS and Wadi Doum, the contractor will most likely establish a satellite camp near Wadi Doum. Orca Gold has the main exploration camp and the "drillers camp" established, both of which will become redundant once the main mine site camp is established. Orca Gold may make these camps available to the contractor who will then relocate the buildings to a suitable location near Wadi Doum. Electricity will be provided for the Wadi Doum camp and operations from generators and water will be trucked in from the distribution point at GSS.

Mining will occur at GSS 24 hrs a day utilising several shifts. The number of shifts per 24-hr period is yet to be determine, but will either be two or three. Wadi Doum and the ore haulage from Wadi Doum to GSS will be day shift only, due to the volumes required to be moved.

Transfer of ore from the ROM Pad fingers to the crusher will be the responsibility of the processing plant, although the contractor will be responsible for secondary breakage of any oversize ore. The primary crusher is a gyratory crusher with a maximum particle size of 900 mm. Secondary breakage will most likely be undertaken by mechanical rock breaker at the ROM pad, however best practice will be to identify and manage oversize at the face where secondary explosive methods can be utilised.

Orca Gold will employ its own mining team to oversee the contract mining operations and to provide technical guidance to the contractor. The owner's team will consist of mine management and technical staff including mining engineers, geotechnical engineers, geologists and surveyors.

Grade control activities will be conducted by a contractor but overseen by Orca Gold. Assaying of samples will be performed by Orca Gold in the onsite laboratory attached to the processing plant.

The contractor will provide its own health and safety advisors, but these will be complimented by the Orca Gold Health and Safety team, of which there will be at least one member dedicated to the mining activities.

The proposed mining licence allows for Orca Gold and its contractors to employ non-Sudanese labour in skilled and management positions when the project commences, however all unskilled positions should be filled with Sudanese employees. Given the current state of the construction and artisanal mining industry in Sudan, it is possible that machinery operators will be available although some form of training programme will be necessary for the size of equipment that will be utilised at the mine.

The mining licence states that Orca Gold and its contractors should then "endeavour to procure that approximately 50% of all technical, administrative, and executive managerial positions at held by

Sudanese nationals". Within 10 years from commencement of operations, it is expected that Sudanese nationals will hold approximately 90% of these positions.

Given the location of the project and lack of any permanent community within 150 km, it will be necessary to provide both drive-in / drive-out and fly-in / fly-out arrangements for all staff. Drive-in / drive-out will most likely be from central points such as Abu Hamad and Atbara, while FIFO will be from Khartoum. To facilitate this, an airstrip will be constructed near the GSS operation during the construction phase of the project and a suitably qualified aviation charter company with high safety record will be appointed to operate a charter service between GSS and Khartoum.

Water will be provided from the Area 5 aquifer, some 100 km to the west of GSS. This water will be utilised for all mine activities including dust suppression. Any water pumped from the pits will be used for dust suppression.

Electricity will be provided from the local Orca grid (generated by LNGgenerators provided by Orca Gold) or from remote generators, for example at Wadi Doum. Where possible, small power requirements will be met with solar / battery arrangements, but these will typically only be used when the cost of connection to the Orca grid is not feasible.

17. RECOVERY METHODS

17.1 Overview

The metallurgical testwork conducted to date, specifically the 2017 and 2018 work completed by SGS Vancouver and Lakefield, has confirmed that the GSS and WD material is amenable to gold and silver recovery via conventional cyanidation techniques and carbon adsorption across a wide range of ore domains and lithologies. Furthermore, a pre-aeration stage gives significant savings in cyanide consumption for Main Transition, Main Fresh and East Fresh ores.

A trade-off study comparing hybrid CIL, leach CIP and leach pumpcell circuits showed that, although the hybrid CIL had the lowest capital cost, over the life of mine the leach pumpcell circuit had the lowest operating costs and the best economics. Therefore, a leach pumpcell circuit has been adopted for design.

The process plant design is based on a robust metallurgical flowsheet designed for optimal precious metal recovery and removal of deleterious elements such as mercury. The flowsheet chosen is based on unit operations that are well proven in the industry.

The key criteria for equipment selection are suitability for duty, reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements, whilst maintaining a layout that will facilitate construction progress in multiple areas concurrently.

The key project design criteria for the plant are:

- Nominal throughput of 6.0 Mtpa with a grind size of 80% passing (P_{80}) 53 to 90 µm depending on ore type.
- Process plant availability of 91.3% supported by the selection of standby equipment in critical areas, reputable western vendor supplied equipment and on-site LNG generated power supply.
- Sufficient automated plant control to minimize the need for continuous operator interface but allow manual override and control if and when required.

The treatment plant design incorporates the following unit process operations:

- Single stage primary crushing with a gyratory crusher to produce a crushed product size P_{80} of approximately 110 mm.
- A crushed ore stockpile with a nominal live capacity of 10,000 wet tonnes. Three apron feeders will be provided underneath the stockpile in the reclaim tunnel to deliver ore to the milling circuit.
- A grinding circuit configured as a two stage circuit with a SAG mill and closed circuit ball mill (SAB). The circuit will produce a P_{80} grind size of 53 µm for oxide and Wadi Doum ores, approximately 75 µm for transition dominant ore and 90 µm for fresh ores. In Year 7 onwards, pebble crushing will be required due to increasing ore competency; provision for this equipment has been allowed in the design.
- Pre-leach thickening to increase the slurry density feeding the leach and carbon in pulp (CIP) circuit to minimise tankage and reduce overall reagent consumption.

- Leach circuit incorporating a dedicated pre-aeration tank for passivation of cyanide consuming minerals and 10 leach tanks in series to provide 45 hr leach residence time.
- A Kemix Pumpcell CIP circuit for recovery of gold onto carbon to minimise carbon inventory, gold in circuit and operating costs. The CIP and elution circuit design are based on daily carbon harvesting.
- 12.5 t split AARL elution circuit, including optional cold cyanide strip, electrowinning, mercury retorting and gold smelting to recover gold from the loaded carbon to produce doré and safely remove mercury.
- Tailings thickening to recover and recycle process water from the CIP tailings.
- Tailings pumping to the tailings storage facility (TSF).

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

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

Figure 17.1 Overall Process Flow Diagram

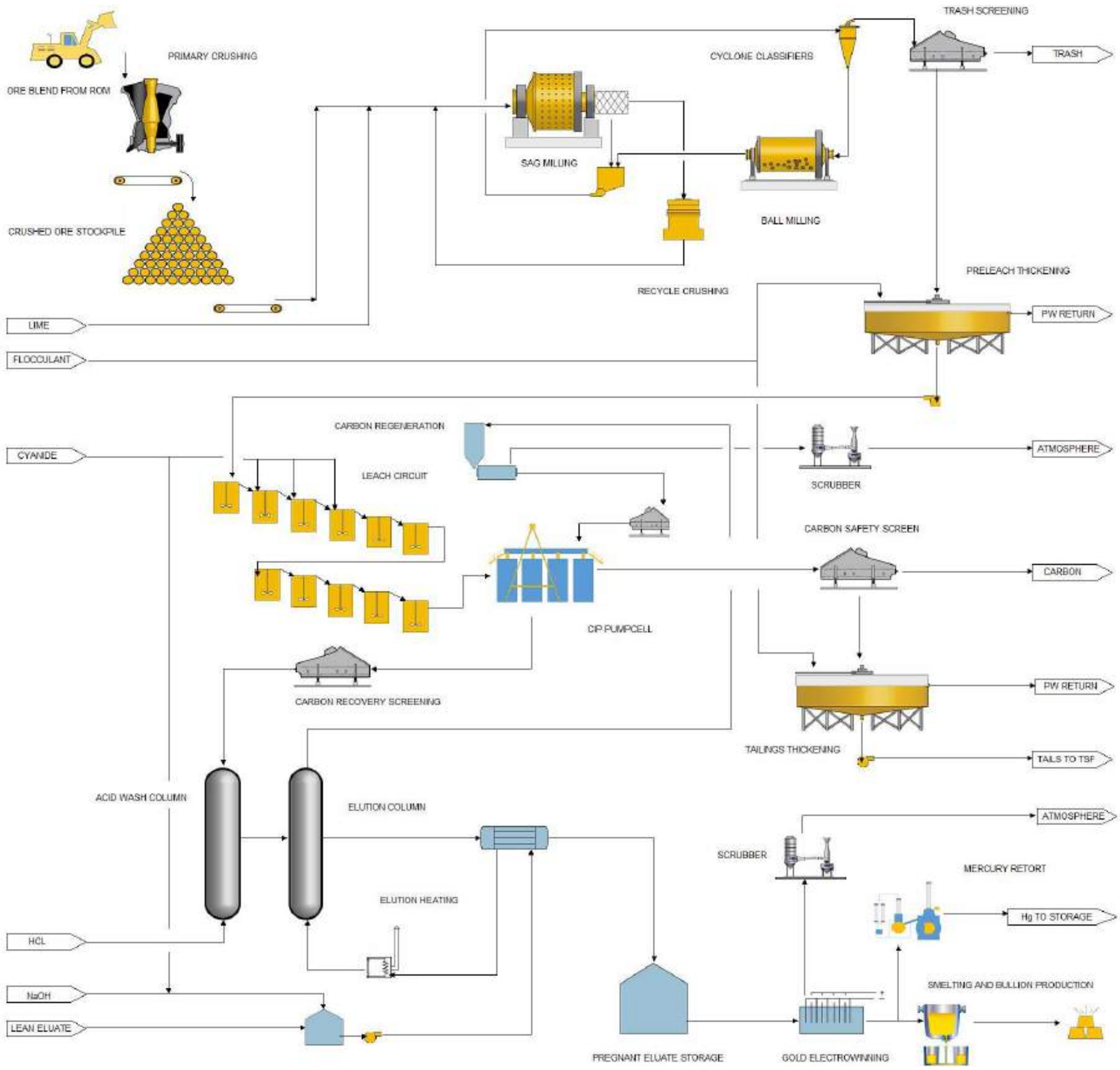


Figure 17.2 Plant Layout

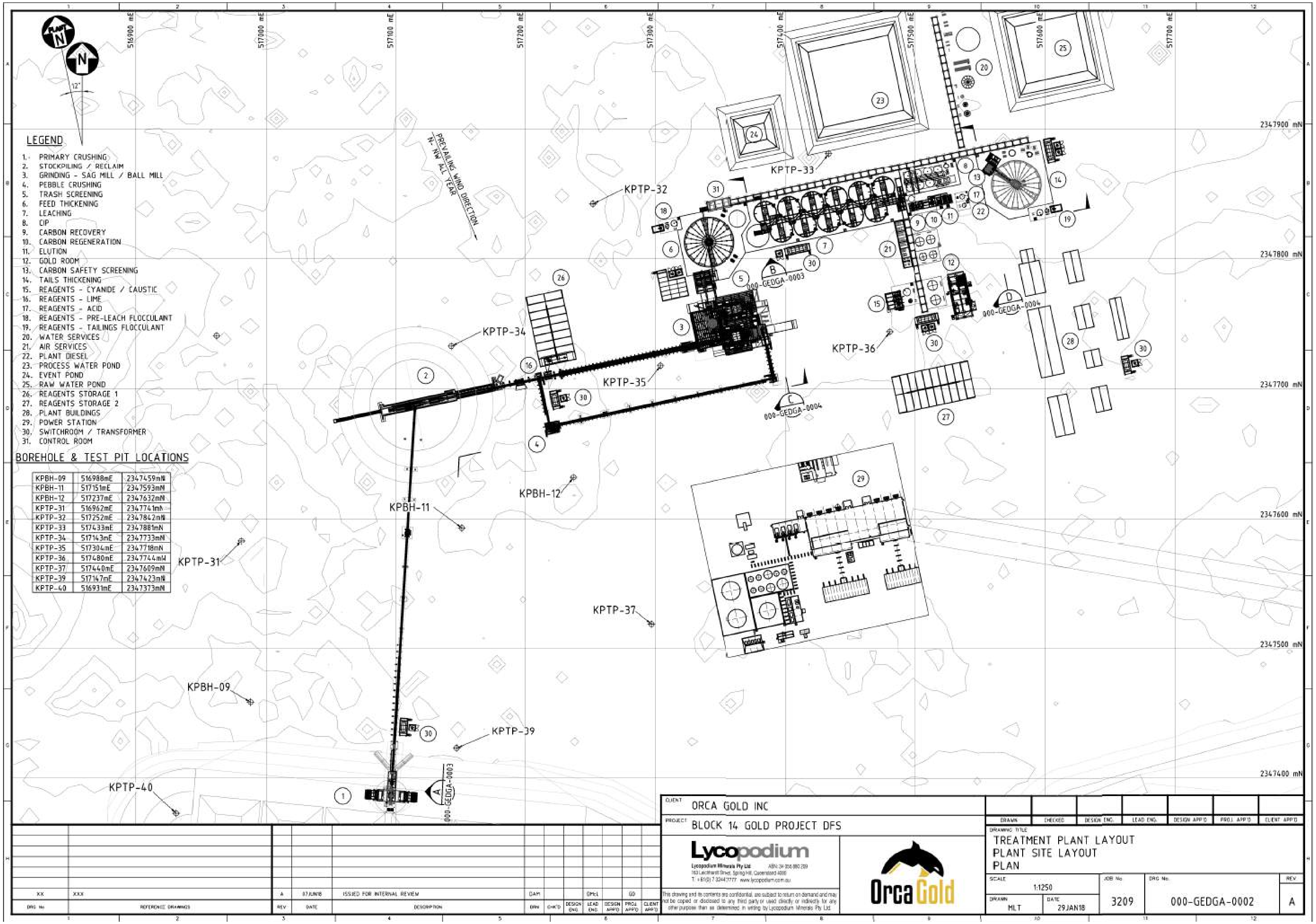
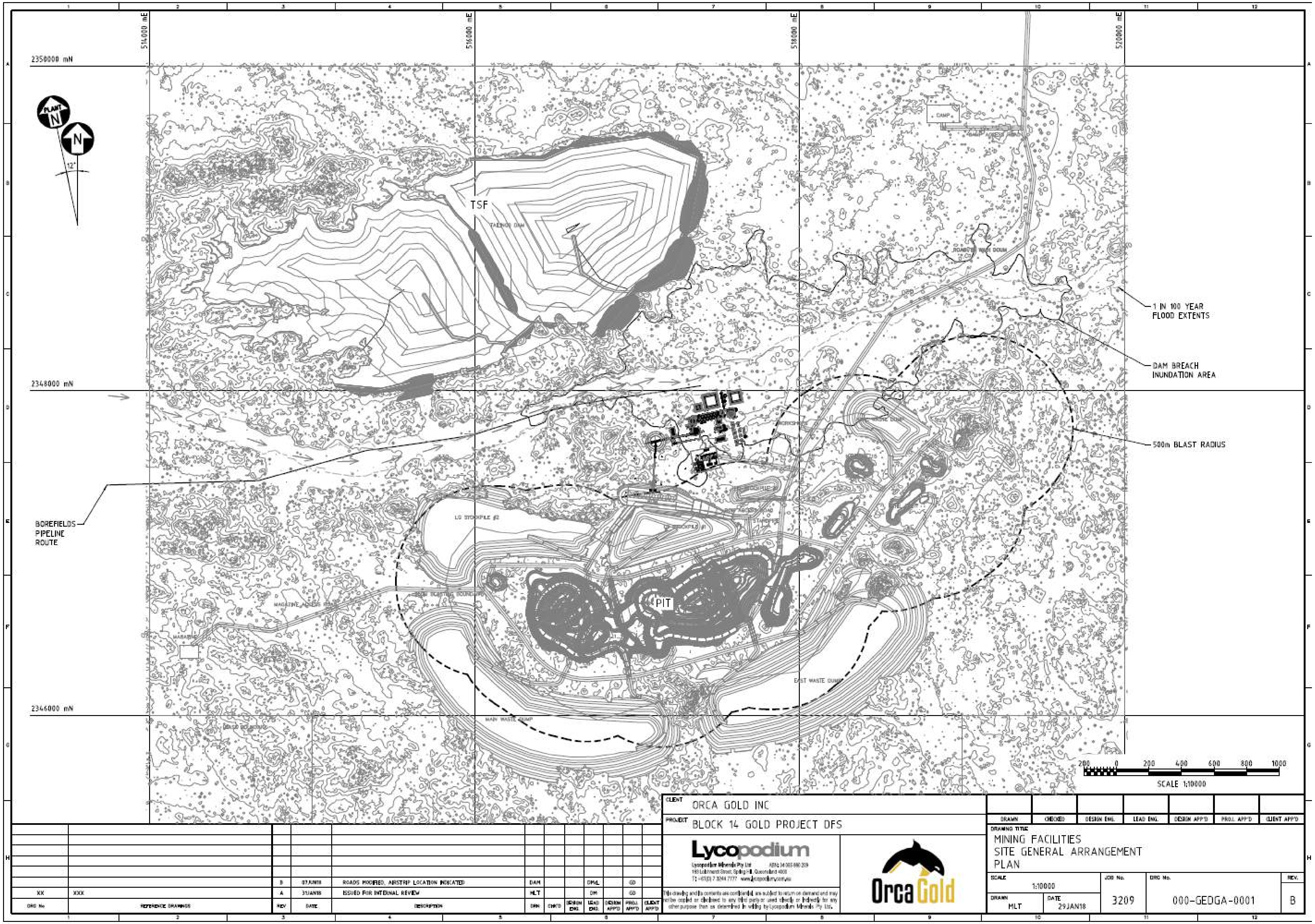


Figure 17.3 GSS Site Plan



17.2 Process Design Basis

The key factors considered in the process plant design and selection of equipment are outlined as follows.

17.2.1 Process Plant

The plant design has been based on a nominal capacity of 6.0 Mtpa of ore and head grade of 1.84 g/t Au and 4.33 g/t Ag, with overall gold recovery 86% and silver recovery 45%. This reflects the highest pregnant solution grade (Au + Ag) over LOM. As the ore has elevated mercury and base metals in some ore types, provision for removal of these deleterious elements has been made in the design.

17.2.2 ROM Pad and Crushing Circuit

The ROM pad will be used to provide a buffer between the mine and the plant. Separate stockpiles will be constructed to allow blending of different ore types to be carried out using the front end loader, to ensure that a consistent grade and hardness will be delivered to the plant.

Due to the abrasive and competent nature of some fresh ores, a vibrating grizzly jaw crusher installation would be high risk in terms of maintaining crushing rate and product size. Therefore, a primary gyratory crusher with an open side settling of 140 mm has been selected for the crushing duty.

An apron feeder has been selected to draw material from the crusher discharge pocket and feed the stockpile feed conveyor. A metal detector located on the crusher discharge conveyor will indicate any magnetic tramp material that may be present with the ore and allow its removal.

17.2.3 Stockpile

Due to the relatively high milling rate and the varying ore types to be treated, a conventional stockpile with three reclaim apron feeders has been specified in order to maximise live storage capacity and size control for the milling circuit.

17.2.4 Milling

Comminution circuit design has been based on achieving the required 6.0 Mtpa throughput rate and grind size for each year, based on the mine schedule. A SAG and ball mill circuit has been selected to reduce crushed product to the nominal circuit P_{80} size of 53 µm for oxide and Wadi Doum dominant blends, 75 µm for transition ore blends and 90 µm for fresh ore blends. In line with the mine schedule, this results in an increasing grind size over the life of mine. In later years of operation, as the ore becomes more competent, a pebble crusher will be required to maintain the throughput rate and provision for this equipment has been included in the design.

17.2.5 Classification

Cyclones of 500 mm in diameter have been selected for the classification duty to minimise wear and reduce the potential for spigot blockages occurring from coarse mill discharge material. These cyclones will require spigot and vortex finder changes over LOM to achieve the varying primary grind sizes required. Design has been based on up to 310% recirculating load to the ball mill.

The pre-leach thickener will allow operation of the grinding and classification circuit to be optimised by operating cyclones at feed densities that maximise classification efficiency, thereby reducing circulating load and overall circuit power consumption.

17.2.6 Trash Screening

Vibrating trash screens have been selected to prevent oversize particles and rubbish from entering the downstream leach and adsorption circuit. Although minimal trash is expected, acceptable trash screen performance will be essential for good carbon management.

17.2.7 Pre-Leach Thickening

A high rate pre-leach thickener ahead of the leach circuit has been selected to thicken trash screen undersize to the design leach circuit feed density of 55% w/w solids. Rheology testwork has confirmed that this density is achievable even for the oxide blends, as there are only trace amounts of clay in the ore.

17.2.8 Leach and CIP Circuit

Head assays and metallurgical testwork has indicated that the Block 14 ores do not show preg-robbing characteristics and therefore a leach CIP circuit can be considered for design.

Metallurgical testwork and trade-off work has indicated that for fresh and transition ores, leaching is generally complete within 32 hr, but oxide and Wadi Doum ores show benefit from longer leach times. Therefore, a leach time of 45 hr was selected based on ten 4,000 m^3 leach tanks at 55% w/w solids density.

As metallurgical testwork on a fresh blend ore demonstrated that there is no gold recovery benefit in using oxygen compared with air; the design is based on air addition, which will be delivered down the agitator shaft.

However, testwork indicated that four hours of pre-aeration reduced cyanide consumption for a number of ore types, therefore, a dedicated pre-aeration tank has been included ahead of leaching for passivation of cyanide consuming minerals.

Carbon loading testwork has indicated that good CIP performance can be expected. A design carbon loading for gold and silver of 6,133 g/t Au + Ag has been used for sizing the CIP and elution circuit, based on the solution tenors achieved in testwork.

A trade-off exercise was conducted to examine the relative benefits of a hybrid CIL, leach CIP or leach Pumpcell plant. While the hybrid CIL had the lowest capital costs, the leach Pumpcell plant had the best NPV due to the lower operating costs. Therefore, the Kemix Pumpcell CIP circuit was selected for recovery of gold onto carbon, to minimise carbon inventory, gold in circuit and operating costs.

17.2.9 Elution, Electrowinning and Gold Recovery

A split AARL elution circuit has been selected to remove absorbed gold and silver from carbon. The split AARL circuit can accommodate additional elution cycles, if required due to the relatively short stripping time of approximately eight hours. A cold cyanide wash has been allowed in order to remove copper from carbon prior to elution, if required.

A 12.5 t batch size is required for the 6.0 Mtpa plant capacity. Based on the calculated carbon movements, a total of seven elution cycles are required each week.

Three parallel 33 cathode electrowinning cells are proposed for the gold room, to provide a high pass efficiency and ensure a low gold and silver tenor in the spent electrolyte returning to the strip solution tank. Due to the high silver content of the ore, the electrowinning capacity is relatively large for the plant throughput, as the reduction of silver and other metals such as lead and mercury consume most of the current applied early in the electrowinning cycle.

A sludging cell design with in-tank washing of the cathodes has been adopted to simplify the cathode handling process. Off-gases from the electrowinning cells will be passed through an activated carbon filter for mercury removal.

Electrowinning cell sludge will be filtered in a pressure filter prior to transfer to a mercury retort for removal of mercury prior to smelting to produce doré.

Barren carbon will be regenerated in a horizontal rotating kiln incorporating scrubbing of off-gases for mercury removal.

17.2.10 Tailings Thickening and Pumping

A high rate thickener has been selected to thicken the CIP tails to maximise process water recovery and reduce the volume of tailings. The tailings thickener also allows the residual free cyanide in the process water that would normally be discharged to the tailings storage facility to be recovered for re-use in the leaching circuit.

The tailings thickener underflow will be pumped to the tailings storage facility.

17.3 Key Process Design Criteria

The key process design criteria listed in Table 17.3.1 form the basis of the detailed process design criteria and mechanical equipment list. Inputs into the design criteria include metallurgical testwork conducted by SGS, Orca advice, comminution modelling by Orway Mineral Consultants (OMC), CIP modelling by SGS and Kemix and Lycopodium calculations and modelling.

Table 17.1 Key Process Design Criteria

Parameter	Units	Oxide & Wadi Doum	Transition	Fresh	Source*
Plant Capacity	tpa	6,036,000	6,036,000	6,036,000	Orca
Gold Head Grade (max)	g Ag/t	2.04	1.84	1.62	Orca
Silver Head Grade (max)	g Ag/t	3.45	4.33	2.44	Orca
Copper Head Grade (max)	ppm Cu	154	144	135	Orca
Mercury Head Grade (max)	ppm Hg	1.20	1.86	1.77	Orca
Design Gold Recovery (Leach)	%	90.5	85.5	83.7	Testwork
Design Silver Recovery (Leach)	%	37.2	45.3	56.7	Testwork
Crushing Plant Utilization	%	85.0	85.0	85.0	OMC
Plant Availability	%	91.3	91.3	91.3	Lyco
Crushing Work Index (CWi)	kWh/t	4.4 to 6.7	5.0 to 11.2	6.8 to 11.3	Testwork
A*b (SMC Test)		40.8 to 113.6	56.3 to 128.0	32.0 to 43.7	Testwork
Bond Ball Mill Work Index (BWi)	kWh/t	8.5 to 12.7	8.4 to 13.3	12.3 to 19.3	Testwork
Abrasion Index (Ai)		0.04 to 0.11	0.11 to 0.34	0.40 to 0.91	Testwork
Grind Size (P_{80})	µm	53	75	90	Orca
Pre-Leach Thickener Flux Rate	$t/m^2.h$	0.83	1.04	1.04	Testwork
Pre-aeration Time	hrs	4	4	4	Testwork
Leach Circuit Residence Time	hrs	45	45	45	Orca
Leach Slurry Density	% w/w	55	55	55	Orca
Number of Pre-aeration Tanks		1	1	1	Lyco
Number of Leach Tanks		10	10	10	Lyco
Number of Adsorption Tanks		7	7	7	Kemix
Cyanide Consumption – Plant	kg/t	0.44	0.49	0.51	Testwork / Lyco
Quicklime Consumption – Plant	kg/t	1.31	1.09	0.72	Testwork / Lyco
Elution Circuit Type		Split AARL	Split AARL	Split AARL	Lyco
Elution Circuit Size	t	12.5	12.5	12.5	Kemix
Frequency of Elution	strips / wk	7	7	7	Kemix / Lyco
Tailings Thickener Flux Rate	t/m2.h	0.83	1.04	1.04	Testwork
Tailings Thickener U/F Density	% w/w	58	62	65	Testwork / Lyco

Source*
"Orca" refers to advice from Orca Gold Inc.
"Testwork" refers to metallurgical testwork conducted at SGS or vendor testwork.
"OMC" refers to advice from Orway Mineral Consultants.

The complete OMC Report No. 7889-02 Rev 2, March 2018, should be referenced for detail of comminution modelling and complete comminution data set.

Note that reagent consumption in the plant includes an allowance for residual cyanide and quicklime supply at 90% active CaO.

17.4 Process Description

17.4.1 Run-of-Mine (ROM) Pad

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

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

17.4.2 Crushing and Ore Stockpile

ROM ore will be transferred from the stockpiles into the crusher ROM pocket by FEL. The facility to truck dump into both sides of the ROM pocket will also be provided. It is not intended that a rock breaker be installed initially, but rather at a future date should it be required for harder and more competent ore. The gyratory crusher will process ore with a maximum lump size of approximately 850 mm.

Ore will be crushed by the gyratory crusher and withdrawn from the ROM discharge pocket by the variable speed primary crusher apron feeder. This apron feeder will discharge ore onto the stockpile feed conveyor. A metal detector will be provided to alert the operator to any magnetic tramp material that may be present. A weightometer will indicate the instantaneous and totalised stockpile feed tonnage and be used to control the apron feeder speed.

The crushing circuit will be serviced by a single dust collection system, consisting of multiple extraction hoods, ducting and a baghouse. Dust collected from this system will be discharged onto the stockpile feed conveyor and then finely sprayed with water for dust suppression.

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

The stockpile feed conveyor will discharge ore onto the crushed ore stockpile. The stockpile will have a live capacity of approximately 10,000 t, which is 13 hr of mill feed at 6 Mtpa. Ore will be withdrawn from the stockpile by two of three variable speed apron feeders. These feeders will discharge ore onto the SAG mill feed conveyor.

A ventilation fan will force air into the concrete stockpile reclaim chamber to ensure fresh air ventilates the upper part of the chamber which would otherwise have limited natural ventilation.

Quicklime, used for pH control in the leach circuit, will be added directly onto the SAG mill feed conveyor. Quicklime will be stored in a lime silo and will be metered onto the belt using a rotary valve.

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

17.4.3 Grinding and Classification Circuit

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

The SAG mill will be a 9.14 m x 5.30 m EGL variable speed mill fitted with a 9.3 MW motor. The SAG mill will be fitted with discharge grates which will allow slurry and small pebbles to pass out of the mill. The SAG mill product will flow to a single deck vibrating screen for pebble separation.

Combined undersize product from the SAG mill discharge screen and ball mill trommel will gravitate to the cyclone feed hopper, where it will be diluted with process water. Slurry will be pumped to the hydrocyclone cluster for classification, by the duty / standby cyclone feed pumps.

The combined cyclone overflow stream will gravitate through a metallurgical sampler, which will be used as the plant feed sampler. Both solids and solution assays will be undertaken on this sample. An in-line particle size analyser (PSA) sampler will follow the cyclone overflow metallurgical sampler to provide feed for the PSA to provide real time sizing information for control of the milling circuit.

Two vibrating trash screens will be installed in a parallel configuration prior to the pre-leach thickener. Two screens have been selected to provide sufficient area for the required volumetric throughput rate. The trash screen undersize will be directed to the pre-leach thickener while trash screen oversize will be collected in trash bins.

Slurry from the cyclone underflow launder will be returned to the ball mill feed chute. The ball mill will be a 6.70 m x 11.3 m EGL fixed speed overflow ball mill fitted with a 9.3 MW motor. Slurry exiting the ball mill will pass through the ball mill trommel and report to the cyclone feed hopper. Reject oversize material from the ball mill trommel screen will be collected in the ball mill scats bunker. Media will be added to the ball mill to maintain the power draw as required, using the ball mill ball charging kibble and hoist.

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

17.4.4 Pebble Crushing

Pebble crushing will only be required for the more competent ores processed towards the end of the mine life. As such, apart from the conveyors and chutes, most equipment described in this area will be included in the financial model as deferred capital.

Oversize from the SAG mill discharge screen will be conveyed to the pebble crusher feed bin via a series of belt conveyors. A self-cleaning belt magnet will be positioned at the head chute of the first conveyor to remove any scrap metal and steel media which could potentially damage the pebble crusher. A pebble crusher feed weightometer will be installed prior to the pebble crusher feed bin.

Downstream of the cross-belt magnet, the pebbles will pass under a metal detector prior to discharging into the pebble crusher feed bin. The feed bin will provide surge capacity ahead of the pebble crusher and allow a controlled feed to be presented to the crusher. Should the pebble crusher not be operational, a diverter gate ahead of the pebble crusher feed bin will allow pebbles to bypass the pebble bin and crusher and feed directly onto the pebble crusher discharge conveyor. Similarly, should the metal detector detect tramp metal (not removed by the cross-belt magnet), the diverter gate ahead of the pebble crusher feed bin will automatically allow pebbles to bypass the pebble bin and crusher and feed directly to the pebble crusher discharge conveyor.

Pebbles will be withdrawn from the pebble crusher feed bin by a variable speed vibrating feeder. The pebble crusher will discharge crushed pebbles directly onto the pebble crusher discharge conveyor which will return the crushed pebbles to the SAG mill feed conveyor.

The pebble crusher discharge conveyor will be fitted with a weightometer for process control purposes.

17.4.5 Pre-Leach Thickening

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

Thickener underflow at 58% solids w/w or higher will be pumped to the leach feed sampler located above the leach feed distribution box using the duty / standby leach feed pumps. This leach feed sample will be used for metallurgical accounting purposes.

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

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

17.4.6 Leach Circuit

Pre-leach thickener underflow will be pumped to the leach feed distribution box via the pre-leach sampler. Tailings thickener overflow water, containing residual cyanide, will be added as required to achieve the design 55% solids leach feed density. The slurry from the leach feed distribution box will flow by gravity to the first tank. If the first tank is offline, the slurry will be diverted to the second tank, via a launder gate system. Sodium cyanide solution will be delivered via a ring main and can be distributed to either of the first two leach tanks using actuated control valves.

The leach circuit will consist of 11 mechanically agitated tanks operating in series. The first tank will be operated as a pre-aeration tank with a residence time of approximately four hours, with the following 10 tanks used for leaching with 45 hr leach residence time. Each tank will have a live volume of approximately 4,000 m^3.

The tanks will be interconnected by launders, with slurry flowing by gravity from tank to tank. Each tank will be fitted with a dual impeller mechanical agitator to ensure uniform mixing and particle suspension. All tanks will be fitted with bypass facilities to allow any tank to be removed from service for agitator maintenance. Low pressure air will be supplied down the central shaft of the tank agitators to provide the oxygen required for gold and silver leaching.

Slurry will flow by gravity through the tanks and report to the CIP circuit feed launder.

The leach circuit will be serviced by six floor sump pumps. Sump pumps will return spillage to a nearby tank.

A cyanide analyser for on-line monitoring of the free cyanide concentration will allow the sodium cyanide dose rate to be optimised. A hydrogen cyanide (HCN) gas monitor will also be installed in the leach area.

17.4.7 CIP Circuit

A typical CIP train is a cascade circuit in which the counter-current movement of carbon is achieved by pumping slurry and carbon upstream for a period of time. However, the Block 14 CIP circuit will be a carousel system in which the feed and discharge points of each tank will be changed and carbon will only be moved when transferring loaded carbon to the elution circuit or returning barren carbon from regeneration. This has a number of advantages over conventional CIP including no back mixing of carbon, smaller tanks and lower gold in circuit.

The feed launder, discharge manifold and internal launder arrangements are integral to the carousel mode of operation. The individual tanks will be connected by an external launder. The feed launder and discharge manifold allow any tank to be either the head or tail tank in the carousel sequence.

Feed slurry will be directed to the head tank while residue slurry will be directed out of the circuit via the residue manifold. Once the desired gold on carbon loadings have been achieved in the head tank, this tank will be isolated and feed slurry will be directed to the next tank in the carousel sequence. The entire content of the head tank will be pumped to the loaded carbon recovery screen, by one of two loaded carbon recovery pumps, to separate the loaded carbon from the slurry. The screened slurry will be returned to the feed launder. Regenerated carbon will be added to the isolated tank which will be brought back on line as the new tail tank in the carousel sequence.

Each CIP tank will be fitted with a Pumpcell mechanism, which allows the functions of pumping, screening and agitation to be combined into a single unit. The Pumpcell consists of a pumping impellor, rotating cage, wedge wire screen, pitch blade turbine and agitator in one tank. The pumping impellor will be used to transfer pulp from one tank to the next. The rotating cage and stainless steel wedge wire screen will retain the carbon within the tank while allowing slurry to be pumped to the next tank in series. The pitch blade turbine will ensure that the slurry remains suspended even when there is no flow through the mechanism, which allows ease of start-up after a prolonged shutdown. The agitator will ensure even slurry and carbon suspension and mixing with the tank.

Slurry from the discharge manifold of the CIP circuit will gravitate to the CIP tails pumps. The CIP tails pumps will transfer slurry to the carbon safety screen to recover any carbon escaping from worn screens or overflowing tanks. Screen undersize will gravitate to the tailings thickener. Screen oversize containing carbon will be collected in the fine carbon bin for potential return to the circuit. HCN monitors will be provided on the CIP deck and near the carbon safety screen.

A CIP tails sampler will be installed prior to the carbon safety screen for metallurgical accounting purposes.

Two vertical spindle sump pumps will be provided in the CIP areas to return spillage and clean up to the CIP feed launder. A CIP area gantry crane will also be provided to facilitate removal of Pumpcell mechanisms for maintenance.

17.4.8 Tailings Thickening and Disposal

Carbon safety screen undersize (CIP tailings) will gravitate to the tailings thickener feed box. Other streams such as acid waste streams will also report to the feed box. Flocculant will be diluted using a static mixer prior to being added to the tailings thickener to enhance solids settling rates. Tailings thickener underflow will be pumped to the tailings transfer hopper. A number of intermittent waste streams, such as scrubber bleed streams and cold cyanide wash liquor, will also report to the tailings transfer hopper to avoid base metal and mercury complexes reporting directly to the process water system. Slurry from the tailings transfer hopper will be pumped to the TSF using two stages of centrifugal pumping.

Two HCN monitors will be installed in the tailings area - one adjacent to the carbon safety screen and the other near the tailings transfer hopper.

Tailings thickener overflow will gravitate to the tailings thickener overflow tank. A portion of the process water from this tank will be returned to the leach circuit by the tailings thickener overflow pump for recycle of residual cyanide. The remainder of the process water will gravitate to the process water pond.

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

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

17.4.9 Elution, Carbon Regeneration and Gold Room Operations

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

- Acid washing of carbon.
- Optional cold cyanide wash to remove copper and/or zinc from loaded carbon.
- Stripping of gold and silver from loaded carbon using the split AARL method.
- Electrowinning of gold and silver from pregnant solution.
- Filtration of electrowinning sludge.
- Removal of mercury using a mercury retort.
- Smelting of retorted products to produce a silver and gold doré.

The elution and gold room areas will operate seven days per week, with the majority of loaded carbon preparation and stripping occurring during day shift. The AARL stripping circuit will be automated and will contain separate acid wash and elution wash columns. The stripping circuit will be sized for a 12.5 t batch of carbon.

Acid Wash

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

Dilute hydrochloric acid, 3% w/w HCl, will be prepared prior to use and stored in the dilute acid make-up tank. During acid washing, the dilute solution of hydrochloric acid will be pumped into the column in an up-flow direction to remove contaminants, predominantly carbonates, from the loaded carbon. This process improves the elution efficiency and has the beneficial effect of reducing the risk of calcium-magnesium 'slagging' within the carbon during the regeneration process.

After the soak period has elapsed, the loaded carbon will be rinsed with treated water. This rinse water will displace any residual acid from the loaded carbon. Dilute acid and rinse water will be disposed of in the tailings thickener. Acid-washed carbon will be hydraulically transferred to the elution column for stripping. Calcium assays on loaded and barren carbon will be conducted to determine the efficiency of the acid wash step.

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

Cold Cyanide Wash, Pre-Soak and Elution

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

The elution circuit will incorporate an optional cold cyanide wash step if copper or zinc loading on carbon becomes an issue. In this case, the operator can select a cold cyanide wash step prior to the solution recirculating and heating step. As part of this step, the cold cyanide wash feed tank will be filled with treated water, cyanide and caustic to form a dilute caustic and cyanide solution. This cold cyanide wash solution will be injected into the column in an up-flow direction. After the soak period has elapsed, the loaded carbon will be rinsed with treated water to displace the residual solution containing base metal cyanide complexes. This waste solution will be directed to the cold cyanide wash product tank. The cold cyanide wash product pump will transfer the cold cyanide wash solution to the tailings transfer hopper.

After the cold cyanide wash, the sequence automatically starts the elution sequence. The split AARL elution sequence will begin with the fill of the elution column and pre-heat of lean eluate solution with simultaneous injection of caustic and cyanide into the lean eluate pump suction. The solution will be recirculated through the heat recovery and primary heat exchangers, through the elution column, through the hot side of the heat recovery heat exchangers and back into the lean eluate tank until a temperature of 95°C is achieved. The sequence will then automatically shift to the elution phase, with the temperature set point raised to 130°C and five bed volumes (BV) of solution pumped from the lean eluate tank, through the heat exchangers and elution column to the pregnant solution tank. Caustic will be added to the pregnant solution tank during this step to ensure that a high enough solution pH is attained for electrowinning.

Following this step, five BV of treated water or barren electrowinning solution will be pumped from the stripping water tank, through the heat exchangers and elution column and into the lean eluate tank to provide lean solution for the next stripping cycle. The temperature set point will be maintained at 130°C for this step.

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

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

Electrowinning

Soluble gold and silver recovery from pregnant solution will be carried out by electrowinning onto stainless steel cathodes. The electrowinning circuit will consist of three electrowinning cells in parallel, each containing 33 cathodes. A dedicated rectifier, per electrowinning cell, will supply the necessary current to electroplate the gold and silver onto the cathode.

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

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

Fume extraction will be provided to remove noxious gases from the cells, with a sulphur impregnated column for mercury removal. In addition to this, a number of gold room vent fans will be provided to ensure there is adequate ventilation inside the gold room.

Gold Room

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

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

Retort product solids will be mixed with a prescribed flux mixture (silica, nitre and borax), prior to being charged into the diesel fired gold furnace. The fluxes added will react with base metal oxides to form a slag, whilst the gold and silver remains as a molten metal. The molten metal will be poured into moulds to form doré ingots, which will be cleaned, assayed, stamped and stored in a secure vault ready for dispatch. A conical shaped vessel will be used for slag collection so that any precious metals prills form in the bottom and can be easily recovered and put directly back in the furnace. Low grade slag will periodically be returned to the grinding circuit, via the SAG mill.

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

Carbon Regeneration

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

The carbon will be heated to 650 - 750°C and held at this temperature for 15 minutes to allow regeneration to occur. Regenerated carbon from the kiln will be quenched in the carbon quench vessel and pumped to the carbon sizing screen using the regen carbon transfer pump. New carbon will be added to the carbon quench vessel to ensure that the carbon is sized over the carbon sizing screen prior to entering the CIP circuit.

The screen oversize (regenerated, sized carbon) will report to the carbon transfer hopper and be returned to the CIP circuit using the barren carbon transfer pump. The quench water and fine carbon (carbon sizing screen undersize) will report to the carbon safety screen via the fine carbon hopper and pump.

The carbon regeneration kiln off-gases will be treated for mercury removal by venturi scrubbing and filtering via sulphur impregnated carbon columns.

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

17.4.10 Reagents

Quicklime

Quicklime will be delivered to site in bulk bags, with the option to handle bulk tanker deliveries if required. Bulk bags will be added to the lime transfer hopper via the bag breaker. The lime transfer blower will transfer the quicklime to the lime silo. A dust collector will be fitted to the silo to minimise particulate emissions when transferring lime into the silo. Quicklime will be metered via a rotary valve directly onto the mill feed conveyor for leach circuit pH control.

Sodium Cyanide

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

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

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

Caustic

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

Hydrochloric Acid

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

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

Activated Carbon

Activated carbon will be delivered in bulk bags. Carbon will be added to the carbon quench tank as required for carbon make-up to the CIP inventory. This addition point will allow attritioning of any friable carbon particles with subsequent fines removal on the sizing screen prior to entering the CIP tanks.

Grinding Media

Grinding media will be delivered to site in steel drums. The balls will be charged to the SAG mill via the SAG mill ball hopper and to the ball mill using a kibble and a fork lift with a hydraulic drum tipper attachment. Media will be added as required to achieve the target SAG and ball mill power draw settings.

Flocculant

Flocculant for use in the pre-leach and tailings thickeners will be delivered to site in 750 kg bulk bags. Each thickener will be provided with a dedicated flocculant mixing and storage system due to the distances between each thickener.

Flocculant bags will be lifted by hoist to a bag breaker on the flocculant feed hopper. The vendor supplied flocculant mixing plant will automatically mix batches of flocculant with raw water and transfer the mixed flocculant to the flocculant storage tank after each mixing cycle is complete.

Flocculant will be distributed to the pre-leach thickener using the variable speed pre-leach thickener flocculant dosing pumps, which will be installed in a duty / standby configuration. A vertical spindle sump pump will be provided to transfer any spillage to the pre-leach thickener.

Flocculant will be distributed to the tailings thickener using the variable speed pre-leach thickener flocculant dosing pumps which will be installed in a duty / standby configuration. A vertical spindle sump pump will be provided to transfer any spillage to the tailings thickener.

Plant Diesel

Diesel will be delivered to the 12 m^3 plant diesel day tank by the mine diesel tanker. Diesel in the header tank will be reticulated to the elution heater, carbon regeneration kiln and smelting furnace on a ring main.

Anti-Scalant

Anti-scalant will be delivered to the plant in totes or bulk containers (IBC). Metering pumps will distribute anti-scalant directly from the IBC to the process water and elution circuits.

Reagents Storage

A minimum of 90 days' stock of reagents will be stored on site to ensure that supply interruptions due to port, transport, or weather delays do not restrict production.

17.4.11 Services

Raw Water

Raw water for the project will be pumped to the plant from the bore field, approximately 100 km south west of the plant site. A number of production bores will discharge into the raw water transfer tank, which will serve as a buffer between the bore pumps and the plant. The duty / standby raw water transfer pumps will transfer raw water into the raw water pond located at the plant. The raw water pond will be 30,000 m^3 in capacity. The raw water pond pumps will transfer raw water into the raw water tank. The total combined raw water capacity is approximately 37 hr at the predicted average continuous demand.

The raw water tank will provide storage for gland water, fire water and raw water services. Duty standby raw water pumps will distribute raw water around the plant for uses such as reagent mixing, cooling water and treated water production. Top up of the process water system will be accomplished by overflow of the raw water tank into the process water pond.

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

Treated water will report to the treated water storage tank and will be pumped to distribution points around the plant, including the elution and gold room areas.

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

Gland Water
Gland water will be supplied from the raw water tank. Duty / standby LP gland water pumps will distribute gland seal water around the plant. The duty / standby HP gland water pumps will be used to supply gland seal water to the tailings transfer pumps which require higher pressure supply.

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

The fire water pumping system will contain:

- An electric jockey pump to maintain fire ring main pressure.
- An electric fire water delivery pump to supply fire water at the required pressure and flowrate.
- A diesel driven fire water pump that will automatically start in the event that power is not available for the electric fire water pump or that the electric pump fails to maintain pressure in the fire water system.
- Fire hydrants and hose reels will be placed throughout the process plant, fuel storage and plant offices at intervals that ensure complete coverage in areas where flammable materials are present.

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

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

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

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

Low Pressure Air
Low pressure air will be supplied by four duty low pressure air blowers. A fifth blower will be provided as a standby. Low pressure air will be reticulated to the leach tanks and delivered down the agitator shaft.

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

LNG
Twelve 120cbm LNG storage vessels will be provided for LNG storage for the power station.

17.5 Control System

17.5.1 General Overview

The general approach to automation and control for the plant will be one with a moderate level of complexity offering the option of local control and remote monitoring or control from a central control room. Instrumentation will be provided within the plant to measure and control key process parameters to minimise operator intervention in standard start-up functions and to provide key monitoring and control to minimise process excursions and maintain steady operation.

The main control room, located adjacent to the leach tanks, will house redundant based supervisory control and data acquisition (SCADA) servers and two operator interface terminals (OIT). A third engineering workstation will be provided for control system overview and modifications. The control room is intended to provide a central area from where the plant is operated and monitored and from which the regulatory control loops can be monitored and adjusted. All key process and maintenance parameters will be available for trending and alarming on the process control system (PCS). A fourth OIT will be located in the crusher control room.

The process control system will be a programmable logic controller (PLC) based SCADA system. The PCS will control the process interlocks and PID control loops for non-packaged equipment. Control loop set-point changes for non-packaged equipment will be made at the OIT.

In general, the plant process drives will report their ready, run and start pushbutton status to the PCS and will be displayed on the OIT. Local control stations will be located in the field, in close proximity to the relevant drives. These will, as a minimum, contain start and latch-off-stop (LOS) push-buttons which will be hard-wired to the drive starter. Plant drives will predominantly be started remotely in the correct sequence after inspecting the local equipment.

The OITs will allow drives to be selected to Local or Remote or Maintenance modes via the drive control popup. Maintenance mode will only be selected by supervisors with appropriate security access (i.e. the control room operator will not be able to select Maintenance mode). Statutory interlocks such as emergency stops and thermal protection will be hardwired and will apply in all three modes of operation. All PLC generated process interlocks will apply in Local and Remote modes. Process interlocks will be disabled or bypassed in Maintenance mode with the exception of critical interlocks such as lubrication systems on the mill.

Local selection will allow each drive to be operated by the operator in the field via the local start pushbutton which is connected to a PLC input. Remote selection will allow the equipment to be started from the control room via the drive control popup. A PLC output will be wired to each drive starter circuit for starting and stopping drives. Status indication of process interlocks as well as the selected mode of operation will be displayed on the OIT.

Vendor supplied packages will use vendor standard control systems throughout the project. Vendor packages will generally have limited interfaces with the PCS such that control and set-point changes may have to be adjusted locally. General equipment fault alarms from each vendor package will be monitored by the PCS system and displayed on the OIT. Fault diagnostics and troubleshooting of vendor packages will be performed locally.

Vendor control panels or vendor supplied PLCs will be utilised for the following vendor packages:

- Primary crusher.
- Apron feeders.
- SAG mill.
- Ball mill.
- Pebble crusher.
- Pre-leach and tailings thickener rake mechanisms.
- Flocculant mixing systems.
- Elution heater.
- Regeneration kiln and scrubber.
- Mercury retort.
- Smelting furnace.
- Low pressure air blowers.
- Compressed air system.
- Instrument air dryers.
- Water treatment systems.
- Sewage pumping stations.
- Sewage treatment system.
- Fire water system.

17.5.2 Control Loops

There will be two modes for loop controlled Control Variables available in the PCS; these are 'Auto Mode' and 'Manual Mode'. In Auto Mode, the Control Variable will be predominantly controlled by the applicable PID loop based on the Process Variable set-point. Auto Mode control may include cascade control loops where the Process Variable set-point is provided by another control loop rather than being directly specified. In Manual Mode, the Control Variable may be entered manually on the

SCADA OIT. Both Process Variable set-point and Control Variable set-point can be entered from the control loop pop up on the OIT.

In general, the plant process drives will, as a minimum, report their Ready, Run and Fault status to the PCS system and will be displayed on the OIT. Local control stations will be located in the field in proximity to the relevant drives. These will as a minimum, contain start and latch-off-stop (LOS) pushbuttons which will be hard-wired to the drive starter.

The OIT will allow drives / valves to be selected to Auto or Remote or Local modes via the drive / valve control popup on the OIT. Statutory interlocks such as emergency stops and overload protection are hardwired and will apply in all modes of operation.

- **Remote mode**: 'Remote' selection allows drives and valves to be controlled remotely from the OIT. This mode allows operators to start / stop or open / close a drive or valve manually by clicking on the relevant pushbutton on the OIT ('Manual mode') or controlled by an automatic sequence or control loop ('Auto mode'). If not in Remote mode, the drive or valve can only be controlled from local pushbuttons in the field ('Local mode').
- **Auto mode**: When all associated drives or valves are selected to 'Auto', this mode allows for the starting of automatic sequences for some systems, e.g. in the elution circuit, where the system may be group started when all associated drives or valves are selected to 'Auto'.
- **Local mode**: 'Local' selection allows each drive / valve to be operated in the field via a local pushbutton, which is connected to the PLC. All process interlocks will apply.

The control of most of the vendor packages will be done via the local control panel with no control or set-point changes from the PCS system. General equipment fault alarms from each vendor package will be monitored by the PCS system and displayed on the OIT. Fault diagnostics and troubleshooting of vendor packages will be performed locally. Digital output can be transmitted from the PCS to the vendor packages where required. The function of these outputs is to enable the start permissive for the packages or to start these packages remotely. For some specific packages (e.g. elution heater) a remote set-point can be entered.

17.6 Area Control Philosophy

17.6.1 Crushing

Feeding of the crusher will be controlled by two sets of traffic lights, which will be interlocked with the primary crusher motor to prevent ore dumping when the primary crusher motor is not running.

Infrared beams will be triggered when the loader tips its bucket into the crusher. This will activate water sprays for dust suppression in the ROM pocket and change all traffic lights to red to prevent another loader or truck from dumping ore in the ROM Pocket until the primary crusher discharge pocket level is low enough. When the primary crusher discharge pocket level transmitter detects a suitable low level, the traffic lights will change to green.

Control of the primary crusher and its ancillaries will be by a vendor supplied control system, which will be interfaced with the PCS to allow monitoring of alarms and operation of interlocks. Key operating parameters will be displayed on the PCS.

A number of blocked chute switches will be positioned under the crusher. Activation of any one of the switches will raise an alarm and stop the crusher.

Primary crushed ore will be measured on the stockpile feed conveyor with a weightometer. A regulatory control loop will adjust the speed of the primary crusher apron feeder to achieve the desired crushed ore withdrawal rate.

A metal detector located on the stockpile feed conveyor will stop the conveyor drive if metal is detected.

A laser level detector will be installed on the stockpile feed conveyor structure, to continuously measure the stockpile level. When the high-high level alarm is activated then the stockpile feed conveyor will be stopped automatically.

17.6.2 Milling and Pebble Crushing

Ore feed to the SAG mill will be measured on the SAG mill feed conveyor with a weightometer. The speed of the selected variable speed apron feeders will be modulated to achieve the desired mill feed rate.

As primary grind size control will be important for achieving maximum leach recoveries, a moderate to high level of process control will be required for the mill and pebble crusher. SAG mill variable speed control of 60 to 80% of critical speed will be provided in order to manage the varying ore competencies and grind size range. The SAG mill will also be provided with standard charge weight measurement and noise monitoring.

Water addition to the SAG mill feed will be controlled in ratio to the ore feed rate in order to maintain a relatively constant mill feed slurry density to optimise grinding efficiency. Water addition to the cyclone feed hopper will be automatically controlled to maintain a constant cyclone feed density.

Flowmeters will be provided on the SAG mill feed and cyclone feed hopper water addition lines. A density gauge and flowmeter located on the cyclone feed line will indicate the cyclone feed density and flowrate and allow calculation of the mass flow to the cyclones.

A level element will measure the slurry level in the cyclone feed hopper. A PID controller will be used to control the speed of the duty cyclone feed pump to maintain the required hopper level.

Cyclone pressure will be maintained within the target pressure band with cyclones turned on / off as required to maintain the target pressure and therefore required grind size.

A particle size analyser (PSA) will be installed to provide real time sizing information for control of the milling circuit.

The recycled pebbles will be measured on the pebble conveyor with a weightometer. A metal detector located on the pebble conveyor will stop the conveyor drive if metal is detected.

The level in the pebble crusher feed bin will be continuously measured by level element. When a low-low level alarm is activated then vibrating feeder will be stopped. The pebble crusher can idle without feed for up to 30 min before the crusher will need to be stopped. When a high level alarm is activated then vibrating feeder will be started. If the pebble crusher is not running, then the vibrating feeder will also be stopped.

The pebble crusher feed chute will be fitted with level detector to continually measure feed chute level. A regulatory control loop on the crusher feed chute level will control the speed of the vibrating feeder regulating ore to the crusher so that it is continually choke fed, thereby maximising fines production and minimising crusher wear rate.

17.6.3 Pre-Leach and Tailings Thickening

Control of the thickener rakes will be carried out using the vendor supplied thickener control PLC. Rakes will be controlled to automatically lift when rake torque exceeds a pre-determined set point and will automatically be lowered in stages as rake torque decreases.

The speed of the thickener underflow pump will be controlled in order to maintain the thickener bed pressure. Output from the bed pressure PID controller will cascade to the flow controller which will regulate the speed of the thickener underflow pump to maintain the set point. A minimum flow rate will be programmed into the flow controller to ensure that sanding of the thickener underflow line does not occur. The thickener underflow density will be measured by a density gauge on the thickener underflow pipe.

The slurry bed level within the thickener will be measured using a bed level detector. A bed level controller will send a remote set point to the flocculant PID flow controller which will adjust the speed of the thickener flocculant dosing pump to maintain the flow set point.

17.6.4 Leaching and CIP

All aspects of the leach and CIP circuit, with exception of the pH control loop, the leach tails cyanide analyser and level control loops in transfer hoppers, will be manually controlled or measured. The plant operators will measure and monitor the following parameters:

- Leach feed density - measured by Marcy scale. The operator will adjust the speed of the tailings thickener overflow pumps to provide the require amount of dilution water to the leach circuit.
- pH - measured using pulp sample from the pre-aeration tank and leach tank 1 by a hand-held pH meter that will be used to cross-check the automatic in tank pH meter.
- Dissolved oxygen level - measured in tanks by hand-held dissolved oxygen meter.
- Cyanide concentration in leach solutions measured on filtrate by manual titration to cross check the online monitoring system.
- Leach feed grind size to provide a cross check on cyclone overflow sample sizing.
- Collection of composite head and tail samples - for assay in laboratory.
- Carbon level - measured by volume from dip sample.
- Observation of slurry level in tanks and condition of Pumpcell screens.
- Sequencing of the Pumpcells, including isolating the head CIP tank once target carbon loading has been achieved and initiating carbon transfer and later returning a batch of regenerated carbon to this tank and sequencing of valves to return this tank into service as the tail CIP tank.
- Temperatures, vibration, etc. of mechanical equipment.

The quicklime addition rate onto the SAG mill feed conveyor will be controlled in ratio to the mill feed rate by a ratio controller which will vary the speed of the rotary valve on the lime silo. The set point on the ratio controller will be adjusted automatically by the pH controller with pH probes located in the pre-leach thickener, pre-aeration tank and leach tank 1. The pH probe used for control will be operator selectable.

A free cyanide analyser will measure the cyanide concentration in the leach circuit to ensure that cyanide addition is optimised.

17.6.5 Stripping and Gold Room Operations

All aspects of the AARL stripping circuit will be automatically controlled, following sequences triggered by timers, tank levels and solution temperatures with the exception of the carbon transfer steps:

- The acid wash and cold cyanide wash rinse steps will operate on a timer to prevent excessive water usage.
- The positive displacement caustic and cyanide metering pumps will operate on timers to ensure the correct dosage of reagents to the cold cyanide wash and pre-soak strip solutions.
- The eluate temperature will be measured by a thermocouple located in the eluate line to the elution column. A temperature controller will modulate the input of energy from the elution heater to maintain the desired eluate temperature. Eluate will only report to the pregnant tank once the target temperature has been achieved.

The critical elution process parameters (temperature and elution column pressure) will be trended by the PCS to assist the operator in the monitoring of the process.

Electrowinning will be manually initiated. An operator will be required to perform the following duties:

- Supervision of electrowinning cell operations, including adjustment of flow split between the cells.
- Sample collection.

Electrowinning parameters (voltage, current) will be displayed on the OIT for recording and trending purposes.

The mercury retort and smelting furnace will be controlled and monitored locally by the gold room operator. The mercury retort will include a vendor supplied PLC and local HMI to control the ramp / soak programs and automatic start-up and shutdown sequencing.

The carbon regeneration kiln temperature will be controlled via a temperature controller mounted in a local control panel. The speed of the screw feeder will be varied to ensure the correct kiln feed rate.

17.6.6 Tailings Disposal

Slurry from the tailings thickener will be discharged into the tailings transfer hopper at a controlled rate. A level element will measure the slurry level in the tailings transfer hopper. A PID controller will be used to control the speed of the tailings pump to maintain the required hopper level. Leak detection will be included on the tailings line via magnetic flowmeters located at either end of the pipeline.

17.6.7 Services

Level elements will detect the water levels in the raw water tanks and pond and the process water pond. High and low level alarms will register in the control room.

The raw water transfer pumps will be controlled, stopped and started remotely using telemetry. Leak detection will be included on this pipeline.

The low pressure air blowers will be controlled by a vendor supplied control panel.

Compressed air for use in the plant will be supplied from a lead / lag set of air compressors controlled from a local control panel. Instrument air dryers will be controlled from a local panel.

Low pressure alarms will be provided for plant and instrument air, low pressure air, process water, raw water, potable water and fire water.

17.7 Metallurgical Accounting

Weightometers will be located on the following conveyors throughout the plant:

- Stockpile feed conveyor, to measure instantaneous and cumulative primary crushed ore tonnage.
- SAG mill feed conveyor, to measure instantaneous and cumulative mill feed tonnage.
- Pebble crusher feed conveyor, to measure the pebbles being discharged from the SAG mill.
- Pebble crusher discharge conveyor, to measure the crushed pebbles being returned to SAG mill (future installation).

The total mill feed tonnage will be calculated based on the sum of the mill feed tonnage and the returning pebble load. This will be used for process control purposes.

Automatic sampling of the cyclone overflow, leach feed stream and the CIP tailings will ensure reliable composite shift samples for leach head grade and tails solution and residue grades.

Density and flow meters on the leach feed and tailings lines will allow the dry tonnage of solids pumped to the leach circuit and TSF to be determined as a cross check on the SAG mill feed tonnage determined from the SAG mill feed weightometer. In conjunction with the leach feed and plant tails samples, the mass flow measurements will allow the gold recovered in the leach / CIP circuits to be calculated.

Regular gold 'in circuit' surveys will allow reconciliation of precious metals in feed compared to doré production.

Water supplied and used in the various areas will be continuously monitored.

Reconciliation of the amount of reagents used over relatively long periods will be achieved by delivery receipts and stock takes. On an instantaneous basis, reagent usage rates of cyanide, elution and detoxification reagents and diesel flow rates to unit operations will be measured (L/min) and accumulated (m^3) using flow meters.

17.8 Plant Layout

The process plant will be located to the north of the pits and south of the TSF. The plant location has been selected to be in close proximity to the pits and TSF, on relatively flat terrain which slopes gently away from the crushing circuit towards to the tailings thickener. The plant avoids the 1 in 100 flood level from the Wadi which is located to the north of the plant.

The plant will be constructed around three major piperacks. The main piperack will run in an east west direction and will be the major pipe route for water distribution and slurry transfer. The secondary piperack will run north south and be used for high and low pressure air, cyanide, caustic, strip solutions and pregnant and barren eluates. A third piperack will run north south to the water services area.

17.8.1 Primary Crushing and Stockpile

The ROM pad will be located at the southern corner of the plant site, which isolates mine heavy vehicle movements to this area and the mine services area. This location will also ensure that any dust carried by the north north-west prevailing winds from the ROM pad will not be blown over the plant or infrastructure. The ROM pad will take advantage of the small natural hill, which avoids the need for excessive earthworks and concrete for construction of the ROM pad and crusher wing walls.

The primary crusher dump pocket has been sized to accommodate double-sided tipping from 777 haul trucks. The gyratory crusher will be mounted in a concrete chamber. Crushed material will discharge to a surge chamber directly beneath the crusher. The surge chamber will have provision for removal of the lower section of the crusher for maintenance.

Dust collection will be provided around the crusher and apron feeder discharge points. Dust suppression sprays will be used at the ROM pocket and over the stockpile.

The stockpile feed conveyor will run in a northerly direction to the stockpile. Replaceable abrasion resistant linings will be provided in all chutes and conveyor skirtboards at all wear points. The stockpile will be constructed on natural ground level with a concrete reclaim chamber and Armco tunnel access via both sides, with backfill to provide a level pad on which ore will be stored. The stockpile will contain approximately 10,000 t of live ore capacity.

The reclaim chamber will be fitted with three apron feeders in line which will generally be run in pairs. Fresh air will be provided around the apron feeders via a ventilation fan.

17.8.2 Milling, Classification and Pebble Crushing

The SAG mill feed conveyor will run in a north-easterly direction towards the SAG mill feed chute. SAG mill ball charging will be performed using a front-end loader which will deliver media to the SAG mill ball hopper, which will discharge onto the SAG mill feed conveyor. Access to this hopper will be from the southern side of the belt.

A lime silo will be provided over the mill feed conveyor. Provision for bulk deliveries or one tonne bags has been made. The bulk lime tanker will drive into the covered lime storage building, located to the north of the mill feeder conveyor, where the lime will be pneumatically transferred into the silo. The transfer facility for bulk bags will also be located within this building. Storage for lime bags will be to the north of the lime transfer and unloading area.

Walkways will be provided on both sides of the SAG mill feed conveyor to allow access to idlers without the need for cranage. Maintenance access to the SAG mill feed conveyor take-up tower has been provided via a series of ladder ways and intermediate platforms from ground level at the eastern end of this conveyor.

The grinding and classification circuit will be located to the north east on a combined concrete and steel structure, over four main levels. Access to the SAG mill and ball mill from ground level will be either be via a set of stairs located at the western end of the main platform, or via a second set of stairs at the eastern end nearest the SAG mill discharge screen. Space for the mill relining machine will be provided on this level. Secondary access to the ball mill will also be possible from the ground level using the stairway on the northern side of the mill area structure.

An intermediate operating level will be located above to allow access to the SAG and ball mill feed chutes and the secondary and tertiary vezin samplers, which are located at the northern end of this platform. The top level will provide access to the cyclone underflow launder and primary cross cut sampler. A walkway will be installed above the cyclones in order to allow access to the vortex finders and overflow launder. Maintenance of the cyclone cluster will be carried out using a jib crane.

The cyclone feed pumps will be located at the eastern end of the bund level to facilitate maintenance access, along with the SAG and ball mill scats bunkers.

The SAG mill discharge screen will also be located at the eastern end of the mill structure. A drive in pebble sump will be provided for occasions where the pebble crusher conveyors are temporarily out of service. This sump pump will also handle spillage from the cyclone feed pumps and hopper.

A second drive in sump pump will be provided at the feed end of the mills on the southern side.

Normally, pebbles will discharge onto the SAG mill oversize discharge conveyor which will travel in a north to south direction. A walkway will be located on the eastern side of the belt for idler access. A belt magnet will be located at the end of this conveyor, over the transfer chute, in order to achieve maximum steel removal efficiency. Access to the transfer chute and magnetic will be via a set of stairs of the western side and a platform around three sides of the magnet.

The pebble crusher feed conveyor will run east to west, with a walkway along the length of the conveyor on the southern side connecting with the top level of the pebble crusher structure. The pebble crushing facilities will consist of five platform levels to allow access to the conveyor head chute, pebble crusher feed bin, vibrating feeder, bypass chute hatches, pebble crusher drives and gap adjustment. The ground level will house the pebble crusher lubrication skid and transfer chute onto the pebble crusher discharge conveyor. The pebble crusher structure will be accessed via stairways on either the north eastern or south western sides.

The pebble crusher discharge conveyor will run south to north, with a walkway provided on the western side. The pebbles will discharge onto the SAG mill feed conveyor prior to the lime addition point. Access to the transfer chute will be provided on both sides of the SAG mill feed conveyor.

17.8.3 Trash Screening and Pre-Leach Thickening

The PSA sampler, PSA and trash screens will be located to the north of the milling structure and connected by a walkway. The structure can also be accessed from ground level on the eastern side.

The trash screens will be located on a level below the primary cross cut sampler to allow gravity flow to the trash screens feed box and the installation of the in-line PSA sampler between these two pieces of equipment. The trash screen structure will be over two platform levels; the upper level will include platform access to the PSA primary sampler, trash screens feed box, isolation valve and screens, while the lower level will provide access to the secondary PSA sampler, PSA, trash screen underpans and thickener feed box. Trash will be collected on the ground level.

The 38 m diameter above-ground pre-leach thickener will be located in a bunded area with pre-leach thickener overflow (milling water) tank, adjacent to the trash screens. The thickener underflow pumps will serve as leach feed pumps and will be located on the periphery of the bund for ease of maintenance and mobile crane access. The dedicated flocculant plant will be located to the west of the thickener.

Pre-leach thickener overflow will be collected in the overflow tank and re-used as milling water. The pumps will be located on the periphery of the bund to facilitate maintenance access.

The thickener bridge will be full span to allow multiple access points. Access to the thickener from ground level will be via the trash screens stairway or via the northern end of the plant near the control room.

The concrete bund will be sloped to a single floor area sump pump for spillage collection.

17.8.4 Leach and CIP Circuit

The leach circuit will consist of eleven 4,000 m^3 agitated tanks in series, running from west to east and adjacent to the pre-leach thickener. Space for a twelfth tank will be included in the layout, at the western (front) end of train. Access to the top of tanks will be via two sets of stairs located at either end of the leach tanks or via the control room, which will be located to the west of the leach tanks. The control room will be situated above the main piperack to give views over the leach tanks, pre-leach thickener and milling circuit.

The pre-leach secondary vezin sampler and leach feed distribution box will be located at top of tank platform level for ease of access. The primary cross cut sampler will be accessible by stairway from this level. Access to launder gates and agitator motors will also be from top of tank platform level.

The top of tank equipment will be serviced by the plant mobile crane. A 90 t all-terrain crane will be provided for this and other maintenance duties.

The leach tanks will be bunded for spillage and splash, with six sump pumps provided on the periphery of the bund for return of spillage to the circuit. The bund floor will slope to each of the sump positions for ease of clean up.

The event pond will be located to the north of the leach tanks to capture any spillage as a result of an uncontrolled release of slurry outside the bund. The 5,000 m^3 event pond has been sized to capture 110% of a single leach tank. Site earthworks and drainage will be installed such that spillage from all major slurry vessels will be directed to this pond. A submersible pump will be used to return spillage to an adjacent bunded area for re-processing.

The CIP circuit will be constructed to the east of the leach tanks. Slurry will gravity flow from the last leach tank to the CIP circuit feed launder. Access to the CIP tanks will be from the stairway at the eastern end of the leach circuit, or from the stairway on the eastern side of the CIP bunded area. The CIP circuit will be a packaged vendor supplied plant complete with handrails, grating, tanks, Pumpcell mechanisms, launders, manifolds and valves. Seven 250 m^3 tanks will be supplied, complete with Pumpcell mechanisms and Pumpcell maintenance bay.

A walkway will run over the top of the feed launder to allow access to the feed slurry valves. A top of tank platform will provide access to the launder gate valves and Pumpcell mechanisms. The Pumpcell mechanisms will be located below the platform steel to reduce potential trip hazards on the platform area above the tanks. Removable grating panels will provide access to the Pumpcell mechanism for inspection or maintenance. A set of stairs will connect the feed launder and top of tank platform levels.

A gantry crane will be provided by Lycopodium to facilitate Pumpcell removal for maintenance.

The loaded carbon recovery pumps and CIP tails pumps will be located on the eastern end of the bund for ease of maintenance access. Two sump pumps have been provided on the edge of the bunded area for spillage removal.

17.8.5 Elution, Carbon Regeneration and Gold Room

The 12.5 t elution circuit and carbon regeneration kiln will be located adjacent to the CIP circuit, to the south. The loaded carbon screen will be situated above the acid wash column with access to the elution area from the eastern end. The acid wash column will have a partitioned area of the area bund, separated such that any acid spillage cannot mix with cyanide solutions and will be separately disposed of by a dedicated sump pump.

The elution column will be directly adjacent to the acid wash column to the west, with the heater skid located to the south. The elution column will be separately bunded.

The carbon regeneration kiln will be situated to the west, along with the carbon quench hopper, carbon transfer hopper and fine carbon hopper. The carbon regeneration kiln will be partially clad to protect the regeneration kiln shell from differential temperature changes and to keep instrumentation and control panels sheltered from airborne sand. Off-gases from the kiln will be scrubbed and contacted with sulphur impregnated carbon for mercury removal. A dedicated sump pump will be provided for this area. An overhead monorail hoist will be used regularly for new carbon addition but will also be available for maintenance of the kiln drive, shell and rollers. Additional lifting facilities will not be required in this area as maintenance of other equipment will be performed in situ.

The stripping water and lean eluate tanks will be located south of the elution area, along the piperack for ease of pumping. The cold cyanide wash feed and product tanks will also be located in this area.

The goldroom building, including electrowinning cells, mercury retort and smelting furnace will be located at the southern end of the plant, inside a secure area with restricted access by authorised personnel only. The strongroom will be enclosed by additional secure walls and roof to provide extra security. The building will be fresh air ventilated with separate extraction of dust, fumes and heat from the smelting furnace and electrowinning cells. Electrowinning cell off gases will be contacted with sulphur impregnated carbon for mercury removal prior to leaving the goldroom building.

The electrowinning cells and cathode cleaner will be located on the upper level and serviced by an overhead gantry crane. There will be stairway access to the upper level.

The mercury retort will have its own carbon adsorption columns post mercury condensers and collectors for final mercury capture prior to exhaust gas exit via the vent stack.

The pregnant solution tanks will be located outside the goldroom building, adjacent to the major north south piperack and within a concrete bund with a dedicated sump pump.

17.8.6 Tailings Thickening and Pumping

The tailings area will be located to the east of the CIP area. CIP tailings will be pumped from the CIP area to the CIP tails sampler. The CIP tails sampler and carbon safety screen will be at the western side of the thickener, in a steel structure to allow gravity flow of tailings through the sampler and the carbon safety screen to the tailings thickener feed box. Access to the tower will be from a stairway on the western side. There will be four platform levels in this tower; the top level will provide access to the primary CIP tails sampler, the next level to the secondary CIP tails sampler, the lower level to the carbon safety screen and bottom platform level to the tailings thickener feed box. The bottom platform level will be linked to the tailings thickener bridge.

The tailings thickener underflow pumps will be situated under the thickener to minimise suction line length.

The tailings thickener flocculant plant will be located on the southern side of the bund. The tailings thickener overflow hopper and pumps and tailings transfer hopper and pumps will be located at the edge of the bund on the north side nearest the main piperack.

The tailings transfer pumps will transport the tailings slurry via a HDPE pipe approximately 2 km to the discharge point at the tailings storage facility, located to the north-west of the plant.

This equipment will be readily accessible by mobile crane for maintenance access.

17.8.7 Reagents

Reagent mixing areas will be located close to their end users. Cyanide and caustic will be situated in a common steel structure at the southern end of the plant adjacent to the piperack, with shared bunding and sump pump. This will be in proximity to both the elution area and leach tanks where these reagents will be used.

Hydrochloric acid and plant diesel will be located at the eastern end of the elution circuit, near to the acid wash column, elution heater and the kiln.

Each thickener will have a dedicated flocculant mixing and storage plant located adjacent to the thickener bunds, to avoid shearing of flocculant by pumping long distances.

Reagent storage will be located to the south of the plant.

Cyanide

Sodium cyanide briquettes in one tonne bulk bags will be transported to the cyanide mixing plant by forklift. A steel structure with access stairs from ground level via the caustic mixing tank, will house the enclosed bag breaker, a mixing tank and a dedicated bag lifting hoist. The structure will be fitted with a roof to provide protection from the weather. Mixed cyanide solution will be pumped to the cyanide storage tank. Cyanide dosing and reticulation pumps will be located near the piperack to minimise pipe runs.

Spillage in the area is pumped via a sump pump into the leach feed distribution box.

Caustic

Caustic in one tonne bulk bags will be transported to the caustic mixing facility by forklift. An enclosed bag breaker will be provided at top of tank level, over the mixing tank. A dedicated hoist and lifting frame will be provided. A roof will be provided for weather protection. Access to the bag breaker will be via a set of stairs on the southern side. Mixed caustic solution will be pumped to the caustic storage tank. Caustic dosing pumps will be located near the piperack to minimise pipe runs.

Hydrochloric Acid

Hydrochloric acid in 1,000 L bulk containers will be transported to the acid mixing area by forklift. Acid will be delivered to the dilute acid make-up tank using a floor mounted peristaltic pump and mixed with raw water to achieve the required concentration. A dedicated air operated sump pump will dispose of any spillage to the tailings thickener.

Flocculant

Dry flocculant will be transported by forklift from the reagents store to each of the flocculant mixing areas. The bag breaker and dry hopper structure will be partially clad for rain and wind protection. The 750 kg bulk bags will be loaded into a bag breaker using the hoist and lifting frame and transferred to the flocculant mixing tank via the screw feeder and blower. Stair access to the top of each mixing tank will be provided. The mixed flocculant will be transferred to each flocculant storage tank via a transfer pump.

Each flocculant mixing area will have a dedicated sump pump.

17.8.8 Air and Water Services

All compressed air services, including high pressure compressors and low pressure blowers, will be housed in a compressor station adjacent to the cyanide and caustic reagents area and along a major piperack. This area will consist of a fully clad steel structure and concrete slab floor. Normal maintenance will be expected to be carried out with the units in situ.

Water services will be located to the north of the plant, along a major north south piperack. The raw water pond will be situated at the eastern end of the plant. Raw water will be supplied from the borefield approximately 100 km away to the south west. The raw water pond pumps will transfer raw water to the raw water tank. This tank will contain a fire water reserve, as well as storage capacity for raw water, gland water and water treatment plant feed.

The process water pond will be located on the western side of the piperack in order to achieve gravity flow of tailings thickener overflow and pre-leach thickener overflow into the pond. Process water will consist of overflow from the raw water tank, decant return water, excess tailings thickener overflow (not used for leach feed dilution) and excess pre-leach thickener overflow (not used as milling circuit water).

The containerised water treatment plant will provide treated and potable water for use around the plant. No other weather protection has been allowed in the water services area.

17.8.9 Plant Buildings

All plant buildings, including offices, warehouse, plant workshop, laboratory, medical clinic and change house will be located at a local high point to the south east of the plant.

18. PROJECT INFRASTRUCTURE

18.1 Water Supply

GCS (Pty) Ltd (GCS), a South African based groundwater consulting company, was contracted by Orca Gold Inc. (Orca) to assist with the exploration and numerical calculations of groundwater resources in the Galat Sufar South Project Area, Block 14.

After completion of the 2014 to 2016 groundwater exploration, it was concluded that the identified aquifers had limited potential for long term groundwater abstraction for large scale mining production.

A potential area approximately 80 km south west of the proposed mining area was identified. SkyTEM Surveys of Denmark, an airborne geophysical contractor specialising in water exploration, were contracted to fly a 5,000 km electromagnetic survey where two old production wells are located - "Station 6 Well" and "Bir Mufta Well". The survey returned positive results over Area 5, which has been followed up with drilling, aquifer testing and water quality analyses.

18.1.1 Historical Groundwater Exploration

Limited reference material for this part of the Arabian Nubian Shield (ANS) and, more specifically, the Nubian Sandstone Aquifer (NSA) was available through desktop surveys. An approach of remote sensing, ground geophysics, ground geology mapping, exploration drilling and pump testing was followed to gain a better understanding of the regional potential for groundwater resources. Between 2014 and 2016, a total of 33 groundwater exploration boreholes have been drilled within eight pre-identified areas across the Block 14 area. Remote sensing and geophysical surveys was used to select drilling targets. In late 2016, 24 hr pump testing was completed on five of the higher yielding boreholes within the HA8 area. It was concluded that the HA8 aquifer would not be able to sustain a water demand of more than 2,000 to 3,000 m^3/day for a period of 12 years.

18.1.2 Water Demand

For the purpose of the Feasibility Study (FS), the maximum and approximate groundwater resource required is as follows:

- To enable processing of 6.0 Mtpa.
- Constant water supply of an average of 12,500 m^3/day or 145 l/s or 4.6 Mm^3/yr.
- Minimum volume over the LOM of 61.5 Mm^3.

18.1.3 Groundwater Resource Exploration in Area 5

Geophysical Survey

Area 5 was chosen for water exploration on the basis that two existing boreholes had for many years reliably supplied water, both for the railway and for the local population.

The depth, standing level of water and the strata intersected by the two wells at Station 6 and Bir Mufta (Figure 18.4) was determined. These observations confirmed the aquifer was hosted in the Nubian Sandstone Formation and pump tests confirmed that both wells can produce groundwater.

An airborne conductivity survey (EM or Electro Magnetic), carried out by SkyTEM (SkyTEM312FAST AEM system). The initial coverage was carried out at a line spacing of 1 km; further infill was later acquired over areas of interest at a spacing of 250 m. The total area covered being around 1,080 km^2. At the same time as the airborne geophysics was being acquired, more geological mapping was carried out, at a reconnaissance scale, to ensure that the geophysical results could be interpreted coherently.

The raw EM data were inverted to provide maps and 2D sections of the ground conductivity and this set of results were used to suggest the first exploratory borehole drilling targets. Initially, the areas of highest conductivity were targeted, stepping out from Bir Mufta as a known aquifer. It was quickly found that the highest conductivities were actually species of basement and not good aquifer.

The model was revised and targets selected from zones of intermediate conductivity which showed strong horizontal layering. This approach proved successful and good aquifer horizons were intersected.

The finalised conductivity models were converted into a 3D voxel grid, from which conductivity surfaces were derived and initial interpretations were used to inform the ongoing exploratory drilling. Refer to Figure 18.3 for some graphical interpretations.

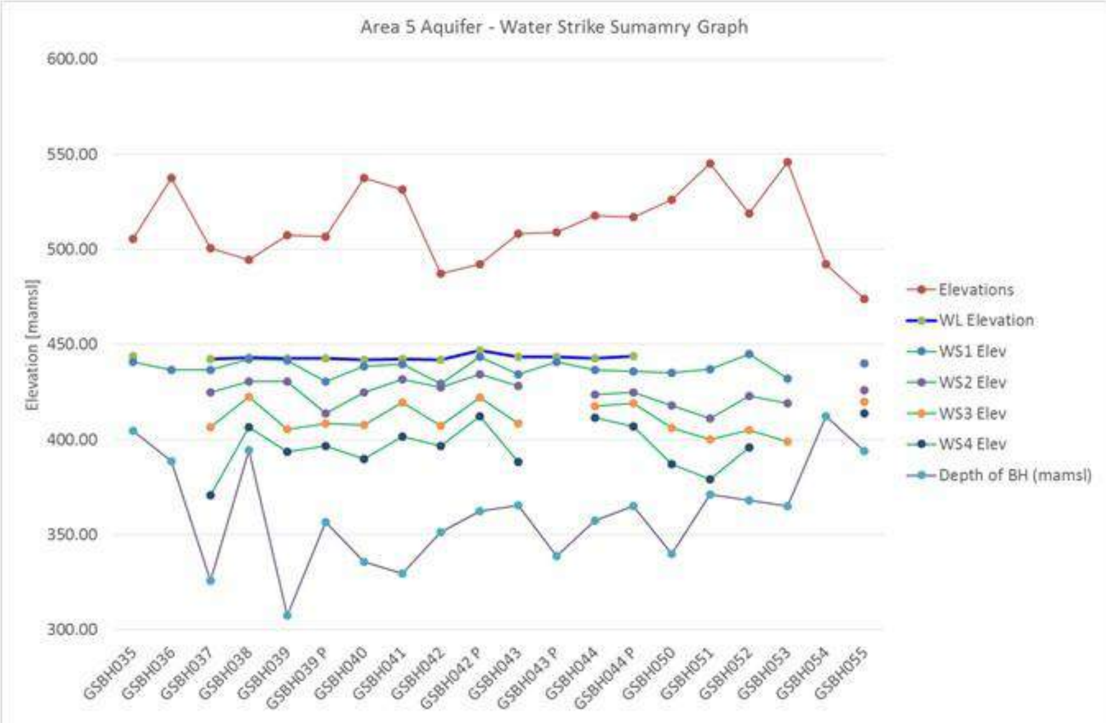
Observations from Drilling Phase

Drilling of 17 exploration boreholes in Area 5 and four test production boreholes commenced in April 2017 and was completed in September 2017. The drilling activities were undertaken by GED and DAL Mining Services, drilling supervision was done by GCS (Pty) Ltd and Orca Gold Staff. An overview of the drilling results and general site and aquifer settings:

- The exploration boreholes were drilled using 10 and/or 8-inch drill bit to install 6-inch pre-collar casing which was advanced to the desired depth using 5 inches drill bit. Solid and perforated uPVC casing of 4-inch sizing was then installed in each completed exploration borehole.
- Exploration boreholes GSBH035 and GSBH036 were unsuccessful. These boreholes were drilled to 101 m and 149 m respectively. The basement granites were intersected in GSBH035. GSBH036 had no sample recovery from 120 m to 149 m.
- An area of approximately 130 km^2 was confirmed / exploited to have significant and constant airlift yields. Airlift yields was measured during drilling by means of a portable V-notch (Figure 18.4). The flow from five boreholes exceeded the volume capacity of the V-notch.

- Water was intersected in boreholes GSBH037 to GSBH055. Boreholes GSBH039, GSBH042, GSBH043, GSBH044 and GSBH54 displayed much higher airlift yield above 10 l/s. The other boreholes had airlift yields between 3.3 l/s and 7.8 l/s with most boreholes around the 5 l/s level. Borehole GSBH037 displayed the lowest yield of 3.3 l/s. Basement granites were intersected in this borehole.
- The geology from the exploration Area 5 is mainly sedimentary sequence with alternating beds ranging from fine to coarse grain sedimentary rocks (sandstones presumably associated with the Nubian Sandstone sequence and Siltstones / Mudstones). The water bearing formations are mainly coarse grained sandstones with abundant quartz chippings.
- The Nubian Sandstone sequence was confirmed to be continuous and has good groundwater potential across the exploited area. Due to the continuous nature and thickness of the course grained sequence, it can be concluded that potentially high volumes of groundwater are available as storage. It is fair to refer to this sequence as the Nubian Sandstone Aquifer as it has similar characteristics when considering reference material. However, the sequence also reflects similarities with typical alluvial basins associated with historical stream flow patterns in the area. More discussions will follow in this report to assess the potential in more detail.
- The following information can be derived from Figure 18.1 below:
  - groundwater was intersected at an average depth of 75 m below surface (range between 34 and 100 m) or 437 mamsl and the main aquifer system (higher yielding zone) around 100 m below surface or 410 mamsl
  - static groundwater elevation is in the order of 443 mamsl and range between 45 and 95 m below surface depending on surface elevation
  - the main airlift yields were achieved in a zone of approximately 40 m thick with limited increases observed in depth. The thickness decreased significantly within the lower yielding boreholes (i.e. < 3 l/s).

Figure 18.1 Simplified Overview of Aquifer Elevations



Borehold collar elevation, static groundwater level elevations and water strike elevations.

- The aquifer is associated with a basin type and layered sequence and can be described as:
  - a shallow sub-outcrop in the east, traces of dolerite and rhyolite was intersected in boreholes GSBH35 at 98 m, GSBH37 at 153 m and GSBH40 at 197 m,
  - the basin dips towards the west and north west at an unknown angle since none of the boreholes to the north-west intersected basement rock,
  - the layered sequence appears to be initially unconfined and evolved into a semi-confined or leaky layered sequence of sandstone and siltstone. Sandstone layers are mostly course grained and greyish / white in colour,
  - later drilling indicated an increase in siltstones within the north-western deeper parts of the confirmed area,
  - two simplified cross section are included (Figure 18.5) based on data received from Harwood, 2017 which was derived from the Airborne Geophysics and geological logs obtained from the boreholes. It can be seen that a major basin exists, striking from south-east to north-west which deepens towards the north-west.

The regional topographical setting in relation to the proposed mining area can be seen from Figure 18.7.

Table 18.1 Area 5 Boreholes

No.	Borehole Name	East (UTM 36N)	North (UTM 36N)	Elevations (mamsl)	Static WL (May 2017) (mbc)	Static WL (Oct 2017) (mbc)	WL Elevation (mamsl)	Complete Date	Depth (m)	Depth of BH (mamsl)	WS1 (m)	WS2 (m)	WS3 (m)	WS4 (m)	WS5 (m)	WS6 (m)	Final Airlift Yield (l/s)	Final Airlift Yield (m^3/hr)
1	GSBH035	461638.41	2312999.76	505.74	61.8	-	443.9	25-Apr-17	101	404.7	65	-	-	-	-	-	-	-
2	GSBH036	458470.69	2312975.27	537.65	-	-	-	27-Apr-17	149	388.7	101	-	-	-	-	-	-	-
3	GSBH037	453853.68	2305018.22	500.66	58.2	58.0	442.4	02-May-17	175	325.7	64	76	94	130	-	-	3.3	11.88
4	GSBH038	449801.62	2305022.18	494.47	51.3	52.2	443.2	06-May-17	100	394.5	52	64	72	88	-	-	5.7	20.52
5	GSBH039	448409.93	2307996.14	507.57	64.8	64.8	442.7	10-May-17	200	307.6	66	77	102	114	126	138	9.7	34.92
12	**GSBH039 P**	**448432.66**	**2307951.73**	**506.60**	**64.0**	**64.2**	**442.6**	**29-Jun-17**	**150**	**356.6**	**76**	**93**	**98**	**110**	**130**	**-**	**11.9***	**42.84**
6	GSBH040	452049.77	2307999.49	537.64	95.6	**95.7**	442.1	16-May-17	202	335.6	99	113	130	148	156	178	5.7	20.52
7	GSBH041	450003.46	2309995.71	531.50	89.2	89.2	442.3	21-May-17	202	329.5	92	100	112	130	160	-	6.8	24.48
8	GSBH042	449817.92	2301979.32	487.46	45.3	45.7	442.1	24-May-17	136	351.5	58	60	80	91	-	-	11.9*	42.84
13	**GSBH42 P**	**449773.97**	**2302005.66**	**492.32**	**45.4**	**46.1**	**446.9**	**03-Jul-17**	**130**	**362.3**	**49**	**58**	**70**	**80**	**90**	**-**	**11.9***	**42.84**
9	GSBH043	446807.00	2304980.00	508.30	64.9	65.7	443.4	28-May-17	143	365.3	74	80	100	120	-	-	11.9*	42.84
14	**GSBH043 P**	**446737.88**	**2304984.74**	**508.92**	**65.6**	**65.6**	**443.3**	**18-Jun-17**	**170**	**338.9**	**68**	**-**	**-**	**-**	**-**	**-**	**11.9***	**42.84**
10	GSBH044	445173.31	2306990.07	517.60	75.0	73.5	442.6	07-Jun-17	160	357.6	81	94	100	106	118	-	11.9*	42.84
15	**GSBH044 P**	**445141.10**	**2307029.86**	**516.98**	**73.0**	**73.5**	**444.0**	**23-Jun-17**	**152**	**365.0**	**81**	**92**	**98**	**110**	**130**	**-**	**11.9***	**42.84**
16	GSBH050	443394.00	2309500.00	526.00	-	84.1	442.0	16-Sep-17	186	340.0	91	108	120	139	178	-	7	25.2
17	GSBH051	441250.00	2312000.00	545.00	-	-	-	20-Sep-17	174	371.0	108	134	145	166	-	-	6	21.6
18	GSBH052	446109.00	2311000.00	519.00	-	75.3	443.7	22-Sep-17	151	368.0	74	96	114	123	-	-	4.8	17.28
19	GSBH053	443668.00	2314000.00	546.00	-	-	-	26-Sep-17	181	365.0	114	127	147	-	-	-	4.6	16.56
20	GSBH054	451658.94	2299537.52	473.89	-	33.0	440.9	27-Sep-17	90	383.9	34	48	54	60	-	-	10.6	38.16
21	GSBH055	450524.76	2296387.04	492.22	-	-	-	29-Sep-17	120	372.2	-	-	-	-	-	-	7.8	28.08

***Airlift measured by V Notch with maximum measuring capacity of 11.9 l/s, boreholes marked with "*" indicated >11.9 l/s.**

Figure 18.2 Regional Overview of Groundwater Exploration

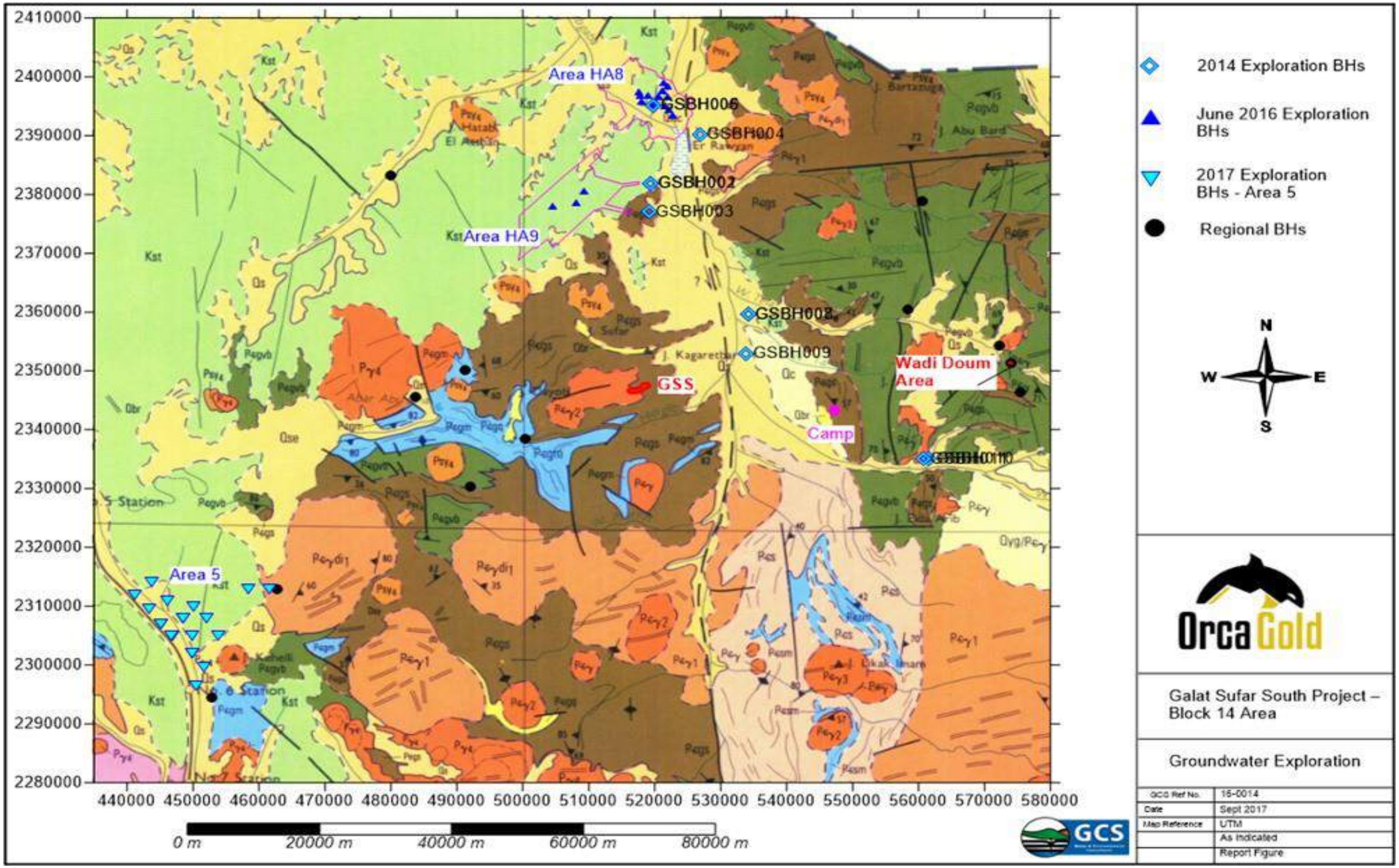


Figure 18.3 Airborne Geophysical Survey for Area 5

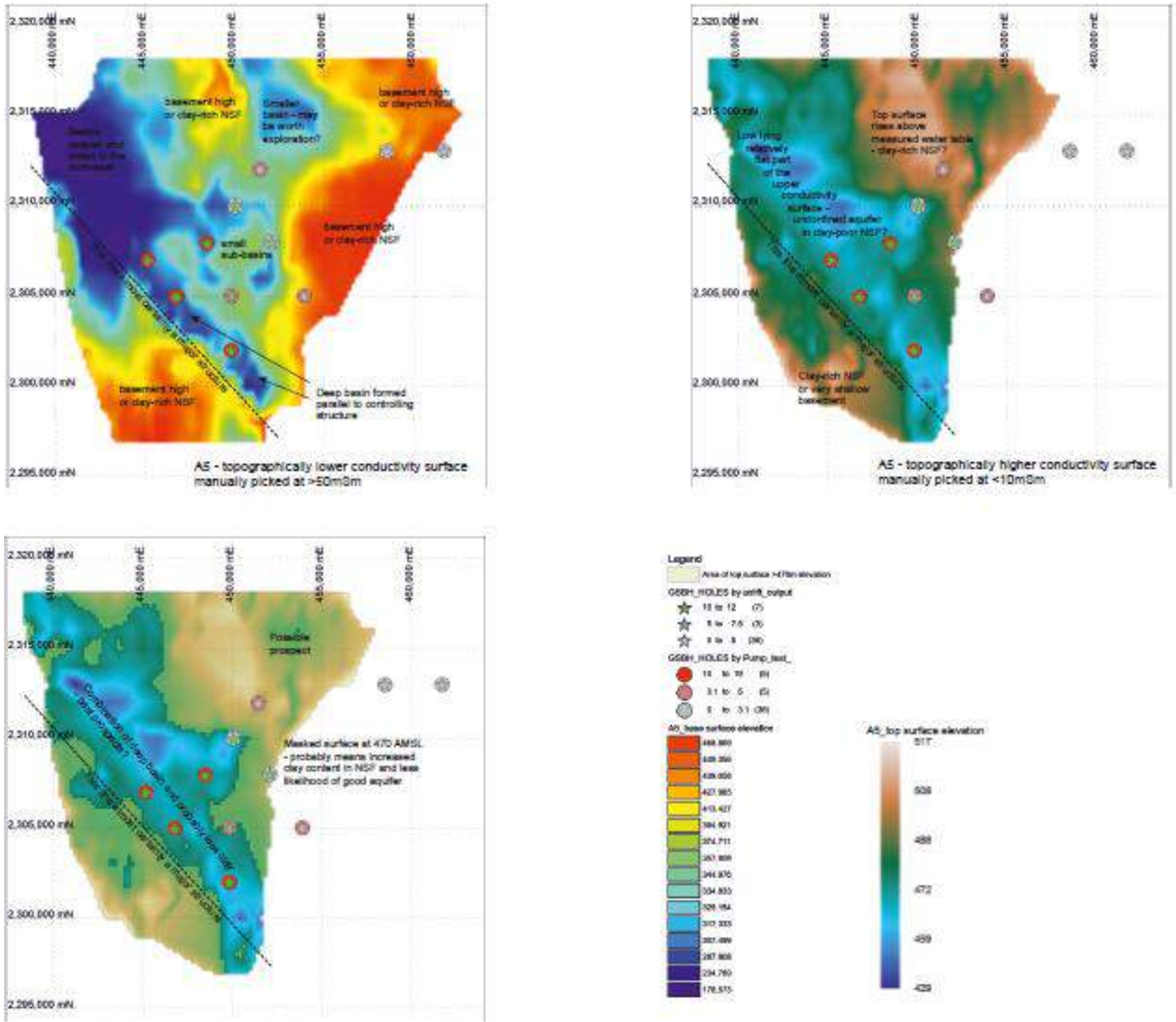


Figure 18.4 Drill Sites, Airlift Yields and Demarcated Aquifer Area

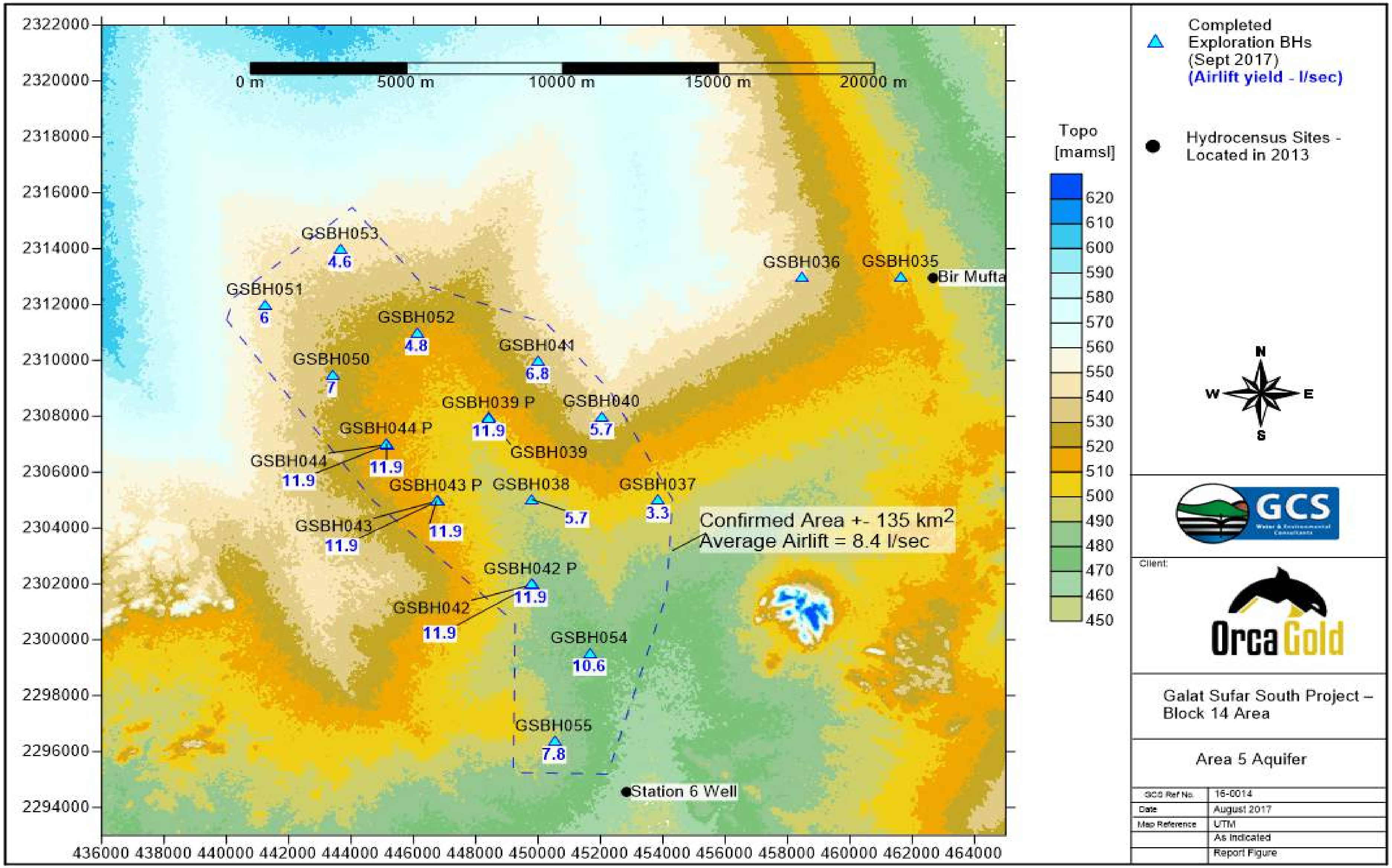
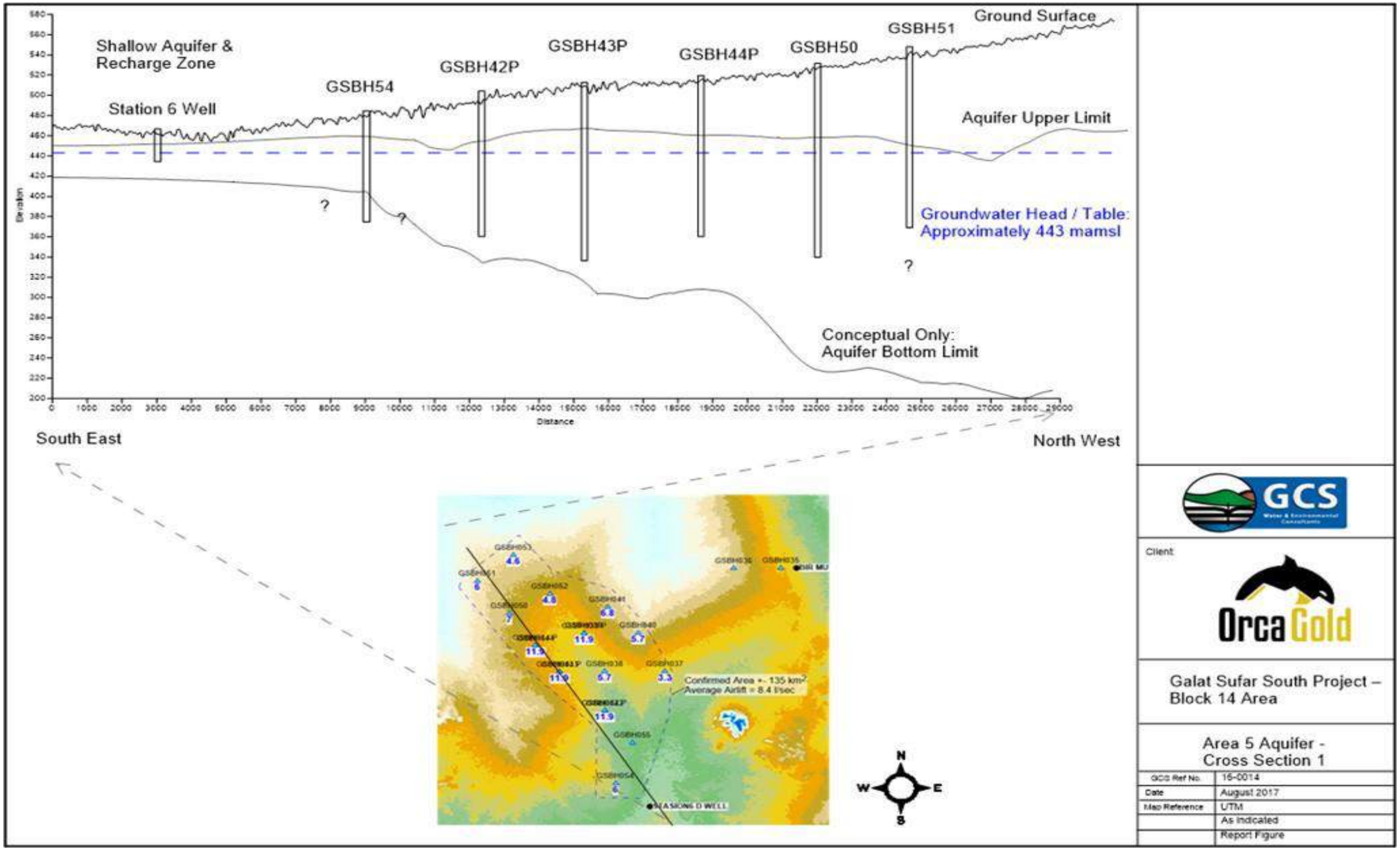
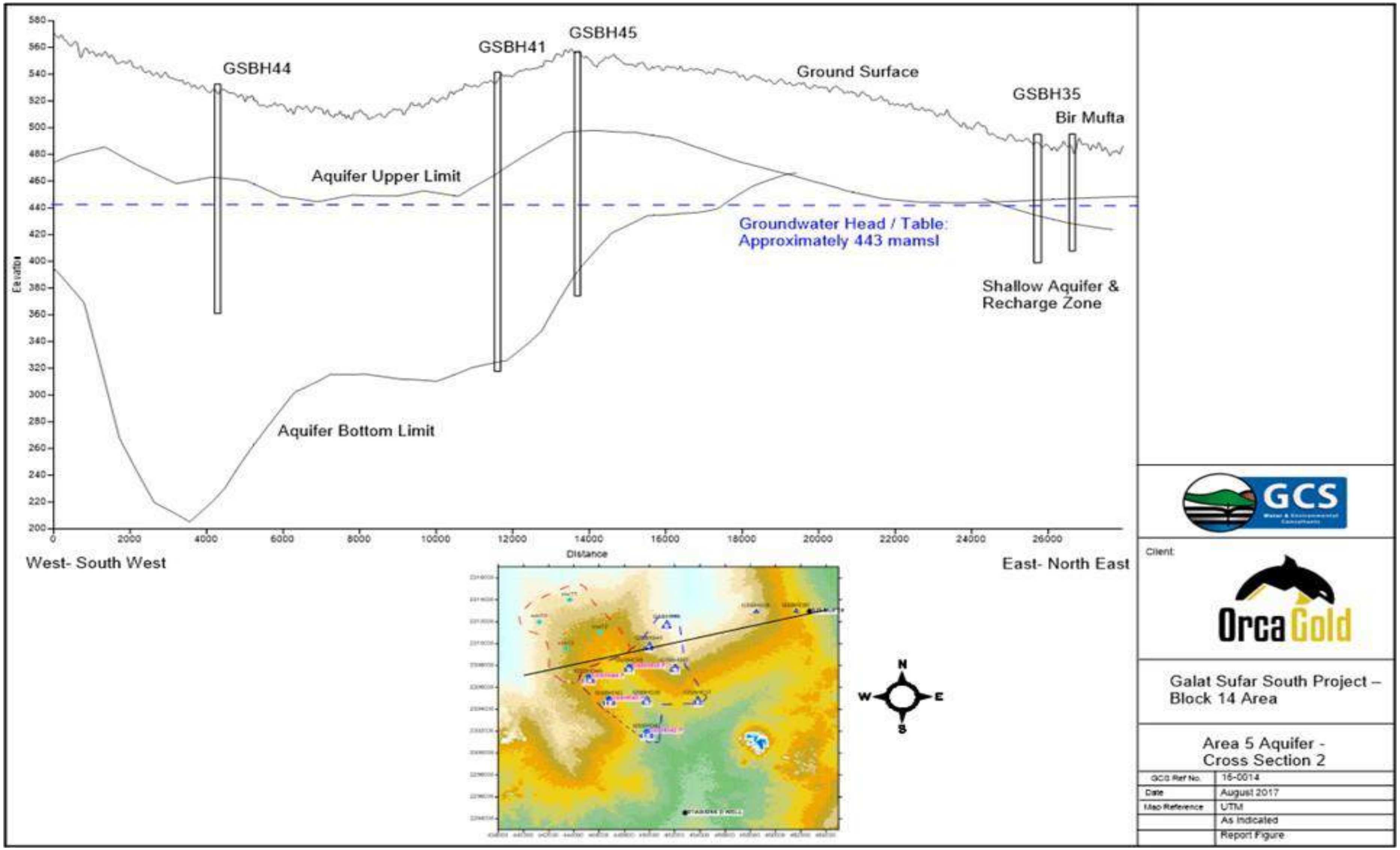


Figure 18.5 SE-NW Cross Section for Area 5



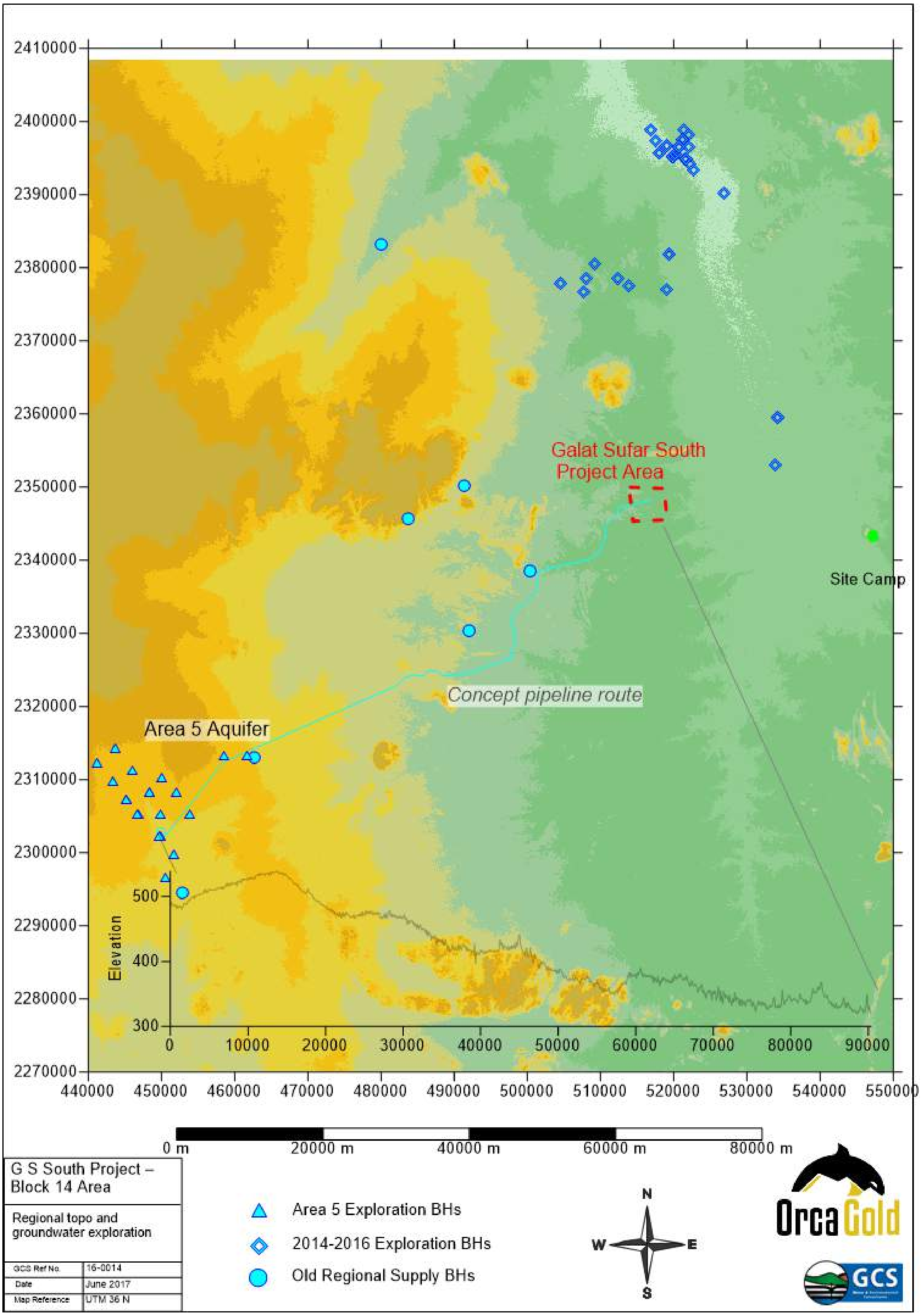
Harwood, 2017

Figure 18.6 WSW-ENE Cross Section for Area 5



Harwood, 2017

Figure 18.7 Topographical Setting of Area 5



18.1.4 Pump Test Results

A total of nine aquifer tests (Table 18.2), consisting of a series of constant discharge rate and recovery tests, were evaluated to obtain initial estimates of hydraulic properties and determine aquifer hydraulic parameters. Four of the tests were conducted from dedicated tests boreholes, where the test borehole was drilled 50 m from the original exploration borehole (these boreholes were labelled with a "P"). The objective was to assess distance drawdown and storage coefficients. The summary of the test results is tabled in Table 18.6 to Table 18.8.

Constant Rate (CR) Test

Constant rate tests were carried out in the boreholes indicated in Table 18.2. It is evident from the table that drawdown within the pumped boreholes were only between 0.5 and 12 m and duration of the tests was between24 and 48 hrs. The drawdown obtained in the boreholes generally suggests minimal well losses and the boreholes are efficient.

A graphical summary of the CR tests is provided in Figure 18.9.

Table 18.2 Summary of Pump Durations

Borehole Name	Pump Test Date	Pump Depth (m)	Pump Rate (l/s)	Pump Rate (m^3/hr)	Pump Rate (m^3/day)	Time Pumped (hr)
GSBH037	-	75	3.1	11.2	268	48
GSBH038	13-Jun-17	65	3.4	12.2	294	36
GSBH039	21-May-17	70	3.6	12.9	309	48
GSBH039 P	**30-Jun-17**	**87**	**19**	**68.4**	**1,642**	**48**
GSBH042	-	55	3.8	13.7	328	24
GSBH042 P	**03-Ju-17**	**70**	**19**	**68.4**	**1,642**	**28**
GSBH043	29-May-17	104	18	64.8	1,555	36
GSBH043 P	**22-Jun-17**	**84**	**13**	**46.8**	**1,123**	**39**
GSBH044 P	**26-Jun-17**	**93**	**17**	**61.2**	**1,469**	**48**

Transmissivity and Hydraulic Conductivity

Table 18.8 supplies a list of transmissivity values as obtained from the pump test analyses. Both Cooper Jacob and Theis Recovery analyses methods were applied. It can be seen from the table that values range between 100 and 1,700 m^2/day. The values were converted to hydraulic conductivity by using an average thickness of 40 m (K=T/thickness; refer to Table 18.3).

Table 18.3 Average K Values Derived from Pump Tests

	m/day
High Zone	
Average High K	40.63
Average Low K	16.25
Total Average	**28.44**
Lower Zone	
Average High K	16.13
Average Low K	9.38
Total Average	**12.75**

Storativity and Specific Yield

Pump test results and field observations indicate rather leaky (semi confined) to unconfined conditions. The following table supplies a range of Sy values derived from the pump test data. This can be regarded as a conceptual and indicative value only (Table 18.4). It can be seen from Table 18.5 that minimal drawdown in the aquifer were observed 50 m away from the production boreholes during the 24 to 48 hr pump tests. These small drawdowns were achived under fairly high pump rates.

Table 18.4 Estimated Specific Yield

Pumping Well ID	Observation Well ID	Distance Between Wells (m)	AVG Rate (l/s)	Test Duration (min)	Q Total (m^3)	Dewater Volume (m^3)	Sy Estimation (%)
GSBH043	GSBH043 P	50	13	2,340	1,825.20	8,992.81	20%
GSBH044	GSBH044 P	50	17	2,880	2,937.60	31,664.64	9%
GSBH039	GSBH039 P	50	18	2,880	3,110.40	44,087.02	7%
GSBH042	GSBH042 P	50	18.4	1,680	1,854.72	14,097.90	13%

Table 18.5 Observation Borehole Data for CR Tests

Borehold Name	Pump Rate (l/s)	Observation	Distance (m)	Static WL (m)	Water Level Obtained (mbc)	Drawdown Achieved (m)
GSBH039 P	**19**	**GSBH039**	**50**	**65.16**	**67.34**	**2.18**
GSBH042	3.8	-	-	-	-	-
GSBH042 P	**19**	**GSBH042**	**50**	**45.26**	**46**	**0.74**
GSBH043	18	-	-	-	-	-
GSBH043 P	**13**	**GSBH043**	**50**	**65.75**	**66.45**	**0.7**
GSBH044 P	**17**	**GSBH044**	**50**	**73.45**	**75.4**	**1.95**
GSBH045	3.1	-	-	-	-	-

Table 18.6 indicates specific capacity between 3.8 and 21.7 (m^3/day/m).

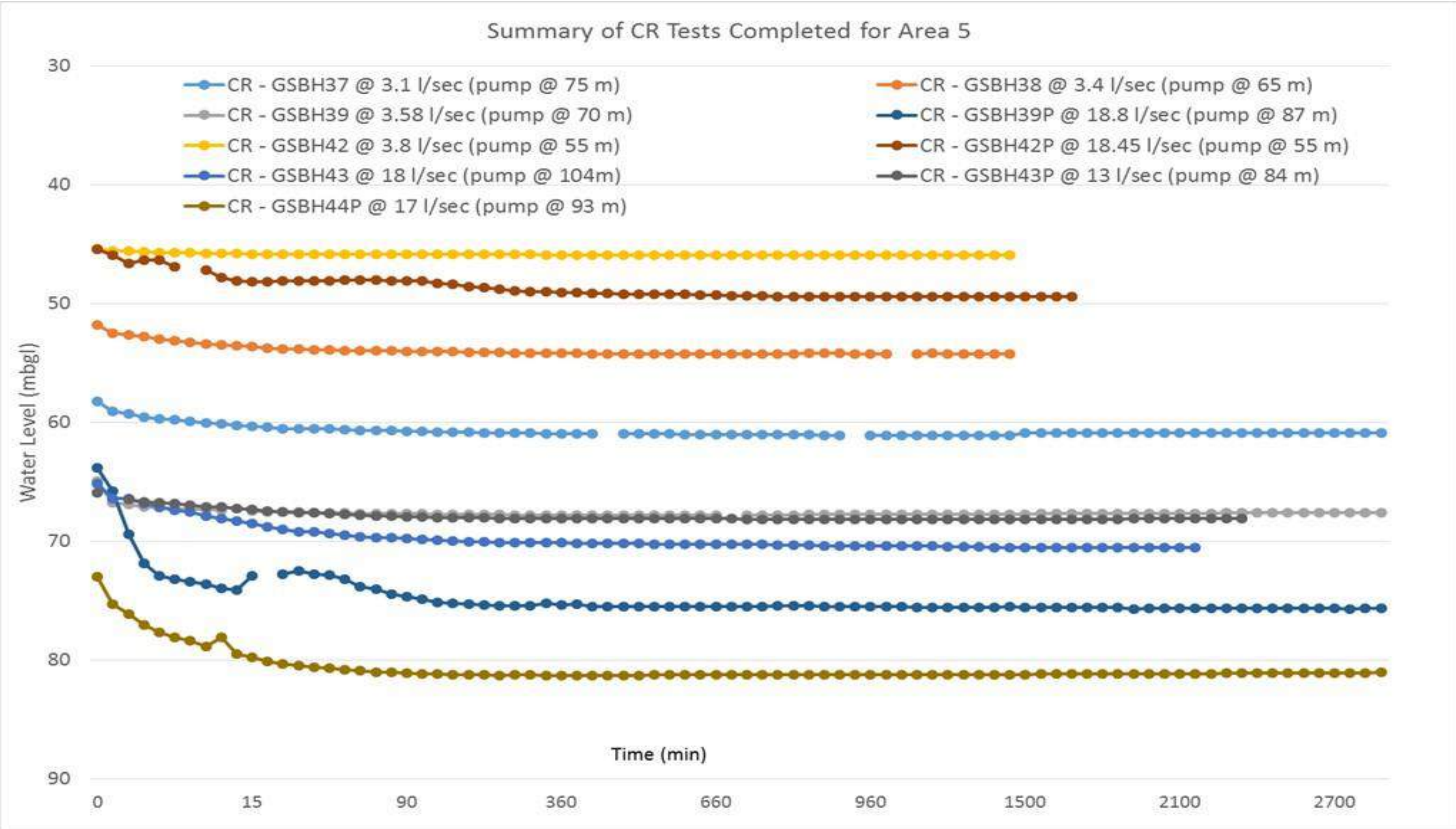
Table 18.6 Constant Rate Tests Summary for Area 5

No.	Boreholde Name	Pump Test Date	Pump Depth (m)	Pump Rate (l/s)	Pump Rate (m^3/hr)	Time Pumped (hr)	Water Level Obtained (mbc)	Drawdown Achieved (m)	Specific Capacity (l/s/m)	Specific Capacity (m^3/hr/m)	Total Vol. (m^3)
3	GSBH037	-	75	3.1	11.2	48	61.1	2.91	1.07	3.84	536
4	GSBH038	13-Jun-17	65	3.4	12.2	36	54.3	2.99	1.14	4.09	441
5	GSBH039	21-May-17	70	3.58	12.9	48	67.8	2.98	1.20	4.32	619
12	**GSBH039 P**	**30-Jun-17**	**87**	**19**	**68.4**	**48**	**64.0**	**11.91**	**1.60**	**5.74**	**3,283**
8	GSBH042	-	55	3.8	13.7	24	45.9	0.63	6.03	21.71	328
13	**GSBH042 P**	**03-Jul-17**	**70**	**19**	**68.4**	**28**	**45.4**	**4.00**	**4.75**	**17.10**	**1,915**
9	GSBH043	29-May-17	104	18	64.8	36	70.6	5.68	3.17	11.41	2,333
14	**GSBH043 P**	**22-Jun-17**	**84**	**13**	**46.8**	**39**	**65.6**	**2.21**	**5.88**	**21.18**	**1,825**
15	**GSBH044 P**	**26-Jun-17**	**93**	**17**	**61.2**	**48**	**73.0**	**8.08**	**2.10**	**7.57**	**2,938**
11	GSBH045	25-Jun-17	130	3.1	11.2	24	112.0	1.58	1.96	7.06	268

Table 18.7 Constant Rate Tests and Aquifer Parameters

No.	Boreholde Name	Recovery To (wl-m)	Recovery Time (hr)	%	T High (m^2/dy)	T Low (m^2/dy)	K High (m/day)	K Low (m/day)	S
3	GSBH037	58.23	0.8	99.7%	235	100	5.88	2.50	-
4	GSBH038	51.7	4.5	86.0%	-	-	-	-	-
5	GSBH039	64.95	1.8	95.6%	345	230	8.63	5.75	-
12	**GSBH039 P**	**65.17**	**2.0**	**99.5%**	**1,750**	**992**	**43.75**	**24.80**	**-**
8	GSBH042	45.37	1.8	92.1%	1,000	620	25.00	15.50	-
13	**GSBH042 P**	**45.33**	**3.5**	**91.0%**	**1,750**	**631**	43.75	15.78	2.46E-05
9	GSBH043	65.43	4.0	90.5%	1,000	720	25.00	18.00	-
14	**GSBH043 P**	**65.84**	**4.0**	**87.0%**	**1,500**	**583**	37.50	14.58	8.10E-04
15	**GSBH044 P**	**73.73**	**3.5**	**87.0%**	**1,500**	**566**	37.50	14.15	7.41E-05
11	GSBH045	110.68	5.5	82.9%	-	-	-	-	-

Figure 18.8 **Summary of the Constant Rate Test and Drawdown Curves**



18.1.5 Water Quality Considerations

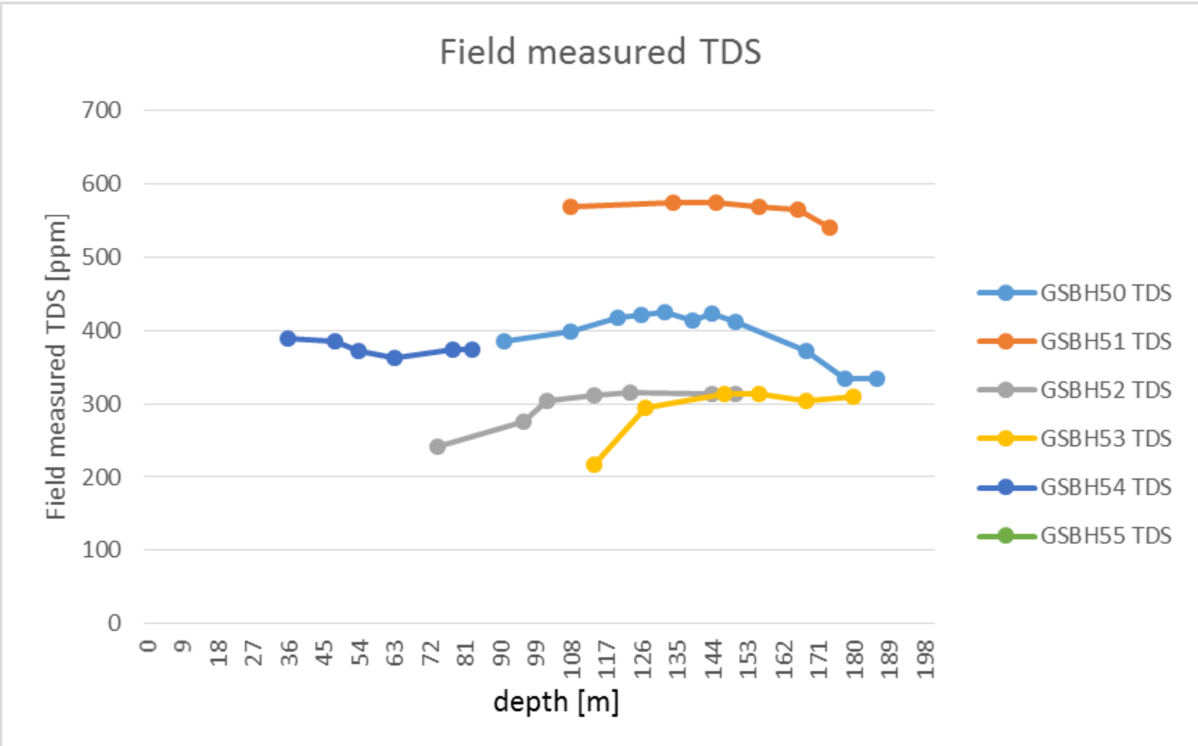
Detailed water quality analyses were completed on three of the exploration boreholes, a summary of the analyses is tabled below (Table 18.8). Electrical conductivity (EC) ranges between 637 and 590 uS/cm (63.7 to 59 mS/m). The water has a good quality and falls within the Target Water Quality Range for Human Consumption for most of the elements (WHO, 2017).

Table 18.8 Summary of the Groundwater Quality Analyses for Area 5

Borehold Name	Ca	Mg	Na	SO_4	Cl	NO_3	T Alk	EC (uS/cm)	pH
GSBH042	56	21.7	83.7	188.4	64	3.8	150	611	7.85
GSBH043	63.9	18.6	76.5	158.1	73.4	25.3	124	637	7.95
GSBH044	59.5	16.6	67.5	146.3	70	27.1	113	590	7.85

TDS (mg/l or ppm) was measured with a field TDS meter during airlift intervals as drilling continuous of some of the exploration boreholes. The results are graphically presented in Figure 18.9, which shows a constant TDS averaging around 400 mg/l.

Figure 18.9 Field Measured TDS During Drilling



18.1.6 Key Hydrogeological Concepts

The following concluding key hydrogeological concepts were derived from the drilling and pump testing phases, as discussed above:

- The geology from the exploration Area 5 is mainly sedimentary sequence with alternating beds ranging from fine to coarse grain sedimentary rocks (Sandstones and Siltstones). The main aquifer zone consists of clean, well sorted, whitish / pale grey and poorly cemented sandstone. This zone appears to occur at depths around 35 m in the lower topographical southern area and 120 m within the higher topographical norther western area.
- This sequence occurs in a basin type depression associated with some degree of faulting. It appears if the basin dips in a north western direction.

- It was established that the airlift yields obtained are directly proportional to the thickness of the well sorted medium to coarse grained sandstone gravels (particle size range from 0.5 to 4 mm). The more this sequence is altered with layers of siltstone and mudstone, the lower the yields. The main zone varies in thickness and an average of 40 m was used for calculation purposes.
- The yields obtained are not directly proportional with depth and it was established that lower yields may occur within the deeper part of the basin due to thicker layers of siltstone. Unfortunately, the boreholes were not drilled deep enough to confirm the basin bottom, however sufficient yields occur at shallower depths.
- The groundwater level is approximately constant in all the boreholes around 443 mamsl.
- The aquifer appears to be semi confined to unconfined in the shallower southern area and more semi-confined in the deeper north western area.

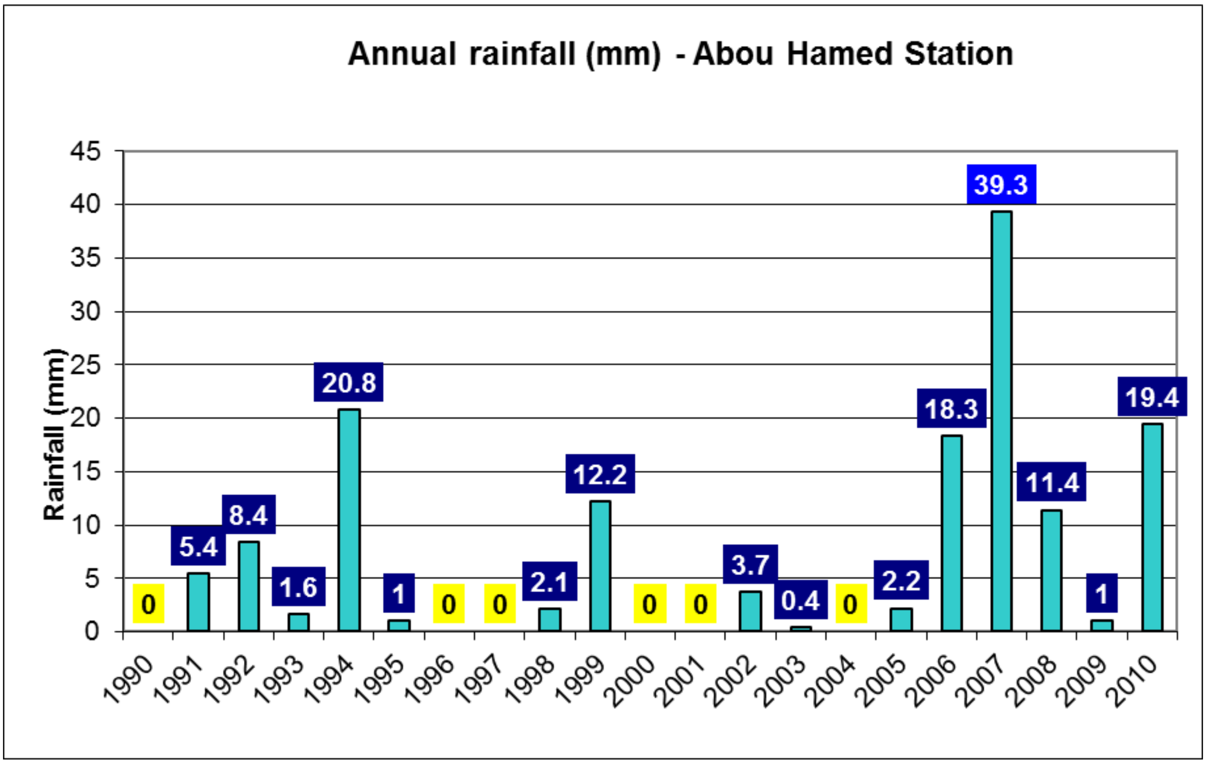
Aquifer Recharge

Considering the good water quality that has been identified and the shallow nature of the southern portion of the Area 5 aquifer, it is highly likely that some aquifer recharge from rainfall occurs.

The northern area of Sudan is typical of desert and arid climate and very low rainfall occur. However, flash floods occur from time to time as a result of thunderstorms and events usually involve 5 to 20 mm of rainfall within a short period of time. It can be seen from Figure 18.10 Abou Hamed's weathered station recorded only 147 mm in 21 years from 1990 to 2010. The annual average rainfall averages 7 mm/y.

It is believed that significant volumes can recharge the shallow groundwater areas during these flash-flood periods.

Figure 18.10 Rainfall Data for the Abu Hamed



Theoretical Aquifer Volume Indication

Based on the range of aquifer parameters obtained from the field work (Table 18.9), a first order indication of aquifer storage potential can be calculated. Due to prevailed uncertainties, it is common practice to allow for a range of expected parameters to enable a better understanding of the potential storage capacity. Based on the available data, the conservative and theoretical aquifer storage is in the order of 100 Mm^3.

The minimum water demand has been determined to be 61.5 Mm^3 for the duration of the life of mine (13.5 years). The groundwater storage volume within the Area 5 aquifer should be adequate to supply the project for the life of mine.

Table 18.9 Range of Indicative Groundwater Resource Available in Area 5

Aquifer Area Confirmed (m^2)	Average Rainfall Recharge per Annum (m)	Average Nett Specific Yield	Average Aquifer Thickness (m)	Theoretical Aquifer Volume (m^3)
100,000,000	-	10.0%	10	100,000,000
100,000,000	-	12.5%	20	250,000,000
135,000,000	0.0015	10.0%	10	135,202,500
135,000,000	0.0015	12.5%	20	337,702,500
135,000,000	0.0015	10.0%	20	270,202,500
135,000,000	0.0015	12.5%	40	675,202,500

Typical Well Field Requirements

To meet the proposed water demand as indicated in Table 18.10, a typical well field will require approximately 20 production boreholes with spacing between the boreholes in the order of 900 m. The distance between the wells was simulated in the numerical groundwater model. The depths of the wells will be in the order of 150 m and installed with 8-inch casing and 6-inch perforated stainless steel screens.

Table 18.10 Well Field Specifications for Area 5

Required Demand For:	m^3/day	LOM (years)	Total (m^3)	Throughput (Mtpa)
	12,500	14	64,000,000	**6.0**
Senario 2: Concept Well Field to Deliver (l/s)	**145**	**Yield / BH (l/s)**	**m^3/day**	**No. of wells required**
Low Average Yield		5	432	29
Expected Average Yield		8	696	18
High Average Yield		10	864	15

18.1.7 Preliminary Numerical Well Field Simulation for the Area 5 Aquifer

Model Limitations and Assumptions

The following assumptions were applied:

- Bulk aquifer properties for the different geological units were used for the preliminary numerical applications, as per the discussion above.
- It is assumed that the aquifer is a continuous unit between the boreholes and that no compartmentalization exits within the identified aquifer zones apart from lower and higher hydraulic conductivity zones.

- The thickness of the main zone varies, for calculation purposes a thickness of 20 to 40 m will be considered.
- Hydraulic conductivity values for the higher yielding areas range between 17 and 40 m/day and for the lower yielding areas 10 to 16 m/day.
- An area of approximately 130 km^2 was confirmed through drilling and this area (as indicated in Figure 18.4) forms the center part of the numerical groundwater model.
- Storage values between 10E-6 and 10E-4 was applied and specific yields range from 5% to 20%.
- Recharge of 1 (north-west) to 2 mm (south) per annum was applied in the numerical groundwater model as an initial figure.
- The areas surrounding the confirmed area forms the outer model grid area and lower hydrogeological values were applied (1 and 2 orders of magnitude lower as per the confirmed area). It is assumed that some degree of groundwater flow exists outside the confirmed area.
- Head boundaries in the form of General Head Boundaries were allocated to the most northern and southern perimeters of the model grid. This is not expected to contribute to the flow of groundwater within the model grid.

Model Construction

The numerical model was constructed using Visual Modflow Pro (Version Build 4.6.0.167), a pre- and post-processing package for the modelling code MODFLOW.

Model Grid

The finite difference model grid consists of 463 740 cells which consist out of 295 rows, 262 columns and six layers.

Definition of the grid and the geometry of the layers was carried out using the VISUAL MODFLOW graphic user interface. Smaller cell sizes (50 m x 50 m) are specified in the areas of the main aquifer zone where a more accurate solution of the groundwater flow equation is required.

The model layers represent:

Table 18.11 Finite Grid Model Layers

Geo Zones / Model Layer	Aquifer	Transmissivity (m^2/day)	Hydraulic Conductivity (m/day)	Specific Yield (%)	Specific Storativity
1-	Overburden windblown sand, clay and silt layers and predominantly sandstone. Layers may vary between unconfined to sem-confined (leaky). Approximately 90 m to 40 thick.	5 to 10	0.2	5	-
2	Semi confined (leaky) zone between upper zone and main aquifer. Mostly fine to medium grained sand and approximately 10 m thick. Some clay lenses exist.	7.5 to 15	1.5	10	3.0E-06
3	Semi confined zone of main aquifer, approximately 30 to 20 m thick.	250 to 1,000	12 to 25	12 to 25	3.0E-05
4	Semi confined zone of bottom of main aquifer, approximately 20 to 30 m thick.	50 to 250	5	12 to 15	8.0E-05
5	Semi confined to leaky layer below main aquifer zone, transition zone to bedrock.	50	1	5	3.0E-06
6	Bedrock layer and solid sandstone.	0.1	0.05	1	3.0E-07

Model Calibration

A steady state numerical simulation was developed using the hydraulic parameters described in the previous sections. A head around 443 - 445 mamsl was achieved over the main aquifer zone which is in line with the measured head (i.e. groundwater levels within the boreholes).

This steady state simulation was used as the initial head for the transient state simulations.

The pump test data was subsequently applied to calibrate the transient state model by inserting pump wells which were allowed to abstract for one to 10 days. The groundwater parameters were adjusted to enable a correlation between the field pump test data and the model simulations.

18.1.8 Conceptual Well Field Design

A conceptual well field design was used to develop an operational borehole field within Area 5. A preliminary groundwater model was constructed to evaluate the combined stresses of abstraction on the aquifer and the borehole arrangement.

To meet the proposed water demand as indicated in Table 18.12, a typical well field will require approximately 20 production boreholes. To simulate the influence of the abstraction wells on the aquifer, a total of 20 abstraction boreholes were specified in the numerical model. The abstraction rate at each of the boreholes was specified at 630 m^3/day, thus totalling 12,500 m^3/day from the 20 boreholes.

Table 18.12 Typical Well Field Specification for the Area 5 Aquifer

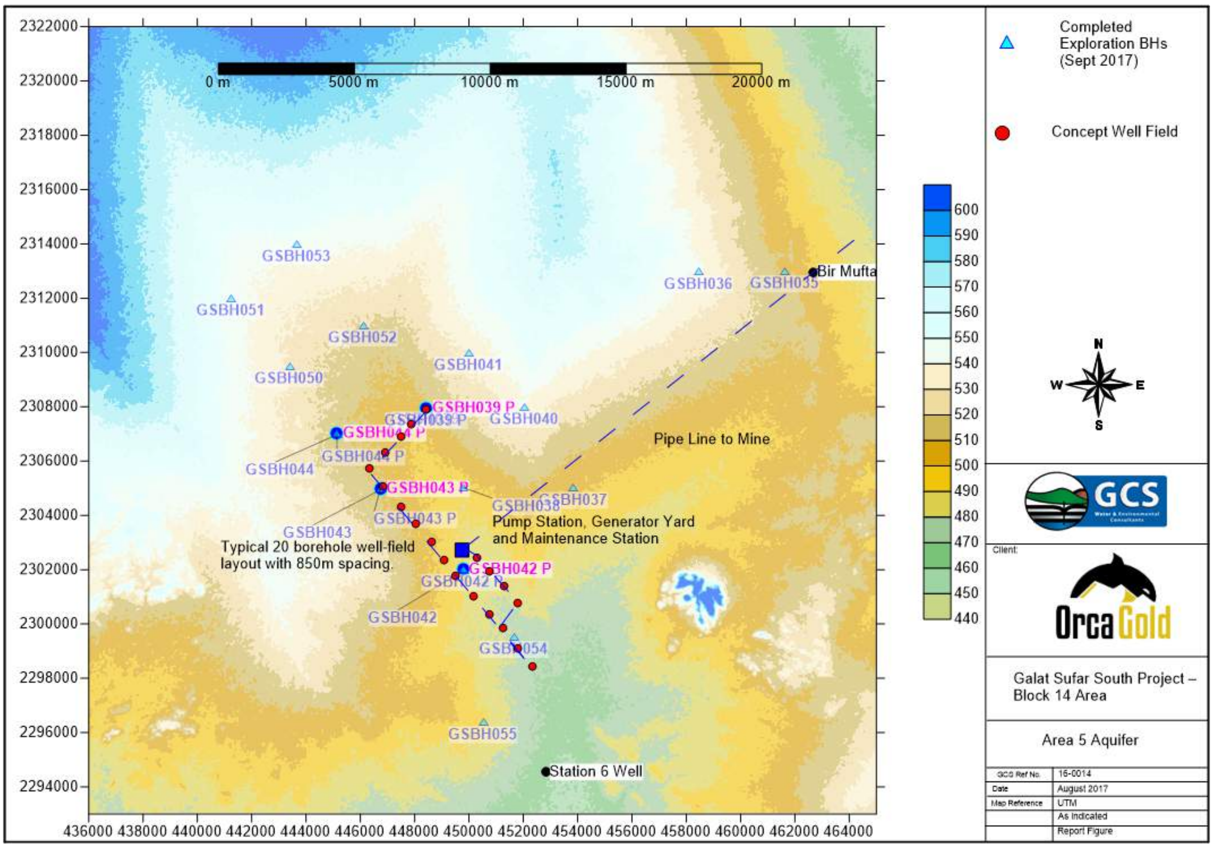
Required Demand for:	**m^3/day**	**LOM (years)**	**Total (m^3)**	**Throughput (Mtpa)**	
145 l/s	12,500	14	64,000,000	**6.0**	
Senario 2: Concept Well Field to Deliver (l/s)	**145**	**Yield / BH (l/s)**	**m^3/day**	**No. of Wells Required**	**BH Spacing**
Expected Average Yield		7.3	631	20	850

The borehole field was designed to intersect the coarse sand (as recognised by the field work) with 120 to 150 m deep boreholes. The provisional borehole design allows for large diameter mud rotary drilling and the installation of stainless steel wedge-wire screen.

The design parameters were:

- Depth of the boreholes will be in the order of 120 – 150 m.
- Drilling diameter will be in the order of 10 to 12-inches or 250 to 305 mm.
- It is recommended that a combination of thick-wall uPVC casing (8 inches or 200 mm) and 316 L stainless Steel V Wire Screens (6 inches or 16 5mm) be used. This will require a reducing connector to connect the 8-inches uPVC with the 6-inches stainless steel screens.
- The required spacing between the boreholes will be in the order of 850 m (Figure 18.11). The recommended spacing is based on the preliminary numerical model simulations and to minimise the overlap of drawdown between the wells.

Figure 18.11 Preliminary Well Field



No.	UTM- X	UTM -Y	Z
1	452339.4	2298449.8	473.3
2	451773.8	2299121.5	473.8
3	451243.5	2299863.9	478.8
4	450748.6	2300358.8	477.1
5	450147.6	2301030.5	488.3
6	449511.3	2301772.9	492.3
7	449087.0	2302373.8	492.5
8	448627.5	2303010.2	503.9
9	448026.5	2303681.8	504.3
10	447496.2	2304318.2	506.1

No.	UTM- X	UTM -Y	Z
11	446824.5	2305060.5	507.8
12	446329.6	2305732.2	510.3
13	446895.2	2306297.8	503.7
14	447496.2	2306898.8	502.6
15	447885.1	2307358.4	505.3
16	448415.4	2307888.7	511.0
17	450289.0	2302444.5	488.0
18	450748.6	2301949.6	482.1
19	451278.8	2301384.0	484.0
20	451809.1	2300783.0	483.6

18.1.9 Uncertainties and Assumptions

The following uncertainties exist:

- Boundaries - it is assumed that the regional sub-surface surrounding the demarcated or confirmed area will supply a certain degree of groundwater inflow.
- It is assumed that limited compartmentalisation exists within the confirmed area.

18.1.10 Conclusion

Drilling and Testing

Drilling of exploration boreholes within the Area 5 indicated medium to high and constant airlift yields and the occurrence of groundwater within an inter-layered basin sequence of course to medium grained sandstone.

Pump testing indicates that the aquifer behaves exceptionally well to high volumes of abstraction within three of the pump test boreholes and average abstraction from the rest. The aquifer parameters obtained are in favour for well field development.

Measured and Indicated Water Resource

A classification scheme is proposed to provide a level of confidence in groundwater resources for industrial use by reporting by semi quantitative methods. The methodology is prescribed in the paper presented by Green, et al, 2005.

The method allows for the following considerations:

- The Relative Water Resources (RWR) – using tested aquifer parameters like aquifer recharge, area, specific yield, aquifer thickness, hydraulic conductivity, etc.
- A call on aquifer knowledge and a rating.
- A rating for aquifer complexity.
- Final rating of the resource by combing the ratings of the three components listed above.

The overall category is subsequently classified as detailed in the following table:

CI	Category
>6	I – Measured
3 – 6	II – Indicated
1 – 3	III – Inferred
<1	IV - Unclassified

The following range of ratings were determined:

- The results for the 61 Mm^3 required volume suggest that future well field development is feasible and that the identified aquifer falls within the **"Measured"** category with a final CI of 10.99. It is highly likely to sustain the proposed 61 Mm^3 water demand over the proposed 13.5 years LOM period.
- For a demand in the order of 80 to 100 Mm^3, the classification can still be regarded as **"Measured"**, with a final CI of 6.0.
- For a demand between 100 and 140 Mm3, the classification is "Indicated" resource.

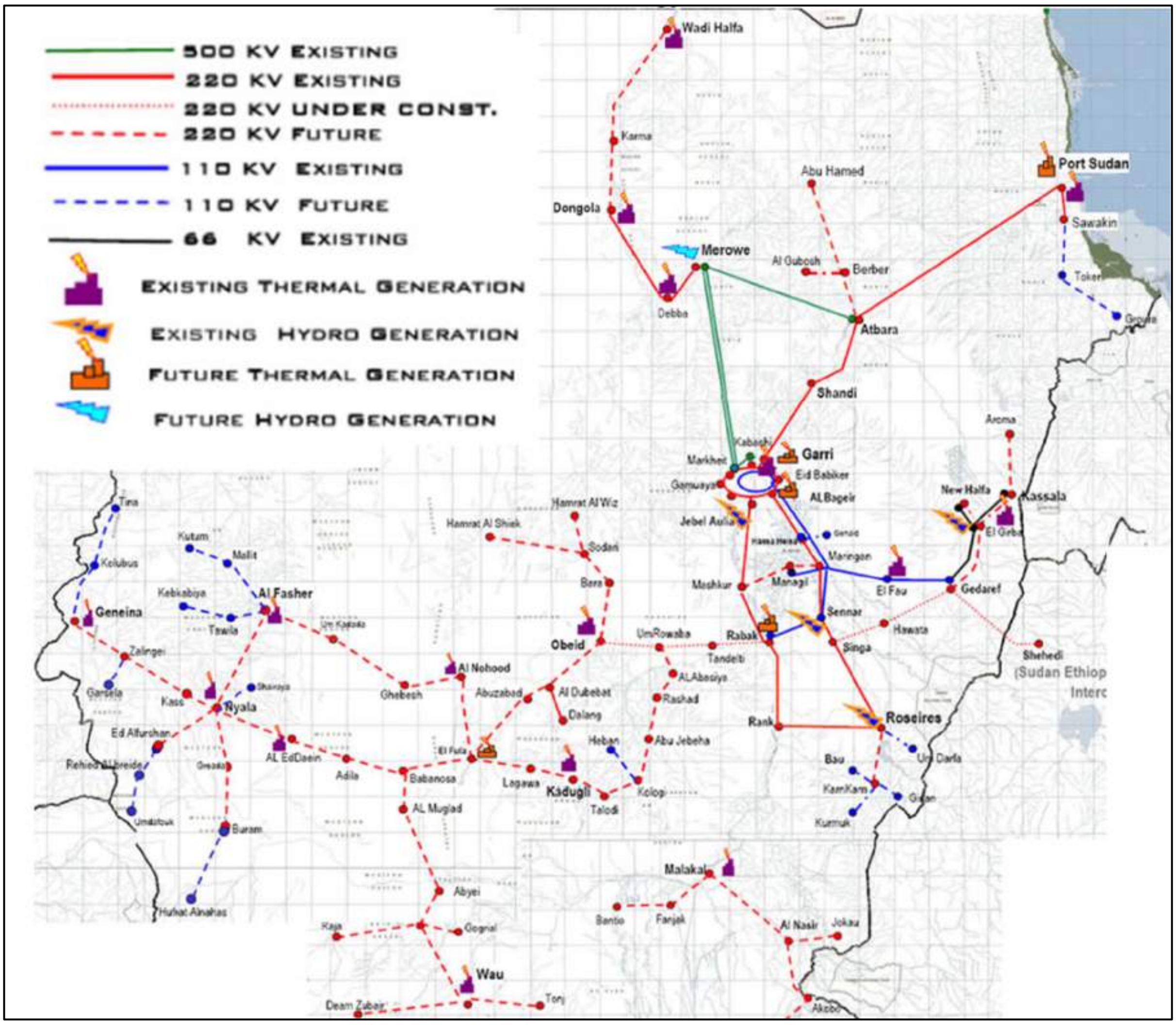
18.2 Electric Power

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

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

Electrical power in the Sudan is produced from both hydro and thermal generation stations, with the transmission network depicted in Figure 18.12. The total output of the generation is approximately 1,500 MW. The actual consumption of domestic and bulk users in Sudan is approximately 1,750 MW, so there is a shortage in the power production of approximately 250 MW. The shortfall in power generation is supplemented from the Ethiopian grid.

Figure 18.12 Sudanese Power Generation & Distribution Network



Due to the power supply shortfall , every factory or industrial complex connected to the power grid utilizes standby diesel generators in case of sudden power cuts, and this would be a necessary addition if the Block 14 project were to utilize a grid connection. It was decided that a grid connection was not viable and self generation would be implemented.

18.2.2 Project Power Supply Demand

The project plant and remote infrastructure power supply requirements have been evaluated based on the study mechanical equipment list, typical load and operating factors by equipment type, and an overall plant availability of 8,000 hr/yr. The summary loads are included in Table 18.13.

Table 18.13 Project Electrical Loads

Area	Connect Load (MW)	Maximum Demand (MW)	Average Demand (MW)	Annual Energy Consumption (GWh)
Process Plant	31.07	23.78	23.01	184.08
Site Infrastructure	1.01	0.47	0.46	3.68
Borefield Infrastructure	0.44	0.35	0.32	2.56
Total	**32.52**	**24.60**	**23.79**	**190.32**

18.2.3 Project Power Supply Options

Power will be generated by an on-site generation plant. In addition to Options 1 & 2, evaluated in the FS, Options 3 & 4 were evaluated in the RFS:

- Option 1: HFO fuelled generators.
- Option 2: LFO (Diesel) fuelled generators.
- Option 3: LNG gas generators.
- Option 4: Solar Hybrid integration with HFO/LFO/LNG options listed above.

The commercial execution model used for the study was a power station under a Build Own Operate Transfer (BOOT) arrangement. In the RFS stage, tenders were solicited from the market for the provision of onsite power generation facilities for Options 3 & 4. Based on the offers tendered the preferred self generation option will utilize multiple LNG fuelled gas engines to provide the total generation capacity. The load profile estimates a 24.6MW peak and 23.8MW average demand. This can be serviced using the twelve 2.5MW Rlls Royce MTU 61V4000 gas engines with Stamford LVS1840T 400V alternators to generate power at 11kV, 500Hz.

The LNG fuel will be transported to the site (in liquid form) from Port Sudan and stored in cryogenic vessels on-site. Waste heat recovered from the operation of the Power Station will be utilized to re-gas the LNG for utilization by the gas generators. All power generation equipment associated with this Power Station will be natural gas-fueled; therefore, no liquid hydrocarbon fuel storage or distribution systems will be required. The Power Plant will be provided with the following configuration operating in an N+2 arrangement:

- 10 + 2 Generators (N+2 Configuration) rated at 2,484 kWe each at site.
- 24,840 kWe + 4,968 kWe installed thermal Generation Capacity.
- 28,340 kWth + 5,668 kWth potentially recoverable thermal output.

Table 18.14 Gas Engine Unit Data

Engine/Alternator (Single Unit Data)	MTU 20V4000 L64FNER / Stamford P80HVSI804X
Gross Continuous Power Rating (COP), kWe @ 1.0 pf	2,535
Net COP (ISO) @ 1.0 pf	2,499
kWe (site) @ 0.8pf	2,484
Total Recoverable Thermal Output (kWth)	2,834
# Gens Running @ Avg. Demand	10
# Gens Reserve @ Avg. Demand	2
# Gens Running @ Peak Demand and Motor Start	10
# Gens Reserve @ Peak Demand and Motor Start	2
Total Gens Installed	12
Total Installed Base kWe (site) kWe	24,840
Total Installed (N+2) (site) kWe	29,808

The relatively low LNG gas delivery to site price and the fuel efficiency makes LNG option the most economical option compared to the HFO and LFO options. A power cost of US$0.1083/kWh was used in estimating operational costs based on the LNG price $9.70/MBTU delivered to site.

18.2.4 Major Equipment Details

Reciprocating Gas Generators

The proposed design incorporates housing the twelve MTU Gas 20V4000L64FNER 500 NOx generators in individual soundproofed 40-foot high-cube ISO Shipping Container for ease of set-up and maintenance. The plant will comprise a N+2 arrangement (two complete units on stand-by for maintenance and emergency response). The engines will be installed in ventilated and conditioned structures that will utilize electric fans to regulate the ambient temperature.

Waste heat recovery from the generator cooling circuit will be utilized to re-gasify the LNG fuel to reduce the amount of electrical heating required for fuel treatment and management. In addition to utilizing the waste heat for positive contribution, this will reduce the amount of heat energy in the cooling water, thus reducing the amount of electricity required to cool the generator water via the remote radiators.

Technical Details:

- MTU Gas 20V4000L64FNER 500 NOx Engines: These engines utilize natural gas at an input pressure between 185 and 250 mbar. T he normal operating speed for these engines when generating power at 50 Hz is 1,500 RPM.
- Stamford P80HVSI804X Alternators: The alternators selected for this project are designed to operate with 97.5 % efficiency (at pf=1). The alternators are sized to provide 11kV, 50Hz power output to the mine site and power plant auxiliary loads at a power factor >0.8 at 2,484kWe continuously.
- Each generator will have a ComAp InteliSys GAS with InteliVision 8 controller system onboard.

The generator equipment will be assembled in San Javier, Spain for Factory Acceptance Testing (FAT), partially disassembled for shipment, delivered to site, and re-assembled and prepared for Site Acceptance Testing (SAT) and Commissioning.

Control System

The power plant management system will be designed specifically for the Block 14 Project. The control system will be able to monitor and integrate all components in the plant to a common interface and control system. This highly automated system will monitor all aspects of the power plant operations and the operation's power demand to provide automated response. Another critical function will be automatic load shedding/adding of non-critical loads in case of an emergency situation. In addition, this system allows the operator to monitor all equipment and control points from the control room and enables remote access for monitoring by factory technicians and support staff via remote Internet access.

Generator Cooling & Waste Heat Recovery

The Generator cooling system will comprise a combination of Waste Heat Recovery (WHR) plate style heat exchangers and table style coolers using a dry cooling method. The system is designed with full cooling capacity in each remote radiator, without relying on waste heat removal. The system is designed with a 10% safety margin for cooling capacity at 48°C. Each generator has a closed loop cooling system with a single dedicated radiator.

The waste heat recovery system utilizes plate heat exchangers located adjacent to the radiators to capture heat and transfer to the WHR loop. This heated water is circulated through the LNG regasifiers and dissipate as much waste heat as possible, reducing the overall cooling load on the remote radiator systems, thus saving energy.

LNG Fuel System

The LNG fuel system will have redundancy and by-pass systems to ensure minimal downtime. The system consists of the following sub-components:

- LNG cryogenic bulk fuel storage.
- LNG cryogenic piping.
- LNG unloading station.
- Natural Gas Fuel distribution piping to and within the power plant.

Electrical Distribution

The power plant will consist of an 11kV paralleling and distribution switchgear system with capacity for future expansion and integration with future capacity and/or demand. The switchgear will be housed in a separate building that is air conditioned to protect the critical equipment.

MV Switchgear

All medium voltage switchgear components will conform to applicable IEC specifications. The primary generators will be connected to 11kV padmount switchgear units in groups of three. The main MV switchgear building will receive the feeders from these four 11kV padmount switchgear units. A total of two 1250A, 11kV feeders will be provided for the customer's use; one feeder for critical processes, and one for non-critical processes to facilitate the ability to load shed in emergency situations but maintain critical processes.

18.2.5 Site Power Infrastructure

The distribution of onsite power from the power station is designed around an 11 kV, 50 Hz reticulation network including both underground and overhead power lines. The plant loadings are broadly dived between two main switchboards, one for the front-end crushing / milling and the second supplying the leach / desorption back end of the process. Site infrastructure services are provided from a high voltage kiosk located adjacent to the plant administration facilities which reticulates via overhead powerlines to the mine services, tailings decant and camp facilities.

The high voltage reticulation of site infrastructure is further detailed in the electrical single line diagrams, Figure 18.13.

18.2.6 Remote Site Power Infrastructure

Remote site power infrastructure for the water supply borefield and transfer water pumping station will consist of two medium-speed diesel generators in a duty / standby arrangement at the pump station. Power is stepped up to an 11 kV, 50 Hz overhead powerline which reticulates out to the multiple boreholes. Small pole mounted step-down transformers and local bore pump controls panels complete the system and provide integrated control of the pumping system.

18.2.7 Plant Electrical Design

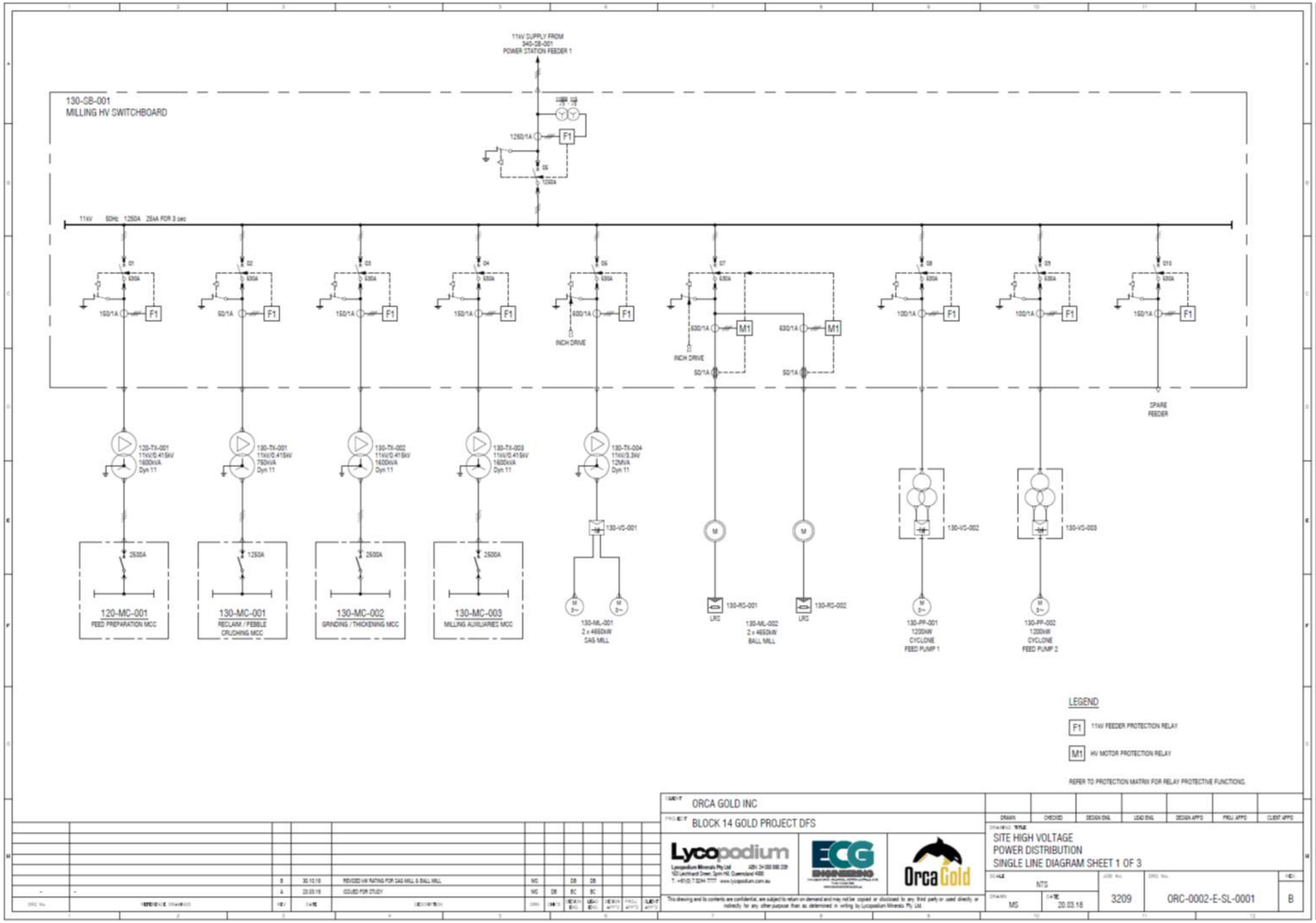
Electrical switchrooms will be based on a panelised room design, elevated to allow bottom entry cabling and ladder support structures. Air conditioned and sealed to operate in the plant operating conditions, these house the high voltage switchgear, motor control centres, variable speed drives and control system hardware. Segregation via internal walls will be provided between low and high voltage equipment.

Power transformers will be located adjacent to switchrooms to minimise cabling lengths and selected with an oil filled design suitable to the environmental conditions and life of mine. Oil containment and fire rated compound walls will provide containment protection to the switchroom and surrounding structures in the unlikely event of a failure.

All switchgear, low and high voltage, will be fully compliant to IEC standards and will be suitable to the site fault levels, providing full arc fault containment and increased personnel safety. Variable speed drives will be either mounted within the MCC or in dedicated floor mounted cabinets in the switchroom dependent on their size and rating.

The process will be provided with a moderate level of automation, suitable to the gold process to allow operation of the plant by a relatively small team. All regularly utilised (or large) valving will be actuated and instrumentation provided to perform high level, flow and analytical control of density, pH and free cyanide. The elution / desorption process will be extensively actuated and sequence automated to provide safe and efficient operation of the process. The control room, located adjacent to the CIL tanks, will provide the primary control system workstations to oversee the plant and house the servers for process control and historical trending.

Figure 18.13 HV Power Distribution – Single Line Diagram



18.3 Tailings Storage Facility

18.3.1 General

The design of the tailings storage facility (TSF) was undertaken to international standards to provide a facility to safely contain the tailings and reduce the potential effect thereof on the environment in the form of dusting, seepage or run-off from the tailings surface during operation and post closure. Provision was made for the effects of seismic events and probable maximum precipitation events during operation and post closure. To support the design and improve the safety of the facility, seepage analysis and stability analysis were done on the embankments. A water balance model was prepared to determine the water volumes retained in the TSF and the available recycle volumes to the plant. If built and operated in accordance with the principles and design concepts outlined in this document, this facility would contain the tailings generated from the project and the effects on the environment would be within acceptable limits as defined by international standards.

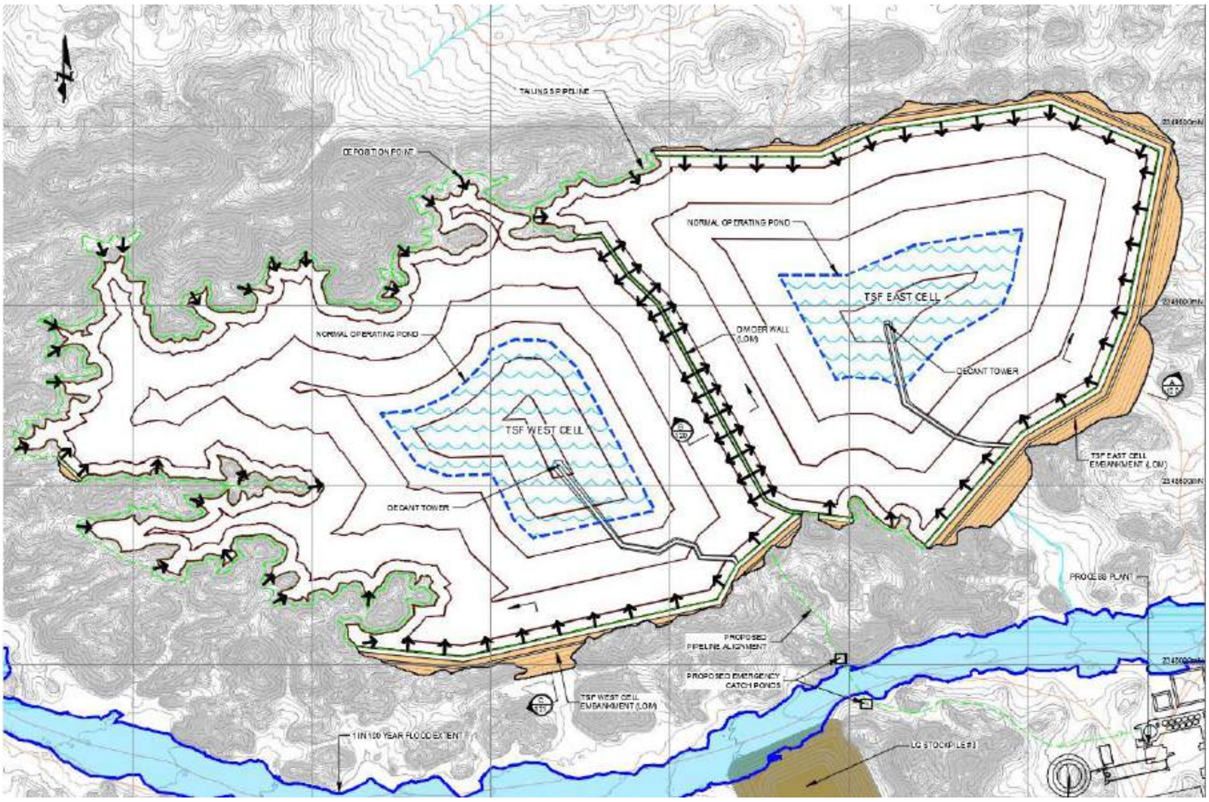
18.3.2 LOM Capacity

The TSF has been designed tailings capacity of approximately 81.5 Mt at a rate of approximately 6 Mtpa after the initial ramp up period.

18.3.3 Location and Embarkments

The proposed tailings storage facility is located within a valley located north of the unnamed Wadi running north of the process plant. The TSF has been designed as two cell paddock facility to allow alternate deposition and subsequent upstream raising. At the final stage of the construction, the total embankment length of the East Cell would be approximately 2.8 km (excluding the divider wall which is approximately 1.0 km long) with a maximum embankment height of approximately 38 m at the southeast corner. The total embankment length of the West Cell would be approximately 1.3 km (excluding the divider wall) with a maximum embankment height of approximately 31 m at the south extremity. The tailings beach surface at full capacity will cover an area of approximately 280 hectares. The general arrangement of the LOM embankment is shown on Figure 18.14.

Figure 18.14 TSF General Arrangement



18.3.4 Tailings Physical Characteristics

The tailings physical testing indicated that the rate of supernatant release for all samples was relatively rapid, with the majority of water released within hours. The expected supernatant release for the samples would be around 18% to 24% of the water in slurry, not accounting for rainfall and evaporation but incorporating the loss of water to re-saturate lower tailings layers. The expected underdrainage recovery could be high (≈8%) with a good underdrainage system due to the high water recovery from the base of the samples in the drained sedimentation tests.

The test indicated that both the tailings samples achieved rapid and high densities from settling alone, particularly the transition and fresh samples. There was appreciable additional improvement due to air drying. Based on the hot, dry climate and large tailings beach area, significant air-drying is expected. A typical achievable density range of between 1.45 to 1.55 t/m^3 is expected.

18.3.5 Tailings Geochemical Characteristics

Based on the tailings supernatant being slightly saline and highly elevated in cyanide with a range of elevated environmentally significant metals and metalloids, a seepage control system comprising a partial basin liner, above-liner underdrainage system and a sub-liner drainage system have been included within the TSF basin across the average supernatant pond extents and over all areas underlain by aeolian and colluvial sand deposits. The proposed partial basal liner, above liner drainage and sub-liner drainage systems are shown on Figures 18.15 to 18.17.

Figure 18.15 Partial Basal Liner

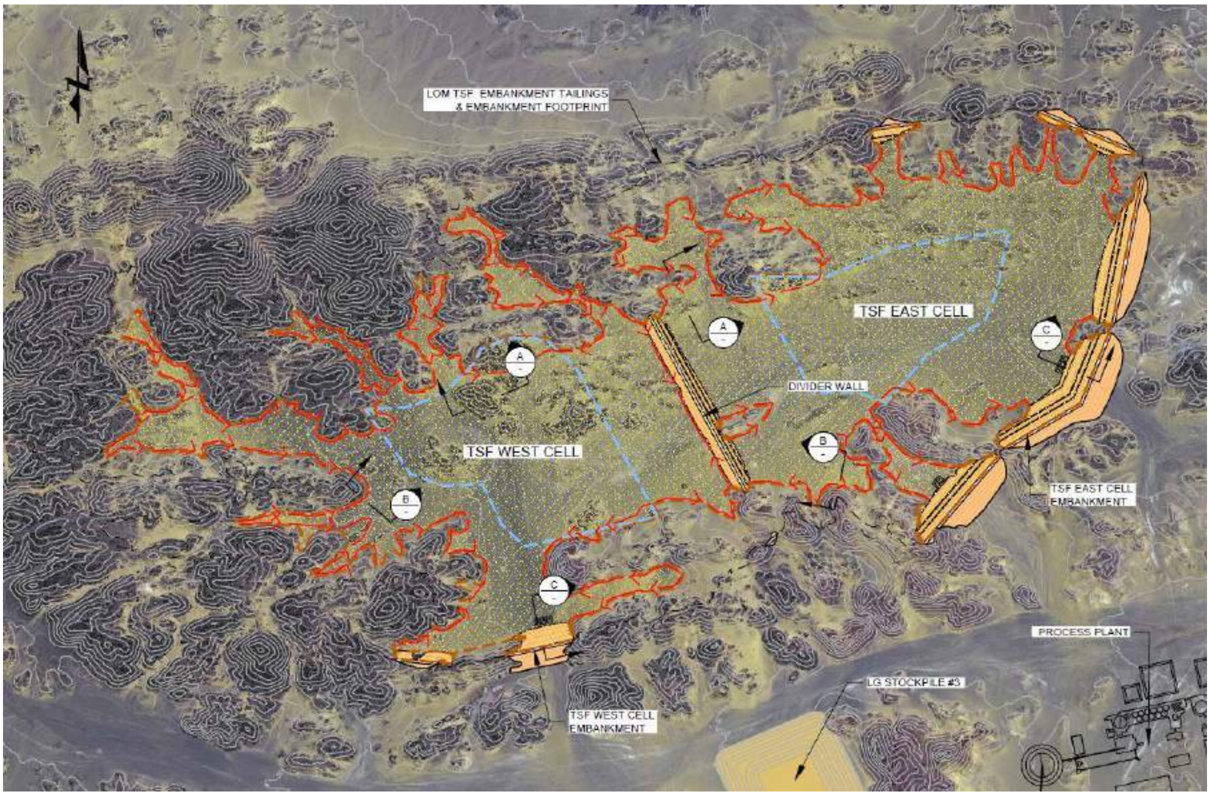


Figure 18.16 Above Liner Drainage System

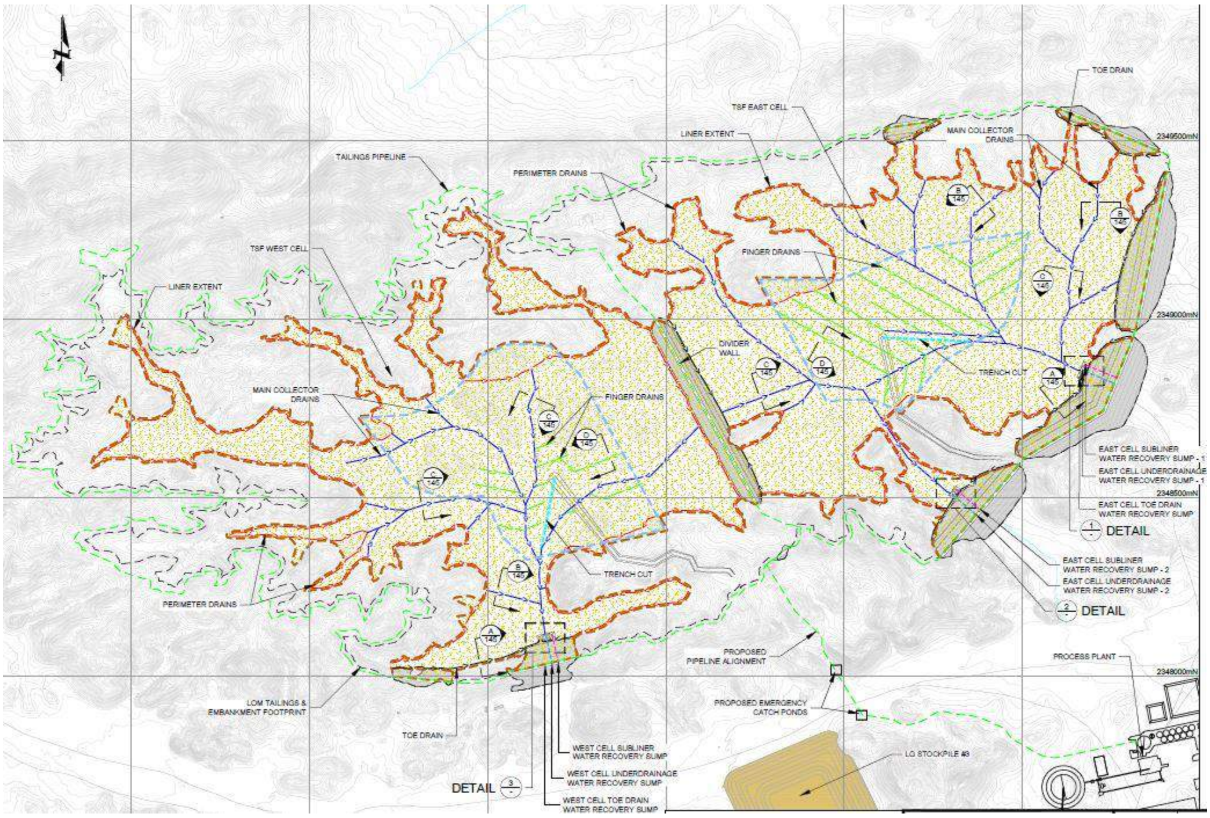
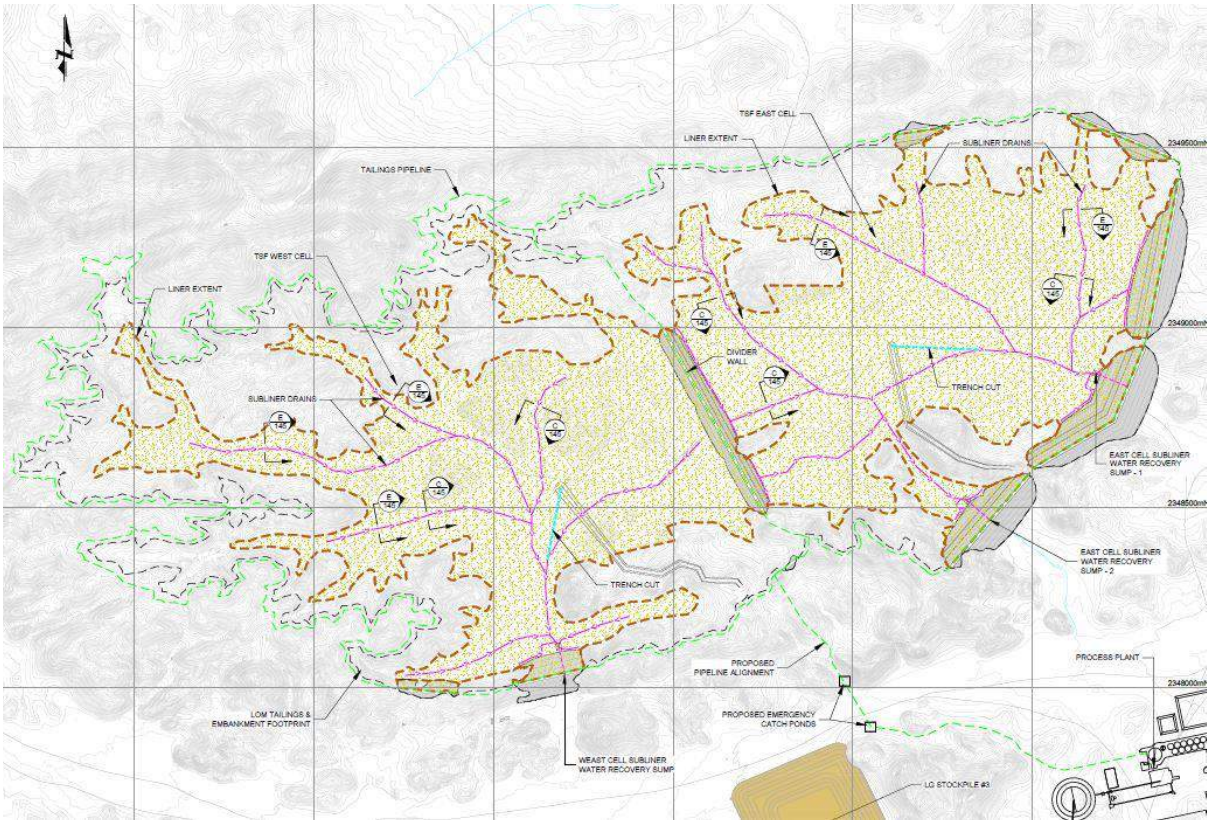


Figure 18.17 Sub-liner Drainage System



18.3.6 Geotechnical Investigation

A geotechnical investigation was conducted at the TSF site which included geotechnical drilling, test pit excavation, in-situ testing and laboratory testing of the foundation and construction materials.

The boreholes indicated a surface layer of aeolian sand with occasional gravel lenses extending to depths of between 0.1 m and 2.6 m, with dynamic cone penetrometer tests conducted through the sand at five borehole locations extending to depths of between 1.1 and 2.6 m before refusing. The sand was underlain by variably weathered andesite and schist, with the depth to medium strength rock varying from 0.5 m to 9.4 m. Falling head permeability tests conducted in the boreholes at depths of between 4.8 m and 15 m and test intervals ranging from 1 m to 3 m in length indicated permeabilities in the range of 3.2×10^{-8} to 3.8×10^{-7} m/s at an average of 1.7×10^{-7} m/s.

Test pits were also excavated across the basin areas, embankment footprint and deposition extents. These were typically located in areas of higher ground and the majority encountered weathered rock outcrops variably covered with a thin layer of aeolian sand. The rock outcrops had a fabric characteristic of schist and was primarily distinctly weathered. In lower lying areas the depth of sand increased up to a maximum of 3.7 m overlying weathered rock.

18.3.7 Dam Failure and Environmental Spill Consequence Category

A dam consequence category assessment has been conducted for the TSF based on the ANCOLD Guidelines to assign a dam failure consequence category and an environmental spill consequence category.

A dam breach assessment undertaken for the TSF indicated plant site could potentially be impacted by tailings run-out with tailings travelling downstream along the main channel for approximately 2.7 km. Hence the dam failure consequence category for the TSF was classified "High A" predominantly based on the adverse impact on the plant should a dam failure event occur.

Environmental Spill Consequence Category is defined as "Very Low" as no permanent water sources which are used for drinking water or agricultural purposes in the potential impact area.

Taking into consideration the extremely dry climate at site the 0.5 m operational freeboard was found to be sufficient to provide the wet season / extreme store capacity recommended by ANCOLD for a High A facility when defining the embankment crest RLs. The estimated storm water storage capacity for each cell was found to be 10 times the Probably Maximum Precipitation inflows. With this excess storm water storage capacity, an operational spillway for the TSF is not considered necessary provided the water management procedures adopted in the design are fully adhered to and the volume of water stored on the TSF is minimised at all times.

18.3.8 Embankment Configurations

The TSF cells will have a starter embankment, one or more downstream raises and subsequent upstream raises up to LOM with an annual rate of rise less than 1.5 m when upstream raising is conducted. The upstream face of the starter embankment and the downstream raises will be lined with 1.5 mm HDPE liner to minimise seepage occurring though the lower portion of the embankment.

The starter embankments have been designed as a multi-zoned dams, with a 6m wide inclined upstream zone constructed of aeolion Sand (Zone B) supported by the downstream structural zone constructed of Wadi Gravel (Zone C1) sourced from local borrows. The downstream raises will also have the 6m wide inclined Zone B, however the structural zone will be constructed of waste rock (Zone C2) sourced from the pit. It is anticipated that overhauling the waste rock for the downstream raises would be more cost effective than developing local borrow pits closer to the embankments. The subsequent upstream raises will be constructed with re-won and compacted tailings with a 8 m wide outer waste rock fill cover.

For ease of liner deployment, the upstream batter slope of the starter embankment has been set at 1V:3H. The downstream batter slope of the starter, downstream raises and the upstream raises has been defined at 1V:2.5H with a 6 m bench constructed on top of the downstream raises.

A divider wall would be required to separate the two cells. The initial raises of the divider wall will be constructed of aeolian sand and local wadi gravel whilst the raises corresponding to the embankment upstream raises will be constructed by centreline technique utilising tailings. Figures 18.18 to 18.20 show the typical details of the TSF embankments and divider wall.

Figure 18.18 Typical Embankment (East Cell)

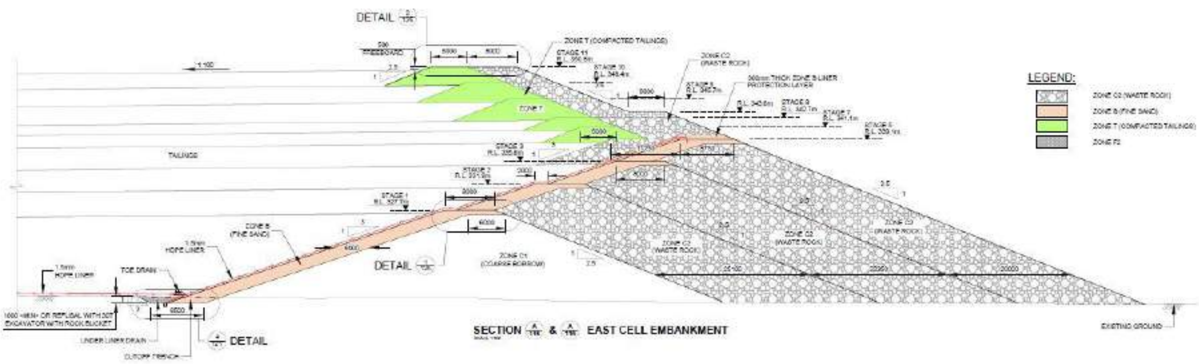


Figure 18.19 Typical Divider Wall

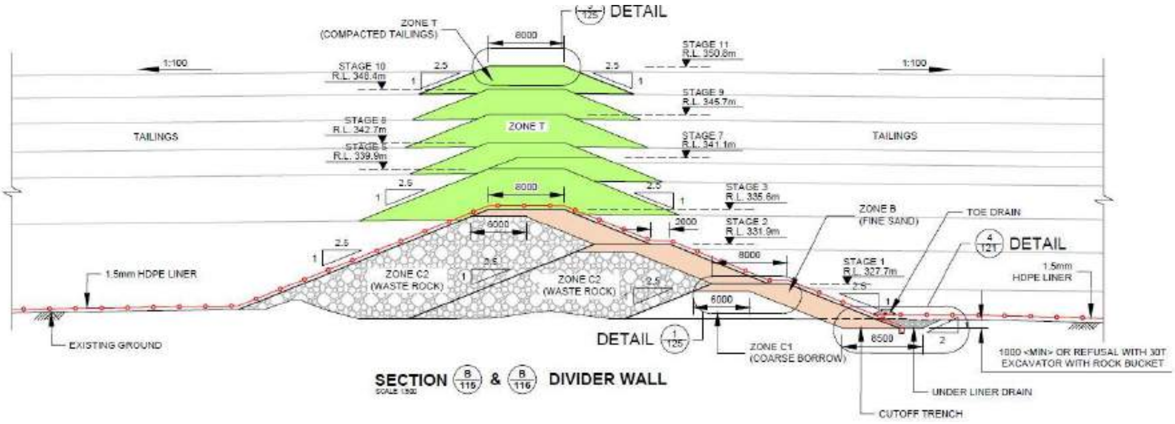
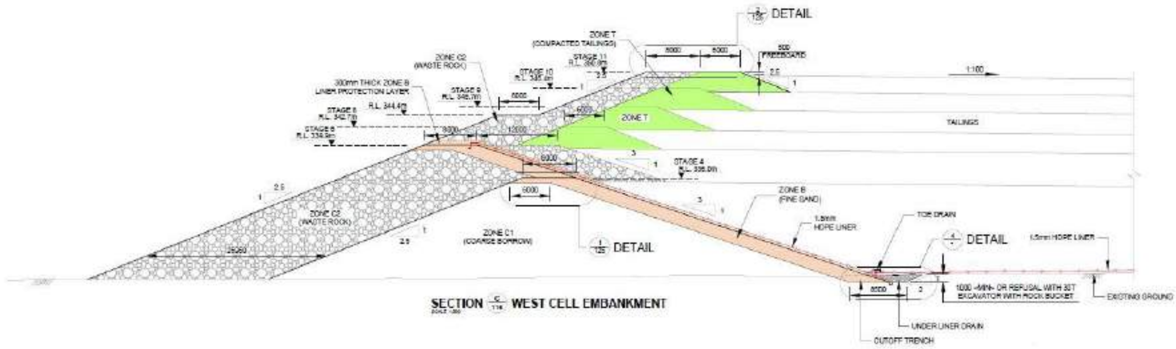


Figure 18.20 Typical Embankment (West Cell)



18.3.9 Embankment Staging

Construction of the embankment will be staged over the LOM with the aim of deferring the capital and sustaining capital costs involved with the embankment construction. The water balance model indicated that supernatant pond under average climatic conditions will be located below the tailings level at the embankment during the life of mine. As such, only 0.5 operational freeboard was considered when defining the embankment crest RLs. The embankment crest RLs were defined so as to maintain the same crest RLs for the upstream raises of the west and east cells and the tailings annual rate of rise less than 1.5 m towards the end of LOM. In order to maintain the same crest elevations for the upstream raises, the annual tailings discharge rates into each cell have been adjusted.

18.3.10 Geotechnical Analyses

Geotechnical analyses of the facility have been conducted to assess the geotechnical stability of the facility and the seepage rates from the facility. The stability assessment indicates that the embankment has a factor of safety in excess of 1.5 under both drained and undrained conditions, with the post seismic stability of the facility also over 1.0 assuming full liquefaction of the stored tailings where water content of the tailings mass equal to or greater than 80% of the liquid limit.

High level deformation assessment indicated that the maximum embankment crest settlement is less than the design freeboard of 0.5 m. Therefore, the MDE event is unlikely to cause severe damage or failure of the TSF embankment.

18.3.11 Tailings Deposition

It was assumed that tailings will be discharged into the facility by sub-aerial methods of deposition utilising banks of spigots located at regular intervals. Sub-aerial deposition allows for the maximum amount of water removal from the facility by the formation of a large beach for drying and draining. Together with keeping the pond size to a minimum, sub-aerial deposition will increase the settled density of the tailings, and hence maximise the storage potential and efficiency of the facility.

Tailings slurry will be pumped to the tailings storage facility via a welded high density polyethylene (HDPE) pipe. This pipe will be located in a HDPE lined bunded corridor designed to contain the spills should the pipeline fail. Where the pipeline crosses the Wadi between the process plant and the TSF, the pipeline will be buried to a depth of at least 1.5 m with a double pipe containment system (outer pipe which can drain to two lined sumps and normal inner tailings pipeline).

The tailings will be discharged into each cell from the embankments, divider wall, perimeter ridge lines and valleys with the aim of pushing the supernatant pond towards the centre of each cell where the decant tower is located. Decant structures will be constructed at the centre of each cell comprising a recovery tower and access causeway to allow supernatant water to be recovered from the facility and recycled to the process plant.

Deposition of the tailings will be carried out on a cyclic basis with the tailings being deposited over one area of the storage until the required layer thickness has been built up. Deposition will then be moved to an adjacent part of the storage to allow the deposited layer to dry and consolidate.

18.3.12 Monitoring and Instrumentation

A comprehensive monitoring programme has been proposed for the TSF to detect a number of potential problems which may arise during operation. The monitoring will include survey pins to check embankment movement in each stage and vibrating wire piezometers to monitor the saturation levels and pore pressure (one of the prime contributing factors to instability of the embankment slopes) within the embankment, tailings close to the embankment and the foundations. Further a number of monitoring bores will also be installed downstream of the TSF for groundwater level and water quality monitoring.

18.3.13 Water Balance Modelling

A water balance model was prepared to estimate the demand for raw water on site, considering the process water demand, losses and gains from the tailings storage facility, dust suppression and potable water demands.

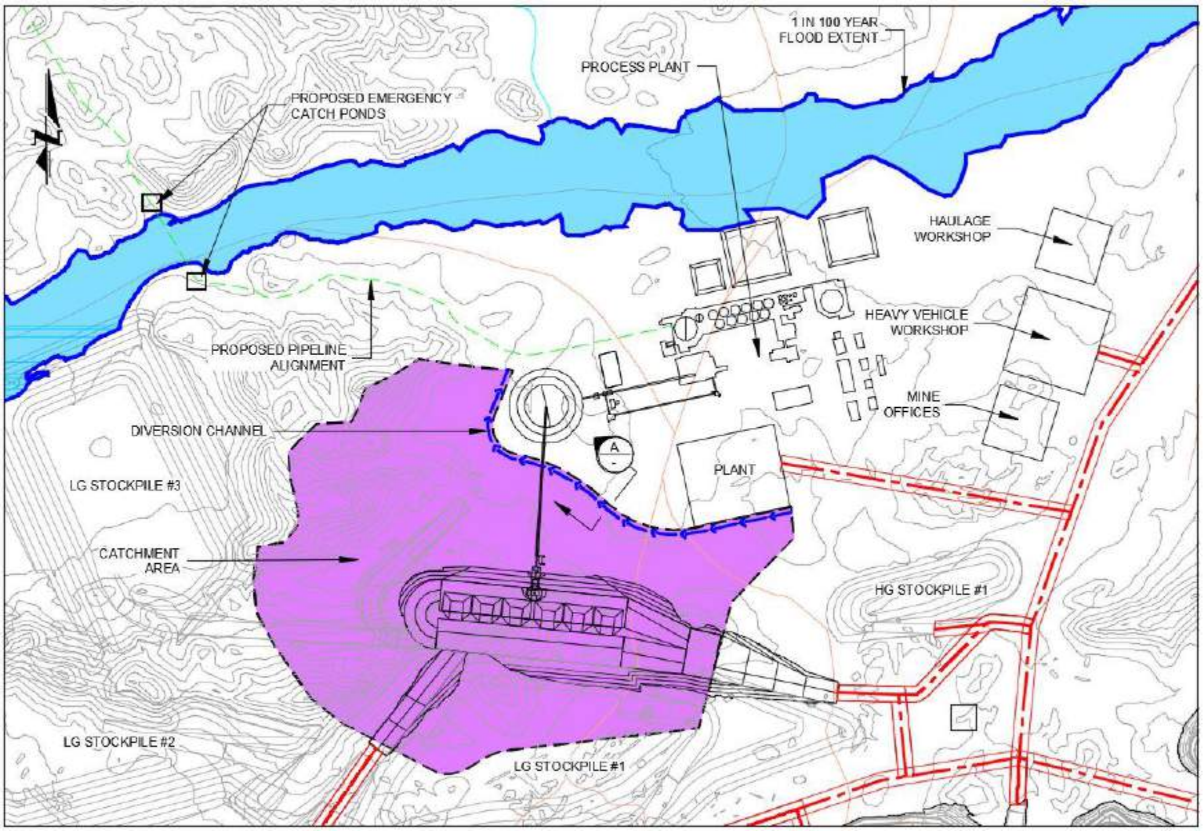
The total annual project shortfall will vary between 2.6 Mm^3 and 3.9 Mm^3 with the maximum shortfalls were predicted in Operation Year 2 and gradually reduce towards the end of the project life. The supernatant pond will be at its minimum level throughout operations; as such the continuous small pond relative to the facility will result in adequate exposed drying beach to achieve higher densities over the life of the project. Water recycled from TSFs will generally be between 3% and 12% of total water in slurry.

18.4 Plant Site Surface Water Management

At the proposed plant site location, the Wadi to the North of the Plant will have an upstream catchment of approximately 2,600 ha which may lead to significant flooding of the plant. Therefore, a detailed flood assessment was undertaken to define the extent of flooding at the plant site location and to define the process plant construction pad elevation which is located above the 1 in 100 ARI 24 hr flood. Localised bunding around some of the process pond would be required to reduce the risk of flooding of these ponds.

The process plant site and majority of the related infrastructure have been located within a valley located north of the mine pits. Hence, a clean water diversion channel has been designed along the southern extremity of the plant site to prevent flooding of the during extreme rainfall events. Figure 18.21 shows the proposed clean water diversion channel and the one in 100 year ARI flood extent in relation to the process ponds.

Figure 18.21 Clean Water Diversion Channel and Bunding Proposed Around the Process Plant



18.5 Fuel Supply and Storage

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

Allowance has been made for two 1,000 m^3 above ground vertical storage tanks for the bulk site diesel storage and 6 x LNG 120cbm storage vessels for the LNG storage.

Fuel storage requirements will be reviewed during the next phase as part of reviewing the BOOT power station tenders. There is deemed to be adequate fuel supply available in Sudan and local transport contractors in the area in order to supply the Project with the required fuel.

18.6 General Infrastructure

18.6.1 Plant Buildings

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

- Plant warehouse and office.
- Reagent storage facility.
- Plant workshop and office.
- Main office building.
- Plant office and control room.
- Clinic and emergency response building.
- Plant security gatehouse.
- Changerooms.

18.6.2 Mine Buildings

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

- Mining office for Orca Personnel.
- Washrooms and ablution block.
- Light vehicle workshop (shared with processing and other departments).

The contractor will be responsible for supplying and constructing the following items:

- Heavy vehicle workshop and GSS and Wadi Doum.
- Heavy vehicle wash pad.
- Parts warehouse.
- Tyre bay.
- Explosives Magazine.
- Go-line and lunch/toilet facilities at both GSS and Wadi Doum.
- Mining/contractor office.

These items have been included in the mobilisation and site establishment cost for the mining contract.

18.6.3 Camp / Accommodation Village

Quotations were solicited from several facility service providers for the design, construction, operation and management of a construction and operations camp with all necessary facilities, on a Build Own Operate Transfer contract basis. The quotations were based on a number of accommodation units to cover the estimated peak number of construction workers, operations employees and visitors anticipated on site at peak manning. Allowance for a camp sewage treatment plant has been included in the capital estimate.

The mining contractor will be responsible for the construction, operation and maintenance of a satellite camp at Wadi Doum. Water will be supplied from the main water point at GSS by tanker and power will be generated by stand-alone diesel or IFO generators.

18.6.4 Communications

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

18.6.5 Security

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

18.6.6 Health and Safety

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

18.6.7 Laboratory

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

18.6.8 Sewage

Site sewage will be managed on site via two vendor supplied sewage treatment plants, (one at the accommodation camp and one at the plant site) with cleaned effluent disposed of via spray fields or soak wells. Treatment plant sludge will be suitable for direct landfill burial.

18.6.9 Roads and Transport

The main supply route to the site will be either via Khartoum or Port Sudan. Roads are tarred between these cities and Abu Hamad (the closest main town to the project area). Access to the Project area from Abu Hamad is via a site access road, comprising a track through the desert 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.

18.6.10 Airstrip

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

19. MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No formal market studies have been undertaken.

A gold price of US$1,350 /oz has been used for the economic analysis, which is 95% of the 3 year trailing average price of $1,410.

Block 14 Project will produce gold doré which is readily marketable on an 'ex-works' or 'delivered' basis to a number of refineries in Europe and Africa. There are no indications of the presence of penalty elements that may impact the price or render the product unsalable.

Payment terms are widely available in the public domain and vary little from refinery to refinery.

19.2 Contracts

The Company has entered into a Memorandum of Understanding (MOU) with Inglett & Stubbs International to develop a Power Station. The MOU defines the steps for finalizing the technical and commercial requirements and development of the formal agreements between the parties.

Refining contracts are typically put in place with well recognized international refinereies and sales are made based on spot gold prices. These contracts typically include fees for transportation of the product from the site, insurance, assaying, refining and an allowance for metal losses during refining. The ability to get a contract in place for the sale of doré prior to start of production is not seen as a risk to the Project.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

Mineesia Ltd, a UK based consultancy, was engaged in 2014 to undertake an environmental impact assessment for the Block 14 Project, including planning and collection of environmental and social baseline data to inform the assessment. This has led to the preparation of an Environmental Impact Assessment (EIA) to accompany the Feasibility Study for the Project. The EIA is based on site visits conducted in 2014, 2017 and 2018 and has been developed from the proposed plan to mine and process material at Gulat Safur South with additional material being trucked in from a satellite mining operation at Wadi Doum.

20.2 Sudan Legal Setting

The EIA for the Project has been undertaken in accordance with the requirements of the relevant Sudanese mining and environmental legislation, as well as international guidelines for best practice, where feasible. A summary of the key legislation considered is provided in Table 20.1.

Table 20.1 Key Sudanese Legislation

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

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

The EIA for the Project was approved in February 2019, under certificate 3/2019. This approval states that the EIA is in full conformity with the Environmental Act 2001, and that the Higher Council of Environment and Natural Resources granted MSMCL approval to construction, commission and maintain.

20.3 International Requirements and Guideline

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

The primary requirement for any International Project is to comply with local and national regulations of the host country. These regulations are often supplemented by standards and guidelines from international financial institutions, particularly with regard to environmental and social components of the Project. The IFC Sustainability Framework, as revised in 2012, with associated Performance Standards on Environmental and Social Sustainability, provide the basis for most EIA assessments. Additional guidance is provided by the Equator Principles, which provide an approach to determine, assess and manage environmental and social risk in project financing.

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

20.4 Project Permitting

Sudan has developed an integrated permitting infrastructure for formal mining operations. The development of the Project hasreceived environmental approval for the construction, operation and maintenance of the Project. The EIA includes an environmental management programme and mitigation measures for potential impacts. Based on the provisions of the various legal requirements and sectoral laws as well as policies of different departments, the impacts of the proposed project were assessed and mitigation measures recommended where appropriate.

The institutions at National Level responsible for the implementation and monitoring compliance to both national and international agreements include:

- The Higher Council for Environment and Natural Resources.
- Ministry of Environment, Natural Resources and Physical Development.
- Federal Ministries of Health, Industry and Agriculture.
- Ministries and Councils at State Level.

The environmental acts and laws provide standards to be applied in assessing probable environmental impacts of the Project. It is important to note here that State Organs and Local laws deal with issues at State or Local levels, while the Federal Acts are more concerned with general directives and set limits and standards to certain environmental concerns without going into details of problems of local nature.

20.5 Project Design and EIA

A revised Terms of Reference (TOR) for the EIA was submitted to the Sudan Mineral Resources Company (SMRC) in November 2017. This TOR was updated from the Revised PEA stage to reflect the current Project. It identifies potential key environmentally sensitive areas and was based on the Project design below:

- Open pit mines – to be developed sequentially using standard open cast mining techniques.
- Processing Plant – material treated with Carbon-in-pulp (CIP) extraction methods, with a throughput of 6.0 Mtpa.
- Waste facilities – comprising waste rock dumps for stockpiling overburden and waste material from the open pit, a tailings storage facility for storage of processed tailings from the CIP processing plant and a domestic waste facility for non-mineral wastes.

- Water supply – including the Area 5 borefield, water pipeline, a reverse osmosis water treatment plant to treat raw water. Initial water requirements are expected to be up to 4.6 Mm^3/a (12,500 m^3/day).
- Power plant – comprising reciprocationg Gas Generators fueled by LNG.
- Associated infrastructure – including haul roads, offices, workshops, ablutions and sewage treatment systems, explosives storage and a laboratory.
- Accommodation camp – to house employees.
- Airstrip.

20.6 Baseline Environmental Setting

20.6.1 Physiography

See Section 5.5.

20.6.2 Climate

As discussed in Section 5.2, the climate is arid with a hot season from June to September, when the maximum daytime temperatures range from 45°C to 49°C and minimum daytime temperatures of 23°C to 35°C. The coolest months of the year are January and February with daytime high temperatures of 28°C and cool nights of approximately 5°C. The project area is hot and dry, with maximum summer temperatures of 52°C. Dry northerly winds prevail, which are strong and hot during summer and cooler in winter resulting in frequent sandstorms (Countryside Studies, 2013). The highest wind speeds are from the NNW direction. The winds result in dusty conditions and there are no days which are completely windless.

20.6.3 Air Quality

Due to the remoteness of the Project area and absence of major industrial activities in the region, air quality at the site is not considered to be affected by any other industries. There are gaseous emissions from vehicles and mining equipment operated by artisanal miners in the locality. These artisanal works are extensive and cover large areas, both in terms of surface (colluvial) mining and shallow hard rock pits. Their activities are transient; moving from one prospect to the next and the land surface is scoured and pockmarked by these workings. There is extensive use of mobile equipment, such as excavators, tractors and trucks together with individuals using hand held metal detectors. These sources of emissions are considered limited and widely dispersed. This dispersion combined with the lack of major industry in the vicinity of the Project region suggests that ambient air quality can be regarded as free from anthropogenic impact. As such, no further data has been collected regarding gaseous constituents.

As a result of the desert landscape around the Project site, the main impact on air quality in the region is naturally occurring suspended particulates (windblown dust) from the desert soils. The relatively flat topography and wind patterns create frequent sandstorms (sometimes termed dust storms) which result in extreme concentrations that create low visibility across the Project area.

In 2014, the Project established a dust monitoring programme using passive samplers installed at the main camp, the drill camp and at the GSS prospect, to provide adequate coverage of the Project area. The drill camp dust sampler was moved to the Wadi Doum prospect in April 2016 to monitor conditions in that area. The results provide a good baseline for dust impacts naturally occurring in the area.

20.6.4 Noise and Vibration

The Project is located in a remote region with no permanent settlements in the vicinity. Potential receptors (excluding persons working on mine project) of noise and vibration in any future mining development are limited to artisanal and illegal small-scale miners in the area, which are themselves already existing sources of noise and vibration. Given the small size, remoteness and impermanent nature of those operations, no noise assessment of them has been undertaken. It is not known if these operations have measures to control and protect their workers from noise and vibration impacts.

20.6.5 Surface Water

As a result of the low rainfall and high rate of evaporation, there are no permanent watercourses on-site or within the local vicinity. However, storms can produce ephemeral floods in wadis and across open ground. Water quality data for wadis are not available due to the intermittent nature of flow. When the wadis do flow, erosion of the soils results in high levels of suspended solids. The nearest open water of significance is the Nile River, near Abu Hamad, approximately 180 km south of the Project site.

20.6.6 Groundwater

As described in Section 18, the Project area is within an arid zone, though groundwater does occur in a well at Talat Abda. The project area is close to the Nubian Sandstone formation which forms part of a regional aquifer system covering Sudan, Egypt, Libya and Chad. Drilling activities have provided evidence of localised groundwater.

Water quality results indicate that groundwater through the area is not continuous, which is supported by evidence that some monitoring wells in the area are dry. Analytical results show that the groundwater samples exceed aesthetic guidelines for a number of parameters. With the exception of Talat Abda, the water is not considered suitable for livestock or irrigation and the presence of high levels of sulphate, chloride and sodium further support this assessment. The high levels of manganese present issues with water use for domestic (non-drinking) purposes. Scaling and staining are likely to be prevalent if this water is used without pre-treatment. Under the criteria available for irrigation use, water from these wells would be classified as 'unsuitable' for irrigation purposes, particularly for sensitive plants such as food crops.

Analysis of water quality from Area 5 indicates that the water is of good quality. Results showed that water mostly conformed to World Health Organisation drinking water standards, although its taste may be poor. Should the water be used for irrigation, it would be classified as good to permissible.

20.6.7 Soils, Pre- and Existing Land Use

The Harmonised World Soil Database (FAO et al, 2009) indicates a mosaic of soil groups within the Project region, namely:

- Arensol - a sandy soil featuring very weak or no soil development.
- Fluvisol - a young soil located in alluvial deposits.
- Leptosol - a very shallow soil over hard rock or unconsolidated very gravelly material.
- Regosol - unconsolidated material derived from freshly deposited alluvium or sands with very limited soil development.
- Luvisol - a soil with subsurface accumulation of high activity clays and high base saturation.

The nature of the soils in the Project area and surrounding region in combination with arid conditions and scarce water resources means that potential soil productivity is of low to nil capacity. Nomadism in the Project area and its surrounding region is considered very limited to none due to the absence of oases, a recognised characteristic of the Nubian Desert.

Mineral wealth has been the only inherent land capacity in the Nile River basin and some adjacent lands throughout history. The development of gold mining over time, from Predynastic (ca. 3000 BC) until the end of Arab gold production times (about 1350 AD), including the Pharaonic period in the remote deserts of Egypt, is well known. Around 250 gold production sites were mined during different periods in the remote region of the Eastern Desert of Egypt, which is now the upper northern and north-eastern Sudan. Gold mining (and to a lesser extent copper and iron) in the above regions and around the greater Project area has a long and important history in the evolvement of various civilisations and cultures in the Nile basin region.

The existing land use is restricted to mining activities, predominantly artisanal gold mining. This artisanal mining primarily comprises predominantly non licenced operations interspersed with small licenced operations. Licencing is being encouraged by the national Government through the issuance of exploration permits to registered companies.

20.6.8 Flora and Fauna

The Project area is situated within the desert ecoregion, which is one of sixteen terrestrial regions within the Nile River Basin. The desert ecoregion contains few visible plants, with most areas covered by bare soil or rock. Some shrubs of *Acacia tortilis* can be found in the true desert, but most plants are only visible in the rainy season and survive the dry season as seeds (as annual plants).

There is considerable variation in vegetation described north of the Project, particularly along wadis. Scanty and stunted vegetation, dominated by *Acacia*, *Tamarix* and *Calotropis* species, has been found along some wadis. Where water is available from the crystalline basement, doum palms (*Hyphaene thebaica*) and sparse thorny shrubs exist. Wadis have been affected by artisanal miners, who have left pits and mounds of earth within the wadis, thereby modifying streamflow (albeit it ephemeral) and vegetation communities within the wadis. None of the plant communities or single plants identified are rare or unusual in a national or international context. None of the species recorded are included in the International Union for the Conservation of Nature's Red List.

The desert habitats in the Project area provide a harsh environment with extreme temperatures, little water and little food. They are capable of supporting only a limited suite of specialist species, although the habitats they inhabit are common and widespread. Wildlife presence is being assessed on an ongoing basis through the use of passive infra-red (PIR) cameras, as well as opportunistic photographs of other wildlife. A database of the wildlife present in the area has been developed, based on lists of fauna that potentially occur there. PIR cameras have been installed at the main camp burn pit and at GSS and these are downloaded monthly.

To date, Golden Jackals (or African golden wolf, *Canis anthus*, as they are now known) have been captured by the PIR cameras (Figure 20.1). Other species recorded by Project personnel in the area, but not captured on any cameras, include the Dorcas gazelle (*Gazella dorcas*), wildcats (the Hausa wildcat, *Felis silvestris hausa* and the East African wildcat, *Felis silvestris rubida*) and Ruppell's foxes (*Vulpes rueppelli*i). No protected species, such as gerbils, have been identified in the exploration area. Several snakes (unidentified) and scorpions have been captured and killed in the camp. There are no known endangered species present in the Project area.

Bird migration movements to and from Central Europe and Asia pass through the Nile valley (approximately 200 km to the west) and along the west coast of the Red Sea (approximately 280 km to the east). The Project area is not within the paths of designated international flyways. Birdlife International does not designate any Important Bird and Biodiversity Areas within the Nubian Desert or adjacent areas. No endangered bird species are listed as occurring within the Nubian Desert or immediately adjacent regions. During operations, the project will observe if site water bodies attract migrating or regional birds, other than those already present. Some migratory birds have been observed on site, such as Little Egret (*Egretta garzetta*) and Abdim's storks (*Ciconia abdimii*). Yellowbilled storks (*Mycteria ibis*) and Sacred ibis (*Threskiornis aethiopicus*) have been observed in the wadis after rains and a species of duck (unidentified) was rescued in the camp. Small passerines have also been observed on site.

Figure 20.1 Golden Jackal Captured by Remote Camera

Source: Orca Gold Inc.

There are no National Parks, Game Reserves, or designated areas of conservation importance or scientific interest in the Sudan desert region spanning from the Nile River to the western hills of the Red Sea.

20.7 Baseline Social Setting

20.7.1 Population

The Project is situated in the remote and largely uninhabited north-western corner of Sudan and therefore does not have a sustained local population in place.

20.7.2 Regional and Local Economic Structure

The low rainfall in the region, combined with thin and poorly developed soils and rocky plateau topography, limit the width of the Nile's floodplain and thus the extent of fertile land, to approximately one kilometre east and west of the Nile. Large-scale irrigation projects are commencing along the Nile floodplain. Outside of these projects, farmers are restricted to small, intensively cultivated fields fed by water diverted from the Nile.

The main source of employment in the vicinity of Abu Hamad arises from artisanal mining and associated support services. The irrigation projects are new operations, so employment opportunities associated with them are low in comparison to mining revenue. Revenue for communities is also generated by fees imposed on the illegal miners, in the form of ore. Workers migrate from time to time, once they have completed exploration activities in their project areas. A small amount of formal employment is provided by mineral exploration companies, including MSMCL.

Abu Hamad is the only significant local population centre, located 180 km south of the Project. Abu Hamad is expanding as a mining equipment maintenance and supply centre for artisanal activities. 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 tribe as staff or as casual labourers.

There are a large number of artisanal miners working in the region. Many come from other areas of Sudan where employment opportunities are scarce. Their works are extensive, covering large areas and comprises predominantly surface mining, although there are a few deep hard rock pits. Their activities are transient, moving from one prospect to the next and the land surface is scoured and pockmarked by these workings. There is extensive use of heavy equipment, such as excavators, shaking tables, tractors and trucks together with individuals using hand held metal detectors. All processing of the material produced by the artisanal miners takes place in Abu Hamad.

20.7.3 Landscape and Visual

No established communities are present in the area that would be potential receptors for landscape changes and visual impacts. The Project area itself is characterised by chains of hills with several hundreds of metres of relief, that are separated by wide, sand filled flats and drainage channels (wadis). Artisanal and small scale miners have been active in the area and have created extensive disturbed areas. These have had a significant visual impact on the natural terrain due to the extent of disturbance. Raised facilities of the Project, such as the tailings storage facility and rock waste dumps, will not impact on the overall relief of the Project area.

20.7.4 Infrastructure

Further to the description in Section 5.4, there is no existing infrastructure such as electricity or water supply present. Supply routes would be through Khartoum and Port Sudan, with tarred roads between these cities and Abu Hamad. Access to the exploration camp and Project area is via desert tracks from Abu Hamad that are used by the artisanal mining community. Routes through the desert are not clearly defined but are heavily used. Two cafeterias are located along the route, which act as supply stops and telecommunication centers for artisanal miners. Within the project area, roads are unpaved.

To the extent relevant to the Project, the infrastructure requirements for the current and planned exploration activities as well as development of the Project are minimal. 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 (where MSMCL has their administration office).

20.7.5 Land Acquisition and Resettlement

Since there are no formal local communities in the vicinity of the project, no resettlement is required. Under Sudanese legislation, the land is owned by the State and the operation’s Concession Agreement gives the company the right to utilize the land. As such, no land acquisition will be required.

20.7.6 Archaeology and Cultural Heritage

The region has been worked historically, with workings dating to circa 4,000 BC. Sites of interest have been observed during the exploration phase, but none have been found within the target resource areas of GSS or WD. However, 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. This Chance Finds Procedure defines a series of steps to minimize physical impacts to cultural heritage by providing a process for conducting an archaeological look ahead-survey, monitoring of ground disturbing activities and responding to any tangible cultural heritage encountered unexpectedly during exploration.

Figure 20.2 Historical Artefacts

Left: Pharaonic tomb. Right: Grinding stone (Source: Orca Gold).

20.8 Potential Environmental Impacts

The EIA includes an assessment of the environmental and social impacts of the Project's planned development. The assessment is based on the site visits and undertaken according to internationally accepted procedures for such assessments. The assessment of impacts includes:

- Gathering of available environmental and social baseline data.
- Analysis of the proposed development and alternatives with regard to potential impacts and risks during construction, operation and mine closure.
- Identification of environmental mitigation strategy.
- Prediction and assessment of impacts of the mitigated project in terms of their magnitude, significance and duration.
- Recommendations for environmental management and monitoring.
- Inclusion of an outline mine closure (decommissioning) plan.
- Collation of the above information into a report.

The EIA thathas been approved determined that while the development of the Project may give rise to some environmental impacts, these can be minimised. Project development and operations will cause several changes to some aspects of the local environment, e.g. visual, air and water quality, use of water resources and removal of wildlife habitat, among others. The Project will use LNG to provide power for construction and operation. This will reduce carbon emissions compared to the use of HFO. The absence of any current social habitat in the concession means that social impact would be minimal. Furthermore, assuming the implementation of the proposed mitigation measures, potential impacts are considered manageable and controllable and therefore would enable effective environmental and social development, operation and closure of the Project.

20.9 Environmental Protection and Management

An Environmental Protection and Management Plan (EPMP) has been prepared for the Project as part of the EIA. MSMCL is committed to managing the impacts of its operations, in conformance with recognised international best practice. The purpose of the EPMP is to ensure that appropriate control and monitoring measures are in place to deal with all significant impacts of the Project. The EPMP has been designed so that it can be regularly reviewed and updated according to company policies and conforms to SMRC guidance. It is also designed to be developed throughout the life of the Project, having commenced with a basic EPMP during the Exploration phase of the Project. A register of impacts is used to track and monitor implementation and progress. The EPMP has been designed so that it can be reviewed and updated on a systematic basis in line with company policies. It is also designed to be developed throughout the life of the Project. A basic plan was developed for Exploration activities, which forms the basis of the EPMP for Construction then Operational Phases of the Project, with the following objectives:

- Establish management targets to be implemented in order to achieve the Project's environmental and social objectives; and.
- Develop a plan that can be systematically audited and reviewed, allowing for continual improvement of the operation.

The Operational Phase of the EPMP will be revised at least 2 years prior to closure in order to incorporate aspects of the Closure Phase.

The key elements of the EPMP include:

- Recognition that sound environmental management is essential to successfully operate the facility.
- Accountability of all staff for minimising environmental risk and assuring compliance with regulatory requirements as well as the Project's environmental objectives.
- Implementation of monitoring programmes to provide early warning of any deficiency or unanticipated performance in environmental safeguards.
- Training and orientation of employees in order to perform their jobs in conformance to sound environmental practices.
- Consideration of environmental factors to be included in all new or modified facilities and the purchase of equipment and materials.
- Establishment of an environmental incident reporting system.
- Development of environmental response plans to provide the basis for response to environmental incidents, including spill prevention and plans for counter measures, monitoring and mitigation.
- Verification of environmental compliance at regular intervals to confirm that management responsibilities are in accordance with environmental procedures; and

- Continuing dialogue with government entities to establish a good working partnership for the Project.

The EPMP includes details of the area of impact, objectives to reduce negative or enhance positive impacts, specific targets adopted to achieve those objectives and definition of responsibilities for implementing the programme. It is a live document that can be reviewed and updated on a systematic basis, in-line with the principles of continual improvement.

20.10 Monitoring

MSMCL has developed and implemented an environmental and social monitoring plan, which commenced during the exploration phase of the Project. The objective of monitoring is to characterise environmental conditions, including groundwater, air quality (specifically airborne dust) and ecology and will continue to observe any changes in the environment through the life of the Project. This information was used to inform the EIA and associated mitigation plans. Data will continue to be collected and analysed to inform the environmental management of the Project and support action levels and response plans for future monitoring of construction, operation and closure phase impacts.

MSMCL has developed and implemented appropriate sampling procedures for water and dust sampling. Passive infrared cameras are used to detect the presence of wildlife.

Groundwater levels are recorded on a monthly basis and water quality samples have been collected and sent for analysis on a six-monthly basis. Groundwater monitoring will continue throughout the life of the Project, with sample locations and frequency adapted to suit project requirements. Interactions with artisanal and small-scale miners are recorded in a diary, along with wildlife observations and any other items of environmental interest.

20.11 Public Consultation

As part of the environmental assessment, public consultation and disclosure is required. In order to ensure that the Project is developed and operated in an appropriate manner, MSMCL endorses the concept that effective engagement with its stakeholders is an essential component of the assessment process and its on-going 'license to operate'. MSMCL is therefore committed to a pro-active programme of communications with all relevant stakeholders. A Public Consultation and Disclosure Plan has been developed by MSMCL for the EIA.

A list of Project stakeholders has been developed, including those at a National, Regional and local level where appropriate. The Project has few local stakeholders due to the remoteness of the location. The closest people to the site are artisanal miners. The Project has engaged with the artisanal miners who under the supervision of the Mining Police have been removed from the Wadi Doum (2017) and GSS areas (2018).

MSMCL has continually held discussions with the SMRC on both exploration and environmental aspects of the Project. Such communication at a Regional level (i.e. with State and urban authorities) will continue throughout the life of the Project, as constant communication will prevent misunderstandings and manage expectations from all parties. A series of small gatherings have been held with distant communities (Geibat peoples and representatives of the Ababda tribes) about the nature of the Project and potential for social assistance. A log of these gatherings has been kept, summarising the numbers of people engaged with, their activities and any issues or concerns they may have with the project.

21. CAPITAL AND OPERATING COSTS

21.1 Introduction

The overall study capital cost estimate was compiled by Lycopodium and includes, in addition to the process treatment facility, the separate cost estimates prepared by others for specific components of the total cost including mining, tailings storage and the raw water borefield. The total cost estimate is presented here in summary format. The various elements of the Project estimate have been subject to internal peer review by Lycopodium and have been reviewed with Orca for scope and accuracy.

The capital cost estimate was developed to an accuracy level range of ±15% to cover engineering, procurement, construction and start-up of the mine and processing facilities, as well as the ongoing sustaining capital costs. The capital cost estimates were developed for a conventional open pit mine, process plant and supporting infrastructure for an operation capable of treating 6.0 Mtpa of material. Power supply and camp construction via a third-party Build Own Operate Transfer and a contract mining scenario have been adopted.

The estimate covers the direct costs of purchasing and constructing the plant 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 have been estimated based on a detailed breakdown of tasks, disciplines and manhours required in each category. Owner's costs have been estimated based on manhour allowances for training and supervision during preproduction and a spare parts allowance based on the equipment supply cost. Contingencies have been assigned by assessing the defining inputs to the item cost basis (scope, supply, installation) and applying the weighted average to define the contingency to that item. Risk amounts are specifically excluded from this estimate. A breakdown of the capital cost estimates is shown in Table 21.1. All costs are estimated in United States Dollars (US$) as at 3Q20.

21.2 Capital Cost Summary

The capital estimate is summarized in Tables 21.1 and 21.2. The initial project capital cost is estimated at US$321.2M, including a contingency allowance of US$33.6M.

Table 21.1 Capital Estimate Summary (3Q20, ±15%)

Main Area	US$'000s
Mine	$15,443
Treatment Plant	$193,492
Generator Plant	$7,100
TSF	$17,301
Camp	$2,506
EPCM	$26,853
Owners Costs	$24,970
Subtotal	**$287,665**
Contingency	$33,557
Grand Total	**$321,222**

The total LOM cost is estimated at US$499.8M, including sustaining capital costs of US$178.6M, as shown in Table 21.2. The Camp and Generator Plant will be financed under a Build Own Operate Transfer (BOOT) contract. The duration of the contracts will be five years.

Table 21.2 Sustaining Capital Estimate Summary (3Q20, ±15%)

Main Area	US$'000s
Camp	$14,198
TSF	$53,733
Generator Plant	$63,112
Process Plant	$35,000
TSF closure	$5,946
Mine Closure	$6,600
Grand Total	**$178,590**

21.3 Direct Capital Costs - Mining

Total pre-production mine development costs have been estimated by Deswik Europe to be US$15.4M, primarily associated with mine contractor site establishment and waste pre-stripping.

21.4 Capital Cost Estimate – Process Plant and Infrastructure

Capital costs for the process plant and infrastructure area were compiled using the methods of calculation described in Table 21.3 below.

Table 21.3 Capital Cost Methods of Calculation

Discipline	Sub Items	Method
Earthworks – Process Plant	Quantities	Quantities for bulk earthworks derived from plant layout. Detailed earthworks for foundations are included in concrete costs.
	Unit Rates	Sourced from multiple regional contractors based on preliminary quantities for this project.
Concrete	Quantities	Quantities derived by area based on area layout and referencing against previous similar projects. Site geotechnical information indicates no special foundation requirements are necessary.
	Unit Rates	Rates sourced from multiple regional contractors based on in-country interviews and preliminary quantities for this project. Rates are inclusive of detailed excavation, formwork, reinforcing steel, batching, placement, and surface finish. Separate rates provided for items such as pre-cast sumps, cast-in platework, concrete surface protection etc.
Steelwork	Quantities	Quantities derived by area based on preliminary structural designs or previous similar projects.
	Supply Rates	Supply rates sourced from multiple regional contractors based on preliminary quantities for this project. Rates are inclusive of workshop detailing, supply, fabrication, painting and packaging onto transport.
	Install Rates	Install rates sourced from multiple regional contractors based on preliminary quantities for this project. Quoted installation hours normalised against Lycopodium database hours to determine productivity factor.
Platework	Quantities	Platework quantities for tanks based on preliminary design datasheet for each tank using sizing from the mechanical equipment list. Minor platework quantities estimated per item.
	Supply Rates	Supply rates sourced from multiple regional contractors based on preliminary quantities for this project. Rates are inclusive of workshop detailing, supply, fabrication, painting and packaging onto transport.
	Install Rates	Install rates sourced from multiple regional contractors based on in-country interviews and preliminary quantities for this project.

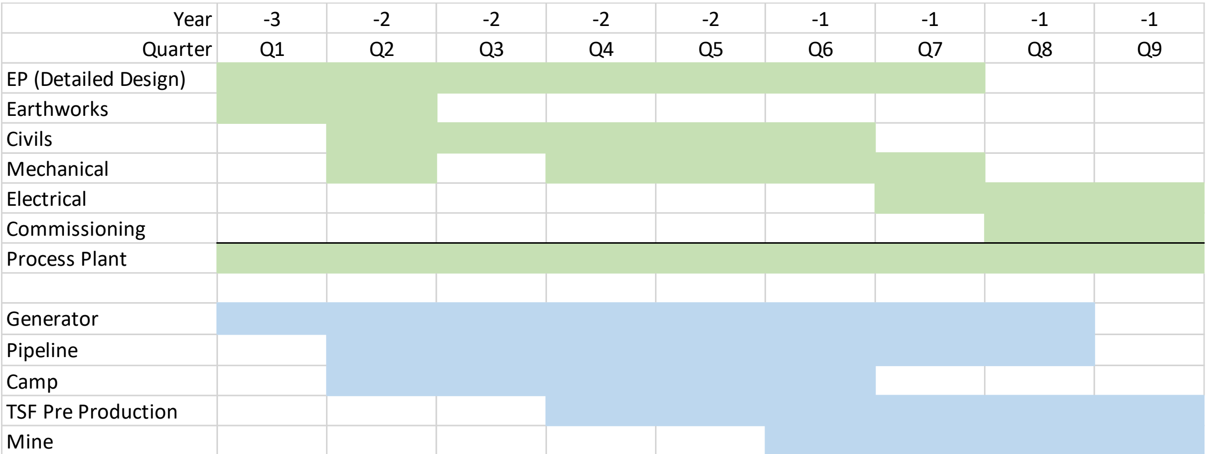
Discipline	Sub Items	Method
		Installation hours obtained for both site erected and shop fabricated tanks.
Mechanical Equipment	Supply Costs – Major Equipment	Multiple budget pricing received from international vendors against datasheets developed specifically for the project. Pricing provided and compiled using native currencies. Preliminary evaluation conducted to select appropriate vendor (not necessarily lowest).
	Supply Costs – Minor Equipment	Minor equipment pricing obtained from Lycopodium database of previous project orders for similar equipment.
	Install Costs	Labour rates sourced from multiple regional contractors based on in-country interviews and preliminary manhours for this project. Labour rates are inclusive of tools and equipment, including cranage. Installation hours estimated by Lycopodium based on previous experience.
Piping	Supply Costs & Quantities	Area piping supply costs factored from the mechanical equipment supply cost based on previous experience and benchmarked against other projects.
	Install Costs	Area piping installation hours factored from mechanical installation hours based on in-country interviews and previous experience and benchmarked against other projects.
Electrical & Instrumentation	Supply Costs	Based on budget pricing from vendors against datasheets developed specifically for the project. Database supply costs used for VSDs and instruments.
	Quantities	Instrument list generated for the project based on previous similar projects. Cable quantities derived from plant layout.
	Install Costs	Install rates sourced from regional contractor based on in-country interviews and estimated quantities for this project.
Buildings	Quantities	Size of pre-fabricated buildings estimated on a m2 basis from previous projects.
	Unit Rates	Supply and installation of pre-fabricated buildings estimated as a m2 rate based on previous similar projects. Building fit-out estimated based on previous similar projects.
Freight	Total Cost	Freight costs based upon previous similar projects and supplemented with a specific Logistics and Route survey conducted for the project in Sudan by Antrak.
Import Duty	Total Cost	All project items assumed to be free of import duty under agreement between Orca and the Sudanese Government.
Vendor Representatives	Total Cost	Estimate prepared based on specialist vendor representatives expected to assist with construction verification and commissioning.
Owners Costs	Total Cost	Costs including: • Opening stocks. • First fill reagents and consumables. • Plant Pre-production labour costs (prior to introducing ore to the plant). • Training. • Laboratory costs. • Spare parts. • Mobile equipment.

Project infrastructure includes mine infrastructure as itemised in Section 18.6.2.

The EPCM cost was determined by a detailed build up of tasks, disciplines and manhours required in of the detailed design, construction supervision and commissioning phases. Expenses such as catering and accommodation for the Engineer's site personnel, as well as site telecommunications costs, are included in the estimate.

The duration of the detailed design and construction phase of the project has been estimated to be 27 months, with the construction of the raw water pipeline the dominant critical path activity.

Figure 21.1 Construction Schedule



The site accommodation camp size has been selected, by taking the manning demands for the various overlapping activities into account. A program of early engineering works is planned to minimise the overall duration. Specific activities will be undertaken related to mining, plant and infrastructure, raw water supply, tailings management and the power station and these will include:

- Hydrogeological and stockpile management studies.
- Preparation of Engineering Specfications.
- Complete engineering design for the water pipeline.
- Specification for long lead equipment packages; Mills, Crusher, Pipe line.
- Geotechincal and concrete aggregate investigations.
- Project Management.

A contingency allowance is included to make specific provision for uncertain elements of cost within the project scope. Contingencies do not include allowances for scope changes, escalation, or exchange rate fluctuations. Contingency has been applied to all parts of the process plant estimate.

The cost of raw water supply to the project is based on the combination of two estimates viz:

- The estimated cost for the drilling of production bores, borehole pumps and piping to a central collection tank which was determined by GCS.
- The estimated cost of the overland pipeline including tankage, transfer pumps and power supply and communications to all facilities within the Borefield and the overland pipeline to the plant site which is included in the Treatment Plant cost in Table 21.1.

Estimated establishment cost and ongoing sustaining cost for the Tailings Storage Facility were estimated by Knight Piesold based on the mine plan, geotechnical assessment, laboratory testwork on representative sample material and feasibility level designs for the TSF as outlined in Section 18.3.

An estimate has been provided by Mineesia of the costs associated with the decommissioning and dismantling the Project. There may be scope for a proportion of these costs to be incorporated as normal operating costs during the life of mine, as facilities and infrastructure become redundant. These costs will be refined in detail in the final Closure Plan, which will be submitted two years prior to closure. The total estimated cost of decommissioning the Project components is in the order of US$6.6M.

21.5 Operating Costs – Mining

Contract open pit mining costs were derived by Deswik Europe from first principles based on equipment required and include pit and dump operations, road maintenance, mine supervision and technical services cost. These costs were then compared with several contractor responses for budget pricing followed by validation against a more detailed budget estimate provided by a well established African based mining contractor. The average open pit operating cost (US$ /t mined) is shown in Table 21.4.

Table 21.4 Average Mining Costs

Mine Area	Mineralized Rock (US$ /t)	Waste Rock (US$ /t)
Main	2.85	2.48
East	2.87	2.52
NEZ	2.36	2.09
Wadi Doum	4.67	4.41
Total	**2.67**	**2.60**

A grade control drilling cost of US$0.26 /t has been included in the above cost for ore tonnes. In addition, a transfer cost of US$7.59 /t will be incurred on ore from Wadi Doum. An elevated cut-off grade strategy has also been implemented to assist with project cash flow and NPV. The average unit rate for rehandle is US$0.87 /t and amounts to approximately US$34.9M over the life of the project.

The diesel price of US$0.60 /l was used.

21.6 Operating Cost – Plant and Infrastructure

Process plant and infrastructure operating costs have been developed by Lycopodium based on a treatment rate of 6.0 Mtpa of ore, with the plant operating 24 h/d, 365 d/y with a 91.3% plant utilization, nominally 8,000 h/y.

The operating cost estimate (OPEX) has been compiled from a variety of sources, including metallurgical testwork, Orca advice, OMC comminution modelling, first principle calculations, vendor quotations and the Lycopodium database.

The OPEX has been divided into multiple cost centres, with fixed and variable costs calculated for each cost centre for each different material type.

The estimate comprises the following major cost centres:

- Plant and related infrastructure power.
- Plant consumables, including mill media and liners, reagents and diesel for fixed plant equipment and plant mobile equipment.
- Plant maintenance materials, including mobile equipment parts.
- Laboratory.
- Plant and administration labour.
- General and administration costs.

The process operating cost includes all direct costs to produce gold bullion for the Project. The battery limits are the ROM feed into the primary crusher (ROM loader by Mining), production of dore and mercury in the gold room and discharge of tailings at the TSF.

Operating costs are presented in United States Dollars (US$), to an accuracy of ±15% and are predominantly based on pricing obtained during the third quarter of 2020.

21.6.1 Summary

Process operating costs have been developed for each major domain and lithology. Operating costs were developed using the plant parameters specified in the process design criteria. Tables 21.5 to Table 21.7 present the operating cost summary by material type. Costs are presented as fixed and variable for ease of presentation.

Table 21.5 Operating Cost per Material Type, Main Zone (3Q20, ±15%)

Cost Centre	Fixed US$'000/y	MZF - PGSS	MZF - FQSP	MZF - VAN	MZT - PGSS	MZT - FQSP	MZT - VAN	MZOX - PGSS	MZOX - FQSP	MZOX - VAN
		Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t
Power (excluding grinding)	2,845	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84
Grinding Power	0	2.06	2.45	2.11	1.82	1.87	1.77	1.94	2.01	1.61
Operating Consumables	0	2.47	3.84	2.67	3.67	3.57	3.08	2.97	3.00	2.80
Maintenance Materials	3,400	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13
Laboratory	810	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07
Process & Maintenance Labour	2,812	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Administration Labour	3,050	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
General & Administration	7,059	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
TOTAL	**19,977**	**5.55**	**7.32**	**5.81**	**6.52**	**6.46**	**5.88**	**5.93**	**6.04**	**5.43**

Table 21.6 Operating Cost per Material Type, East Zone & Wadi Doum (3Q20, ±15%)

Cost Centre	Fixed	EZOX - SRR	EZOX - VAN	WDF - QPR	WDF - VAN	WDF – VC1	WDF – VC3	WDT	WDOX
	US$'000/y	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t
Power (excluding grinding)	2,845	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84
Grinding Power	0	1.63	1.63	2.49	2.58	2.58	2.67	2.58	1.63
Operating Consumables	0	2.73	2.79	3.52	3.72	3.88	3.87	5.60	3.85
Maintenance Materials	3,400	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13
Laboratory	810	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07
Process & Maintenance Labour	2,812	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Administration Labour	3,050	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
General & Administration	7,059	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
TOTAL	**19,977**	**5.38**	**5.45**	**7.04**	**7.32**	**7.49**	**7.56**	**9.20**	**6.51**

Table 21.7 Operating Cost per Material Type, North East Zone (3Q20, ±15%)

Cost Centre	Fixed US$'000/y	NET - PGSS	NET - VAN	NEOX - FQSP	NEOX - PGSS	NEOX - VAN	LOM
		Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t	Variable US$/t
Power (excluding grinding)	2,845	0.84	0.84	0.84	0.84	0.84	0.84
Grinding Power	0	1.46	1.46	1.71	1.71	1.71	2.11
Operating Consumables	0	3.07	2.83	2.86	2.79	2.80	3.27
Maintenance Materials	3,400	0.13	0.13	0.13	0.13	0.13	0.13
Laboratory	810	0.07	0.07	0.07	0.07	0.07	0.07
Process & Maintenance Labour	2,812	0.00	0.00	0.00	0.00	0.00	0.00
Administration Labour	3,050	0.00	0.00	0.00	0.00	0.00	0.00
General & Administration	7,059	0.00	0.00	0.00	0.00	0.00	0.00
TOTAL	**19,977**	**5.56**	**5.31**	**5.60**	**5.53**	**5.54**	**6.41**

21.6.2 Power

Power consumption has been calculated from the gross power required to achieve the desired grind size on each material type, based on OMC comminution modelling, plus the remaining installed power from the mechanical equipment list, with suitable drive efficiency and utilization applied. The total power draw was used to calculate power costs, based on an LNG fuel price of US$9.70 /MBTU, at a unit price of US$0.1083 /kWh.

This power price is based on an LNG on-site power station and supported by vendor quotation and stated fuel consumption data. A summary of the power cost for the plant by plant area is shown in Table 21.8. Table 21.9 details the key variable costs associated with the mills and pebble crusher for each major domain and lithology.

Table 21.8 Process Plant Power Costs by Area (3Q20, ±15%)

Plant Area	Total Power (US$/t)
Feed Preparation	0.13
Grinding Area	0.24
Mills / Pebble Crusher	Variable
Pre-Leach Thickening	0.05
Leaching & CIP	0.28
Tailings	0.11
Elution / Goldroom	0.03
Reagents	0.01
Water and Air	0.28
Water Supply, TSF	0.07
Plant Buildings & Camp	0.09
Mine Services	0.01

Table 21.9 Mill and Pebble Crusher Power Costs by Material Type (3Q20, ±15%)

Material Type	Power (US$/t)	Material Type	Power (US$/t)
MZF - PGSS	2.05	EZT - VAN	1.72
MZF - FQSP	2.45	EZOX - FQSP	1.59
MZF - VAN	2.11	EZOX – PGSS	1.67
MZT - PGSS	1.82	EZOX – SRR	1.63
MZT - FQSP	1.86	EZOX – VAN	1.63
MZT - VAN	1.77	WDF – QPR	2.49
MZOX - PGSS	1.94	WDF - VAN	2.57
MZOX - FQSP	2.01	WDF – VC1	2.57
MZOX - VAN	1.61	WDF – VC3	2.57
EZF - FQSP	2.55	WDT	2.57
EZF - PGSS	2.28	WDOX	1.63
EZF - SRR	2.99	NET - PGSS	1.46
EZF - VAN	2.51	NET - VAN	1.46
EZT - FQSP	2.00	NEOX - FQSP	1.71
EZT - PGSS	1.99	NEOX - PGSS	1.71
EZT - SRR	1.90	NEOX - VAN	1.71

21.6.3 Operating Consumables

In general, costs have been built up from first principle estimates, with budget quotations obtained for major reagents and consumables and adjusted to a DAP (Delivered at Place) price. The diesel price has been based on vendor quotation with allowance for shipping, wharfage and transport to site.

Reagent and consumables consumption rates have been based on metallurgical testwork, calculations, vendor supplied data and/or modelling. Diesel consumption rates for plant mobile equipment have been based on the selected mobile equipment fleet and designated operating hours. Pricing and usage rates for a small number of minor consumables have been based on the Lycopodium database of recent gold projects.

A summary of the reagent and consumables costs by area is presented in Table 21.10. Table 21.11 details the key variable costs associated for each major domain and lithology.

Table 21.10 Process Plant Operating Consumable Costs by Area (3Q20, ±15%)

Plant Area	Total Cost (US$/t)
Primary Crushing	0.02
Mills / Pebble Crusher	Variable
Leach / CIP / Thickeners	Variable
Elution / Goldroom	0.24
Fuel	0.33

Table 21.11 Variable Operating Consumable Costs by Material Type (3Q20, ±15%)

Material Type	Mills / Pebble Crusher (US$/t)	Leach / CIP / Thickeners (US$/t)	Material Type	Mills / Pebble Crusher (US$/t)	Leach / CIP / Thickeners (US$/t)
MZF - PGSS	0.58	1.35	EZT - VAN	1.07	1.48
MZF - FQSP	1.87	1.43	EZOX - FQSP	0.72	1.52
MZF - VAN	0.90	1.23	EZOX – PGSS	0.68	1.52
MZT - PGSS	1.14	2.00	EZOX – SRR	0.70	1.49
MZT - FQSP	1.17	1.87	EZOX – VAN	0.70	1.56
MZT - VAN	1.11	1.44	WDF – QPR	1.18	1.80
MZOX - PGSS	0.63	1.80	WDF - VAN	1.22	1.95
MZOX - FQSP	0.65	1.83	WDF – VC1	1.22	2.12
MZOX - VAN	0.52	1.75	WDF – VC3	1.26	2.06
EZF - FQSP	2.17	1.39	WDT	1.22	3.84
EZF - PGSS	1.17	1.36	WDOX	0.70	2.62
EZF - SRR	3.03	1.47	NET - PGSS	0.82	1.72
EZF - VAN	2.20	1.47	NET - VAN	0.82	1.47
EZT - FQSP	1.47	1.75	NEOX - FQSP	0.63	1.69
EZT - PGSS	0.95	1.56	NEOX - PGSS	0.63	1.62
EZT - SRR	1.16	1.52	NEOX - VAN	0.63	1.63

21.6.4 Labour (Processing / Maintenance and Administation)

Labour rates and overhead costs for Sudanese workers have been supplied by Orca based on current in country rates. Expatriate costs were agreed by Lycopodium and Orca, based the Lycopodium database and Orca experience. The labour rates are based on a skill level and consist of a base salary and the required overhead allowances. Overheads include such items as work permits, health insurance, social security and production bonuses.

Manning numbers were agreed by Lycopodium and Orca based on similar projects in Africa and with an eight week on four weeks off roster for expatriates and a six week on two weeks off roster for Sudanese workers. As there are no local townships in close proximity to the mine site, all workers will be accommodated in single person's quarters on site. As the laboratory will be run on a contract basis, labour costs are included as an all up rate in the Laboratory cost centre.

The costs of accommodation, messing and worker transport are included in the General and Administration costs.

Table 21.12 presents a summary of the labour costs.

Table 21.12 Process Plant Labour Costs (3Q20, ±15%)

Area	No. of People	Total Cost (US$/y)	Total Cost (US$/t)
Administration – Expats	8	1,501,816	0.25
Administration – Sudanese	68	1,548,241	0.26
Process & Maintenance – Expats	12	1,413,772	0.23
Process & Maintenance – Sudanese	67	1,398,682	0.23
Contract Laboratory	19	-	-
TOTAL	**174**	**5,682,510**	**0.97**

21.6.5 General and Administation Costs (Excluding G&A Labour)

General and administration costs were developed jointly by Lycopodium and Orca and have been based on information from Orca's existing in country exploration facilities and similar African operations. A quotation for camp management, catering, cleaning, laundry and pest control was obtained from RAI International.

Accommodation and messing costs for all process plant, site administration, laboratory and mining personnel have been included in these costs under the contract area.

General and administration costs are presented by major area. The site office category includes items such as communications, stationery, computers and software and office equipment. Financial includes legal and banking fees, as well as external auditing and accounting costs. Personnel encompasses first aid and medical costs, safety supplies, recruiting, training, recreation and entertainment and professional memberships. Transport includes all FIFO costs for expats as well as transport for Sudanese workers from major townships. Contract includes camp catering and cleaning and external consultants.

This information is summarised in Table 21.13.

Table 21.13 General and Administration Costs (3Q20, ±15%)

Area	Total Cost (US$/y)	Total Cost (US$/t)
Site Office	533,760	0.09
Insurances	900,000	0.15
Financial	156,000	0.03
Government Charges	20,500	0.003
Personnel	311,601	0.05
Transport	310,000	0.05
Contract	4,485,629	0.74
Community Relations	102,000	0.02
Other	240,000	0.04
TOTAL	**7,059,490**	**1.17**

21.6.6 Maintenance

Maintenance materials costs have been calculated as a percentage factor of the direct capital costs on an area-by-area basis. The costs include wear items such as screen panels, pump liners, valves, rubber linings, etc., but exclude crusher and mill liners which are included in the operating consumables cost centre. An allowance for maintaining plant mobile equipment (tyres, oil, etc.) has also been included based on Lycopodium experience.

Maintenance material costs are summarised in Table 21.14.

Table 21.14 Plant Maintenance Materials Costs (3Q20, ±15%)

Area	Total Cost (US$/y)	Total Cost (US$/t)
Feed Preparation	308,000	0.051
Milling / Pebble Crushing	1,493,604	0.247
Pre-Leach Thickening	156,340	0.026
Leaching & CIP	244,283	0.040
Elution / Goldroom	172,638	0.029
Tailings	157,605	0.026
Reagents & Air Services	290,569	0.048
Infrastructure	287,644	0.048
Electrical	6,101	0.001
Mobile Equipment	582,510	0.097
Contract & General	456,000	0.076
TOTAL	**4,155,294**	**0.688**

21.6.7 Laboratory

The laboratory will be run as a contract operation, that will provide all equipment, labour and operating consumables required to process the agreed number of plant, grade control and water samples. Quotations were based on a detailed sample analysis list.

Table 21.15 presents a summary of the laboratory costs.

Table 21.15 Laboratory Costs (3Q20, ±15%)

Area	Total Cost (US$/y)	Total Cost (US$/t)
Laboratory Analysis	1,203,504	0.20
TOTAL	**1,203,504**	**0.20**

21.7 Exclusions

The following items have been excluded from the operating cost estimate:

- All sunk costs.
- Government monitoring and compliance costs.
- All Orca head office costs and corporate overheads.
- Withholding taxes and other taxes, such as GST or VAT.
- Escalation beyond the date of the estimate.
- Financing costs.
- Foreign exchange fluctuations.
- Interest charges.
- Political risk insurance.
- All costs associated with areas beyond the battery limit of the study.
- Land compensations costs.
- Subsidies to local communities.
- Licence fees.
- Royalties.
- Contingency.
- All mining and exploration costs, except power costs for mining services within Lycopodium scope of work, Orca owned mine light vehicle costs and grade control sample assay costs.
- Maintenance costs of all mine, haul and plant access roads.
- Gold refining costs.
- Mercury treatment / stabilisation costs after closure.
- Bullion transport costs.
- Bullion marketing costs.
- Bullion insurance in transit costs.
- Tailings storage costs, including future lifts and rehabilitation, which are included in sustaining capital.
- Tailings dust suppression costs.
- Any rehabilitation or closure costs.

22. ECONOMIC ANALYSIS

22.1 Introduction

The economic analysis is based on Probable Reserves and mine schedule as per Table 22.4.

An 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 mined and processed tonnages and grades for the CIP feed, process recoveries, metal prices, operating costs and refining charges, royalties and capital expenditures (both initial and sustaining).

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

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

22.2 Model Inputs and Assumptions

The model inputs and assumptions used in the economic analysis are summarized in Table 22.1 and, unless otherwise stated, is used in the model.

Table 22.1 Model Inputs and Assumptions

Model Inputs	Unit / Value
Base Currency	US$
Base Date	3rd Quarter 2020
Money Terms	August 2020
Sudan Royalty (charged against Revenue)	7%
Sudan Tax Rate	15%
NPV Discount Rate	5%
Metal Price Scenario – Fixed	US$1,350 / oz
Refinery Charges & Shipping	US$4 /oz
Assumptions	
Capex excludes Finance Charges & Fees	
Capex excludes Pre-production Investigations	
Capex Amortisation/Depreciation based on no salvage value	
Capex excludes Escalation	
Tax paid on an Annual basis	

22.2.1 Capital Cost Expenditures

Pre-production capital expenditures are defined in Table 22.2. Sustaining capital for the Plant, Mining and TSF expansion costs have been phased over the life of the project and detailed in Table 22.3.

Table 22.2 Pre-production Capital Expenditure

Item	Unit	Total	Year		
			-3	-2	-1
Mine	US$'000	15,443			
Process Plant	US$'000	193,492			
Power Generation	US$'000	7,100			
TSF	US$'000	17,301			
Camp	US$'000	2,506			
EPCM	US$'000	26,853			
Owner	US$'000	24,970			
Construction Sub Total	**US$'000**	**287,664**			
Contingency	US$'000	33,557			
Construction Total	**US$'000**	**321,222**	**12,210**	**112,333**	**196,673**

Table 22.3 Sustaining Capital Expenditure

Item	Unit	LOM
Camp	US$'000	14,198
TSF	US$'000	53,733
TSF Closure	US$'000	5,946
Generator	US$'000	63,112
Plant	US$'000	35,000
Mine Closure	US$'000	6,600
Sustaining Total	US$'000	**178,590**

22.2.2 Royalties

Royalties at 7% have been included for the LOM and will be charged against the revenue.

22.2.3 Cost of Sales

Cost of Sales includes freight and refining costs. A value of US$4.00 /oz gold recovered has been allowed for in the model.

22.2.4 Depreciation

Depreciation is calculated using the units of production method starting with first year of production and can be summarized as follows:

- Initial pre-production capex depreciated over the total LOM, based on units of production.
- Capitalised pre-production costs (i.e. cumulative exploration and PEA costs) to date depreciated over the total LOM, using estimated total capitalised pre-production costs of US$60.9M up to date (30 Jun 2020).
- The annual sustaining capital is assumed to be largely repairs and maintenance.

Remaining sustaining capital items (i.e. TSF) each depreciated separately based on respective remaining LOM.

22.2.5 Inflation

Inflation has not been included in the cash flow analysis.

22.2.6 Operating Costs

Annual fixed and variable costs, as per Sections 21.5 and 21.6, are included in the cash flow.

22.3 Financial Model

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 for the pre-tax and the post-tax model. Figure 22.2 shows the annual and cumulative post-tax cash flow.

The pre-tax and post-tax financial results of the project are summaries in Table 22.4. On a pre-tax basis, the project has a Net Present Value (NPV) of US$723M at a discount rate of 5%, an Internal Rate of Return (IRR) of 38.5%; on a post-tax basis the NPV is US$606.8M at a discount rate of 5%, the IRR is 33.3% and the discounted payback period is 2.9 years following commencement of production.

Figure 22.1 Cumulative Cash Flow

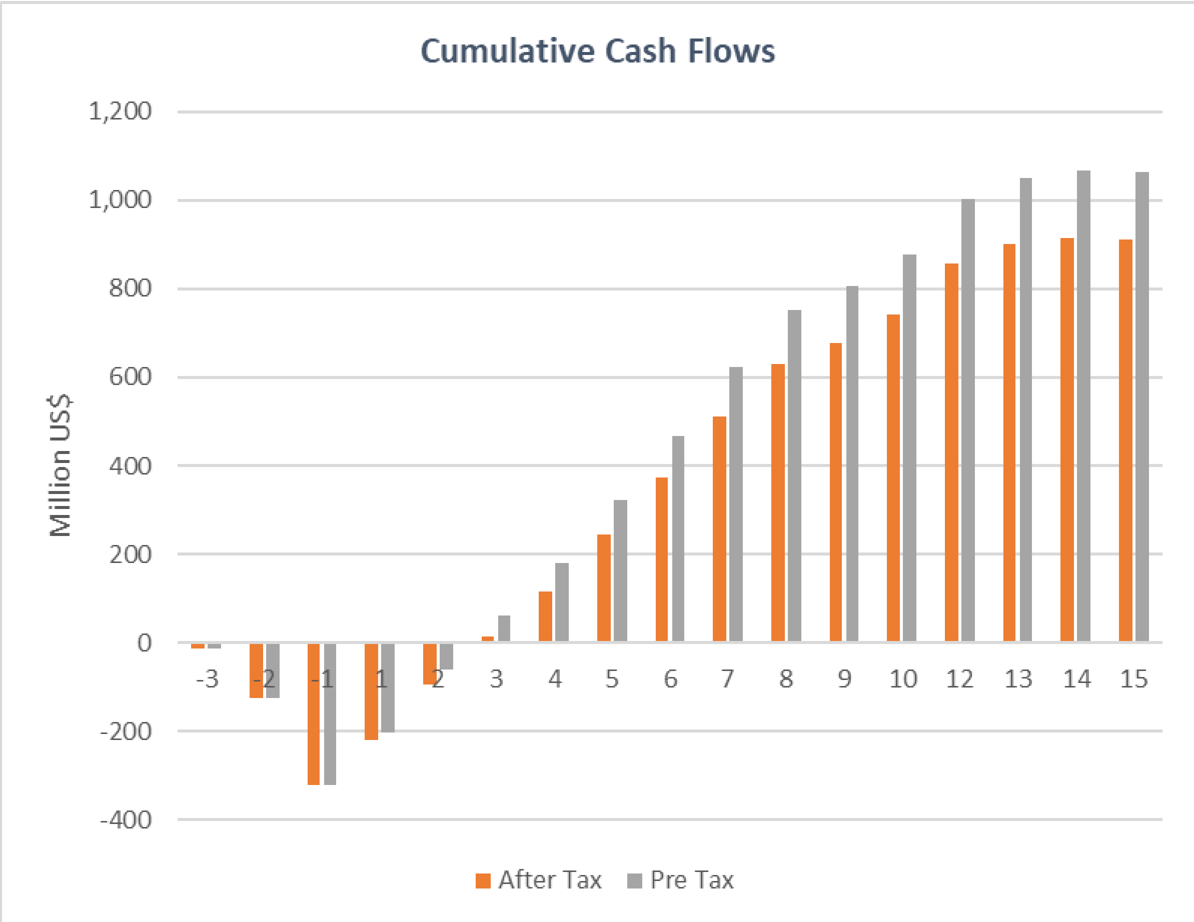


Table 22.4 Financial Model (Operations)

Block 14 - FS Study vs. 1s.xlsx			FS 11a	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Mining																				
	Main Total Tonnes	'000 t	22,923.0			14.5	2,835.0	1,250.4	3,434.0	5,908.1	5,068.3	3,896.5	516.1	-	-	-	-	-	-	-
	Main Total Grade	Au g/t	1.28			0.68	1.24	1.52	1.20	1.13	1.35	1.37	1.87	-	-	-	-	-	-	-
	Main Total Grade	Ag g/t	1.84			0.72	1.54	2.16	2.02	1.81	1.78	1.81	2.56	-	-	-	-	-	-	-
	East Total Tonnes	'000 t	50,266.9			205.8	3,303.4	7,454.8	5,902.9	3,818.3	6,769.8	6,018.2	12,164.9	4,628.8	-	-	-	-	-	-
	East Total Grade	Au g/t	0.99			0.80	0.93	0.93	0.96	0.91	0.95	0.96	1.06	1.18	-	-	-	-	-	-
	East Total Grade	Ag g/t	1.10			0.60	0.78	0.83	1.15	0.97	1.11	1.11	1.25	1.43	-	-	-	-	-	-
	NEZ Total Tonnes	'000 t	4,165.7			10.3	797.5	530.2	-	0.9	194.6	2,261.8	370.5	-	-	-	-	-	-	-
	NEZ Total Grade	Au g/t	0.850			0.97	0.85	0.75	-	0.40	1.87	0.80	0.78	-	-	-	-	-	-	-
	NEZ Total Grade	Ag g/t	0.79			0.51	0.54	0.57	-	0.28	0.98	0.81	1.38	-	-	-	-	-	-	-
	WD Total Tonnes	'000 t	2,587.6			120.0	528.9	600.0	600.0	600.0	138.6	-	-	-	-	-	-	-	-	-
	WD Total Grade	Au g/t	2.36			2.51	1.85	2.38	2.37	2.73	2.55	-	-	-	-	-	-	-	-	-
	WD Total Grade	Ag g/t	5.69			3.89	2.69	5.60	6.63	7.15	8.62	-	-	-	-	-	-	-	-	-
	Total Ore Tonnes	'000 t	79,943.2			350.6	7,464.9	9,835.4	9,936.9	10,327.2	12,171.3	12,176.5	13,051.6	4,628.8	-	-	-	-	-	-
	Total Ore Grade	Au g/t	1.11			1.38	1.10	1.08	1.13	1.14	1.15	1.06	1.08	1.18	-	-	-	-	-	-
	Total Ore Grade	Ag g/t	1.44			1.73	1.18	1.27	1.78	1.81	1.48	1.28	1.30	1.43	-	-	-	-	-	-
	Total Waste Tonnes	'000 t	119,248.1			936	14,252	14,987	19,256	24,521	21,651	14,234	8,088	1,322	-	-	-	-	-	-
	Total Ore Tonnes	'000 t	79,943.2			351	7,465	9,835	9,937	10,327	12,171	12,176	13,052	4,629	-	-	-	-	-	-
	Global Strip Ratio	W:O	1.49			2.7	1.9	1.5	1.9	2.4	1.8	1.2	0.6	0.3	-	-	-	-	-	-
						1,287	21,717	24,823	29,193	34,848	33,822	26,411	21,140	5,951	-	-				
Processing																				
	Stockpile Rehandle Tonnes	'000 t	40,148.6			-	533	449	375	716	392	307	578	3,457	6,000	6,000	6,000	6,000	6,000	3,343
	Oxide Tonnes Processed	'000 t	14,782.8				3778.5	2577.4	1250.4	667.4	377.8	444.7	135.3	836.7	105.4		2.8	26.2	1580.1	3000.0
	Oxide Grade Processed	Au g/t	1.03				1.4	1.4	1.2	1.0	1.2	1.0	0.8	0.7	0.8		0.6	0.5	0.5	0.5
	Oxide Grade Processed	Ag g/t	1.19				1.6	1.6	1.7	0.9	0.8	1.6	1.7	0.8	0.6		1.4	0.5	0.6	0.6
	Trans Tonnes Processed	'000 t	18,635.3				878.5	2576.7	3113.5	2363.0	665.0	1030.7	412.1	224.6				2608.1	4419.9	343.1
	Trans Grade Processed	Au g/t	0.98				1.3	1.3	1.4	1.2	1.3	1.0	0.9	0.8				0.6	0.6	0.6
	Trans Grade Processed	Ag g/t	1.11				1.2	1.2	2.0	1.6	1.0	0.7	0.7	1.3				0.4	0.7	0.7
	Fresh Tonnes Processed	'000 t	46,525.1				143.0	645.9	1636.1	2969.6	4957.1	4524.6	5452.5	4938.7	5894.6	6000.0	5997.2	3365.7		0.0
	Fresh Grade Processed	Au g/t	1.19				1.7	2.2	1.7	1.9	1.7	1.6	1.6	1.2	0.7	0.7	0.7	0.7		0.6
	Fresh Grade Processed	Ag g/t	1.66				2.2	4.8	3.6	3.4	2.4	2.0	1.8	1.5	0.9	0.9	0.9	0.9		0.6
	Total Tonnes Processed	'000 t	79,943.2				4,800.0	5,800.0	6,000.0	6,000.0	6,000.0	6,000.0	6,000.0	6,000.0	6,000.0	6,000.0	6,000.0	6,000.0	6,000.0	3,343.1
	Total Grade Processed	Au g/t	1.11				1.43	1.44	1.45	1.51	1.64	1.48	1.49	1.14	0.74	0.68	0.68	0.62	0.55	0.49
	Total Grade Processed	Ag g/t	1.44				1.57	1.79	2.38	2.43	2.15	1.72	1.69	1.39	0.87	0.95	0.95	0.72	0.69	0.59
	Total Recovered Au ozs	'000 ozs	2,341				193	229	230	235	251	227	232	175	116	109	109	100	89	47
	Total Recovered Ag ozs	'000 ozs	1,195				26	79	122	164	168	129	124	100	60	69	68	53	27	6

Table 22.5 Financial Model (Operations)

Block 14 - FS Study vs. 2d.xlsx		FS 11a	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Description	Unit	Total																		
Capital Costs		Early Works																		
Pre-Production CapEx		Yes																		
Construction Total	US$ '000	321,222	12,216	112,333	196,673															
Sustaining CapEx																				
Sustaining Total	US$ '000	178,590	-		-	18,816	22,369	31,761	22,966	17,611	6,049	6,827	3,500	21,094	3,500	7,696	-	3,855	9,247	3,300
		70,212																		
Profit/Loss		499,812																		
add: **Revenue**																				
Gold Revenue	US$ '000	3,144,436			-	258,855	307,236	308,289	315,155	337,221	304,895	311,335	234,933	155,680	146,601	146,509	134,831	119,590	63,305	-
Silver Revenue	US$ '000	22,818			-	503	1,500	2,321	3,129	3,210	2,467	2,376	1,910	1,154	1,317	1,296	1,008	511	115	-
Revenue	US$ '000	3,167,254			-	259,359	308,736	310,610	318,284	340,432	307,362	313,711	236,843	156,834	147,917	147,806	135,839	120,101	63,420	-
less: **Selling Costs**																				
Selling Costs	US$ '000	(14,144)			-	(876)	(1,229)	(1,404)	(1,594)	(1,677)	(1,425)	(1,425)	(1,100)	(705)	(712)	(708)	(613)	(463)	(213)	-
less: **Royalties**																				
Royalties	US$ '000	(221,708)			-	(18,155)	(21,612)	(21,743)	(22,280)	(23,830)	(21,515)	(21,960)	(16,579)	(10,978)	(10,354)	(10,346)	(9,509)	(8,407)	(4,439)	-
less: **Op Costs - Mining**																				
Op Costs - Mining	US$ '000	(549,366)			(4,938)	(55,945)	(63,551)	(75,030)	(95,397)	(95,560)	(73,939)	(65,577)	(19,754)	58	58	58	58	58	32	-
Stockpile Rehandle	US$ '000	(34,929)			-	(464)	(391)	(326)	(623)	(341)	(267)	(503)	(3,008)	(5,220)	(5,220)	(5,220)	(5,220)	(5,220)	(2,909)	-
Op Costs - Process (Fixed)																				
Op Costs - Process (Fixed)	US$ '000	(133,776)				(9,868)	(9,868)	(9,868)	(9,868)	(9,868)	(9,868)	(9,868)	(9,868)	(9,868)	(9,868)	(9,868)	(9,868)	(9,868)	(5,498)	-
Op Costs - Process (Variable)																				
Op Costs - Process (Variable)	US$ '000	(518,247)				(27,885)	(35,018)	(38,425)	(38,355)	(39,485)	(39,483)	(42,035)	(43,373)	(44,672)	(37,666)	(38,371)	(39,573)	(35,240)	(18,665)	-
Op Costs - Process (Fix & Var	US$ '000	(652,023)				(37,752)	(44,885)	(48,293)	(48,222)	(49,353)	(49,351)	(51,903)	(53,241)	(54,539)	(47,534)	(48,239)	(49,440)	(45,108)	(24,163)	-
		8.16				7.87	7.74	8.05	8.04	8.23	8.23	8.65	8.87	9.09	7.92	8.04	8.24	7.52	7.23	-
less: **Op Costs - G&A**																				
Administration Labour	US$ '000	41,350				3,050	3,050	3,050	3,050	3,050	3,050	3,050	3,050	3,050	3,050	3,050	3,050	3,050	1,699	-
General & Administration Co	US$ '000	95,707				7,059	7,059	7,059	7,059	7,059	7,059	7,059	7,059	7,059	7,059	7,059	7,059	7,059	3,933	-
Op Costs - G&A	US$ '000	(137,057)				(10,110)	(10,110)	(10,110)	(10,110)	(10,110)	(10,110)	(10,110)	(10,110)	(10,110)	(10,110)	(10,110)	(10,110)	(10,110)	(5,633)	-
equals: **Operating Profit**	US$ '000	**1,562,965**				**136,057**	**166,959**	**153,705**	**140,059**	**159,561**	**150,756**	**162,235**	**133,052**	**75,340**	**74,045**	**73,242**	**61,006**	**50,852**	**26,096**	**-**
Net Cash Flows, before tax	US$ '000	**1,063,153**	**(12,216)**	(112,333)	(196,673)	**117,242**	**144,590**	**121,944**	**117,093**	**141,951**	**144,707**	**155,408**	**129,552**	**54,246**	**70,545**	**65,546**	**61,006**	**46,997**	**16,849**	**(3,300)**
NPV Pre Tax	US$ '000	**723,008**																		
IRR Pre Tax		**38.5%**																		
Operating Profit	US$ '000	**1,562,965**			-	**136,057**	**166,959**	**153,705**	**140,059**	**159,561**	**150,756**	**162,235**	**133,052**	**75,340**	**74,045**	**73,242**	**61,006**	**50,852**	**26,096**	**-**
Depreciation		**(513,187)**			-	**(33,008)**	**(41,186)**	**(44,706)**	**(48,404)**	**(54,228)**	**(49,510)**	**(51,345)**	**(38,745)**	**(29,248)**	**(27,542)**	**(28,849)**	**(26,549)**	**(26,069)**	**(13,800)**	**-**
Other Sustaining		**(47,546)**				**-**	**(3,500)**	**(3,500)**	**(3,500)**	**(3,500)**	**(3,500)**	**(3,500)**	**(3,500)**	**(3,500)**	**(3,500)**	**(3,500)**	**-**	**-**	**(9,246)**	**(3,300)**
equals: **Taxable Profit**	US$ '000	**1,002,231**				**103,050**	**122,273**	**105,498**	**88,155**	**101,834**	**97,746**	**107,390**	**90,807**	**42,592**	**43,004**	**40,893**	**34,457**	**24,783**	**3,050**	**(3,300)**
Tax Rate	15%																			
Taxes Payable	US$ '000	**(150,830)**				**(15,457)**	**(18,341)**	**(15,825)**	**(13,223)**	**(15,275)**	**(14,662)**	**(16,109)**	**(13,621)**	**(6,389)**	**(6,451)**	**(6,134)**	**(5,168)**	**(3,717)**	**(457)**	**-**
Net Cash Flows, after tax	US$ '000	**912,324**	**(12,216)**	**(112,333)**	**(196,673)**	**101,784**	**126,249**	**106,119**	**103,870**	**126,676**	**130,045**	**139,299**	**115,931**	**47,857**	**64,095**	**59,412**	**55,838**	**43,279**	**16,392**	**(3,300)**
NPV After Tax	US$ '000	**606,781**																		
Discount																				
IRR After Tax		**33.3%**																		
Cash cost per ounce Au	US$	**676**				**637**	**613**	**674**	**746**	**708**	**679**	**643**	**583**	**693**	**665**	**672**	**735**	**772**	**790**	
All-in sustaining cost per ounce Au	US$	**751**				**738**	**714**	**815**	**847**	**781**	**709**	**676**	**606**	**879**	**701**	**747**	**740**	**820**	**797**	
Payback period - Pre Tax	Years	**2.5**																		
Payback period - After Tax	Years	**2.9**																		

22.4 Financial Summary

The results of the financial model are summarized in Table 22.6.

Revenue generated per domain is shown in Figure 22.2; where total oxide provides 19%, total transition provides 21% and total fresh provides 60% of the total revenue.

A breakdown of the total cash costs is shown in Figure 22.3. Table 22.6 shows the breakdown of the LOM cash costs and unit costs per tonne processed.

Table 22.6 Financial Model Summary

Description	Units	LOM
Feed Tonnage	Mt	79.9
Waste Rock	Mt	119.3
Total Mined	Mt	199.2
Stripping Ratio	W:O	1.49:1
Feed Grade Processed (average)	g/t	1.11
Gold Recovery (average)	%	82.0
Production Period	Years	13.6
Gold Production	'000 oz	2,341
Annual Gold Production (Yr 1-7)	'000 oz/y	228
Annual Gold Production (average)	'000 oz/y	167
Silver Production	'000 oz	1,195
Pre-production Capital Cost	US$M	321
Sustaining Capital Cost	US$M	179
Total Capital Cost	US$M	500
Gross Revenue	US$M	3,167
Total Operating Costs	US$M	1,604
Operating Profit	US$M	1,563
Mining Cost / t Mined	US$ /t	2.67
Process Cost / t Processed	US$ /t	8.16
Cash Cost / oz Au recovered	US$ /oz	676
AISC / oz Au recovered	US$ /oz	751

Figure 22.2 Revenue Generated per Material Type

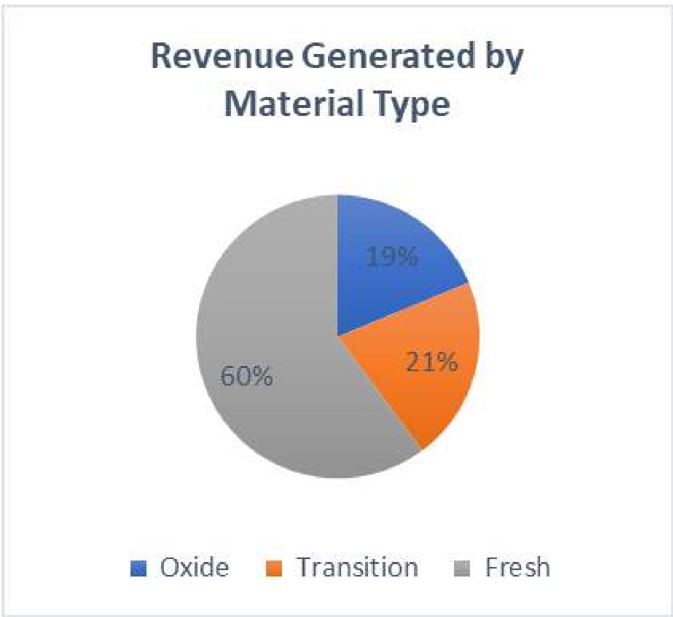


Figure 22.3 Operating Expense Split

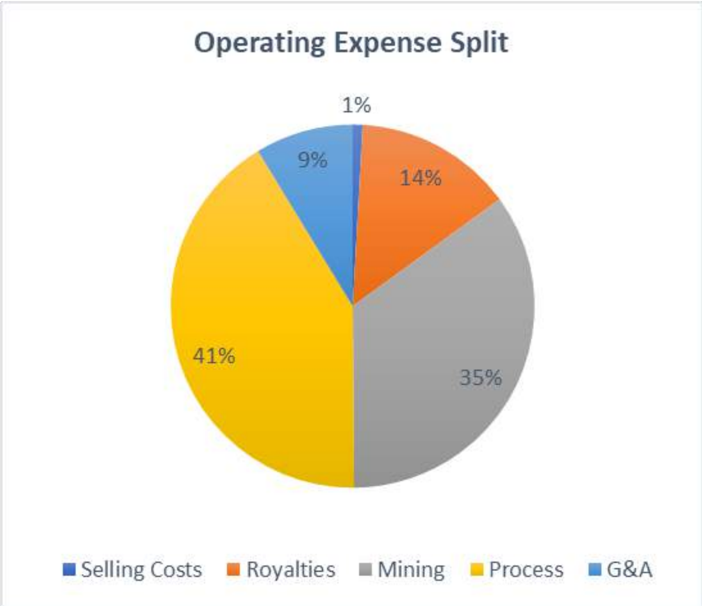


Table 22.7 Cash Cost and Unit Cost Summary

Description	LOM ($/oz)	LOM ($/t processed)
Silver Credit	(10)	(0.29)
Mining	233	6.81
SP Rehandle	15	0.44
Processing	279	8.16
G&A	59	1.71
Refining	6	0.18
Royalties	95	2.77
Total Cash Cost	676	19.78
Sustaining Capital	71	2.08
Closure	4	0.13
All-in Sustaining Costs	751	21.99

22.5 Single Parameter Sensitivities

Figure 22.4 shows the changing post-tax $NPV_{5\%}$ and IRR for varying single parameter sensitivities for revenue, pre-production and sustaining capital costs, mining, plant and G&A operating costs and revenue / gold recovery. Figure 22.4 also shows the post-tax IRR sensitivity to parameters that the NPV is most sensitive revenue / recovery.

Figure 22.4 NPV and IRR Sensitivity

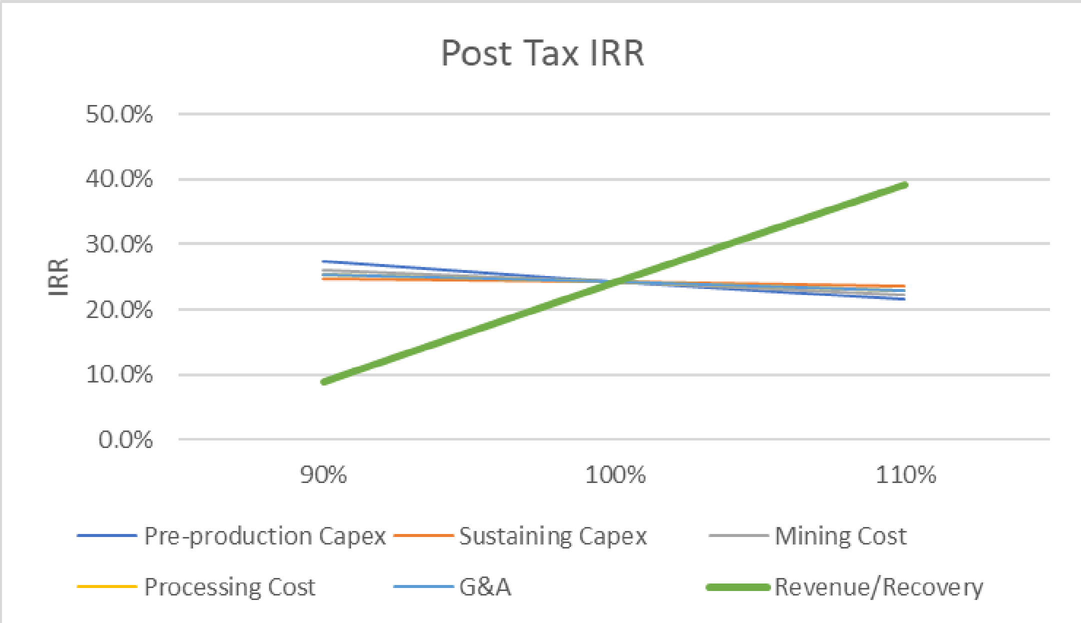


Table 22.8 shows the sensitivity of the NPV and IRR with gold price and discount rate.

Table 22.8 Gold Price NPV and IRR Sensitivity

NPV	Low	Base	3Yr Trailing		Spot[1]
Discount	1,250	1,350	1,410	1,750	1,950
5%	467	607	691	1,166	1,445
6%	427	560	639	1,090	1,356
7%	390	516	592	1,021	1,273

IRR	1,250	1,350	1,410	1,750	1,950
	27.5	33.3	36.8%	55.4%	66.0%

[1] 31 August, 2020

Fuel prices used LNG - US$9.70 /MBTU and diesel US$0.60 /l. The impact of 10% change in fuel price on the NPV and IRR is shown in Table 22.9 .

Table 22.9 Fuel Price NPV and IRR Sensitivity

	-10%	Base	+10%
LNG Price $/MBTU	8.73	9.70	10.67
Diesel (DFO) Price $/l	0.54	0.60	0.66
$NPV_{5\%}$	623	605	586
IRR	34.0%	33.3%	32,6%

23. ADJACENT PROPERTIES

Managem International operate a pilot plant scale operation 100 km south of Block 14 at Gabgaba and are known to be exploring for gold mineralization in the Block 15 exploration licence. Resources are said to be in the range of 6 Moz gold but no data on the project is published.

Tahe Minerals are operating a small gold mine 200 km to the west of Block 14 in the Wadi Halfa area. There is no publicly available information on the project.

24. OTHER RELEVANT DATA AND INFORMATION

There is no additional information or explanation required in order to make this report understandable and not misleading; all relevant information has been summarized in the Feasibility Report and Appendices.

25. INTERPRETATION AND CONCLUSIONS

25.1 Mining

The mining of the GSS and WD deposits has been shown to be technically feasible through conventional open pit mining methods. Pit optimizations show that both deposits have economically viable material under the assumed economic and physical parameters.

A total of 79.9 Mt of crusher feed at a gold grade of 1.11 g/t is contained within nine pits across the two deposits, along with 119.2 Mt of waste, resulting in a strip ratio of 1.49:1.

A combined mining schedule demonstrates that a processing rate of 6.0 Mt can be sustained for 14 years, including an initial ramp up period in Years 1 and Year 2. A pre-production period of six months is utilized to strip waste from both deposits and stockpile minor amounts of crusher feed, ready for the commissioning of the processing plant. The pit at WD is mined for the first five years and blended with material from GSS. An elevated cut off grade over eight years maximises early returns by blending the significantly higher-grade material from the WD deposit and gain access to the higher grades at depth in the Main and East pits. The last six years rehandle the low-grade stockpiles.

25.2 Mineral Processing and Recovery Methods

A comprehensive program of metallurgical testwork was conducted at SGS to identify the optimum processing route, generate recovery equations and provide data for plant design. Mineralogy, comminution, leach, carbon modelling, thickening and rheology tests were conducted on a large range of ore lithologies and domains.

The Block 14 ores are amenable to a conventional gold processing scheme incorporating crushing, SAG and ball milling, cyanide leach, carbon adsorption, elution, electrowinning and gold smelting. The comminution circuit and gold recovery plant design is based on a throughput rate of nominally 6.0 Mtpa.

Trade-off studies have identified that there is economic benefit in including a pre-aeration stage ahead of cyanide leaching to reduce cyanide consumption rates and that the Kemix Pumpcell CIP circuit provides the optimum carbon adsorption circuit design.

The proposed process route is a proven and robust concept with very little associated risks, apart from the typical operational hazards stemming from the reagents that are used in the process. The plant design will take cognisance of this fact and good engineering practise and industry standards will be applied to design a safe operating plant.

25.3 Major Infrastructure

25.3.1 Selected Water Supply Option

Seventeen exploration boreholes and four test production holes have been drilled within the Area 5 aquifer, which indicated high and constant airlift yields with the occurrence of groundwater within an inter-layered basin type sequence of course to medium grained sandstone.

A numerical model was constructed using Visual Modflow Pro. A conceptual well field design was used to evaluate the aquifer potential. The aquifer was pumped for a period of 10, 12 and 15 years with positive results.

A classification methodology prescribed by Green, et al, 2005, has been used to evaluate the Water Resource. The aquifer contains 100 Mm^3 in the Measured Resource category and a further 40 Mm^3 in the Indicated Resource category.

25.3.2 Selected Power Supply Option

Based on the trade-off carried out, self-generation using LNGis the preferred option with the least associated risk.

25.3.3 Roads

Surface haul roads have been designed with surface haulage distances estimated for the various deposits to allow for the calculation of mining costs.

No additional site access road networks or off-site infrastructure will be developed for this project from Abu Hamad (the closest main town to the project area) due to the use of desert tracks that are capable of handling the loads from heavy vehicles.

25.4 Environmental

There are currently no objections to the development of the Project. There are no receptors in the area, with no officially recognised human settlements in proximity. The Project has undertaken an environmental assessment and data collection is continuing. The arid conditions and sparse vegetation of the concession provide a difficult environment, which is primarily nocturnal. The use of remote infrared cameras provides the opportunity to record these nocturnal animals. Other wildlife records are captured through daily observations, to the extent possible. Climate data and weather data has been collected and compared, to provide reliable data for the EIA and design teams. Water data from the existing boreholes and Talat Abda well has been collected and there are no other known sources of potable water and few potential water users who legally reside in the vicinity. Social data continues to be collected, through monitoring of artisanal miner's activities. All information collected will be used in managing social and environmental impacts associated with the Project.

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 mining activities, including exploration, in such a way as to mitigate the negative impacts. MSMCL have initiated environmental measures through their Exploration Statement, which is specific to exploration work and includes an EPMP to mitigate impacts associated with the work. In accordance with the Concession Agreement, the EPMP is a dynamic document that is revised and updated as the Project progresses.

The EIA has determined that while the development of the Project may give rise to some environmental impacts, these can be minimised. Project development and operations will cause several changes to some aspects of the local environment, e.g. visual, air and water quality, use of water resources, removal of wildlife habitat among others. The absence of any current social habitat in the concession means that social impact would be minimal and restricted to within the mine complex itself. Furthermore, the implementation of mitigation measures, as proposed in the EIA, will result in any changes being considered manageable and controllable. Therefore, the development, operation and closure of the Project could be undertaken in an effective environmental and social manner.

25.5 Economic Analysis

Based on a gold price of US$1,350 /oz, the Project has a pre-tax IRR of 38.5% and pre-tax $NPV_{5\%}$ of US$723M. The Project has a post-tax IRR of 33.3% and a post-tax $NPV_{5\%}$ of US$607M.

The Pre-tax undiscounted payback period is 2.5 years and Post-tax, undiscounted payback period is 2.9 years from start of production.

The NPV and IRR are most sensitive to the metal price / recovery, pre-production capital costs and plant and mine operating cost.

25.6 Risks and Opportunities

25.6.1 Risks

The following risks were identified; a formal Risk Response Plan will be developed during the Definitive Feasibility Study to mitigate these risks:

- Mining contract rates may be higher than expected given the remote location.
- Skilled labour will be difficult to source in the local area, meaning that initially expatriate operators and trainers will be required, although experienced operators can be found in Egypt. This has the possibility of increasing contract mining rates and plant operating personnel for the first several years.
- Due to the location of the plant, transport and logistics of fuel and reagents might be problematic and sufficient buffer capacity and storage will be designed in the event of logistical problems and non-delivery of items to the project.
- The current Direct Capital Cost estimate has been shown to be within a level of accuracy range of ±15 %. The risk is that the direct cost might increase by 15% once more detail engineering and development is completed. Based on Figure 22.4 it can be seen that the project will be able to absorb this increase at the current (September 2020) metal spot price.

25.6.2 Opportunities

Mining

Material haulage from Wadi Doum could be improved if long distance haulage units which can be loaded at the face are utilized. This would negate the need for ROM stockpiles and re-handling of the crusher feed, potentially saving US$0.40 – 0.50 /t of crusher feed from Wadi Doum.

A review of the hydrogeology will lead to a greater understanding of the effects of groundwater on the pit walls (if any) and could allow for further steepening.

Mineral Processing

One opportunity has the potential to reduce operating costs or improve recovery and will be investigated with additional test work:

- Undertake an economic trade-off / optimization study on increasing silver recoveries by increasing carbon movement.

Power Supply

Alternative energy sources can be investigated in future project phases including solarPV. A trade-off between capital cost and operating cost savings should be carried out. Currently power costs account for 3% of the plant operating costs.

Tailings

There is the potential to construct the starter embankment from overhauled Run of Mine waste if the mining schedule and truck availability permits. This could reduce the cost of the initial construction of the tailings storage facility.

26. RECOMMENDATIONS

26.1 General

The Feasibility Study has demonstrated a strong project. During the period of financing a range of early works will reduce the timeframe required between project award and project completion for a relatively minor investment.

Lycopodium would carry out carry out early works engineering services covering:

- Preparation of Engineering Specfications.
- Complete engineering design for the water pipeline.
- Geotechincal investigations.

ISI Prime would complete the power station design to enable the long lead item specifications to be completed.

26.2 Additional Recommendations

26.2.1 Environmental

By initiating the EIA process early, results could be used to improve the design, as well as maximising the benefits of the EIA without incurring excessive costs. In accordance to continual improvement processes, there are several strategies that can be used to support the Project, such as:

- Ongoing monitoring of wildlife presence in the Project area, such that management measures can be adapted to reflect changing conditions.
- Further to the improvement to carbon emissions as a result of using LNG, the potential for renewable power supplies is to be examined, as renewable technology continues to improve
- Maintain a grievance procedure to identify and pre-empt potential issues.

26.2.2 Mining

- Complete hydrogeological study for GSS to confirm presence, volume and effects of ground water and follow up with updated geotechnical review.
- Review and revise stockpile management practices to allow better grade definition within stockpiles.

26.2.3 Mineral Processing and Metallurgical Testing

- Conduct a Trade-Off Study to ascertain the benefits of increasing silver adsorption stage efficiencies onto carbon by moving more carbon per day to reduce gold loadings, which will allow more silver to adsorb.

26.2.4 Water Supply

- Further work to confirm potential recharge of groundwater will include isotope analyses and additional water quality analyses.
- Long duration 30-day pump tests to be completed to act as AST (aquifer stress tests) and improve calibration of the numerical groundwater model.

26.2.5 Tailings Storage Facility

- Complete a geophysical survey (or geotechnical investigations) along the alignment of the main embankment to confirm the depth of aeolian sand below the embankment footprint to allow refinement in the location of drainage systems and the cut-off trench design.
- Conduct geological mapping of the TSF basin to define the main structural features within the basin to ensure they are accounted for in the detailed design.
- Conduct additional drilling and packer testing to confirm the in-situ permeability and variability of the permeability of the foundation rocks in the areas not covered by the HDPE liner in the TSF.
- Complete any outstanding sterilisation drilling within the TSF basin to confirm that there are no exploration targets below the facility.
- Optimise the design of the TSF embankment to match the mining fleet to be utilised by the selected mining contractor to provide construction efficiencies.

26.2.6 Process Plant Geotechnical Design

- Complete geotechnical investigations and geotechnical design checks at the actual location of the key process infrastructure including the crusher, mills, tanks, and thickener.
- Finalise the concrete aggregate testing which has commenced to ensure suitable aggregate sources (both fine and coarse) have been located and develop and test potential concrete mix designs for the early construction works.
- Confirm that the earthworks contractor has allowed for appropriate equipment to crush, screen and wash the concrete aggregates to meet the required specifications.

27. REFERENCES

i. Ahmed et al, 2000, Keraf Shear Zone, North East Sudan: Geodynamic Characteristics of the Nile Craton-Nubian Shield Boundary. PhD Thesis, technical University of Berlin, Unpub, 141pp.

ii. Arab News, 2017. News article “Sudan aims to increase gold production in 2018”, published 28 Dec 2017, www.arabnews.com/node/1215661/business-economy

iii. Johnson, P.R., Andresen, A., Collins, A.S., Fowler, A.R., Fritz, H., Ghebreab, W., Kusky, T. and Stern, R.J., 2011. Late Cryogenian-Ediacaran history of the Arabian-Nubian Shield: a review of deposition, plutonic, structural and tectonic events in the closing stages of the northern East African Orogen. Journal of African Earth Sciences, 61, 167-232.

iv. Davies, B.M, McDonnell, L. and Salih, M. 2017 Galat Sufar South – an update on trap site environment and district targeting. Renaissance Geology, Unpub. Report for Orca Gold Inc., 54pp.

v. Mason, R., November 2017, Interpretation of structure at Galat Sufar South, from field

vi. mapping and drill core data. Tectonite Geology, Unpub. Report for Orca Gold Inc., 48pp.

vii. Seago, R., 2017, Structural Appraisal of the Planned Open Pit Areas.

viii. Galat Sufar South and Wadi Doum, Block 14, N Sudan. SRK Consulting, Unpub. Report for Orca GoldI Inc, 46pp.

ix. GCS (2013). Initial Hydrogeological Assessment, Orca Gold - Sudan - Galat Sufar South (GSS) Project –Block 14 Area.

x. GCS (2016). Follow up groundwater exploration for the Orca Gold - Sudan - Galat Sufar South (GSS) Project –Block 14 Area.

xi. GCS (2018) - Groundwater Exploration, Groundwater Resource Estimation and Preliminary Well Field Design for the Galat Sufar South Project, Sudan – Block 14 Area.

xii. Hydrogeology in the Service of Man, Mémoires of the 18th Congress of. The International Association of Hydrogeologists, Cambridge, 1985. Re-evaluation of Hydrogeological and Hydrochemical Aspects for Nubian Aquifer in Sudan. H.H. ELKADI1 and M.E. ABD ELSHAFI Faculty of Earth Sciences, King Abdulaziz University, Jedda, Saudi Arabia, 2 Rural Water Corporation, Sudan.

xiii. Nov. 2014. Vol. 2. No.7 ISSN 2311-2484, International Journal of Research In Earth & Environmental Sciences 2013- 2014 IJREES & K.A.J. Evaluation of Aquifer Characeristics for Selected New Method of the Um Ruwaba Formation: North Kordofan State, Sudan. Elhaga.B *1; ELZIENS.M*2 ANDLISSANN.H*3, *1Department of Civil Engineering, College of Engineering, King Khalid University, Saudi, *2 Faculty of Petroleum & Minerals, Alneelain University, Sudan, *3 Faculty of Earth Sciences and mining, University of Dongola, Sudan.

xiv. SRK (2017), “Geotechnical Feasibility Study of the Block 14 project, Sudan”

xv. Cube Consulting (2018), “Orca Gold – LOM Schedule and Stockpile grade range analysis”

xvi. Knight Piésold (2018), “Tailings Managment Design, Definitive Feasibility Study Report”

28. QP CERTIFICATES

I **Geoffrey Alexander Duckworth** hereby state:-

1. I am employed as a Senior Consultant – Process, with the firm Lycopodium Minerals Pty Ltd, 153-163 Liechardt Street, Spring Hill, Queensland 4000 Australia, specialising in minerals processing engineering consulting, contracting for a wide variety of clients, and have been so employed since 2008.
2. This certificate applies to the technical report with an effective date of 31st August 2020, and titled "Feasibility Study, NI 43-101 Technical Report - Block 14 Project, Republic of Sudan".
3. I am a practising Metallurgical Engineer and registered Fellow of the Australasian Institute of Mining and Metallurgy.
4. I am a graduate of the Royal Melbourne Institute of Technology with a Bachelor of Engineering (Chemical) 1974 and post graduate of the University of Queensland Australia with a M Eng Sc degree (1979) and PhD (1982), both in Mining and Metallurgy. I have practiced my profession continuously since 1974.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I visited the Galat Sufar South Project between 3rd October and 9th October 2016. The purpose of the visit was to review site location, access, infrastructure requirements and allocation of layout areas for the project.
7. I am responsible for sections 1.16, 1.17, 1.20, 1.21, 1.23, 1.24, 17, 18.2, 18.4, 18.5, 18.6, 21, 22, 25 and 26.
8. I am independent of Orca pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Orca or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

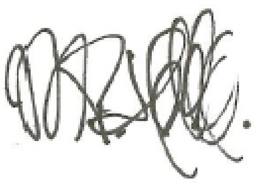
Dated this 8th September 2020

Geoffrey Alexander Duckworth

I **Michael Peter Hallewell** hereby state:-

1. I am a consulting Metallurgist, with the firm of MPH Minerals Consultancy Ltd, 8 The Gluyas, Falmouth, Cornwall, TR11 4SE.
2. This certificate applies to the technical report with an effective date of 31st August 2020, and titled "Revised Feasibility Study, NI 43-101 Technical Report - Block 14 Project, Republic of Sudan".
3. I am a practising Metallurgical Consultant and 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.
4. I am a graduate with a B.Sc (Engineering) degree in Minerals Engineering from the University of Birmingham, UK. I have 37 years practical experience in Minerals Processing as Plant Manager, Consulting or Senior Metallurgist in precious metals, base metals and ferrous metals industry.
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 have not visited the project site.
7. I am responsible for Sections 1.12, 13, and 25.6.2 of the report.
8. I am independent of the issuer as described in section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Orca or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 8th day of September 2020



Michael Peter Hallewell

MPR Geological Consultants Pty Ltd

ABN: 98 152 948 957

I **Nicholas James Johnson** hereby state:

1. I am a consulting Geologist, with the firm of MPR Geological Consultants Pty Ltd, 19/123A Colin Street, West Perth, WA 6005, Australia.
2. This certificate applies to the technical report with an effective date of 14th September 2020, and titled "Revised Feasibility Study, NI 43-101 Technical Report - Block 14 Project, Republic of Sudan".
3. I am a practising a practising Geologist and registered Member of the Australian Institute of Geoscientists.
4. I am a graduate of the Latrobe University, Melbourne, Australia with a Bachelor of Science (Honours) degree in Geology (1988). I have practiced my profession continuously since 1988.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I visited the Galat Sufar South Project between 17th January and 21st January 2014. The purpose of the visit was to review the exploration practices and project geology.
7. I am responsible for sections 1.13 and 14.
8. I am independent of Orca pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Orca or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th day of September 2020 at Perth Australia.

Nicolas Johnson
Consulting Geologist
MPR Geological Consultants Pty Ltd

I **Pieter Ferdinandus Labuschagne** hereby state:-

1. I am a consulting Hydrogeologist, with the firm of GCS (Pty) Ltd and situated in the Durban office in South Africa (4A Old Main Road, Kloof, 3610).
2. This certificate applies to the technical report with an effective date of 31st August 2020, and titled "Revised Feasibility Study, NI 43-101 Technical Report - Block 14 Project, Republic of Sudan".
3. I am a practising Hydrogeologist and registered Member of the South African Council for Natural Scientific Professions – SACNASP (Pr.Sci.Nat.400386/11).
4. I am a graduate of the University of the Free State, Bloemfontein, South Africa with a Master's of Science degree in Hydrogeology (2004). I have practiced my profession continuously since 1998.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I visited the Galat Sufar South Project between 23 November and 2 December 2016. The purpose of the visit was to review the groundwater exploration drilling and aquifer testing.
7. I am responsible for sections 1.9.1, 18.1 and 26.2.4.
8. I am independent of Orca pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Orca or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 7th day of September 2020 at Kloof, South Africa.

Pieter Ferdinandus Labuschagne

I Carl Steven Nicholas hereby state:-

1. I am a Chartered Environmental Consultant, with the company of Mineesia Limited, 4 Mace Farm, Cudham, Kent, TN14 7QN, UK.
2. This certificate applies to the technical report with an effective date of 31st August 2020, and titled "Revised Feasibility Study, NI 43-101 Technical Report - Block 14 Project, Republic of Sudan".
3. I am a practising Environmental Consultant and registered Member of the Institute of Materials, Minerals and Mining.
4. I am a graduate of Imperial College, London, UK with a Masters in Environmental Diagnosis, with a Bachelor of Science (Honours) degree in Biodiversity Conservation and Environmental Management. I have practiced my profession continuously since 2005, and have 15 years practical experience in Environmental Impact Assessments for mining projects.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
6. I visited the Galat Sufar South Project between 24th November and 3rd December 2014. The purpose of the visit was to review the baseline conditions and establish priorities for environmental management for the project,
7. I am responsible for sections 1.19, 20 and 25.4.
8. I am independent of Orca pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of Orca or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with N143-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this%ay of September 2020

Carl Steven Nicholas

I **Chris Reardon** hereby state:-

1. I am the Country Manager for Orca Gold Inc and live at 99 Kings Mill Way, Denham, UB9 4BT, England. (I was previously a Principle Mining Engineer, with the firm of Deswik Europe Ltd).
2. This certificate applies to the technical report with an effective date of 14th September 2020, and titled "Revised Feasibility Study, NI 43-101 Technical Report - Block 14 Project, Republic of Sudan".
3. I am a practising Mining Engineer with over 20 years of relevant experience in open pit mining operations, 14 of which have been in open pit gold mines.
4. I am a graduate of the University of Queensland, Australia with a Bachelor of Science degree in Geology (2004). I have practised my profession continuously since 2005.
5. I am a Member of the Austrailan Institute of Mining & Metallurgy and a Member of the I am a practising Metallurgical Consultant and 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.
6. 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.
7. I visited the Galat Sufar South Project between 3rd and 8th October 2016. The purpose of the visit was to review the site layout, geological setting and logistics associated with the development of an open pit mining operation. I have visited site on a number of occasions in 2018 and 2019.
8. I am responsible for sections 1.14, 1.15, 15, 16, 21.3, 21.5, 25.1, 25.6.1, 25.6.2 and 26.2.2.
9. I am not independent of Orca pursuant to Section 1.5 of NI 43-101.
10. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th September 2020 at Amersham, England.

C.AR

Chris Reardon

I Timothy Rowles hereby state:-

1. I am employed as Regional Manager, with the firm Knight Piésold Pty Limited Level 1, 36 Cordelia Street, Brisbane, QLD 4101, AUSTRALIA.
2. This certificate applies to the technical report with an effective date of 31st August 2020, and titled "Revised Feasibility Study, NI 43-101 Technical Report - Block 14 Project, Republic of Sudan".
3. I am a current Fellow of the Australian Institute of Mining and Metallurgy (Member Number 227249) and have been awarded the status of Chartered Professional (CP) in the field of Environmental Engineering.
4. I am a registered Professional Engineer Queensland (RPEQ) in good standing with the Board of Professional Engineers of Queensland in Australia (No 10166).
5. I am a graduate of the Royal School of Mines, Imperial College, London, with a Bachelor of Science in Environmental Geology in 1996.
6. I graduated from the University of Manchester with a Masters Degree in Earth and Environmental Science in 1998.
7. I have practiced my profession continuously since 1999.
8. 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.
9. I have visited the Block 14 project site between 5th and 7th October 2016.
10. I am responsible for sections 1.17.3, 18.3, 25.6.2, 26.2.5 and 26.2.6.
11. I am independent of Orca pursuant to Section 1.5 of NI 43-101.
12. I do not beneficially own, directly or indirectly, any securities of Orca or any associate or affiliate of such company.
13. I have not had prior involvement with the property that is the subject of the Technical Report.
14. 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.
15. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 07th September 2020

Timothy Rowles