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NI 43-101 Technical Report
Preliminary Feasibility Study

Segilola Gold Project
Osun State, Nigeria
Thor Explorations Ltd

Effective Date: 16th October, 2017

Qualified Persons:

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1. SUMMARY

1.1 Property Description and Ownership

The property comprises mining license ML41 and exploration license EL19066. ML41 previously covers an area of (17.2km^2; 1,720ha) and is wholly contained within the larger EL19066 covering an area of 135 Cadastral Units (27.0km^2; 2,700ha).

The Segilola Gold Project is 100% owned by Segilola Resources Operating Limited (SROL), a wholly owned subsidiary of Thor Explorations Ltd (Thor). It is fully permitted, with the mining licence renewed in September 2016 for a period of 25 years.

1.2 Geology and Mineralisation

The Project area is located in the crystalline basement complex rocks of southwestern Nigeria within the Upper Proterozoic rocks of the Ilesha schist belt which formed part of the Pan African mobile belt. At Segilola, gold mineralisation is localised within structural "compartments" defined by the intersection of two main controlling features: a westerly-dipping footwall calc-silicate suite of rocks and sub-vertical shear zones. Gold mineralisation is associated with stacked set of steep westerly-dipping, north-trending pegmatitic quartz-feldspar veins that intrude variably deformed gneissic rocks and schist. The vein system, which outcrops in places, extends over a strike length of about 2,000m and down dip to nearly 400m from surface. The gold itself is often coarse and visible in diamond core. There are opportunities to extend the known resource both along strike and down-dip.

1.3 Exploration and Drilling

The project is in the advanced exploration and resource development stages with nearly all past activities relating to the definition of the gold mineralisation now complete.

From April to August 2017, Thor completed a 15 hole diamond drilling program that was designed to test for potential in the areas external to the existing Indicated Mineral Resource. The program was successful in identifying four high grade down-dip trends beneath the Indicated Mineral Resource. Thor management believes that further work is justified to convert the substantial amount of Inferred Resource and Unclassified (unreported) material.

1.4 Mineral Processing and Recovery Methods

ROM ore will be delivered from the mine to the processing plant, which consists of a conventional crushing circuit and a single stage grinding circuit to achieve a target grind size P80 of 106 microns. The plant will operate on a 365 day/year, 24 hour/day operating cycle with a design plant availability of 91.3% for a nominal ore throughput of 62.5 tph resulting in a throughput rate of 500,000 tonnes per annum. Gold is to be extracted by conventional carbon in leach, to produce a gold dore via elution, electrowinning, and smelting.

Life of mine average gold recovery is estimated to be 96% resulting in life of mine production of 430,100 ounces from the currently stated Probable Reserves.

1.5 Mineral Resource Estimate

The Preliminary Feasibility Study (PFS) includes an updated Mineral Resource model completed by Auralia Mining Consulting Pty Ltd (Auralia). The Mineral Resource statement was previously announced by the Company (refer to Company press release dated September 11, 2017) and is shown in Table 1.

Table 1 – Mineral Resource Estimate reported using an optimised pit shell

Type	Within Whittle Shell (Open Pit Resource) (0.64g/t Au cut-off)			External to Whittle Shell (Underground Resource) (2.5g/t Au cut-off)			Total		
	Tonnes	Au g/t	Ounces	Tonnes	Au g/t	Ounces	Tonnes	Au g/t	Ounces
Indicated	3,926,000	4.3	539,000	111,000	4.7	17,000	4,037,000	4.3	556,000
Inferred	835,000	5.1	137,000	1,195,000	4.4	169,000	2,030,000	4.7	306,000

Thor's management believes that the Mineral Resources remain open to expansion and that there is an opportunity to improve the classification of the Inferred Mineral Resources with infill drilling. There is no certainty that such additional drilling will result in expanding or upgrading the classification of the Inferred Mineral Resources to Indicated category.

1.6 Mineral Reserve Estimate

The Mineral Reserve has been estimated in accordance with NI 43-101 guidelines, which excludes Inferred Resources, and are shown in Table 2.

Table 2 – Mineral Reserve

Classification	Cut off	Tonnes	Grade	Ounces
Probable	g/tAu	(t)	(g/tAu)	(Au)
	0.64	3,345,000	4.2	448,000

1.7 Mining

The mine is to be developed in three stages, incorporating two interim stage pits. Mining operations will be carried out using conventional drill and blast and load and haul mining methods with 3.5Mt of ore and 62.0Mt of waste being extracted over a period of seven years

A detailed mining schedule has been developed that requires minimal pre-stripping prior to plant commissioning. Production will initially commence from the high grade northern pit, which outcrops at surface, and along with the Stage 2 pit which commences after four months will return an average head grade of approximately 7.0g/t for the first 15 months of operation. Stage 3 commences in month 32 with a cut back of the southern wall of the Stage 2 pit to the final pit design.

A summary of the life of mining schedule is shown in Table 3.

Table 3 – Mining Schedule Summary

Mining Physicals		year	0	1	2	3	4	5	6	7	8
INDICATED ORE	kbcm	1,253.225	-	334.153	154.405	279.272	95.436	64.107	167.795	158.056	-
	kt	3,344.9	-	891.0	412.3	745.7	254.7	171.2	448.0	422.0	-
	g/t	4.17	-	4.78	3.75	4.33	4.55	2.91	2.78	4.73	-
	kg	13,936.3	-	4,262.8	1,544.9	3,230.7	1,160.2	497.3	1,245.3	1,995.2	-
INFERRED ORE	kbcm	70.1	-	1.3	9.0	5.7	6.0	17.6	15.9	14.5	-
	kt	187.1	-	3.4	24.1	15.3	15.9	47.0	42.5	38.8	-
	g/t	4.60	-	3.19	3.24	2.43	2.97	5.55	5.28	5.21	-
	kg	861.5	-	10.9	78.2	37.1	47.3	261.2	224.6	202.2	-
WASTE	kbcm	23,250.4	-	9,523.8	3,506.0	2,450.3	2,635.9	2,652.5	1,861.2	620.7	-
	kt	62,015.1	-	25,382.7	9,360.8	6,539.1	7,023.9	7,081.8	4,969.5	1,657.4	-
All Material (Ore+Waste)	kbcm	24,573.8	-	9,859.2	3,669.5	2,735.3	2,737.3	2,734.2	2,044.9	793.3	-
	kt	65,547.1	-	26,277.2	9,797.2	7,300.0	7,294.6	7,300.0	5,460.0	2,118.2	-
Stage 1	kbcm	6,723.8	-	6,412.0	311.8	-	-	-	-	-	-
	kt	17,929.6	-	17,097.2	832.4	-	-	-	-	-	-
Stage 2	kbcm	9,525.7	-	3,447.2	3,357.7	2,389.5	331.3	-	-	-	-
	kt	25,409.4	-	9,180.0	8,964.8	6,380.0	884.6	-	-	-	-
Stage 3	kbcm	8,324.2	-	-	-	345.8	2,406.0	2,734.2	2,044.9	793.3	-
	kt	22,208.2	-	-	-	920.0	6,410.0	7,300.0	5,460.0	2,118.2	-

The overall pit wall angles are based on recommendations from independent geotechnical consultants Peter O'Bryan & Associates. A further work program is planned for geotechnical, hydrological and general operational aspects such as dewatering studies and slope depressurisation. The results of these programs will be considered in the DFS.

The Company intends to engage an experienced mining contractor for the drill, blast, load and haul operations. There are a number of well-established regional and international mining contracting companies in West Africa. For the purposes of the PFS the Company obtained quotes from two contractors with relevant and current operating experience and benchmarked these quotes with other regional contractors.

1.8 Project Infrastructure

Electrical power will be generated on site by the use of diesel powered generators. A total of three 1.6 MW generating sets will be installed and operated on a two duty, one standby basis.

The treatment of the ore will result in the production of approximately 500,000t of tailings per annum. The tailings will be pumped as a slurry to a tailings storage facility ("TSF"). The TSF will comprise single circular storage with an area of 24.6 ha. The TSF will be equipped with a centrally located decant tower which will enable water released from the tailings, and collected rainwater, to be returned to the plant for re-use. The TSF will be designed to international standards.

The primary source for process water at the Project will be decanted from the TSF, with raw water to make up the balance. Raw water will be supplied from a newly constructed water storage dam on a local creek. The water storage dam will be equipped with a spillway capable of allowing the excess water to be discharged to the river downstream of the dam. The spillway is likely to be in use for a large part of year, as the run of river flow will exceed the plant requirements.

1.9 Environmental Studies, Permitting and Social or Community Impact

The Project has an existing EIA which has been approved by the Federal Ministry of Environment (Certificate issued 13 March 2013, FMENV/EIA/00871). The EIA approval is conditional on compilation of: (1) Environmental Management Plan ("EMP"); (2) Environmental and Protection and Restoration Plan ("EPRP"); and (3) Community Development Agreements ("CDAs"), which are required to be completed prior to operations commencing on site.

Both the EMP and CDAs are currently being developed and will be in place prior to the construction phase. The EPRP has recently been approved by the Ministry of Mines and Steel Development (22 August 2017, Ref. No. MMSD/S/39/S.657).

The Project does not require physical resettlement, however compensation for economic displacement and land acquisition for development activities is necessary. A Resettlement Action Plan is being developed to guide this process. The Project provides considerable opportunity for improvement of socioeconomic conditions in the local area. Currently the local area and communities are underserved by social services and infrastructure and therefore the Project will look to enhance sustainable socio-economic development opportunities wherever possible.

To date the Project has maintained good relationships with local stakeholders and there is a common understanding of the Project development process. Community representation on the CDA committees has been established

A Project closure plan has been approved as part of the EPRP. Closure costs are estimated at $5m.

1.10 Capital Costs

Total initial capital cost for the Project is estimated at US$71.4M including a 10% contingency factor.

Capital cost estimates (initial and sustaining) by area are summarised in Table 4 (numbers may not sum exactly due to rounding).

Table 4 – Capital Cost Summary

Area		0	1	2	3	4	5	6	7	8	9
Mining Contractor Capital Costs		$ 4.68	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Site Establishment, Clearing and Drainage	$USD M	$ 1.81									
Mining Contractor Mobilisation	$USD M	$ 2.88									
Processing Capital Costs		$ 51.99	$ -	$ -	$ 0.33	$ -	$ 0.76	$ -	$ -	$ -	$ -
Direct Plant Costs	$USD M	$ 35.27									
EPCM Contract	$USD M	$ 10.13									
First Fills and Spares	$USD M	$ 3.00									
Earthworks - Tailings and Water Storage Dams	$USD M	$ 3.60			$ 0.33		$ 0.76				
Other Project Capital Costs		$ 8.24	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Accomodation, Roads, Fencing	$USD M	$ 1.78									
Owner's Facilities and Equipment	$USD M	$ 5.61									
Owner's Vehicles/Mobile Plant	$USD M	$ 0.85									
Sustaining Capital	$USD M	$ -	$ -	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 0.50	$ -	$ -
TOTAL US$M, Rounded		$ 64.91	$ -	$ 1.00	$ 1.33	$ 1.00	$ 1.76	$ 1.00	$ 0.50	$ -	$ -
Contingency		$ 6.49									
Estimate + Contingency		$ 71.40	$ -	$ 1.00	$ 1.33	$ 1.00	$ 1.76	$ 1.00	$ 0.50	$ -	$ -
Pre-strip	$USD M	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Closure Costs	$USD M	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 5.00	$ -
Capital Reclaim	$USD M	$ -	$ -	$ -	$ -	$ -	$ -	$ 19.01	$ 48.82	$ -	$ -

1.11 Operating Costs

Mining operating costs are based on quoted budget pricing from West African mining contractors obtained as part of this PFS. Mining operating costs are inclusive of drill and blast, load and haul, fuel, all labour and equipment maintenance as well as contractor costs, geotechnical, rehabilitation and dayworks.

Processing operating costs are based largely on the estimates detailed in the Revised Bankable Feasibility Study (RBFS) completed by previous project owners Ratel Group Ltd in 2012, with adjustments made to reflect current conditions.

Life of mine operating costs are summarised in Table 5.

Table 5 – Operating Cost Summary

LoM Project Operating Costs	Total $M	$/oz	$/t
Mining	$ 183.5	427	54.9
Processing	$ 60.0	139	17.9
G&A	$ 18.3	42	5.5
Refining	$ 2.1	5	0.6
Cash Operating Cost	$ 263.8	613	78.9
Royalties	$ 7.7	18	2.3
Total Cash Cost	$ 271.5	631	81.2
Sustaining Capital	$ 11.6	27	3.5
Corporate G&A	$ 10.3	24	3.1
All-in Sustaining Cost	$ 293.4	682	87.7

1.12 Economic Analysis

The key inputs and assumptions used in the economic analysis are summarised in Table 6. Note that the economic analysis treated all Inferred material contained within the PFS pit design as waste in accordance with NI 43-101 reporting guidelines.

Table 6 – Economic Analysis Inputs

Economic Inputs	Units	Value
Mining Recovery	%	95%
Mining Dilution	%	110%
Processing Throughput	tpa	500,000
Processing Recovery	%	96%
Sell Price	oz	1,250.00
Royalty (TML)	% sell price	1.5%
TML Royalty Cap	$	$ 4,000,000
Royalty (Ratel)	% sell price	1.5%
Ratel Royalty Cap	$	$ 3,500,000
Nigerian Government Royalty	% sell price	0.0%
Industrial Levy	% payroll	1%
Discount Rate	%	8%
Corporate Income Tax (CIT)	% profit	30%
CIT Holiday Period	years	5.0
Education Tax (EDT)	% profit	2%
Capital Reclaim	%	95%
g:oz conversion	g/oz	31.1034768

The results of the economic analysis are summarised in Table 7.

Table 7 – Economic Analysis Outputs

Economic Outputs		Units	Value
NPV @ 8%	Pre tax	$M	$ 121
	Post Tax	$M	$ 119
IRR	Pre tax	%	53%
	Post Tax	%	53%
Payback		Years	1.8
Pre-production Capital		$M	$ 71.4
Gold Production	Years 1-3	koz	81
	Years 4-7	koz	47
Production Cost	LOM C1	$/oz	$ 613
	LOM AISC	$/oz	$ 682
Mine Life		Years	7.0
Probable Mineral Reserves	Tonnes	Mt	3.34
	Grade	g/t	4.2
	Ounces	koz	448
LOM Processing	Tonnes	Mt	3.34
	Grade	g/t	4.2
	Recovered Oz	koz	430
Processing Recovery		%	96%

Economic modelling on various gold prices was performed on both a pre-tax and post-tax basis, with the results summarised in Table 8.

Table 8 – Segilola PFS Project Economics - Gold Price Sensitivity

Gold Price	1,050	1,150	1,250	1,350	1,450
Pre Tax					
NPV 8%	55	88	121	155	188
NPV 5%	69	105	141	178	214
IRR	29%	41%	53%	65%	78%
Payback	2.8	2.4	1.8	1.3	1.2
After Tax					
NPV 8%	54	87	119	150	181
NPV 5%	68	104	138	172	206
IRR	29%	41%	53%	65%	77%
Payback	2.8	2.4	1.8	1.3	1.2

Notes: Economics have been centred on a base case using an 8% discount rate and a gold price of $1,250/oz. Economics based on 100% equity financing with contractor mining. Payback period calculated on an undiscounted basis starting from production start. West African peers commonly use 5% NPV and these figures are quoted for comparison.

The project benefits from a fiscal incentive regime. Under the key incentive provisions of the Industrial Development (Income Tax Relief) Act (IDITRA") and the Mining Act, the Project would benefit from a 5 year tax holiday followed by an accelerated capital allowance of 95% of mining expenditure. Following this period, the Project would be subject to standard Corporate Income tax of 30% on total profit.

1.13 Interpretation and Conclusions

The results of this Preliminary Feasibility Study indicate that work on the Segilola Project should progress to a more detailed study. The current Mineral Reserve and mining schedule show a project that is robust and will generate significant returns. The Mineral Resource and ongoing exploration also point to a potential expansion to the Project.

Existing exploration drilling and Resource modelling has indicated that there is a potential to undertake mining at the Project via underground mining methods upon completion of open cut mining. Upgrading Inferred Mineral Resources to Indicated and extending the Resource through additional exploration drilling would be required before underground mining could be realistically considered.

The inputs and ultimate outputs for this Preliminary Feasibility Study are considered to have an accuracy of ±25%.

1.14 Recommendations

During the PFS a number of areas were identified as requiring further review to bring the Project level of detail up to the standard required of Definitive Feasibility Study (DFS), whilst also allowing opportunity to further optimise and improve overall Project results. The areas requiring additional or updated detail include, but may not be limited to:

- Detailed topographical survey
- Capital Cost Estimates- re-evaluating plant design and supplier options
- Operating Cost Estimates- potential processing plant changes and detailed discussions with potential mining contractors
- Geotechnical Drilling and Analysis
- Hydrogeological studies
- Detailed Metallurgical Testwork- based on any processing plant changes
- Review of EPCM vs Lump Sum Turnkey (LSTK) contract for the main project infrastructure
- Pit Designs- updated to suit final optimisation and equipment selection
- Tailings Storage Facility Design and Analysis
- Surface Infrastructure Design- updated, detailed processing/office compound
- Construction/Mining/Processing Applications
- Mine closure process and costs
- Environment and HSE management, monitoring and resourcing
- Community and social benefits, funding, monitoring and resourcing

2. INTRODUCTION

The Segilola Gold Project is located approximately 120km north east of Lagos in the Osun region of Nigeria.

The Segilola deposit, formerly known as the Iperindo Reef, was first discovered during the working of elluvial deposits in 1945 and was later the subject of small scale open cut mining operation which is estimated to have produced less than 2,000 ounces gold. During the mid-1980s the Nigerian Mining Corporation (NMC) conducted the first drilling of the known gold mineralisation with the completion of 33 diamond holes spread over a strike length of about 900m.

Tropical Mines Ltd (TML) subsequently acquired the project with NMC retaining a 20% interest through a joint venture company called Pineridge Nigeria Limited. The initial drilling was followed up in the mid-1990s with the drilling of an additional six diamond holes by the new joint venture partner at the time – Hansa GeoMin. Early resource estimates by these companies ranged from 270,000 to 340,000 ounces at grades between 6.0 and 10.9 g/tAu.

In 2006 CGA Mining Ltd (CGA) identified the project as one with the potential for a medium to high grade open pit resource and in early 2007, through a newly formed wholly-owned Nigerian subsidiary – Segilola Gold Ltd (SGL) entered into a joint venture with TML. During the subsequent two years SGL completed a total of 12,203.52m of diamond drilling in 121 holes. In 2011 an additional 36 diamond drillholes for 3,705m were completed.

The current Mineral Resource Estimate block model used for this Preliminary Feasibility Study is based on two phases of diamond drilling that were completed in 2009 and 2011. This resource estimate was modelled on the information from 157 boreholes totalling 15,908.85 metres collected from the two drilling programs. The historic drilling program by Hansa was poorly controlled and assayed, so this information has was not used in the current Mineral Resource Estimate.

A review of tenement status with respect to any legal or statutory issues was not conducted. Thor has advised that there are no known title impediments to the operations and that all project tenements are in good standing.

2.1 Terms of Reference and Report Purpose

Thor Exploration Ltd commissioned Auralia Mining Consulting Pty Ltd (Auralia) to complete a Preliminary Feasibility Study (PFS) for the Segilola Project. This report outlines the work completed as part of the PFS, and details the results of this work. This PFS was completed in accordance with the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

2.2 Cautionary Notes

Auralia Mining Consulting (Auralia) has prepared this Preliminary Feasibility Study report for the use of the Client (Thor Explorations Ltd) and for the intended purposes as agreed upon between the two parties. No responsibility is accepted for the use of this report, either in whole or part, in other contexts or for other purposes.

Any use which a 3rd party makes of this document, either in whole or part, is the sole liability of the 3rd party. Auralia accepts no responsibility, liability or any potential attempt at damages, covering this statement

Auralia has based this report on quoted data, 3rd party expert reports, information available to the Authors and by investigations of published and unpublished data, as well as on data, information and discussions provided by the Client and their advisors.

Auralia has no reason to believe that any facts have been purposefully withheld, and that to the Author's knowledge the information provided was true, accurate and complete and not incorrect, misleading or irrelevant in any aspect. Auralia takes no responsibility for any errors in such provided data and information, nor any potential erroneous work resulting from the data and information provided, should it prove to have been incorrect or erroneous.

Any assessments made in this report are based on the conditions indicated from published sources and the investigation described. No warranty is included that actual conditions will conform exactly to the assessments contained in the report, and Auralia accepts no responsibility or liability covering this statement.

2.3 Sources of Information

In preparing this report the authors were solely reliant on information provided by Thor Explorations Ltd. This included information on the status on the property, the drillhole database, wireframes, geological information and data, previous Mineral Resource Estimates and previously completed reports on the project (Revised Bankable Feasibility Study and others), geotechnical and environmental reports and other information. Sources of information are referenced in full in the relevant sections of this report.

2.4 Site inspections

The Independent Qualified Person, Anthony Keers (MAusIMM, CP Mining), Director at Auralia Mining Consulting Pty Ltd visited the site in May 2017. Whilst on site Anthony met with key personnel, took part in a site tour and inspected the existing historic excavations, drill core, surface conditions and existing infrastructure.

3. RELIANCE ON OTHER EXPERTS

Items 13, 17 and 18 have been replicated or summarised from the Revised BFS completed by Ratel, a previous owner of the project, in 2012. There have been no material changes to this information since that time. In addition, the following is a list of key contributors upon which the Qualified Person has relied for information provided in the relevant areas:

- Geotechnical- Peter O'Bryan and Associates
- Taxation- Thor appointed tax advisers (BDO Stoy Hayward)
- Environmental and Social- Thor
- Tailings Storage Facility – DE Cooper & Associates
- Hydrology and Hydrogeology – Peter Clifton & Associates

The Qualified Person is satisfied that there is a reasonable basis for reliance on the information provided.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The Segilola Gold Project is located in Osun State, Nigeria. The project site is situated within 600m of a sealed road, 18km south of Ilesha, the local government centre, with a population of 300,000. Ilesha itself is located on a sealed dual carriageway, 120km north east of Lagos (the principal international entry port and major commercial centre of Nigeria with an estimated population of 16 million).

The area of Ilesha is located at 7°20' N and 4°45' E within the tropical zone of Nigeria (Figure 1). The Segilola Gold deposit is located some fifteen kilometres south-east of the town of Ilesha, between Odo Ijesha and Iperindo villages in Atakunmosa East Local Government area of Osun State, in the south-western region of Nigeria (Figure 2). Osun State is endowed with several mineral resource deposits including talc, kaolin, granite, clay, gold and feldspar. There are also indications of the occurrence of cassiterite, columbite, aquamarine and mica in the State. Traditionally, the people of the State engage in agriculture while a reasonable number of them are traders and artisans. Other occupations of the people include making of hand-woven textiles, tie and dye clothes, leather work, calabash carving, soap making, wood carving and mat-weaving. The population census of 1991 put the population of the State at 2.2 million.

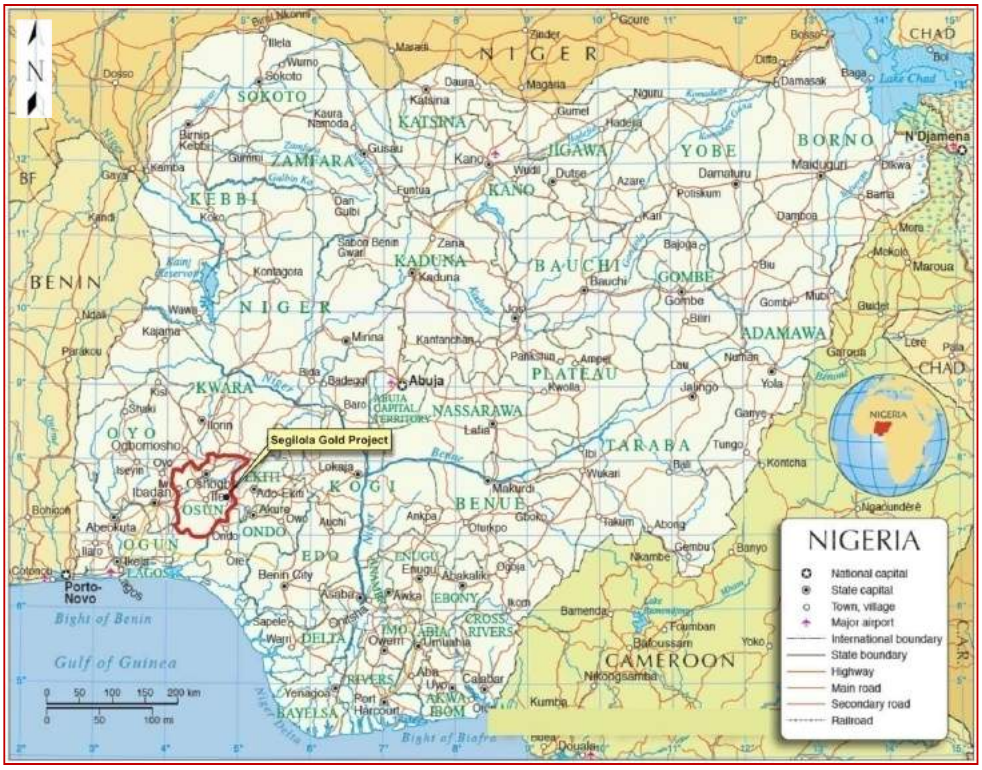


Figure 1 - Location Map -Nigeria

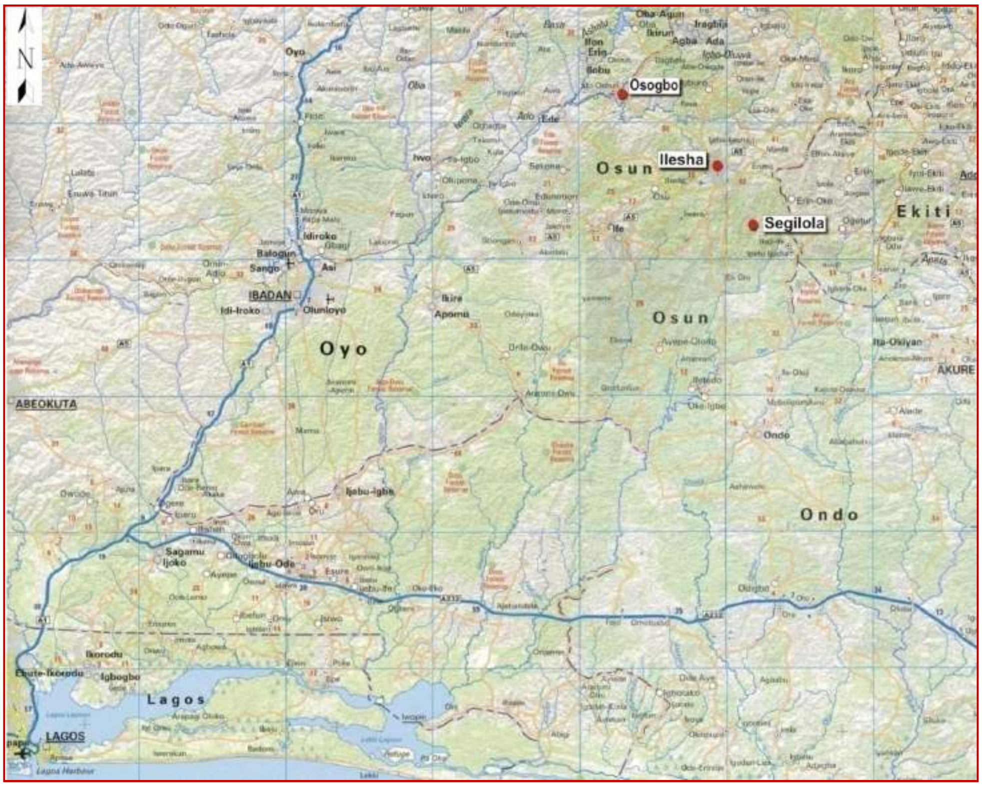


Figure 2 - Location Map – Osun State

The surrounding settlements, which consist of three villages, are inhabited by about 10,000 people. The villages are linked by minor roads and footpaths. The inhabitants of the villages are mainly farmers with family holdings.

The property comprises mining license ML41 and exploration license EL19066. ML41 covers an area of (17.2km^2; 1,720ha) and is wholly contained within the larger EL19066 covering an area of 135 Cadastral Units (27.0km^2; 2,700ha).

- ML41 was renewed in September 2016 for a period of 25 years and, as of the date of this report, ML41 is in good standing with its statutory requirements, with all the fees fully paid;

- EL19066 was originally granted on 25 September 2014; EL19066 is renewable twice for a period of 2 years each, with the first renewal application granted with effect from 25th September 2017 to 24th September 2019. As of the date of this report, EL19066 belonging to Segilola Resources Operating Ltd is in good standing with its statutory requirements with all fees fully paid.

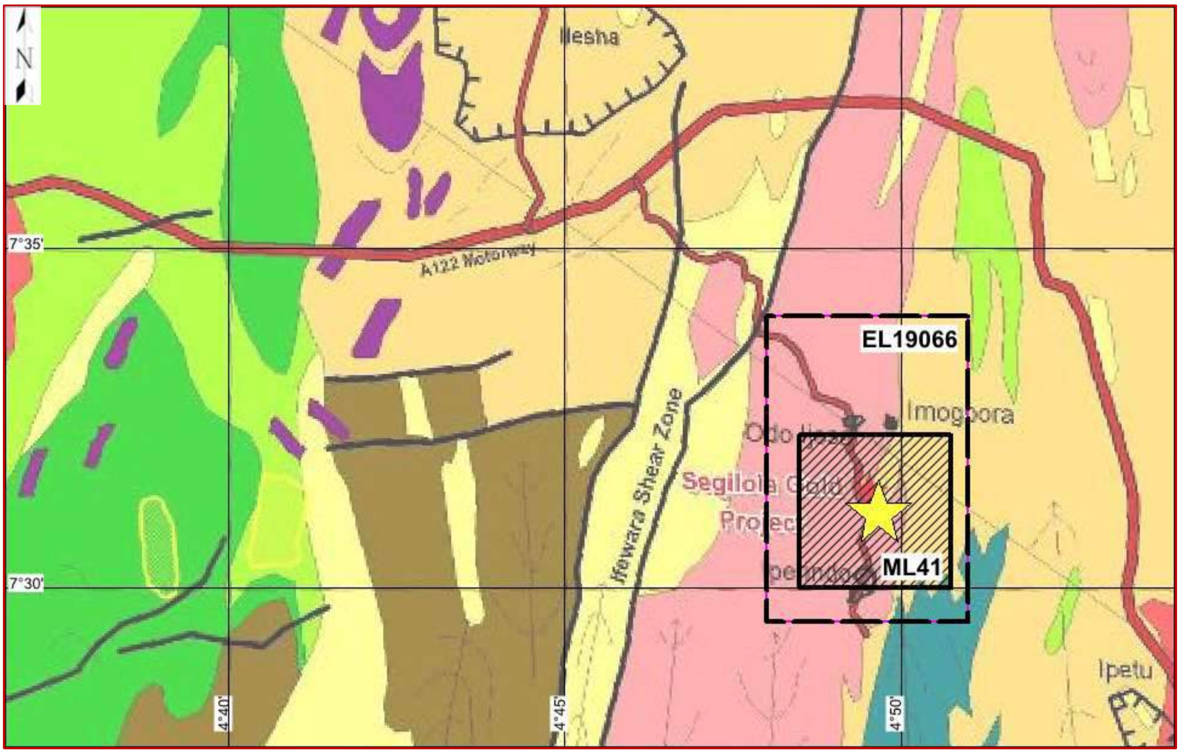


Figure 3 - Tenement Location Plan

4.2 Ownership

The tenements, which underlay the resource, are registered to and owned by Segilola Resources Operating Limited (SROL) which is a 100% owned Nigerian subsidiary of Thor Explorations Ltd.

In August 2016, Thor acquired a 100% interest in the Segilola gold project through the acquisition of 100% acquisition of Segilola Resources Operating Limited ("SROL") and through the acquisition of its joint venture partner Segilola Gold Limited ("SGL") from Ratel Group Limited ("Ratel"), a wholly owned subsidiary of RTG Mining Inc.

The tenements are now registered in the name of SROL who have a 49% interest in the Project. The issuer's rights to the property are through its 100% acquisition of SROL and 100% of SGL.

The Qualified Person has not reviewed legal issues regarding land tenure, surface rights, access and permitting nor independently verified the legal status or ownership of the Property, and is, therefore, wholly reliant on information from Thor. Thor has stated that, to their knowledge, there are no known issues with the current licenses. The surface rights are locally owned and permission to access the license area has been obtained, and this is a requirement when lodging a license application. All necessary permits are in place for the current activities being carried out. There is currently no known reason why permits and approvals for future activities, including mining operations, will not be granted.

4.3 Permits and Approvals

The Mines Department of the Ministry of Mines and Steel Development is the Federal Government's agency for policy making, implementation of laws and regulations governing solid minerals exploration, exploitation use or exportation. Licence holders/operators are therefore expected to maintain safety and environmental standards at all times and keep all records required of them in accordance with regulations.

The author and contributors are unaware of any legal issues regarding land tenure, surface rights, access and permitting and has relied upon information supplied by Thor Explorations Ltd. author has not viewed originals or copies of licence documents for ML41 and EL19066 issued by the Nigerian Mining Cadastre.

The surface rights are locally owned and permission to access the licence area has been obtained. This is in fact a requirement when lodging a licence application.

Thor reports that all necessary permits, including the Environmental permit, are in place for any planned future activities.

4.4 Encumbrances, Royalties and Taxes

Companies engaged in mining activities are liable for a corporate tax of 30% of their taxable profits. They are also liable for education tax of 2% on taxable profits. A value-added tax of 5% is payable in respect of taxable goods and services. Certain goods and services are, however, exempted from VAT. The most significant of these exemptions in the context of mining is goods that are exported.

Minerals obtained in the course of mining or exploration are subject to the payment of royalty at a rate of 3% for gold. The Minister may, upon the approval of the Federal Executive Council, defer payment of any royalty on any mineral for a specified period.

Annual service fees are payable in respect of all mineral titles. In addition to this the holder of a mining lease is required to pay surface rent at a yearly rate to be determined by the minister with respect to lands used by it for mining operations.

The holder of a mineral title enjoys the following tax incentives:

- a tax exemption for the first three (3) years of operation, which period may be extended for another two (2) years;
- capital allowance of 95% of qualifying capital expenditure incurred in the year of investment;
- annual indexation of the unclaimed balance of capital expenditure by 5% (only applicable to mines that commence production within five (5) years of enactment of the Act);
- carry-over of losses indefinitely;
- deduction of the mine reclamation costs and pension contributions from assessable profits;

- exemption from customs and import duties on approved plants and machinery, equipment and accessories imported specifically and exclusively for mining operations;
- subject to the prior permission of the Central Bank of Nigeria, retention of a portion of earned profits in an external account for use in acquiring spare parts and other inputs required for its mining operations where such equipment will not be readily available without the use of such earnings;
- expatriate quota and resident permit in respect of the approved expatriate personnel;
- personal remittance quota for expatriate personnel, free from any tax imposed by any enactment for the transfer of external currency out of Nigeria;
- free transferability of dividends or profits, payments in respect of servicing a foreign loan and foreign capital in the event of sale or liquidation of mining operations in any convertible currency;
- freedom from expropriation, nationalisation or acquisition by any government of the federation unless the act is in the national interest or for a public purpose and under a law that makes provision for payment of fair and adequate compensation and a right of access to the courts for the determination of the investors' interest or right and the amount of compensation to which he or she is entitled; and
- the right to a dispute settlement procedure under UNCITRAL Rules.

4.4.1 Pioneer Status

Pioneer Status is a tax holiday granted to qualified (or eligible) Mining Companies anywhere in Nigeria. The IDITRA provides for the grant of tax holiday for a period of 3 years, which may be extended at the end of the initial 3 years for a period of 2 years upon satisfactory compliance with certain conditions.

4.4.2 Additional Information

The Mines Department of the Ministry of Mines and Steel Development is the Federal Government's agency for policy making, implementation of laws and regulations governing solid minerals exploration, exploitation use or exportation. The operators are therefore expected to maintain safety and environmental standards at all times.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

The project site is situated within 600m of a sealed road, 18km south of the regional centre Ilesha, with a population of 300,000, which is itself located on a sealed dual carriage way, 120km north east of Lagos (the principal international entry port and major commercial centre of Nigeria, estimated population, 16 million). The old mine site with only a remaining core shed is just 600 meters from the tarred road and can be reached by foot or by four-wheel drive vehicles throughout the year.

5.2 Climate

A humid tropical climate predominates with a mean average annual rainfall in excess of 1400mm which is concentrated in the rainy season from March to November with a break during August. The mean maximum and minimum temperatures across the project region are over 34°C (in the month of February) and 18°C (in the month of December) respectively. The highest relative humidity range within the project area is between 81% and 87% corresponding to the wettest months (April through October).

Although situated in a zone of tropical rain forest, the vegetation is mainly moderate to dense secondary forest and bush re-growth, due to intensive farming.

5.3 Local Resources

An office, accommodation, core logging and sample preparation facility are all housed within a single secured compound on the outskirts of the town of Ilesha which located approximately 25km north of the Segilola project area.

5.4 Physiography

The general area topography is undulating with elevations ranging between 300m and 580m above mean sea level. Locally, north north-easterly striking steep valley incisions are developed. Within the exploration permit area, the topography shows a general slope towards the south. The lowest levels within the permits are within the area of the village Iperindo. The topographic relief within the immediate vicinity of the Segilola Gold deposit area is gently undulating with a 30m range in elevation.

Vegetation types of the area comprise mainly of cash crop plantations (kola nuts, cocoa, banana/plantain) together with secondary forests and bush fallows.

The project area lies within the crystalline Basement Complex rocks of southwestern Nigeria. Groundwater availability in the Basement Complex is very unpredictable. The crystalline rocks by their nature are impervious but fracturing, fissuring, jointing and weathering may impose secondary aquifer characteristics on these rocks, thus making them favourable to groundwater storage.

Recoverable groundwater often occurs in the weathered mantle covering the basement. Groundwater in the Basement Complex is essentially unconfined and the water table is restricted to sub-basins, which may be hydrologically isolated locally. Borehole yields vary widely in this area, the average safe yield of the successful ones being of the order of only few hundred litres per hour.

There are few perennial rivers, but there is a dense network of smaller seasonal tributaries. The drainage system of the project area flows north into the Osun River and south towards into the Oni River. The watershed cuts across in the northern parts of the tenements.

Weathering is typically tropical and penetrates down to 15 m depending on the parent rock types and the morphology. Where exposed, the rocks are reddish brown and are decomposed to clay minerals with quartz relics. Fresh rocks are found in the steep north-south striking valleys whereas the heavily weathered meta-sediments occur at higher levels. In general, saprolite can be reached within less than one meter apart from the alluvial terraces or other sedimentary cover.

5.5 Operating Season

Operations are possible throughout the year

5.6 Operations Infrastructure

The Segilola Project is proximate to key operating infrastructure such roads, electrical power, process water supplies and a ready source of labour from the nearby towns of Iperindo and Ilesha. There is sufficient space within the tenements to enclose future possible waste dumps, tailings storage facilities and processing plant.

6. HISTORY

The area is well known for its gold output from eluvial placers. Modern mining of the alluvial and eluvial deposits began in 1942. Official records state an annual production of about 23,000oz for the early years. The Segilola deposits, formerly known as the Iperindo Reef, was first discovered during the working of the eluvial deposits in 1945. The reef was subsequently investigated through a shaft and an adit.

From 1949 to 1969, the reef was worked by small scale local operators in an open cast down to a depth of about 5 metres from the surface and 300 metres along strike. However, a yearly production of only 220 ounces is confirmed for 1969 in the official records. The operators processed the ore with a second-hand stamp mill together with a ball mill and tables acquired from Ghana. The recovery of gold was very low. This accounts for the prospects of additional gold recovery from the tailings and ore dumps, which are about 40,000 tonnes with an average grade of 10g/t Au.

Geological exploration on the Segilola tenements has been carried out at various times and stages by:

- BRGM, France 1976
- Polish Engineers and Geologists 1981
- Nigerian geologist and surveyors 1981 - 1983
- Dr. R.W. Boyle (A Canadian gold specialist) 1984
- Nigerian Mining Corporation 1984 - 1987
- Nigerian Mining Corporation (core drilling) 1987 - 1992
- Pineridge Nigeria Ltd. (clearing and resampling old adits) 1992 – 1996.

Between 1984 and 1987 the Nigerian Mining Corporation (NMC) completed 33 diamond holes along the strike length of the lode. These holes were pre-fixed F-IG and numbered sequentially from F-IG1. The mineralisation was tested with mostly one hole on each section, with drilling commencing in the north and progressing to the south. Sections were roughly spaced at 25m.

In 1996 Hansa GeoMin negotiated and signed agreements with Tropical Mines Ltd and Pineridge Nigeria Ltd for a joint-venture based on the exploration and the development of five Exclusive Prospecting Licences (EPL) and one Mining Lease (ML). The EPLs cover the known gold deposits of Ilesha-Iperindo and neighbouring areas. The rediscovered Ilesha-Iperindo lode gold deposit was renamed as Segilola.

Between 1997 and 1999 Hansa GeoMin drilled seven diamond holes. Four of these holes were prefixed NIG and three were prefixed TIG. The drilling campaign was mainly designed to check and evaluate the Pineridge study on the deposit. Hansa started core drilling on the property in February 1998 with local contractors, Geo Core Drillers (GCD) and the state owned Nigerian Mining Corporation (NMC). Neither contractor was able to achieve the average drilling progress as stipulated in the respective contracts, mainly due to logistical problems and used drill equipment.

Hansa drilled 3 types of holes:

- Twinholes: To compare and check results of the old-boreholes.
- Deep holes: To demonstrate the vertical extent of the mineralisation and intersect the ore body at around 130m depth
- New holes: To step-out of the drill-hole covered zone of the Segilola Gold deposit to clarify the lateral extension of the mineralisation
- The core drilling program totalling 895m was completed in September 1998.

During this period Hansa GeoMin also resurveyed and relogged the available core for the original NMC holes. HGC renumbered the original holes with BH prefixes. The assay results of three of their twin hole compared very well to those of the respective NMC holes. Unfortunately, as a result of the re-sampling, most of the ore intersection core no longer remains.

In the early 1990s a group of local Nigerian investors formed a joint venture with NMC to further explore and subsequently exploit the Segilola Gold deposit. As a result, the Project tenements were transferred to TML, an incorporated joint venture company formed between PNL (80%) and NMC (20%).

In May 2007, TML entered into a JVA with CGA Mining Limited ("CGA"), a Toronto Stock Exchange and Australian Stock Exchange listed gold mining company, granting CGA the right to a 51% stake in Segilola, subject to certain operational milestones being reached. CGA entered into this agreement via its wholly-owned Nigerian subsidiary SGL. CGA immediately embarked upon a detailed drilling program of the known mineralised zone declaring a maiden resource estimate in 2009 after which a bankable feasibility was completed.

In 2009, CGA transferred SGL to its affiliate, a Toronto Stock Exchange listed entity called RGL (originally referred to as Ratel). The same CGA management team remained as overseers of the Project. In 2010 Ratel Mining Limited completed a Feasibility Study which was not publicly released. In 2012 Ratel Group Limited completed a Revised Bankable Feasibility Study (RBFS) following the completion of additional in-fill and resource extension drilling. Development of the Project was delayed due to a dispute between TML and RGL regarding earned interest in the Project. The RBFS was not publicly released.

In late 2012 SRK Consulting (UK) Limited issued a Minerals Expert Report on the SGP for Nigerian Gold Mining ("NGM") that reviewed the outcomes of the Revised Bankable Feasibility Study prepared by RGL. This report was not publicly released.

In 2016 Thor commissioned Odessa Resources Pty Ltd ("Odessa") to calculate and report an updated classified Mineral Resource estimate for the Segilola Project in order to support the execution of a Share Purchase Agreement by Thor Explorations Ltd for the Segilola Gold Project.

6.1 Previous Resource Estimates

Prior to 2006 there were two separate unpublished and unclassified resource estimates. These are summarised in Table 9.

Table 9 - Historic Resources Estimates

Company	Date	Tonnes	Grade	Oz Au
Pineridge (Nig) Ltd	1992	1,062,541	10.1	346,500
Hansa GCG	1999	1,400,000	6.0	270,000

In 2009, Odessa Resources Pty Ltd produced an updated Mineral Resource estimate for CGA based on drilling completed during 2009. This is shown in Table 10.

Table 10 - 2009 CGA Resource Estimate

	Indicated			Inferred		
Lode	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces
100	3,107,000	3.9	393,000	472,000	5.1	78,000
200	551,000	7.3	129,000	20,000	6.5	4,000
300	-	-	-	298,000	1.6	15,000
400	-	-	-	25,000	11.1	9,000
Total	3,658,000	4.4	522,000	815,000	4.1	106,000

In 2016, Odessa Resources Pty Ltd produced an updated Mineral Resource estimate for Thor based on existing data in order to support the execution of a share purchase agreement. This is shown in Table 11.

Table 11: 2016 Mineral Resource Estimate (1g/t Au Cut-off grade)

Classification	Indicated		
	Tonnes	Grade	Ounces
Indicated	4,581,000	3.8	555,000
Total	4,581,000	3.8	555,000

7. GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The project area is located in the crystalline Basement Complex rocks of southwestern Nigeria within one of the main “schist belts” known as the Ilesha Schist Belt. Schist belts in Nigeria occur as north-south trending domains of Upper Proterozoic (Eburnean 2,000Ma) meta-sedimentary, meta-volcanic and intrusive sequences that are oriented parallel to the boundary between the West African Craton and the Pan African Province (Figure 4). These schist belts are deeply infolded into a migmatite-gneiss-granite basement of Archean to Lower Proterozoic age and have been intruded by granitoids of the Pan African (600Ma) orogenic suite.

The Ilesha Schist Belt has a north-south strike extent of over 200 km and a maximum width of 60 km in the south. It is followed for much of its length by the regional Ifewara-Zungeru or Ifewara Fault or Shear Zone. This is a dextral strike-slip structure, which may have been active for a lengthy period, being from the Proterozoic to the Mesozoic. There is a marked structural contrast between rocks to the east of the Ifewara Shear Zone (where the Segilola Project is located) and to its west.

Primary gold mineralization in the schist belts commonly occur in quartz veins within several lithologies. In the Ilesha district mineralized lodes occur in fractures, folds and foliation planes at the lithological contact of amphibolites, talc tremolite schists and biotite schists. Quartz veins are foliated and commonly contain interleaved mica and feldspar with minor amounts of pyrite, pyrrhotite, chalcopyrite, magnetite and limonite.

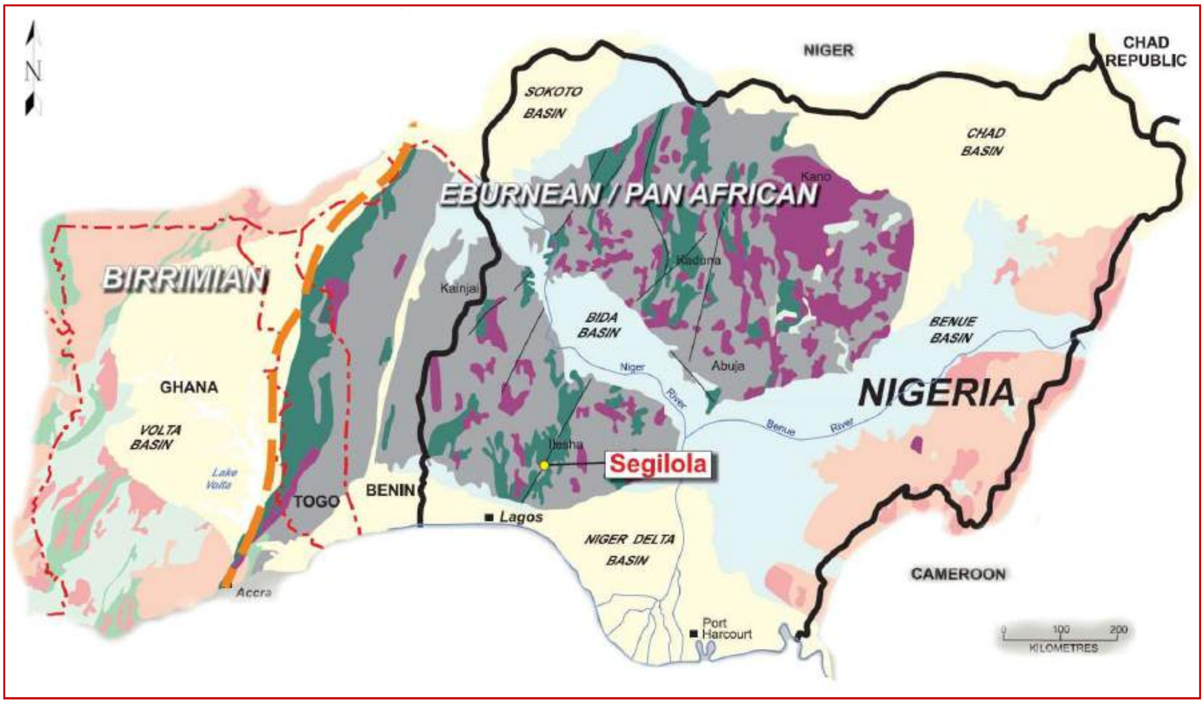


Figure 4 - Generalised Regional Geology Map

There are a number of academic papers dedicated to the Ilesha Schist Belt in the Ilesha area (Elueze, 1982; Elueze, 1986; Oyinloye, 1998; Caby and Boesse, 2001). Although all of these were published prior to the release of the latest aeromagnetics in 2008, some geophysical information seems to have been available, as the structures in Figure 5 show good correlation with the new data. Papers by Oyinloye and Steed, 1996; and Oyinloye, 2006 relate specifically to the Segilola (or Iperindo) gold prospect.

According to Caby and Boesse, 2001, an Archaean basement with U-Pb zircon ages of 2,600 Ma outcrops to the west of the area, of Figure 5, and possibly also outcrops within the area shown as hornblende gneiss, and structurally underlies outcropping rocks. It is typically of grey gneiss, and includes lenses of orthogneiss of tonalitic-granodioritic composition, along with some lenses of amphibolite. It is extensively intruded by probable Pan-African age granodiorite sheets.

Caby and Boesse recognise orthogneiss units of late Palaeoproterozoic age (~1,850 Ma by U-Pb), mostly outside the area of Figure 5, but which may include the orthogneisses around the Segilola Gold Project.

An extensive series of aluminous schists, quartz schists and quartzites are interpreted as Proterozoic sediments. They frequently display preserved sedimentary bedding. This broader sedimentary unit may be interlayered with syn-kinematic orthogneiss after felsic intrusives. Biotite-garnet-plagioclase schists and gneisses may represent greywackes, and more calcareous quartzites, schists and tremolitic calc-silicate layers also occur. Metavolcanics and metaporphyries of dacitic composition are also recognisable within this broad grouping.

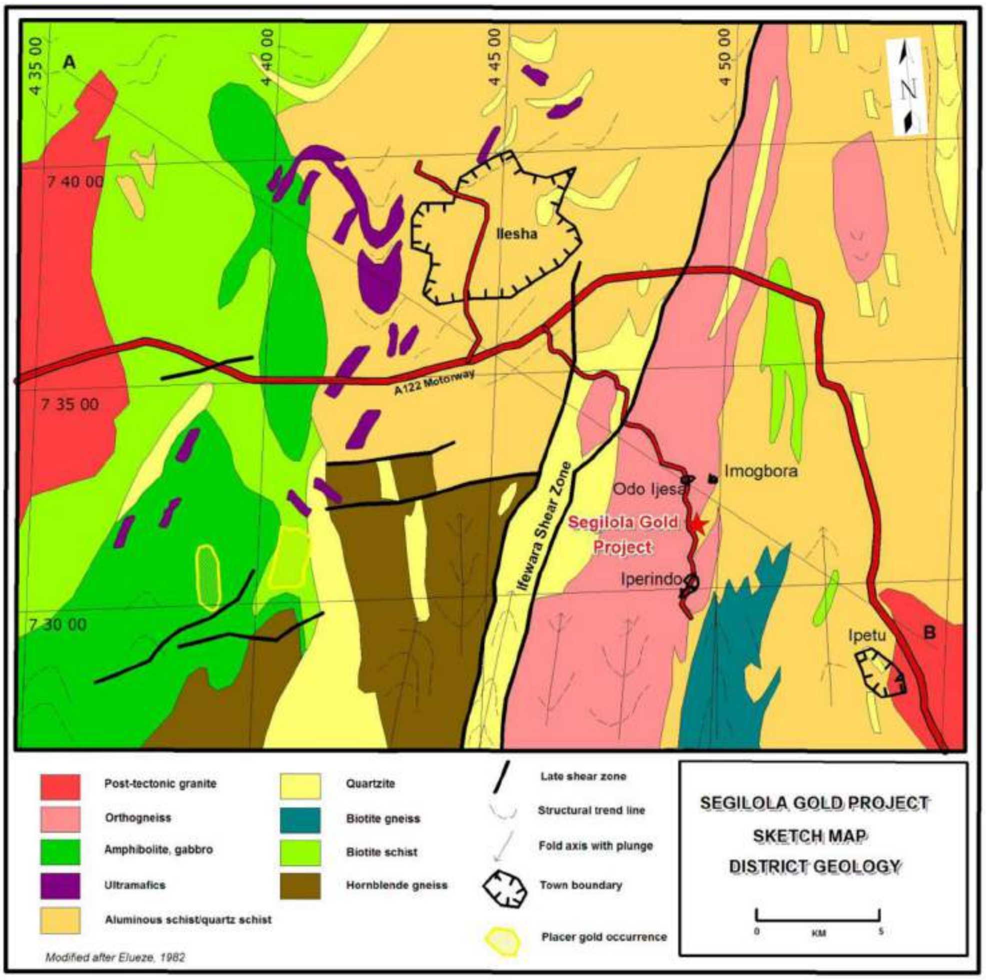


Figure 5 – Segilola District Geology

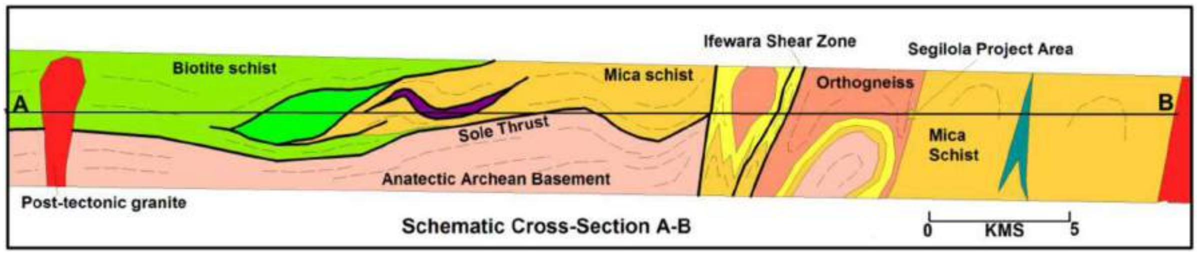


Figure 6 – Cross-Section A-B from Figure 5 (adapted from Caby and Boesse, 2001)

A large belt of mafic and ultramafic rocks, known as the Mokuro Massif, occurs to the west of the Ifewara Shear Zone (Figure 5). It includes meta-pyroxenite and gabbro units and melanocratic amphibolites. Talc-tremolite schist outliers are also mapped as interlayered with the aluminous schist (meta-sedimentary unit). Caby and Boesse interpret the massif as a large, strongly boudinaged, differentiated mafic sill, emplaced within the schists and quartzite prior to regional metamorphism; the outliers are interpreted as similarly boudinaged bodies. The contacts are tectonic, and it is interpreted as a largely flat-lying lens, underlain by schists (Figure 5 and Figure 6).

Late-kinematic Pan-African intrusives of granitic to granodioritic composition occur (Figure 6). They have traditionally been referred to as the "Older" granites in order to distinguish them from the Mesozoic intrusives seen elsewhere within the Nigerian shield. Caby and Boesse observed more widespread minor bodies of aluminous granites, with abundant related pegmatites, east of the Ifewara Shear Zone.

7.1.1 Regional Structure and Metamorphism

Caby and Boesse (2001) distinguish two main deformation events, D1 and D2, both of Pan-African age. D1 produced generally flat-lying foliations with fold axes typically trending from 120-150º. The recumbent attitudes are extensively preserved to the west of the Ifewara Shear Zone, and are interpreted by Caby and Boesse as a thrust stack, developed above an Archaean basement. Associated stretching lineations trend 040-080º, which is interpreted as the overall sense of tectonic movement, probably top-to-northeast. The D2 event has produced, to the east of the Ifewara Shear Zone, upright refolding of the D1 foliation at various scales, with sub- horizontal fold-axes trending 010-030º. Steep, ductile, north-northeast trending shear zones were also formed during this event, and host a range of mylonitic rock types and Pan-African pegmatites. West of the Ifewara Shear Zone, the D2 event is represented by large basin and dome structures deforming the D1 foliation. Mylonites within the Ifewara Shear Zone exhibit consistent flat dextral sense of shear indicators. Caby and Boesse suggest that although it may have been initiated as a lateral ramp during the D1 thrusting event, the shear zone was mainly active during the D2 event, under conditions of decreasing temperature, from sillimanite to greenschist (retrogressive) facies metamorphic conditions. It is locally cut by Phanerozoic east-northeast trending brittle faults, with minor dextral offsets.

The Pan-African metamorphic event in NW Africa is generally of high temperature and medium to low pressure type. Lower temperature assemblages are commonly preserved in synformal schist belts, whereas amphibolite-granulite facies assemblages occur in adjacent antiforms (Caby and Boesse, 2001). The extent to which granulite facies metamorphism might be pre-Pan-African is uncertain.

The extent of partial melting within the banded grey gneisses which are interpreted as Archaean in the Ilesha area implies temperatures ≥700º C. For the interpreted metasedimentary sequence, assemblages of quartz – muscovite – biotite - (± staurolite ± garnet ± sillimanite) suggest maximum metamorphic temperatures of 550º to 620º and pressures of 4.5 to 5.0 kbar.

7.1.2 Regional Mineralisation

The Ilesha area has accounted for a significant proportion of Nigeria's limited gold production. Although Segilola is the largest known bedrock resource in the area, significant alluvial-eluvial occurrences are known elsewhere, particularly around Itagunmodi, 15-20 km west of Segilola. This is within the amphibolite belt to the west of the Ifewara Shear Zone (Figure 5). According to Elueze (1986) the placer material has been derived from quartz veins and stringers particularly from contacts between biotite-rich rocks and amphibolites and talc-tremolite schists. Pyrite with minor arsenopyrite, pyrrhotite, pentlandite and chalcopyrite has been observed in both veins and wall rocks.

7.2 Local Geology

7.2.1 Introduction

The lithological units within the general Ilesha area include; variably migmatised gneiss, biotite and biotite-hornblende-gneiss with intercalated amphibolites, coarse porphyritic biotite-and biotite-hornblende-granite and quartzites.

The acidic biotite-and biotite-hornblende-gneiss rock is mineralogically similar to coarse-porphyritic biotite and biotite-hornblende gneiss mentioned above. The basic amphibolites occur mainly as lenses and inclusions in the migmatite gneiss.

The stratigraphy of the project area consists of the quartzite schists, the gneissic sequence and of the surficial Quarternary and alluvial sediments.

The geneissic sequence is subdivided into:

- undifferentiated para-gneisses and
- orthogneisses.

The orthogneisses underlay topographic highs with rounded tops, which are often poor in vegetation. These tops can be easily identified in aerial photos, satellite imageries and due to their rounded shapes some of them can be even identified in the topographic map. These orthogneisses dominate in the western parts of the EL19066 but they are also found in the Kajola Ridge at the eastern margin of the EL.

The orthogneisses are located below the quartzite schists. The northernmost outcrops of this unit have been found at the lower eastern side of Kajola Ridge. In the area of Kajola, the orthogneisses form the whole eastern flank of the ridge.

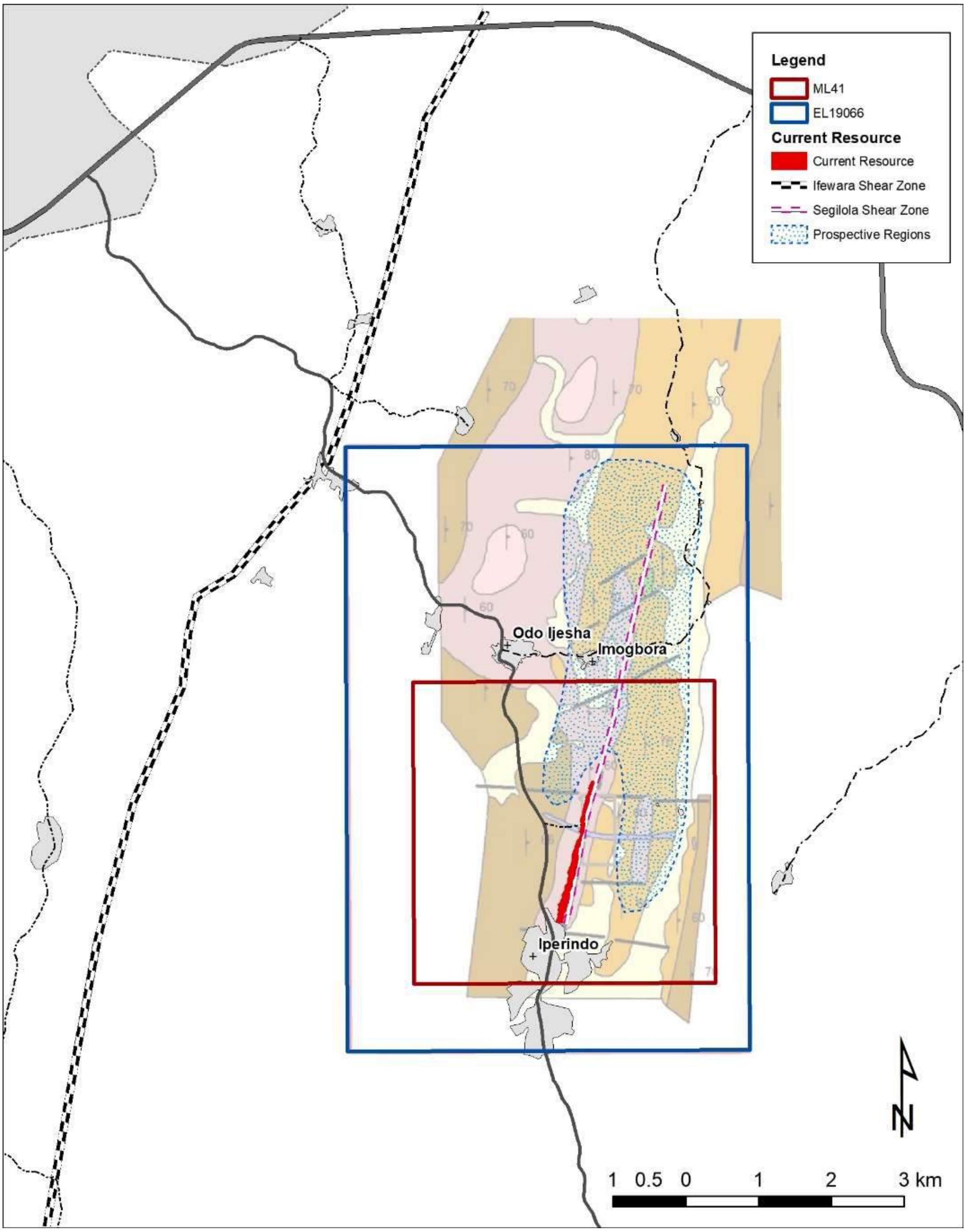


Figure 7 – Geology of the Segilola Project Area

East of Iperindo, the quartzite schists are restricted to the western flank of the ridge. These could be interpreted as contact metamorphism from the intrusion of the now orthogneiss. Another outcrop of garnet bearing schists/gneisses is located in the steep valley of Arafa River. This outcrop is isolated and disappears below the adjacent quartzite hills.

Within the orthogneisses, different degrees of metamorphosis can be observed. The orthogneisses located within the permit are intersected by quartz-feldspar pegmatoids of different size and the foliation ranges from weak to medium intensity. The texture of the orthogneisses north of Ijimo is more a granitic one, but a metamorphic overprint caused a weakly developed foliation.

The undifferentiated gneisses are often found at the base of steep valleys, which are located between quartzite ridges.

There is minimal weathering.

7.2.2 Geology

Outcrop in the area is poor, and mapping is largely based on topographic patterns and rare (commonly artificial) outcrops. Hansa interpreted a NNE-trending, generally steep-west dipping alternating sequence of various types of gneiss and probably a meta-sedimentary unit made up of quartzite and quartz mica schist. They distinguish orthogneiss (which they also call “granite gneiss” and “pegmatoidal gneiss”), biotite gneiss and undifferentiated gneiss. The term “orthogneiss” is usually meant to indicate gneiss derived from an igneous (granitic) progenitor – whether that is implied by Hansa is uncertain. Another interpretation would be that these are zones of more intense migmatization (metamorphic re-melting). The extensive biotitic gneisses are implied to be paragneisses (derived from sedimentary progenitors). Presumably the undifferentiated gneisses are those which could not obviously be allocated to the other two groups. They describe their “Quartzite Schist Series” as an intercalation of quartzite and quartz mica schist, with lesser meta-sedimentary types like garnet-biotite schist. By this interpretation the Segilola prospect lies at the boundary of a biotite gneiss (the Hangingwall) and a quartzite-schist zone (the Footwall lithologies).

According to Oyinloye (2006), in the immediate area of the gold prospect, the host biotite gneiss is a medium grained, foliated rock, consisting of quartz, biotite and K-feldspar, with minor plagioclase and hornblende and accessory apatite, monazite, ilmenite and zircon. Pyrite is the primary sulphide with pyrrhotite, sphalerite and galena occurring as minor sulphides. Whole rock analysis of 17 unaltered and unmineralised gneiss samples by Oyinloye and Steed (1996) suggested they are S-type granitoids (derived from partial melting of sedimentary source rocks).

In deeper drillholes the gneiss sequence passes down through a zone of highly foliated biotitic schist – presumably a high-strain zone – before passing into a calc-silicate sequence (the Footwall Lithologies). Getsinger (1988) described one of these rocks as calc-silicate gneiss, with grey layers of quartz, microcline and subordinate plagioclase, with green layers of diopside, blue-green amphibole, epidote and minor garnet, perhaps with some magnetite. The local presence of scapolite suggests high pressures of CO2. Zones of massive carbonate occur within the sequence and are presumably marble; although it is possible they are carbonate veins. Whether the calc-silicates are related to a contact metamorphic environment (i.e. are skarns) is not yet obvious from the drilling. They are at least locally sulphide-bearing, so have some potential for gold-(copper) mineralisation. However, the drilling has located no gold mineralization in the Footwall rocks within the Segilola project area

In detail, the SGL/RGL drilling shows that gold mineralization occurs in fractured pale to dark grey coloured smoky quartz veining, sheared pegmatite and silica/chlorite/carbonate alteration along a single robust shear zone in mainly biotite ortho-gneiss. The shear is located 2 to 15 metres in the hangingwall of a well-defined contact with calc-silicates and biotite schists. The tabular but slightly sinuous mineralised shear strikes 12Eº for over 2,000 metres and dips 58º to 72º west. It varies in width from <2 metres to over 20 metres and is over 200 metres deep and open in depth.

7.2.3 Structure

In the broader tenement area of Figure 7, the prevailing strike of metamorphic foliation and banding is to the NNE, with dips predominantly steeply to the west, but locally dip steeply east. Stereographic projections compiled by Hansa show a preponderance of measured foliations dipping steeply to 270-290º or 090-110º. Mapped joints, however, cluster with steep dips striking ~005º and ~185º. The former is more to the northwest than would be expected from the regional data, and may just reflect the limited number of measurements.

Trends suggesting large scale folding are evident in satellite imagery, although none have been definitively interpreted in the immediate tenement areas. Minor folds of foliation and veins are common. According to Oyinloye and Steed (1996) the axes of these typically plunge to the north – they do not indicate if this observation is based on a significant number of measurements.

The drilling has shown that there are two styles of minor faulting affecting the gold mineralized shear at Segilola.

Firstly, an east-west orientated strike-slip fault located at the northern extremity of the deposit. At around northing 832300N in particular, is a regional dextral wrench fault with over 50m of displacement. This occurs where the northern mineralization starts pinching out and will not affect mining.

A series of at least two parallel steeply west dipping faults (F1 and F2) that run sub-parallel to slightly oblique to the footwall contact have been noted. The footwall contact and the faults diverge gradually towards the south (Figure 10). Fault F1 appears to have truncated the main footwall vein system (200) north of 831500N as the fault merges with the footwall calc-silicates. The northern high-grade lodes (100) appears to be the northern continuation the hangingwall lode (400) that is developed to the south.

The north-central part of the deposit also hosts dolerite intrusions, but these are very irregular in nature and appear to have no effect on the mineralization.

From the 2017 drilling data, several notable geological features were either observed or interpreted that have an impact on the resource – particularly at depth.

All the historic drilling was focussed within a narrow corridor of the vein. In the 2017 program, collars were positioned further to the west and thus provided full intersections of the hanging wall rocks sequence. The mine sequence is illustrated in Figure 8.

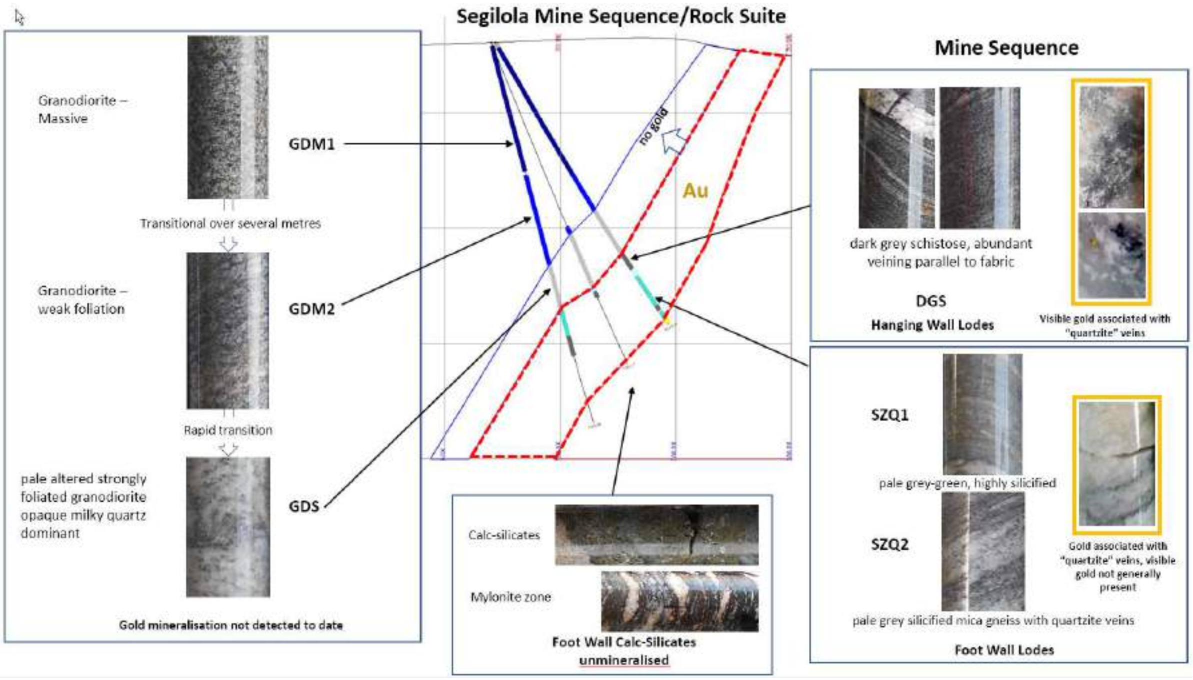


Figure 8 – Provisional Mine Sequence/Geotechnical Domain Scheme

The rocks transition from massive to weakly foliated granodiorites in the west, through a gneissic (GDM2) unit to intensely foliated and sheared rocks (DGS, SZQ) towards the footwall calc-silicate sequence which, itself is typically of mylonitic character. Higher grades and greater thicknesses are developed adjacent to a 5-20m thick zone of intense quartz-carbonate flooding (GDS) that is located in the hanging wall sequence (Figure 9).

Two styles of gold mineralisation are observed:

- Narrow, 1-3m thick, "hangingwall lodes" within silicified biotite schists. These lodes, locally contain 50-200 micron grains of gold, and are developed in the hanging wall to the main (footwall) lode. These lodes appear to have different controls as compared to the footwall lode and have a more vertical continuity over shorter strike-lengths.

- Wide, up to 15m, "footwall" mineralisation within a characteristically grey-green, strongly silicified zone of biotite schists and gneisses

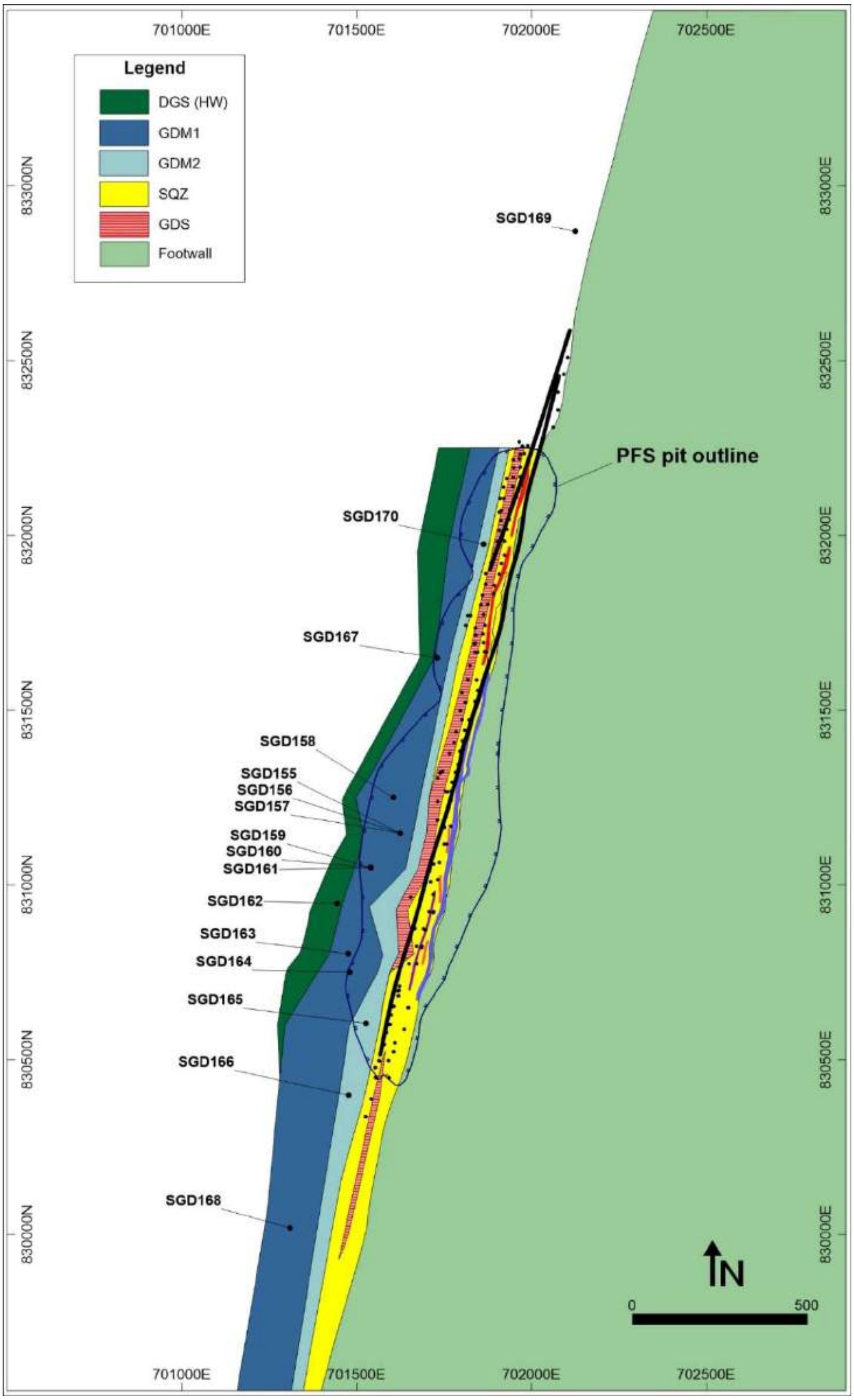


Figure 9 - Plan view of rock model showing location of GDS unit

Several notable features have been interpreted from the 2017 drilling data:

- The apparent southerly-plunge of the main ore body is not due to a plunging shoot but rather is the result of the intersection of sub-vertical fault (F1) and the westerly-dipping footwall (Figure 10 to Figure 13). These two surfaces diverge slightly towards the south thus creating a shallow south-plunging intersection vector. Mineralisation above the intersection is robust and predictable in its location. Below this intersection line, mineralisation is either very weak or absent in the zone currently drill-tested.
- To the north, the two above features converge with the fault passing into a mylonitic footwall. In this area, the main (footwall) lode is absent (faulted out).
- The northern high grade lode is an offset to the hangingwall lodes located south of the off-setting fault.

In summary:

The southerly plunging nature of the southern part of the resource has been confirmed. However, the plunge is due to structural intersections and not due to a plunging dilatant shoot. The plunging nature of the system is illustrated in the longitudinal section (Figure 14).

Mineralisation is predictable to the south at depth, based on current wide-spaced drilling, and further infill drilling between 830600N and 830950N is warranted in the future.

The high-grade hangingwall lodes provide the best opportunity for identifying an underground resource. Rather than follow the shallow southerly plunge of the main lode, the hangingwall lodes appear to be more vertically controlled. The main target area is located over a 500m strike length near to and south from the juncture of the main fault zone and the mineralised lodes (Figure 15 to Figure 17).

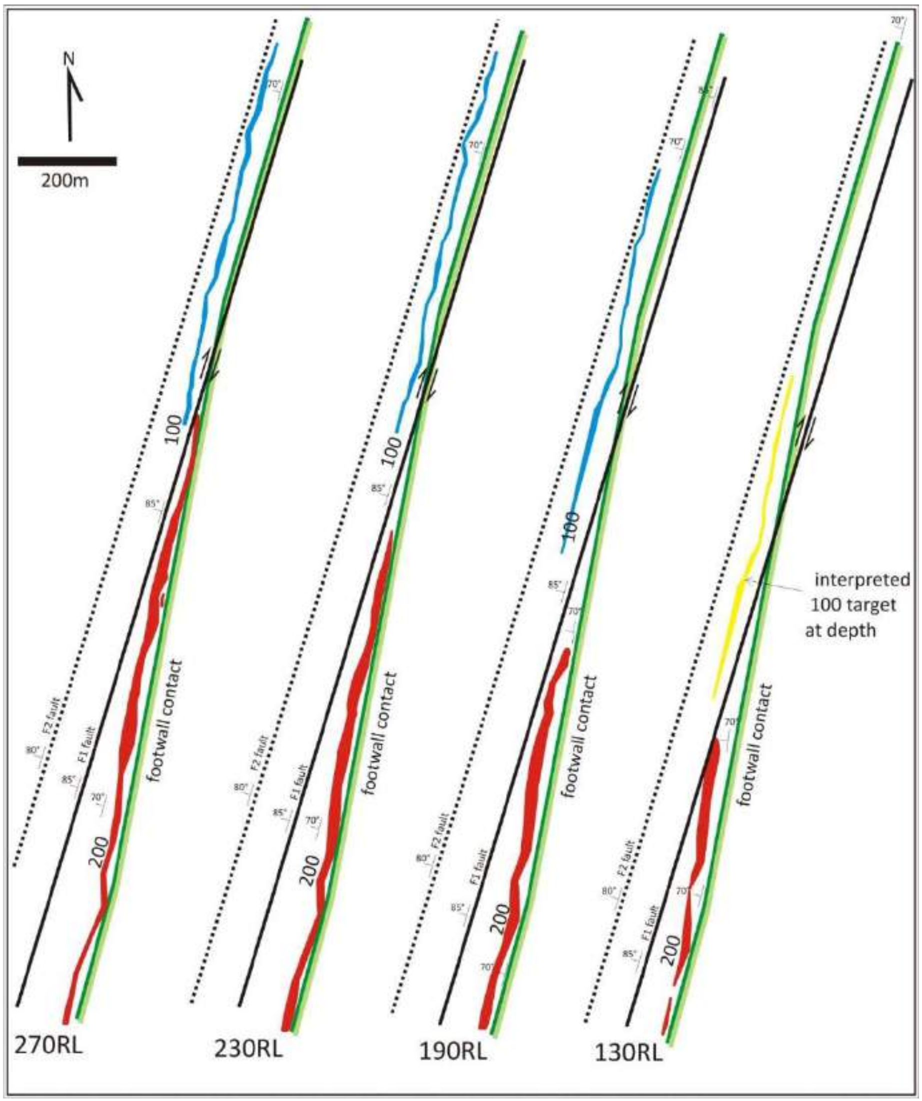


Figure 10 - Schematic plan views at increasing depths showing structural interaction of the lodes, footwall and sub-vertical strike-slip faults

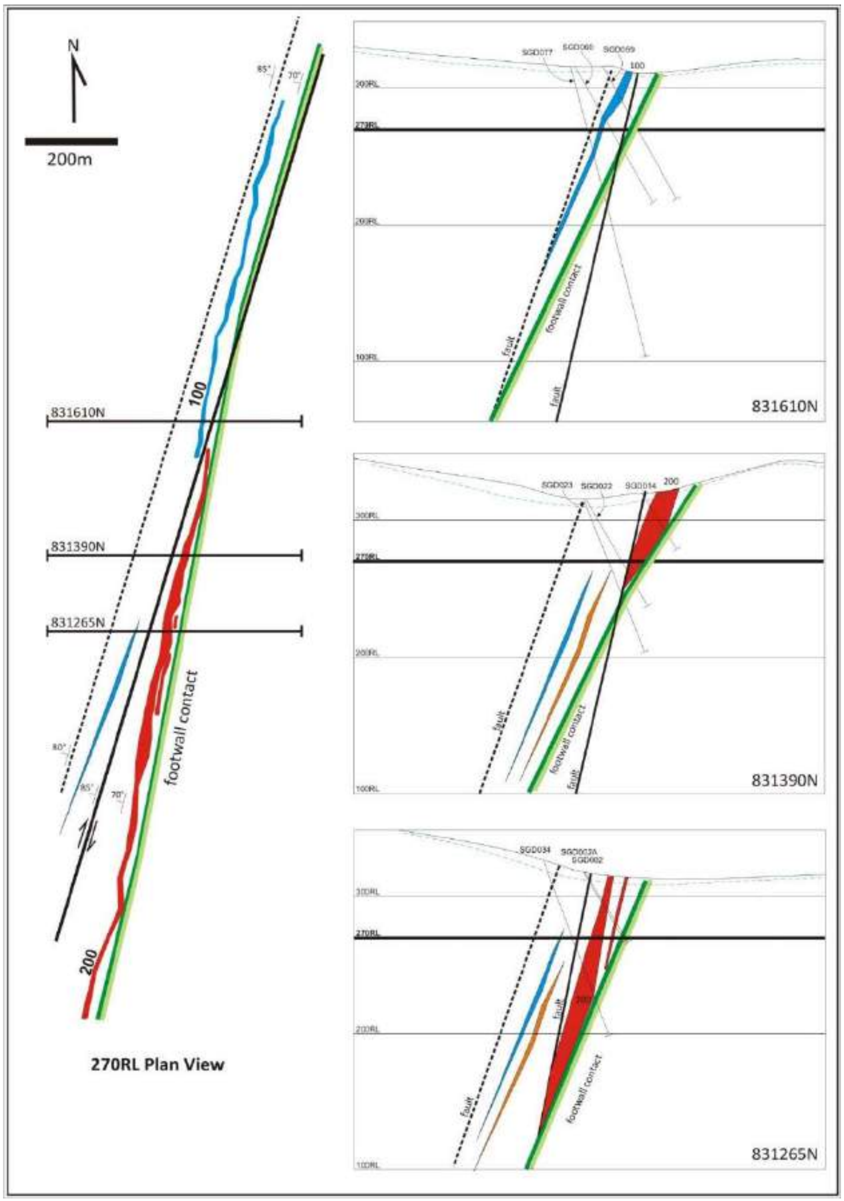


Figure 11 - Schematic cross sections at 270RL along strike showing structural interaction of the lodes, footwall and sub-vertical strike-slip faults

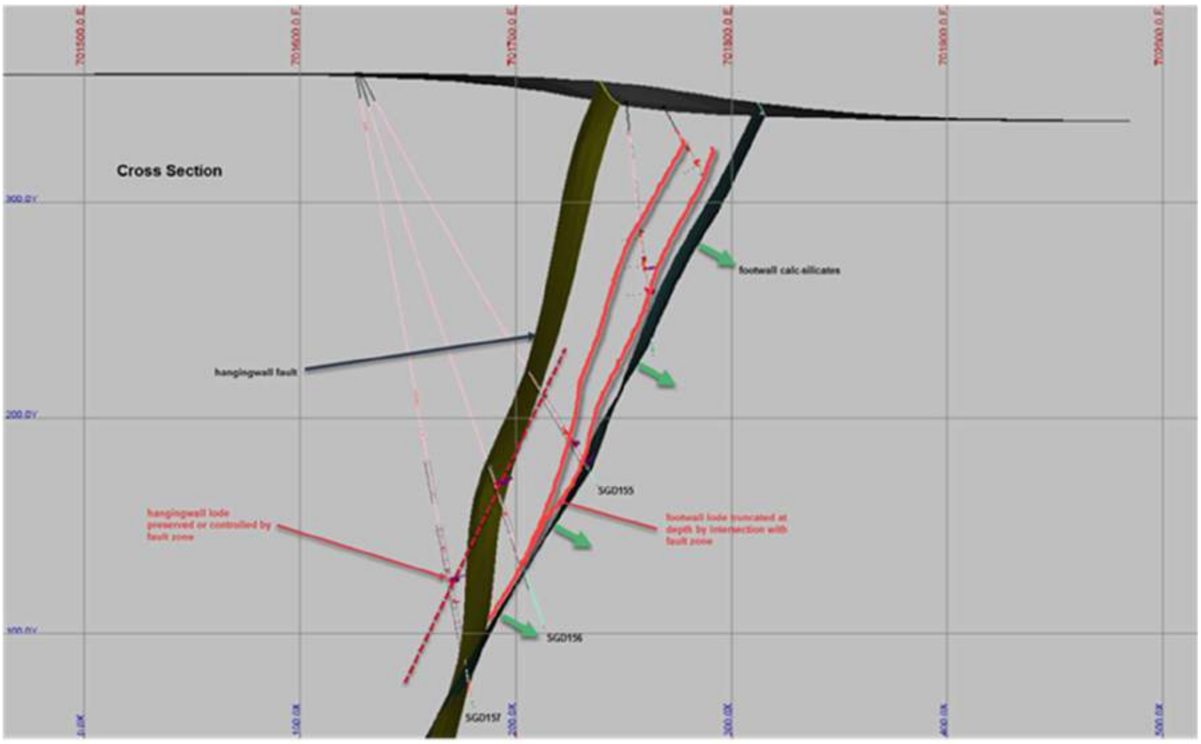


Figure 12 - Cross section showing structural compartment created by intersection of fault and footwall

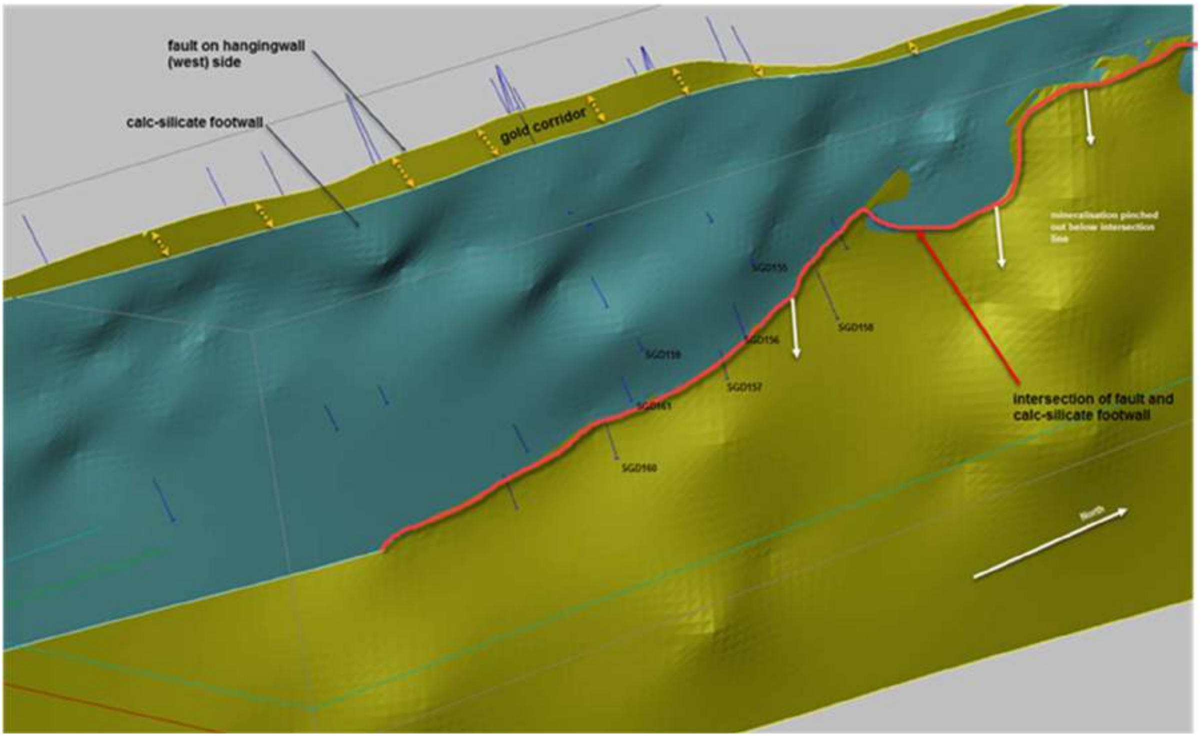


Figure 13 - Oblique view showing intersection vector (red) created by intersection of fault (yellow) and footwall (green)

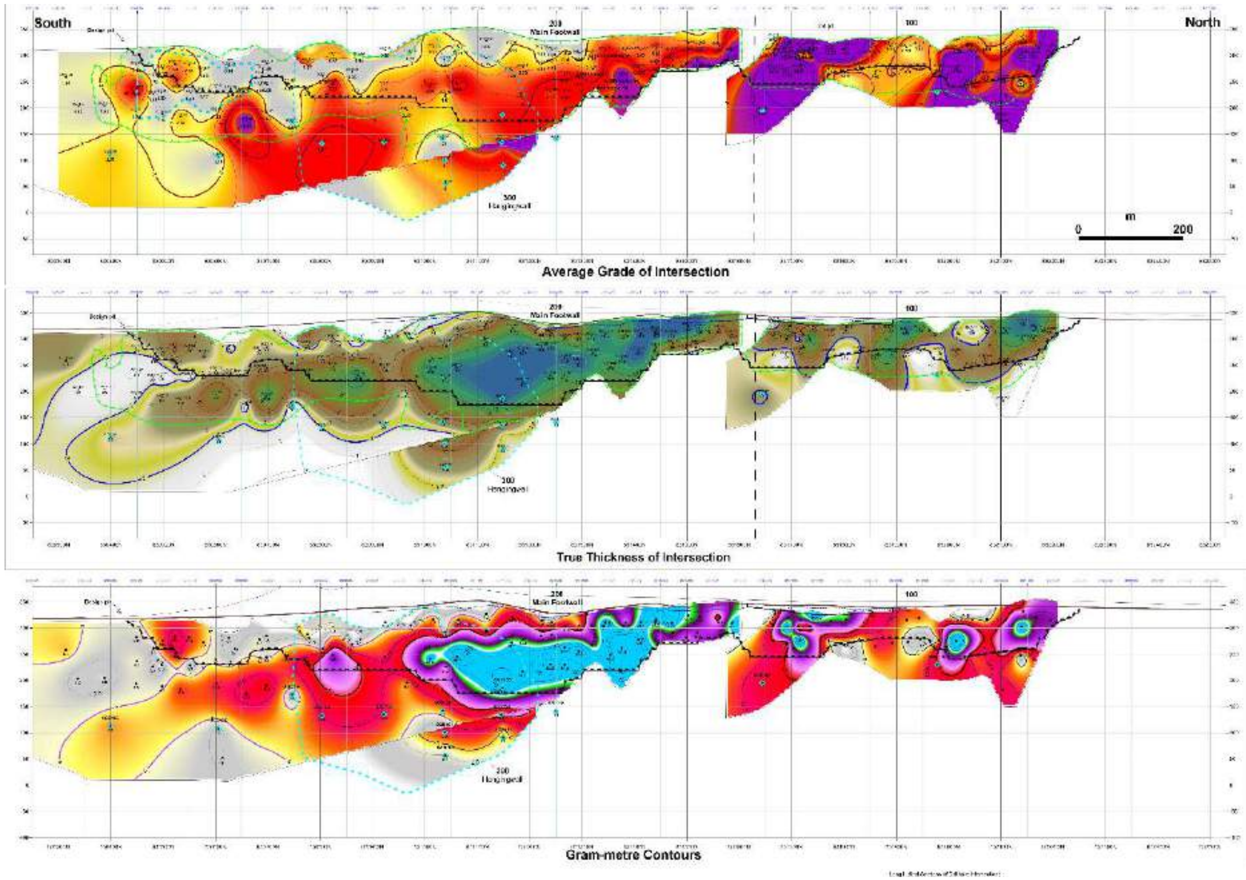


Figure 14 - Longitudinal section of 100 and 200 lodes

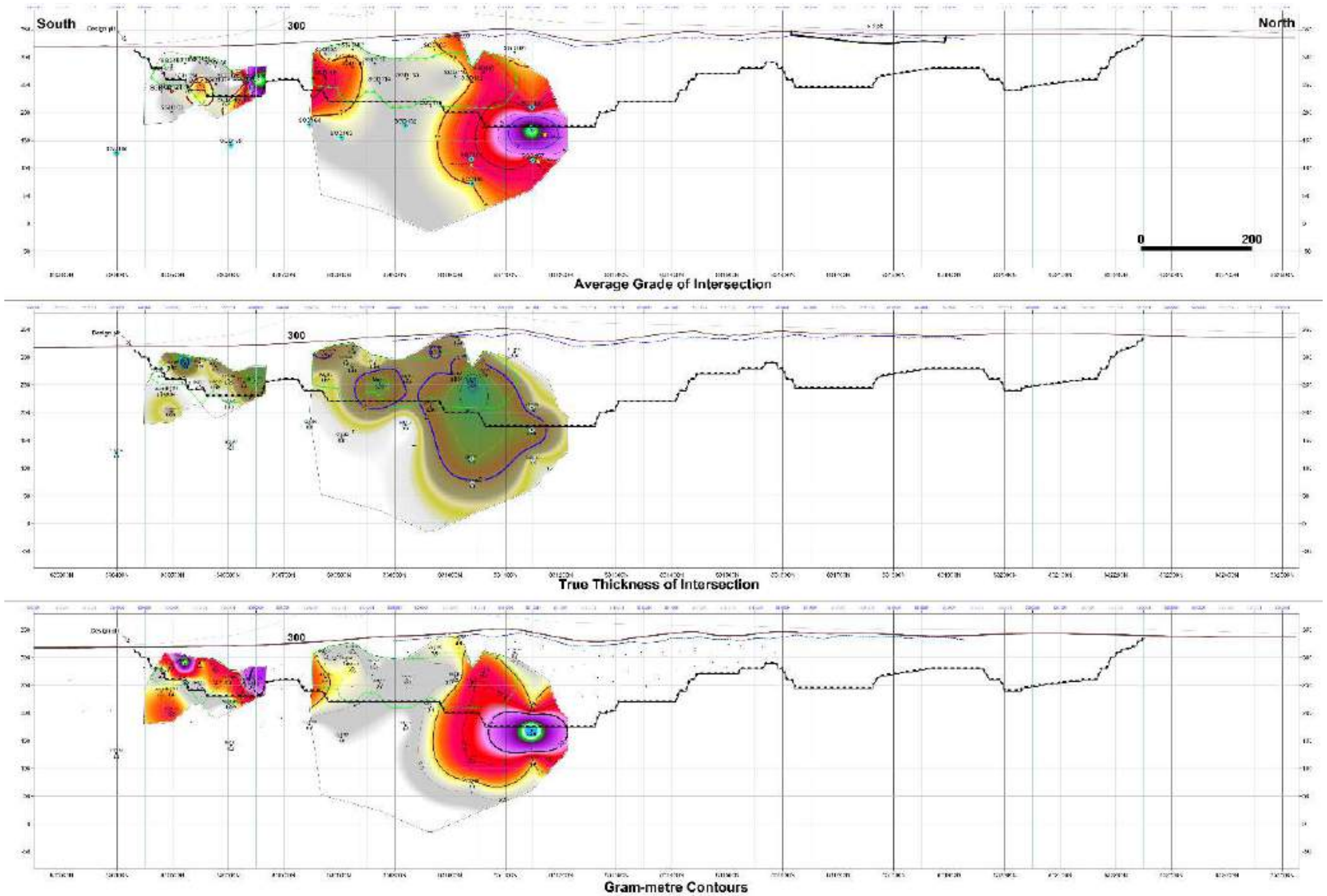


Figure 15 - Longitudinal section of 300 Lode

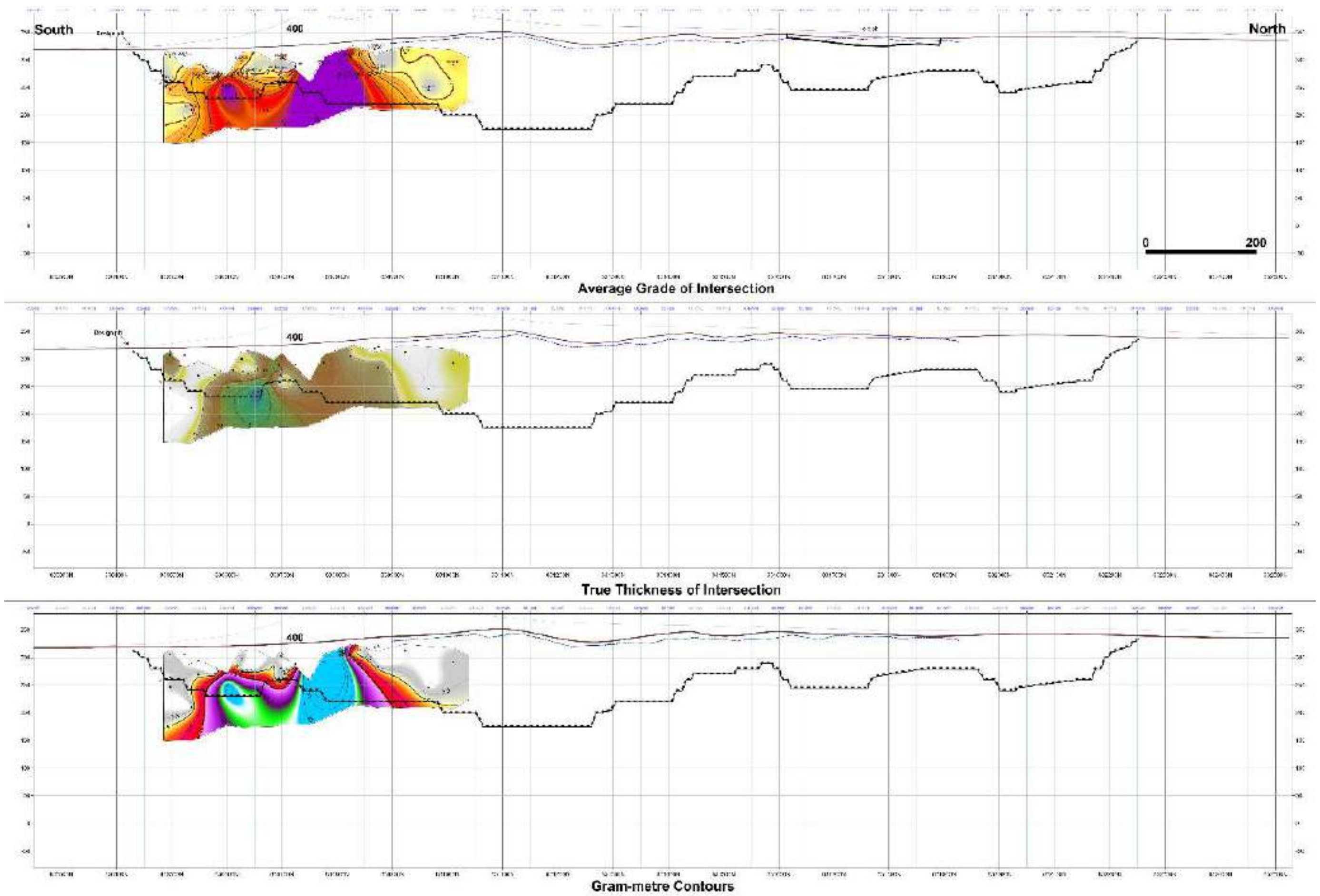


Figure 16 - Longitudinal section of 400 Lode

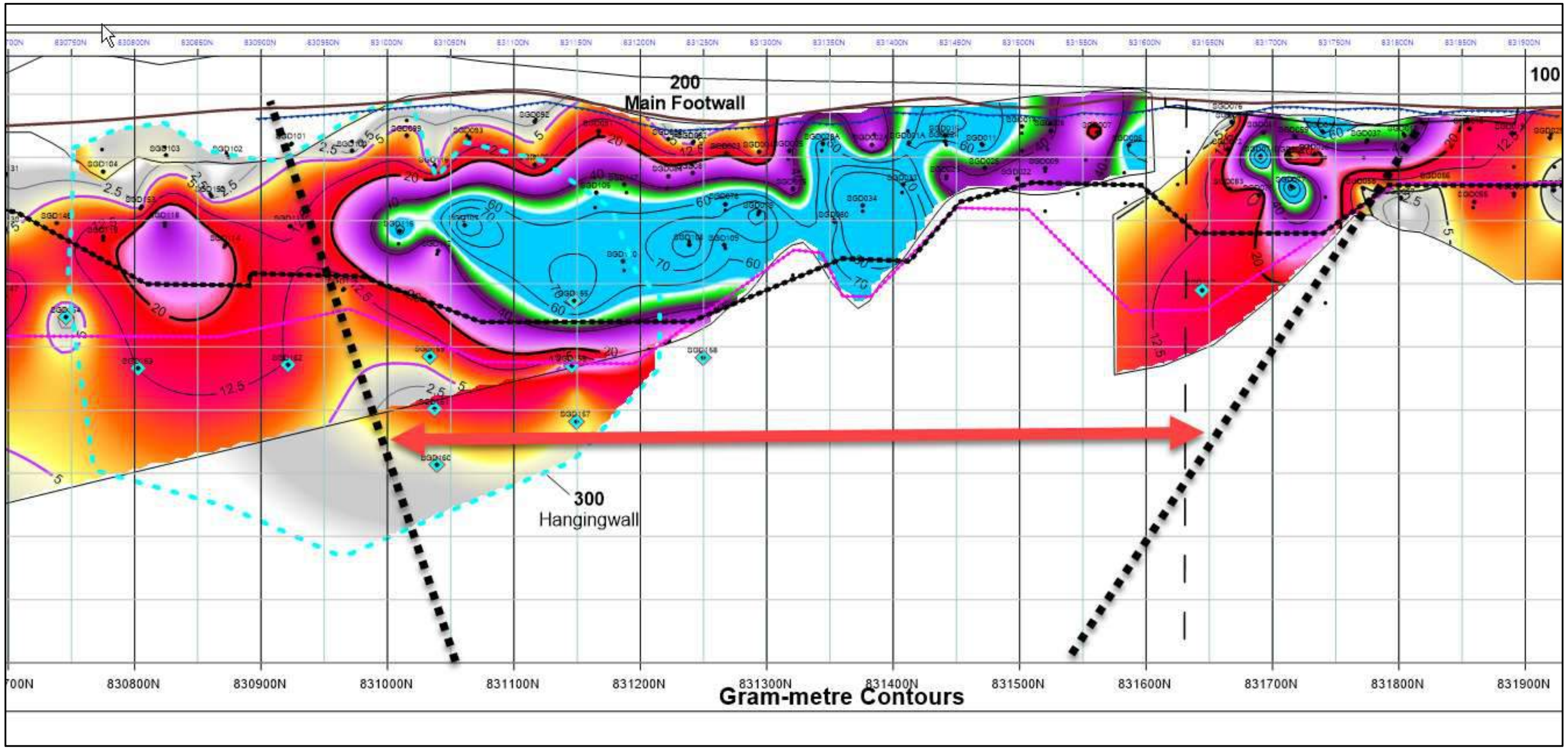


Figure 17 - Composite Gram metre longitudinal section showing likely location of hangingwall mineralisation at depth

7.3 Mineralisation

7.3.1 Macroscopic Characteristics

The mineralised lodes generally comprise highly silicified fine-grained foliated silicified biotite gneiss typically intruded by both discordant and concordant pegmatitic quartz-feldspar veins. Typical specimens of these rocks are shown in Figure 18.

Figure 18 - Pegmatitic quartz-feldspar vein (SEG001) and silicified biotite gneiss (SEG002) from drillhole SHD155

Figure 19 - Visible gold within pegmatitic quartz-feldspar vein

Shearing, fracturing and alteration influence the locality of gold mineralization. This relationship has generated multiple zones of gold mineralization hosted by shears now represented by chlorite and calcite alteration, together with quartz veining and pyrite development (Figure 20 and Figure 21).

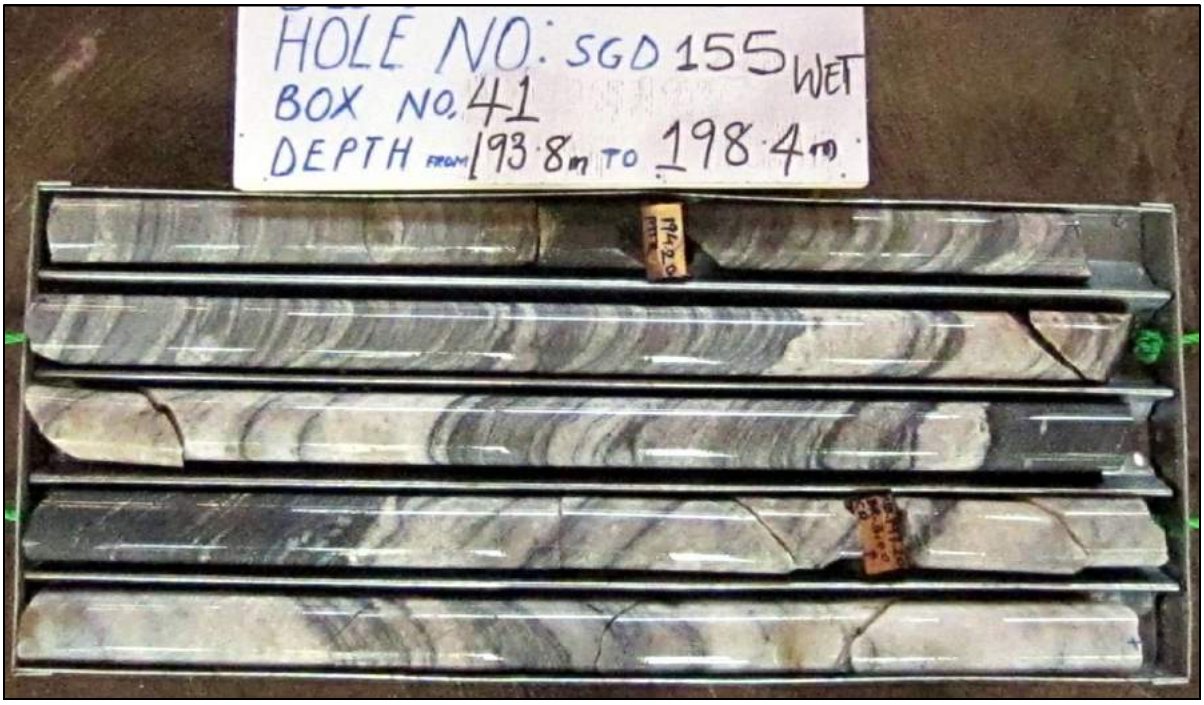


Figure 20 - Intense silica-carbonate alteration of mineralised zone in hole SGD155 (refer to Figure showing detailed log)

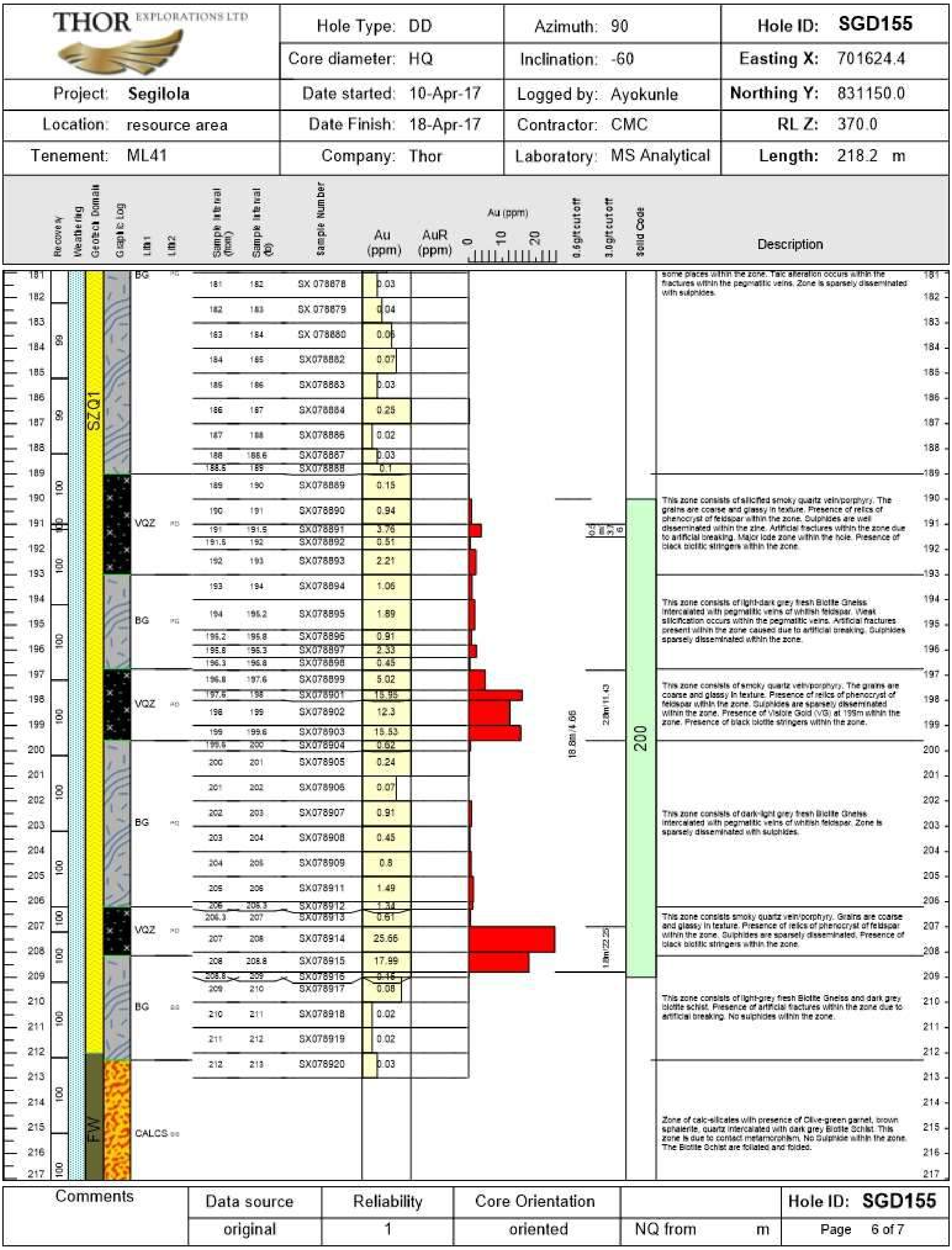


Figure 21 - Geological log of Footwall (200) lode in SGD155

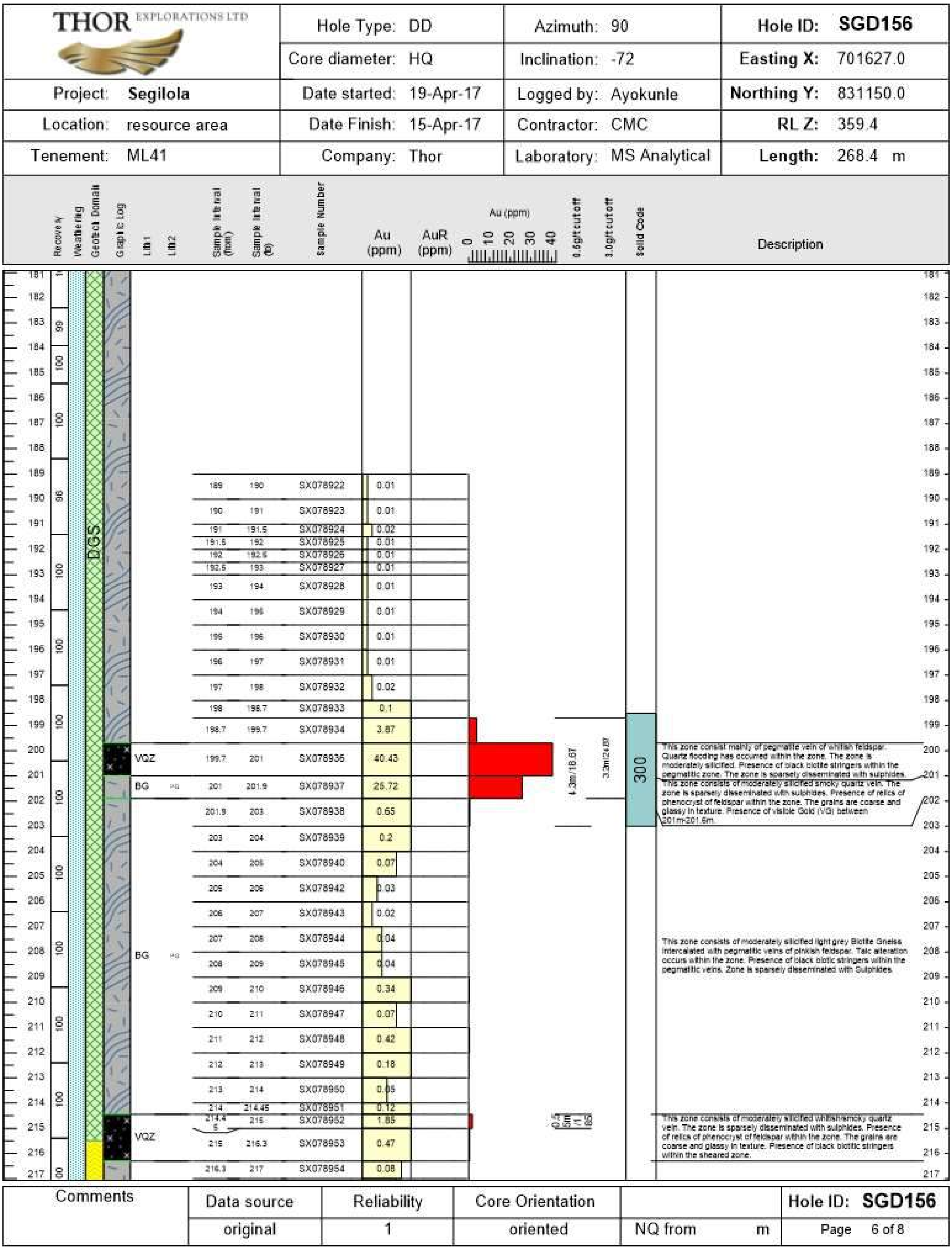


Figure 22 - Geological log of Hangingwall (300) lode in SGD156

7.3.2 Alteration

Macroscopic observations suggest a bleaching in the gneisses around the mineralised areas. In part, this may be because more muscovitic gneisses seem commonly to occur in these zones – presumably this is an alteration effect (phyllic or sericitisation). Silicification is also evidenced by the development of grey quartz, particularly within these bleached rocks. Alteration is patchy and rarely obvious more than 10 m or so from mineralised veins.

Oyinloye and Steed (1996) carried out whole-of-rock and trace analyses of ~19 hanging wall drill core samples, ~15 mineralised (vein) samples and ~12 footwall samples. Their main conclusions were that SiO_2 and K_2O are slightly enriched in the hanging wall relative to unaltered rock, and that K_2O is more enriched in the footwall, and SiO_2 less so. This suggests mild silicification and phyllic alteration in both hanging and footwalls, as is macroscopically observed. In their petrographic work they note the common presence of calcite, both in veins and as alteration. However, these observations are at odds with their analyses which show depletion in both CaO and MgO relative to unaltered rock. They also note depletion of P_2O_5 in altered rocks (relative to already low values in unaltered rock). Apart from gold, minor elements with enrichment include sulphur, zinc, lead, copper and rubidium.

7.3.3 Regolith and Weathering

Satellite imagery and topographic maps of the area suggest the presence of a paleaosurface forming topographically higher areas. According to Hansa GeoMin Consult, these surfaces typically show evidence of lateritic weathering, although the nature of this depends on the rock type. Hansa have mapped alluvial areas along drainage valleys.

At the Segilola resource area itself, drilling indicates total oxidation usually extends to less than 10m, normally 2 to 3 m. More susceptible units (faults and shears) are sometimes totally oxidised at depths of more than 60m. Oxidation logging has been recorded on handwritten log sheets, and thereafter transferred to the digital database.

7.3.4 Mineralogy

Minor sulphides, typically pyrite, are associated with the lode. Macroscopic observations show that sulphide grains and blebs are often aligned with foliation, commonly following either biotite-rich laminae or near pegmatite boundaries.

There is also, however, a common generation of pyrite occurring along fractures or as quartz-pyrite tension gashes, highly discordant to the foliation. Cursory examination suggests most of tension gashes do not contain pyrrhotite. These either relate to a late episode of mineralisation, or to remobilization of sulphides.

Visible gold is commonly logged as occurring both in altered wall rock (usually gneiss) and in quartz-feldspar veins. Native gold occurs with petzite (a silver gold telluride) within pyrite and quartz veins. Typical size of native gold blebs is about 10-50 microns (Figure 23). Gold, either as native grains, flakes and blebs occurs together with gold-on-pyrite in alteration zones, along tension gashes, hair-like fractures, joints and minor faults.

In 2017, two samples, one of hangingwall quartz-feldspar vein mineralisation (SGD156, ~40g/tAu) and one of footwall lode silicified gneiss (SGD155, ~20g/tAu) were prepared for polished section mineralogical study and conducted by Townend Mineralogy Laboratory (Perth).

Sample 1: SGD156 201.15m to 201.2m (refer Figure 22)

The drill core is dominantly composed of a very coarse, slightly deformed and slightly recrystallised quartz containing around 10% fresh potash feldspar (microcline) of 1-2mm. There is no evidence of a preferred orientation, particularly by the micas. The rare biotite is partly chloritized.

The two slides contain only traces of sulphides. These are present as pyrite occurring as rare, rather angular, linear grains or clusters once composite with chalcopyrite. A single flake of molybdenite separately measured 400µ.

There were at least four tellurides detected, usually in close association with gold. The main non-gold telluride was altaite (PbTe). Coarsest grains to 200µ may be in contact with the gold telluride calaverite, only rarely was altaite in contact with gold. An unnamed lead bismuth telluride was noted with calaverite and tellurobismuthite (Bi2Te3) was also identified.

Additional observations included:

- Gold was present as almost pure metal (<2 %Ag) and as the gold telluride, calaverite (AuTe2).
- Eighteen particles of gold were detected in the two slides. Sixteen of the gold particles are mainly hosted by quartz.
- Six gold grains were between 15µ and 50µ, with a 25µ example uncommonly hosted by mica.
- Ten gold grains range between 50µ and 100µ. One of these of 80µ occurs as a composite with calaverite and a little altaite.
- Two gold grains measured 370µ in quartz and the other, a 105µ grain was attached to calaverite.
- Coarse gold particles that were visible macroscopically in the original core were not present in the slide
- Eight examples of calaverite were detected.
- Five examples were discrete in quartz measuring between 30µ and 120µ. Other gold grains occurred as composites with other minerals and had grain sizes of between 40µ and 150µ. One 100µ gold grain occurred as a composite with coarse altaite and a PbBi telluride.

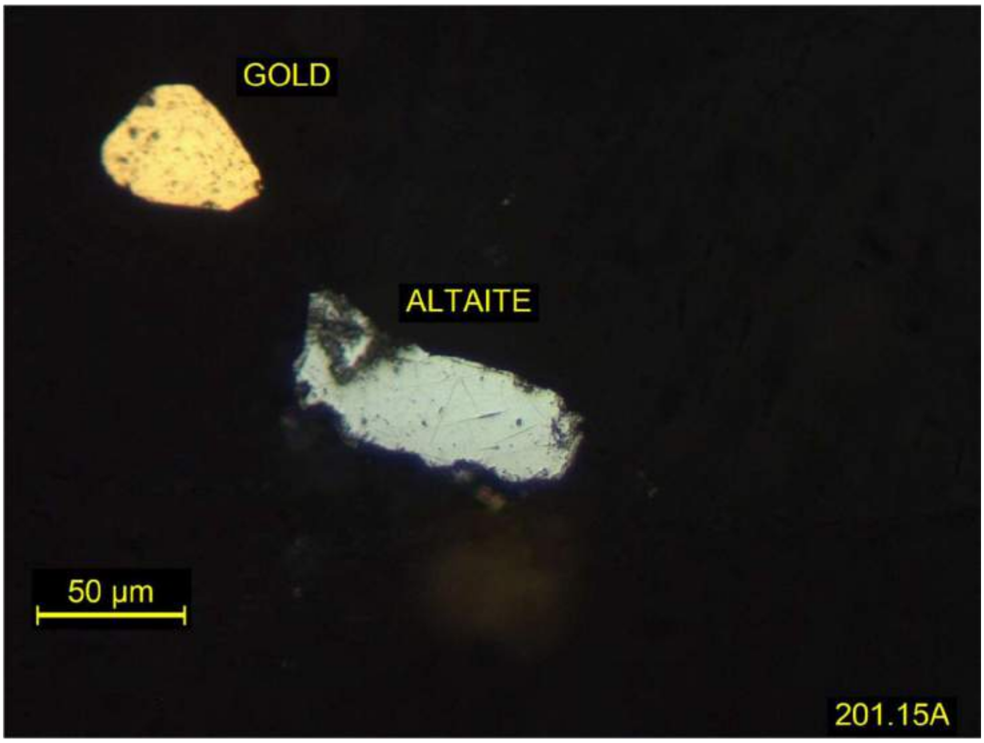


Figure 23 - Polished section SGD156

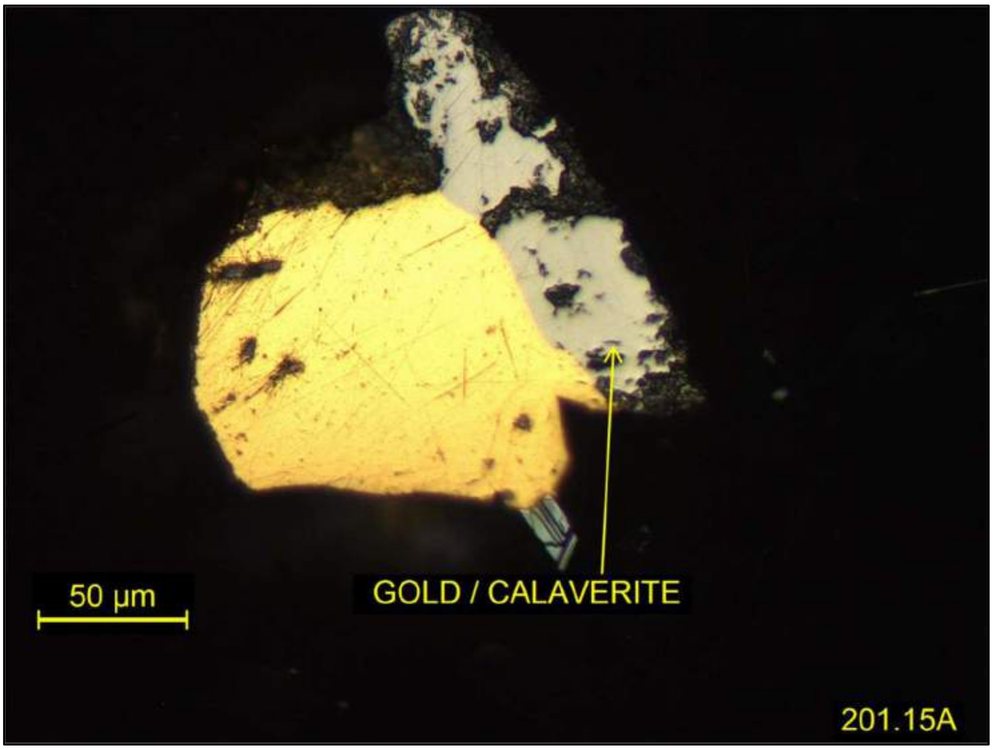


Figure 24 - Polished section SGD156

Sample 2: SGD155 207.40m to 208.20m (refer Figure 21)

The quartzolite is essentially identical to Sample 1. The gneiss, which has a similar quartz feldspar mineralogy also contains well oriented fresh biotite and muscovite. In addition, there are examples of oriented fibrous sillimanite.

The ores in the gneiss consisting of sulphides and rare magnetite. Pyrite is the most common sulphide, present as anhedral single grains, also composite with chalcopyrite, pyrrhotite and sphalerite plus a vein-like mass. The other sulphides are single occurrences. The magnetite occurs as a fractured association of 300µ overall. The drill core shows that there is more concentration of oriented sulphides further from the contact, not in the slide. A detailed optical examination of the 6X 2.5 cm slide area failed to detect gold.

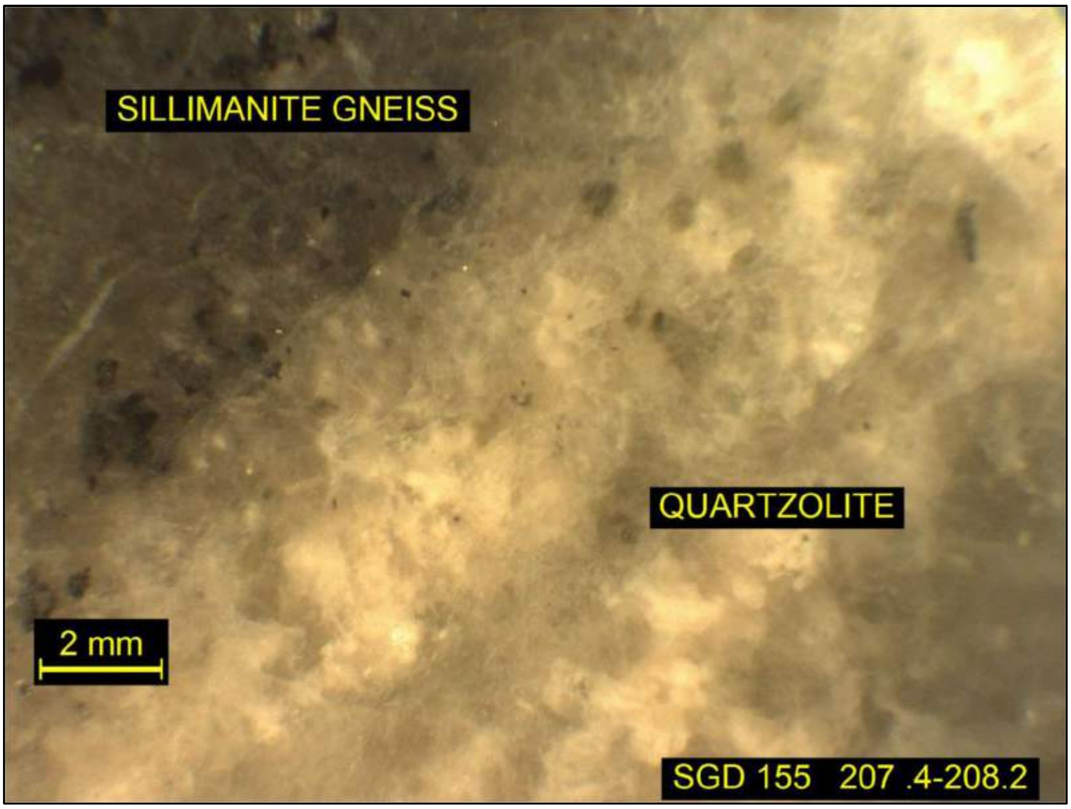


Figure 25 - Polished section SGD155

8. DEPOSIT TYPE

8.1 Deposit Morphology

The Segilola deposit comprises a series of sub-parallel, steeply dipping gold-bearing veins. The mineralized zone forms an elongated body varying from 2 metres to over 20 metres thick, is 2,000 metres in strike and between 70 and 200 metres in depth.

Four main lodes are developed in the hangingwall schists above the footwall calc-silicate sequence (Table 12).

Table 12 - Lode nomenclature and description

Lode/ Model Code	Description	Estimated Average True Width (m)	Interpretation	Grade Character (g/t Au)
100	Northern High Grade Lode	4	northern continuation of 400 lode	8.0
200	Main Footwall Lode	10	developed only south-east of oblique strike-slip fault	3.5
300	1st Hangingwall Lode	3	developed only south-east of oblique strike-slip fault	3.6
400	2nd Hangingwall Lode	2	southern continuation of northern high grade lode	14.0
102	minor lode	3	footwall to 100 lode	3.4
500	minor lode	1.5	developed only south-east of oblique strike-slip fault	5.5
600	minor lode	1	developed only south-east of oblique strike-slip fault	12.0

The lode and structural elements are shown in Figure 26. Figure 27 shows a cross section through the central part of the resource area.

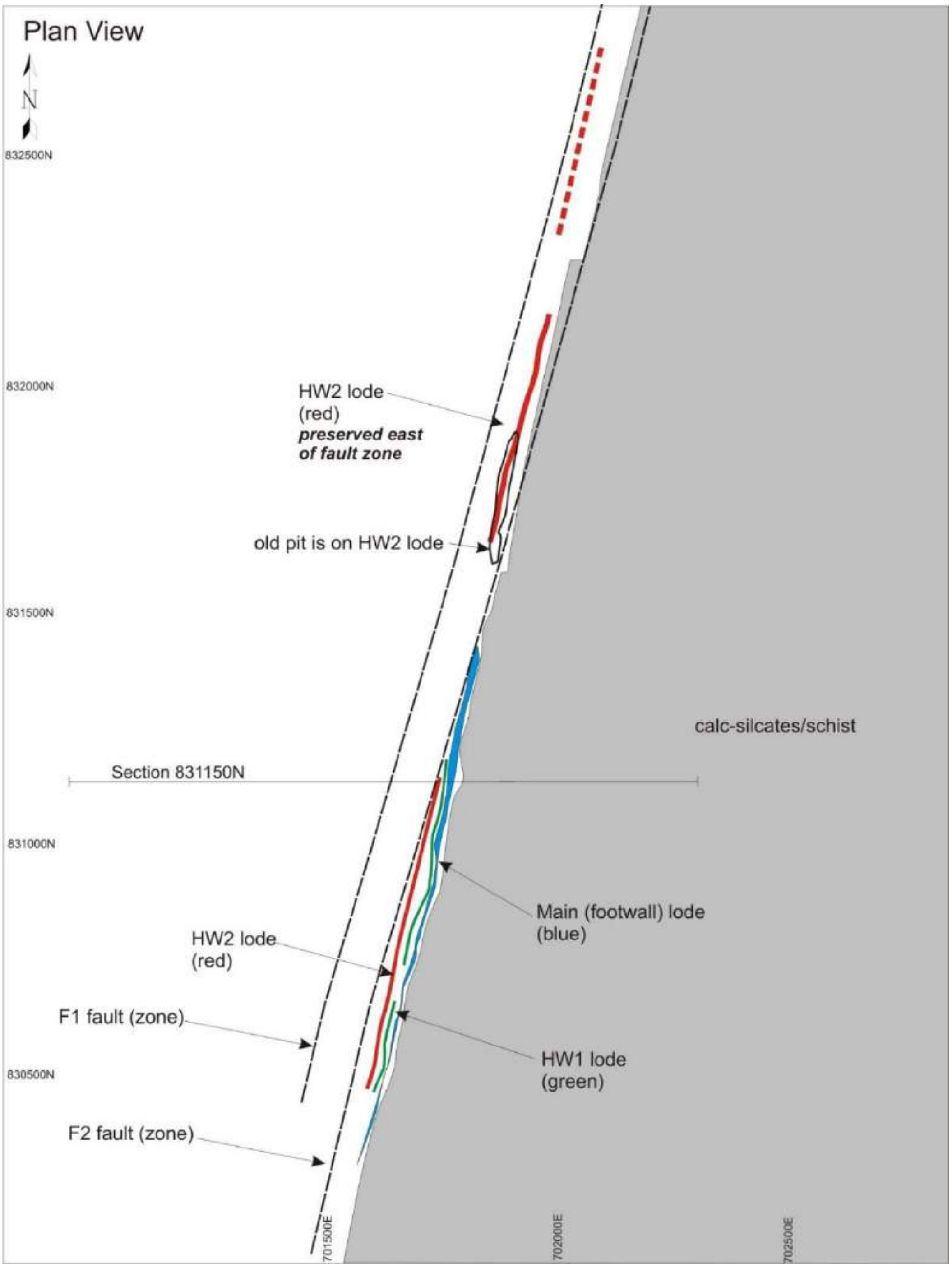


Figure 26 - Plan view map showing mineralised lode arrangement and nomenclature

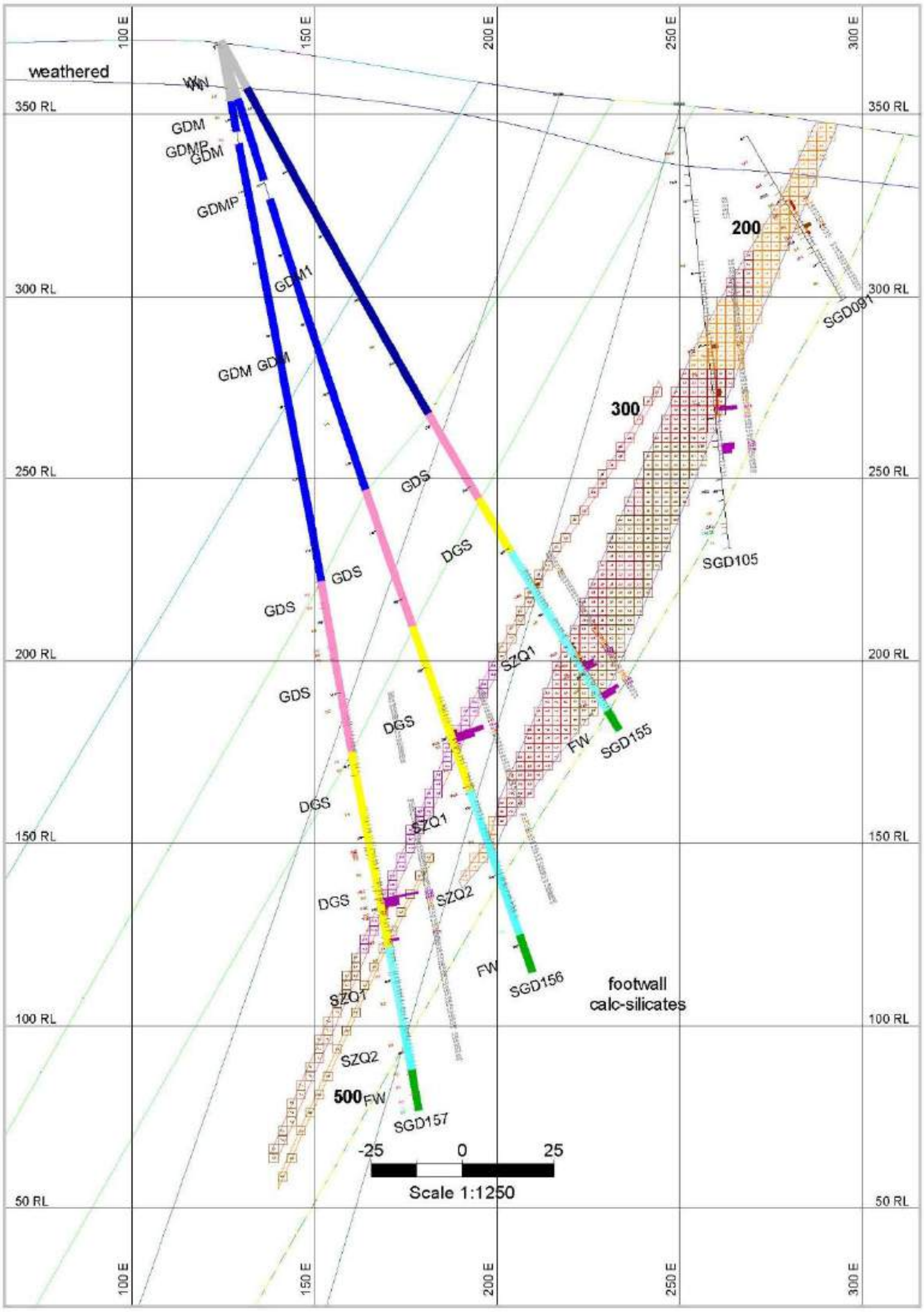


Figure 27 - Cross Section 831150N

9. EXPLORATION

9.1 Introduction

Exploration activities on the Segilola Gold Project have included geological mapping, soil sampling, trench sampling and drilling. Drilling programs have been conducted diamond core (DC) drilling techniques. From 1998 to 2014, all exploration activities were completed by either Hansa Geomin or CGA Mining. From early 2017, exploration has been undertaken by Thor Explorations, following its acquisition of Segilola Gold Project, comprising the mining lease and surrounding exploration licence.

The focus on the previous exploration programs (1998-2014) has been to test the strike length of known mineralisation on mostly 25m-spaced drill sections with the objective of producing a definitive resource estimate. In addition, drilling was completed on the northern and southern extremities to expand the resource in these directions. Limited trenching was also completed to identify the surface expression of the mineralized shear.

The most recent drilling on the project was commissioned by Thor and commenced in April 2017, mainly with the intention of testing the down dip extension of the mineralised zone. The exploration concept is to define potential high-grade extensions capable of supporting an underground mining operation to compliment the currently proposed open pit operation.

9.2 Historical Exploration and Mining Activities

Active modern mining of the alluvial and eluvial deposits began in the region around 1942. The Iperindo reef was first discovered during the working of the eluvial deposits in 1945.

1947: Taken up by Odutola Brothers (one of the larger indigenous businesses, initially cocoa and palm-oil traders), who commenced a limited underground development (comprising a shaft and adit).

1950s: The prospect was sold to a Mr Gomra, an Abadan-based Lebanese expatriate. He commenced developing the surface workings, with excavations being approximately 5 m wide, 15 m deep and 300 m long. Ore was treated by a stamp battery, with manual panning of the crushed material. When operations ceased in 1969, tailings were estimated at 40,000t. Recent sampling suggests this material could have grades of over 5g/t gold, with some samples reportedly exceeding 9 g/t gold.

1953: Investigation of the district by the Geological Survey of Nigeria (de Swardt, 1953).

1965-1966: The Ilesha S.E. (Sheet 243 S.E.) of the Nigeria 1:50,000 Series, was compiled from aerial photography and ground control by the Government of Canada as part of an aid program with the Government of Nigeria. The map sheet has a contour interval of 50 ft. This appears to have been the topographic control used by NMC and Ijesa GeoMin Mining Development Corporation Limited (IGMDC).

1970: Property acquired by Obokun Minerals Developments Limited (OMDC), a local company. They rehabilitated the plant, but operations ceased due to internal company problems.

1976: BRGM was invited by NMC to map and carry out geochemical surveys of the property.

1981: NMC invited Polservice, a Polish consulting company, to undertake a geological review, petrographic and metallurgical studies, and to prepare “reserve” estimation.

1982: The government-owned NMC acquires the two ML’s from OMDC. They carry out an eluvial drilling program.

1983: NMC carries out detailed geological mapping, surveying and soil geochemistry. Old trenches are cleared out and additional trenches excavated.. Six drillholes totalling 447 m was drilled.

1984: NMC carries out additional exploration work, followed by 1,298 m of core drilling in 13 holes.

1986: NMC issued a new EPL.

1987: NMC carries out a further 1,280 m of core drilling in 15 holes.

1992: PNL carries out a detailed pre-investment study and compiled all the data.

1994: PNL enters into a joint venture with NMC.

1995: TML incorporated as a joint venture company (owned NMC 20%, PNL 80%). NML issued Temporary Mining Licence TMiL 19706.

1995: A preliminary assessment report was prepared for NMC and PNL by Neil Cole of N.H.Cole and Associates Private Limited, in October, 1995.

1996: Ilesha Gold Fields Limited, a company incorporated in Alberta, Canada, but managed out of Sydney, Australia, by Neil Cole and others, entered into an agreement to earn 75% equity in the property, and commenced fund-raising activities to fund a preliminary feasibility study. This agreement collapsed. They referred to the project as the “African Queen” property.

1996: TML 19706 assigned to TML and approved for 21 years and a 3 year temporary title.

1997: TML signs a joint venture with Hansa GeoMin Exploration Limited, a German-based mineral exploration company, active across West Africa. Hansa operated through their consultancy company Hansa GeoMin Consult – results were reported in the name of the joint venture company IGMDC. IGMDC between 1997 and 1999 resurveyed the licence; rehabilitated, extended, mapped and sampled the underground crosscut; rehabilitated several old trenches and dug new trenches; all trenches were mapped and sampled; completed ground geophysical and geochemical surveys; drilled three twin diamond holes, two deep boreholes and two additional drillholes (totalling 895m). They completed a statistical study of the assay results, sampled the tailings, completed petrographic and fluid inclusion studies and carried out a “reserve” estimate.

1999: TML-Hansa GeoMin Joint venture terminated.

2007: SGL (then a wholly owned subsidiary of CGA Mining Ltd - CGA), on 25 May 2007 acquired the right to earn up to 51% undivided interest in the tenements. A detailed drill-out of the known mineralised zone commenced.

2011-2012: SGL (now a wholly owned subsidiary of Ratel Group Ltd - RGL) embarked on a 4,200mpetrographic drilling programme to test the southern and northern strike extensions of the already delineated mineralized.

2015-2016: Acquisition of Project by Thor Explorations

2017: Thor undertakes soil sampling program and 4,000m diamond drilling program

9.3 Survey Coordinate System

The coordinate system used for all data collection and surveying on the Segilola property is the Universal Transverse Mercator (UTM) projection, Zone 31 North, using the World Geodetic System 1984 datum (WGS84).

9.4 Trenching

Both Hansa Geomin and CGA Mining undertook several trenching programs. These trenches were typically about 50cm deep and were focused mostly on the outcrop of the gold-bearing vein system. The historic trenching does not provide any useful information outside the immediate vicinity of the drilling coverage. Thor has not carried out trench sampling to date.

9.5 Geophysics

During 1997 and 199, IGMDC contracted Terratec, of Heitersham, Germany, to carry out a ground magnetic survey. The survey totalled about 400 line-km, on a 100m x 10m format. The survey lines were located by DGPS. An EDA Omni Plus proton magnetometer was used, with sensors at 2 m and 1 m heights to permit measurement of magnetic gradient as well as total field. Small-scale images of a number of products were provided. They are represented by a grey scale image of magnetic analytical signal data generated by IGMDC and processed using an analytical signal algorithm. Analytical signal is a data processing method suitable for low geographic latitudes, where magnetic field lines dip at low angles. Unlike other methods such as reduction-to-pole in such environments, it is independent of anomaly trends. It produces high amplitude responses at anomaly contacts.

The mineralisation occurs near and parallel to contacts between gneiss (noisy data, low total magnetic intensity) and the schist/sediment sequence (quite data, moderate TMI). The gaps in the data occur at villages, power lines (northeast of area) and where survey lines are too far apart (>130 m) Figure 28.

IGMDC also carried out trial ground radiometric surveys for which some raw data are available. Hand-held scintillometer readings for approximately 2,920 locations with count readings for K, U and Th channels exist. Only locally do the traverses cross the mineralised zone, and there is no obvious strong association with mineralisation in any of the channels. Highest counts in the K channel are over areas mapped as metasediments, and are presumably a reflection of micas within that unit; K counts seem much lower over the gneissic rocks. U and, to a lesser extent Th, have their highest values to the west and northwest of the mineralisation. Again, response seems to be controlled by lithology, with particular gneissic zones traverses over lithologies in which contents of radiogenic elements are known. Radiometrics is likely to be a useful tool in lithological mapping in this environment, used in conjunction with magnetics. (perhaps orthogneiss or younger intrusives) returning stronger counts. The significance of the magnitude of the scintillometer readings is not obvious without some control

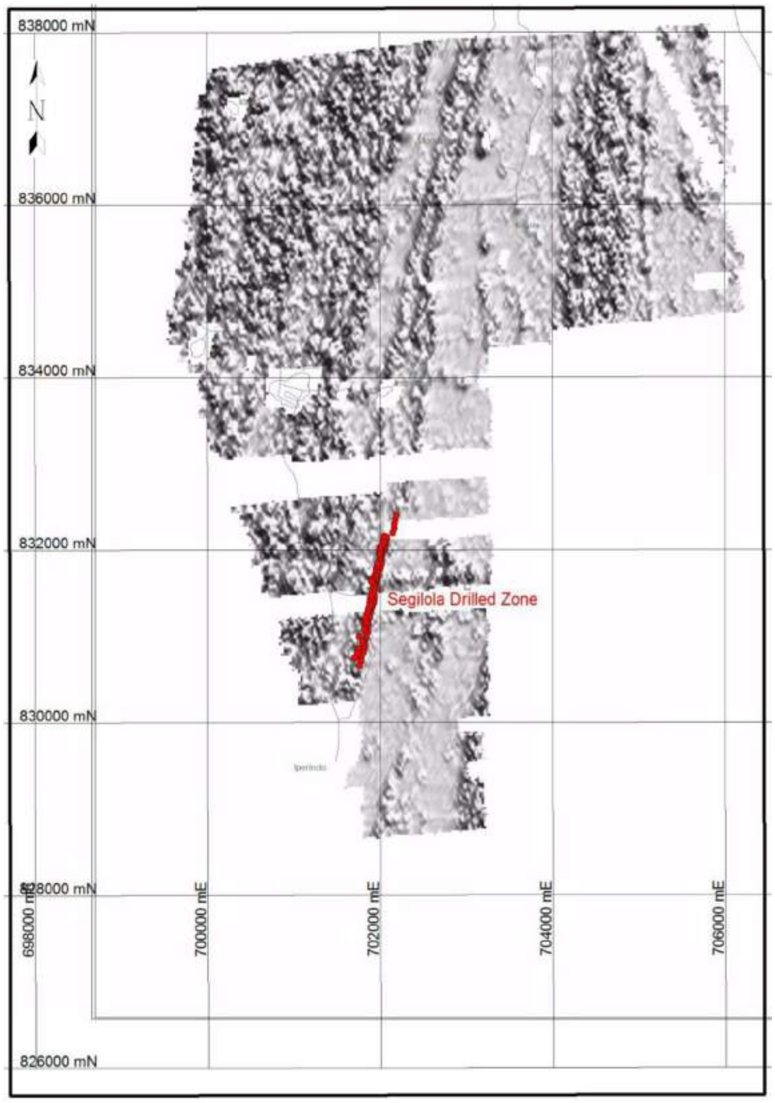


Figure 28 - Grey-Scale Image of Ground Magnetics

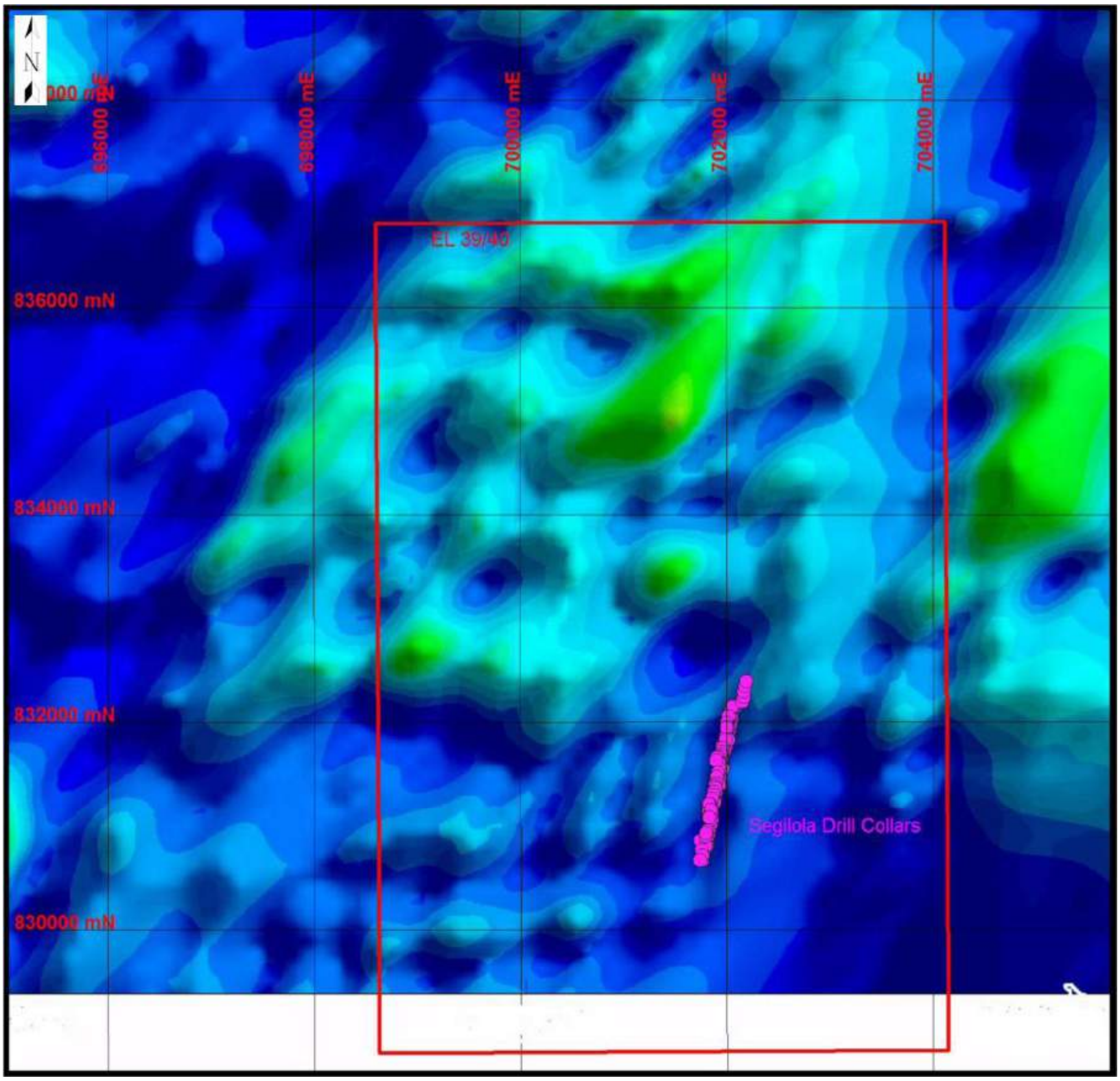


Figure 29 - Image Processed Airborne Total Magnetic Intensity Data, 2008 Survey

In 2008, data used to compile a total magnetic intensity map for the Ilesha (243) 1:100,000 area (part of a national survey) was released. Data was collected by Fugro Airborne Surveys using a Scintrex CS2 Caesium Vapour magnetometer. Flight line spacing was generally 500 m, locally infilled to 250 m (the writer has not interrogated the data sufficiently to know line spacing in the area of interest). Flight lines trend at 135° and sensor mean terrain clearance was 75 m. Radiometrics were flown concurrently, and DTM information derived from a radar altimeter was also generated. Figure 29 shows the total magnetic intensity around the area of EL40/39. A number of regional-scale faults are evident, including the Ifewara Shear Zone to the west of the tenement, and a sub-parallel to parallel structure in the east of the tenement. Several structures trending at ~060° are also apparent, including one that crosses the northern part of the drill-tested area.

9.6 Soil Geochemistry

IGMDC carried out a substantial soil geochemical program. It has only been possible to locate the raw data for a portion of the total survey, but there appears to have been in the order of 500 samples taken. The samples were taken on irregular lines 25 m to more than 200m apart, with samples taken every 10 m along the lines, with every 5 contiguous samples composited, so that plotted along-line sample interval is 50 m. This was done in order to make each analysed sample more representative of its surroundings, and to reduce noise in the data. Samples were taken from an average depth of 0.4 m, and sieved to -80 mesh. Samples were assayed with a detection limit of 0.1 ppb gold. The extents of this survey are shown in Figure 30.

Relocation/re-sampling of anomalous zones, and extension of the survey would be an important early step in on-going exploration at Segilola. Some infill sampling was carried out by SGL/RGL, to establish whether the gold mineralized structures, tested by the 2009 diamond drilling extended north and south of the existing mineralization. A total of 422 soil geochemical samples were collected during October to early December 2010 and analyzed for gold at the SGS Laboratory in Ghana. Soil samples were collected every 25 metres along east-west orientated lines spaced 200 metres apart. To the north, there were 10 lines, each 200 metres apart, thereby testing a northerly strike distance of 2 kilometres. The location of Iperindo Village in the south only allowed the sampling of 2 lines. The soil sampling within the village limits, suggests contamination.

Experience of similar soil sampling programs carried out similar geological, geomorphological and climatic conditions in Ghana have shown that 90ppb Au in soils will be of interest and will require exploration follow-up. A value of over 200ppb Au in soils indicates bedrock gold mineralization that has good potential to be economic. The results of the northern sampling indicated potential bedrock gold scattered in small zones along the full 2,000 metres strike extent. These areas should be tested by trenching and channel sampling. The southern sampling showed that the main mineralized zone as indicated by drilling does continue southwards into Iperindo village and possibly beyond.

In 2016-2017 Thor undertook soil geochemical program, comprising 261 samples, in an attempt to reproduce several of the historic gold in soil anomalies. This program is ongoing and results will be compiled on completion.

The various phases of soil geochemical coverage are shown in Figure 30 and Figure 31.

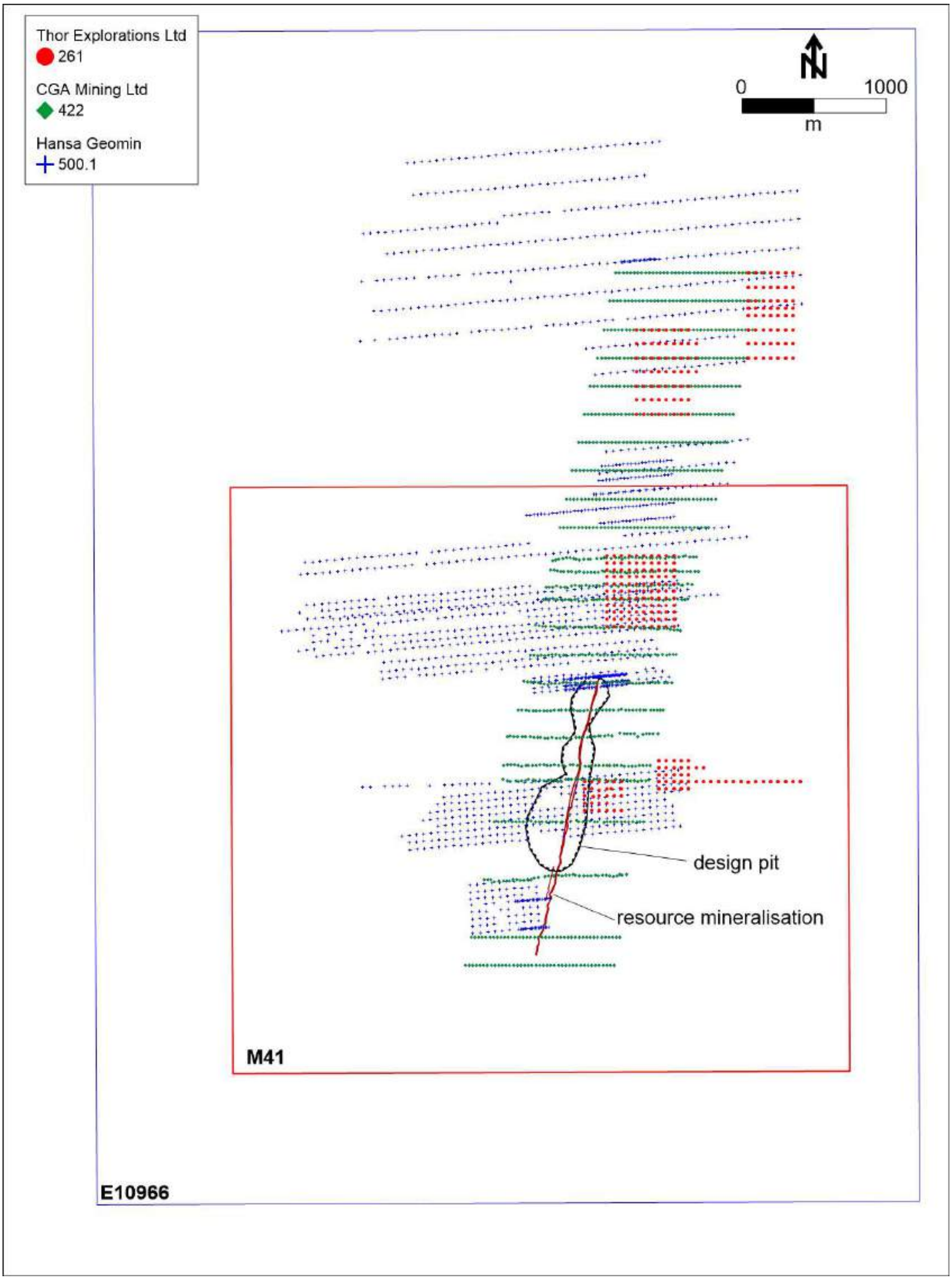


Figure 30 - Soil Geochemical Survey Coverage

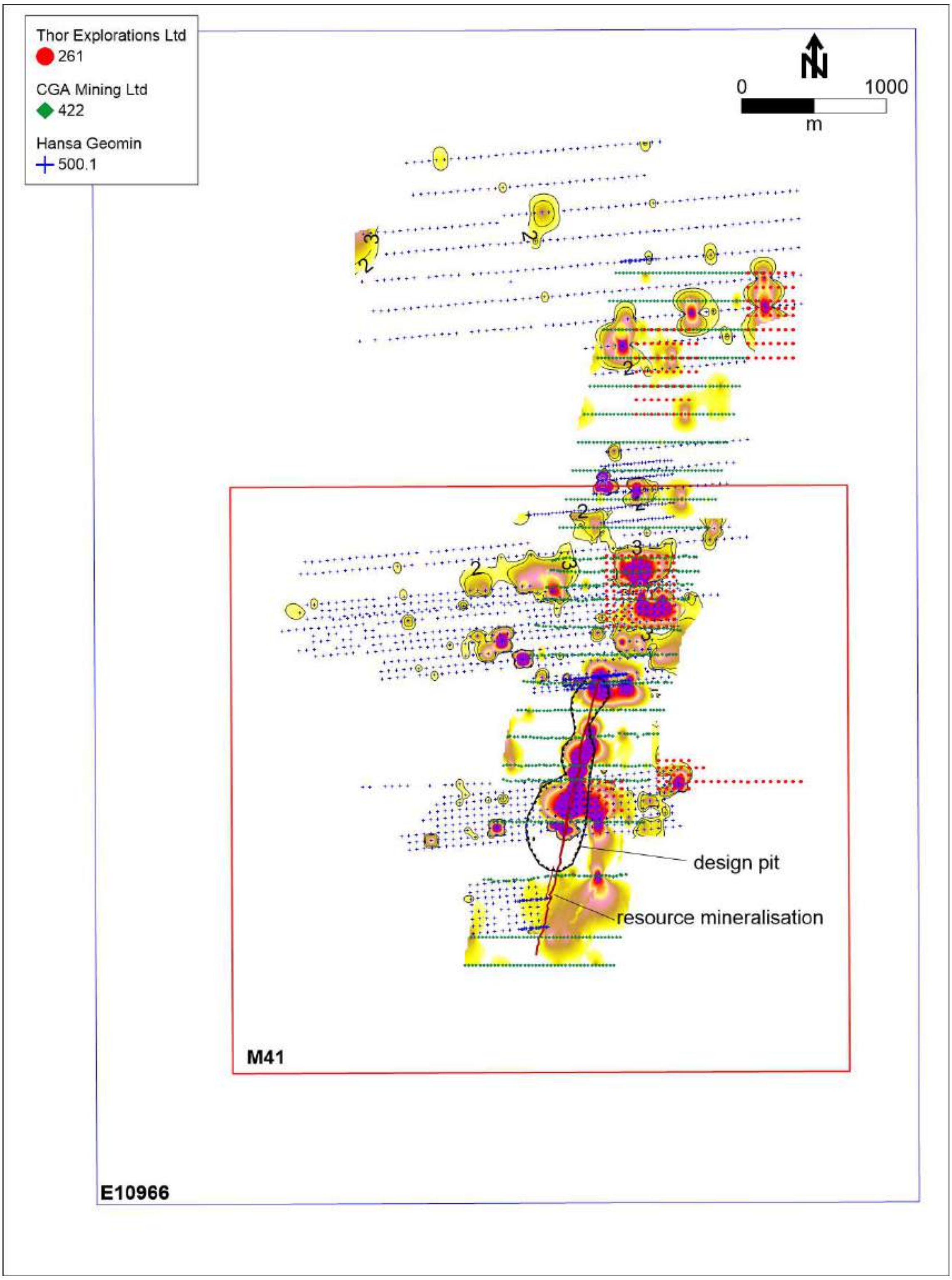


Figure 31 - Soil Geochemical Survey Coverage showing composite gold in soil anomalies

10. DRILLING

10.1 Introduction

The Segilola deposit has been drilled systematically over a strike length of over 1700m. The average strike of the lodes is 010° with an average dip to the west of 65-70°. Holes are located on mostly 25m spaced sections (Figure 5). Where possible, the holes were inclined at -60° to the east. However, due to access problems many holes, particularly towards the south, were inclined up to -90° in order to intersect the lode.

Summaries of the drilling and sampling statistics relevant to the resource estimate are shown in Table 13 and Table 14 respectively.

Table 13 - Segilola drilling statistics

Company	Year	Metres Drilled	No. Holes	Type	Contractor	Comments
CGA/SGL	2008	2,560	32	NQ/DD	Tanylag/Spektra	included in resource estimate
CGA/SGL	2009	9,644	89	NQ/DD	Spektra Geotech	included in resource estimate
CGA/SGL	2011	3,705	36	NQ/DD	GeoHydro	included in resource estimate
Thor	2017	3,533	13	HQ/DD	Century Mining Co.	included in resource estimate
Sub Total		19,442	170			included in resource estimate
Hansa GeoMin	1998	3,857	40	NQ/DD	Geo Core Drillers	excluded from resource estimate
Thor	2017	596	3	HQ/DD	Century Mining Co.	excluded from resource estimate
Sub Total		4,453	43			excluded from resource estimate
Project Total		23,894	213			

* excludes incomplete/abandoned holes

Thor completed two exploration holes outside the resource area. The results of the last exploration hole of the program, below the current resource, had not been received at the time of the resource estimate.

A total of 3,326 assay intervals that fall within the resource area are considered to be representative.

Table 14 – Segilola Sampling Statistics

Sample Type	Total
Fire Assay total	3,326
Fire Assays constrained by model	1,484
1m composites constrained by model	1,323
Screen Fire Assay	11
Standards	202
Blanks	199
Field Duplicates	70
Density Measurements	1,216

Sample intervals varied from 0.20m to 3m. However, within the "lode as logged" the sample interval is predominantly 1m.

The historic data relating to previous exploration campaigns conducted by Hansa Geomin was also imported and reviewed in 3D. Although this data appeared to be spatially correct and the assays results were consistent if not better than the SGL twin-holes, it was decided to exclude it from the resource estimate on the following basis:

- Lack of any QA/QC information on sampling and analytical data
- Lack of downhole survey information that could be verified
- Most of the original core intersections were missing due to full-core sampling

10.2 Drilling Procedures

The drilling procedures used by Thor and CGA are detailed in this section. Thor largely continued with the industry-standard drilling, logging and QAQC protocols and procedures that had been established by CGA. These include systematic drillhole planning, site preparation, drilling, sampling and logging.

10.3 Collar Mark-out and Survey

Drillholes site are initially located using a hand-held GPS. Once the sites are located qualified surveyors, Sphero Grid Surveys (NIG) Ltd, accurately position the planned drill location using a ProMark 2 GPS and Kolida Digital Total Station DGPS. The system utilises a base receiver set up on a control point with a separate rover receiver used for the survey of drillhole collar. The data from both instruments is post processed in Dell and Compac Laptops using Astech Solution Software, Kolida Downloading Software and Autocad software. Due to local terrain difficulties, collars may be repositioned to allow better access for the drilling rig.

On completion of each drillhole, the process is repeated to obtain the final surveyed collar position.

Two control points with iron pegs set in concrete have been established on the Property. Validation checks of the control points are routinely undertaken along with some earlier surveyed holes. Approximately one in ten existing drillholes have been resurveyed with the same instrument as a check on a prior field surveys. Results indicate accuracy of the collar locations is within ±0.3 m for Easting, Northing and RL.

All drillhole collar coordinates are recorded in X,Y,Z using WGS84, UTM Zone 31N .

10.4 Downhole Surveys

Downhole survey measurements were made in each drillhole to determine the spatial position and bottom of the hole. Downhole surveys were carried out by Spektra personnel using a Flexit SmartTool Downhole Survey System. Surveys were generally acquired at 25m spaced intervals downhole.

Century Mining Company (CMC) drillers used a digital single shot Reflex camera with shots taken at between 25 and 30m intervals.

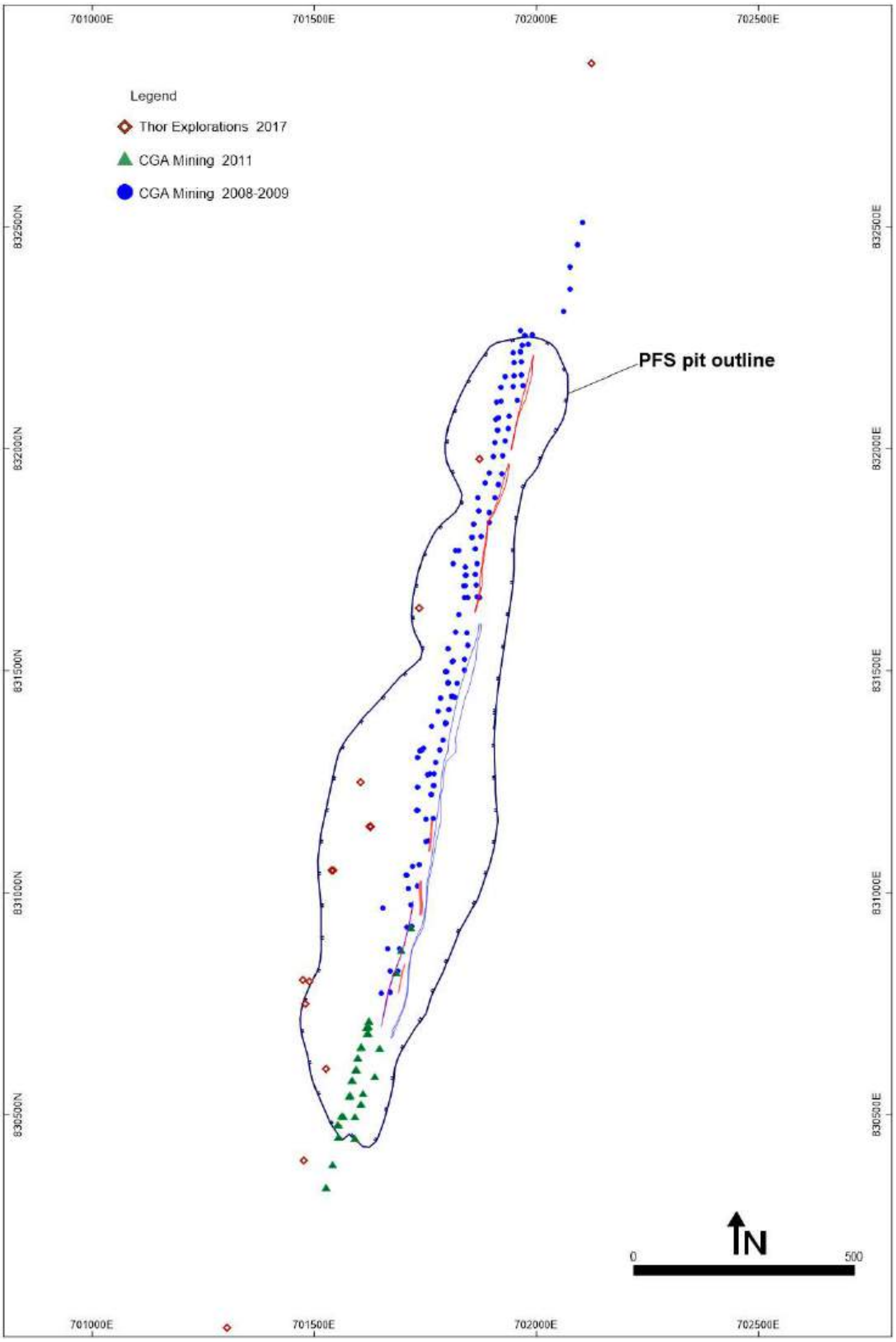


Figure 32 – Drillhole Location Plan

10.5 Hole Logging Procedures

All drilling is logged to a standard that is appropriate for the category of resource which is being reported.

All measurements taken in the field, as well as all data obtained from sampling and logging of drillholes, are captured directly to paper with the data then transferred to Excel spreadsheets. All parameters are logged using codes specific to the project. The logs are checked daily by the senior geologist for completeness and accuracy.

Relevant non-geological data such as hole ID, declination, azimuth, hole depth, core diameter, date, water ingress, etc., are recorded on datasheets.

Core recovery during the drilling program was measured systematically on all core. The average core recovery from within the resource wireframes is over 90%.

10.5.1 Summary and interpretation of all relevant results

A total of 170 drillholes have been completed and this number of holes is considered adequate to define the geometry of the mineralised system. This drilling has generated 3,326 individual assay intervals. Those assay intervals that occur within the resource area are considered sufficient to enable an accurate estimate of the average gold grade within the defined geometry. The data indicate a highly continuous linear steeply dipping quartz vein as the only host for gold in the system. All available data have been used to derive this interpretation. The mineralised quartz has been intersected in nearly every hole and as such the interpretation as described above is considered to be robust. A total of 1,484 assay intervals that fall within the reported resource volume are considered to representative.

10.5.2 Accuracy and reliability of the results

Down hole smearing of gold mineralisation is a major concern in the drill sampling of narrow vein high-grade gold deposits. Such smearing can be mitigated, if not totally avoided, by using the diamond core drilling method. All drilling has been carried out using diamond coring and as such sampling is considered to be highly reliable with assay results producing a true and accurate indication of the in situ grade and geology.

10.5.3 Sample Length And True Thickness Of The Mineralisation

Gold mineralisation is developed within a linear vein that dips at 65-70° towards the west. The vein varies in thickness from 3m to 15m true width. The dominant sample interval is 1m. This sample interval is considered to be optimum to accurately define the mineralised envelope. In order to limit the effect of significantly higher grade intervals influencing the average grade estimate, a top cut of 60g/tAu has been applied.

11. SAMPLE PREPARATION, ANALYSES AND SECURITY, CORE SAMPLING PROCEDURES

11.1 Core Sampling Procedures

Sampling procedures for the half core samples involved marking the sample boundary on the core then cutting or breaking the core at that boundary. A diamond saw was then used to cut the core lengthways along the sample interval. One half was sent for analysis, the remaining half initially retained in the core tray.

The following procedures were followed:

- Sampling is always half-core and is generally made at one metre intervals.
- Sampling commences at significant geological boundaries that are considered to represent a distinct change in grade. Such boundaries could be structural, lithological or alteration zone contacts. The sample lengths either side of this boundary should not be less than 0.5m and no more than 2m and be adjusted to return to even 1m sampling intervals as soon as is geologically sound.
- Where barren zones have been clearly identified, at the discretion of the senior geologist, half core is sampled over 5m on both sides of the ore zone at 1m intervals.

11.2 Sample Preparation

Once intervals for sampling are recognized these intervals are noted on the drill log. An aluminium tag (or a core marker) showing the sample number and depth from and to, is then wired or riveted in to the core tray at the start of the interval.

Before the core is cut it is turned to ensure that the veins are cut at the optimum angle. If there is more than one vein set and these are at different orientations then the core is turned to allow cutting of the main auriferous veins at the optimum angle. If the core is relatively soft, friable or likely to shatter it is wrapped in masking tape to ensure that the sample does not disintegrate under the core saw. The core is then cut down the orientation line.

For controlling the quality of performance of the principal laboratories standard samples were added to each batch of samples. Standards were inserted into the sample stream at typically after every 10th (CGA) or 20th (Thor) sample.

All samples were weighed upon receipt (method code PWE-100). Core was dried, crushed to 70% passing 2mm, split (250g sub-sample) and pulverized to 85% passing 75µm (method code PRP-910). Laboratory QAQC involves the use of internal lab standards using certified reference material, blanks, splits and duplicates as part of in house QAQC protocols.

Certified reference materials, having a good range of values, were inserted blindly and randomly. Results highlighted that sample assay values were accurate and that contamination had been minimised.

11.3 Logging Procedures

Rock type, texture, veining, weathering, alteration and structure were recorded on paper logs then later transferred to Excel spreadsheets.

Core recovery and RQD data was recorded for all diamond holes and was entered into an Excel spreadsheet.

11.4 Sample Dispatch

Drill samples were submitted to the laboratory as loose pieces of core contained within appropriately numbered plastic bags. The following procedures were followed:

- Consolidate all samples for one hole at site. Place the sample numbers under one submission form only (i.e. one submission number).
- Weights are recorded for individual samples then are put into manageable loads of large polyweave sacks.
- Senior personnel transported the sample batch to DHL couriers in Lagos for delivery to:
    - CGA: SGS Laboratory in Ghana
    - Thor: MS Analytical Laboratories (Vancouver) – airfreight

11.5 Sample Security

Prior to dispatch the diamond core was stored at the exploration office in Ilesha. The office and sampling facilities are located within a single, walled compound which has a gated entrance manned continuously by a security guard.

Samples are packed onto an independently owned and operated vehicle by senior staff technicians under the supervision of senior staff geologists.

11.6 Gold Analyses

Sample preparation and analyses were undertaken by either SGS Laboratories in Tarkwa, Ghana (Table 15) or MS Analytical Laboratories (Vancouver). Both laboratories meet ISO9001:2000 requirements.

Table 15 – Analytical Methods

Laboratory	Elements	Method	Code	Detection
SGS Tarkwa	Au	Fire Assay	FAA 505	0.01ppm
SGS Tarkwa	Ag, Cu, Pb, Zn, As, Mo, Sb	Aqua Regia	ARA 155	
MSAnalytical	Au	Fire Assay	FAS-221	0.01ppm
MSAnalytical	Total Au	Metallic Screen Fire Assay	MSC-150	0.05ppm
MSAnalytical	SG	water immersion	SPG-410	

SGS internal laboratory quality control procedures involved the analysis of 691 (12%) duplicates (AuR) of which 128 received a second check (AuS). Refer to Figure 33.

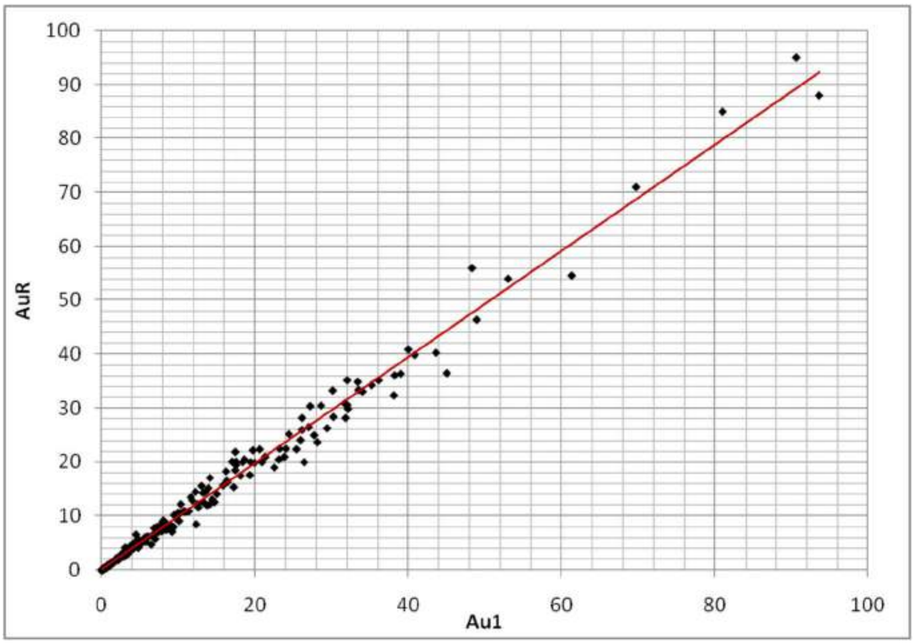


Figure 33 - Scatter Plot of Laboratory Check Assays (Au1 versus Au-R)

11.7 Density Measurements

Bulk density determinations were taken by CGA for 1,071 individual full-core samples. Core from every hole was selected, with the determinations carried out on site using the immersion method, with weight in air and weight in water used to determine the displacement and then density. An average density of 2.67 was obtained from all samples, and this average was used in the resource model as a basis for determining tonnages. There was no variance in bulk density between the lodes and host rock. During Thor's 2017 drilling campaign a further 145 density measurements were conducted by MS Analytical Laboratories from core samples mostly obtained at depth and outside the indicated resource limits. These samples averaged 2.65. Figure 34 and Figure 35 show the distribution and range of the bulk density sampling.

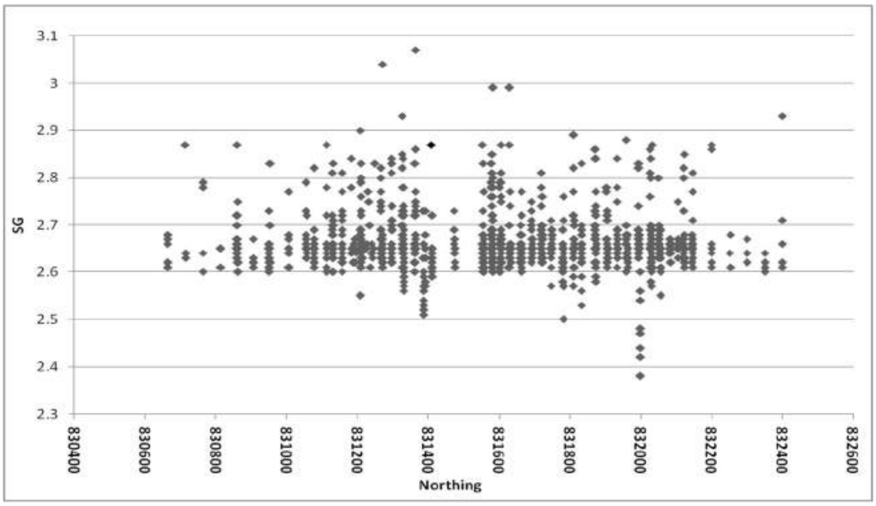


Figure 34 - Variation in Bulk Density along Strike

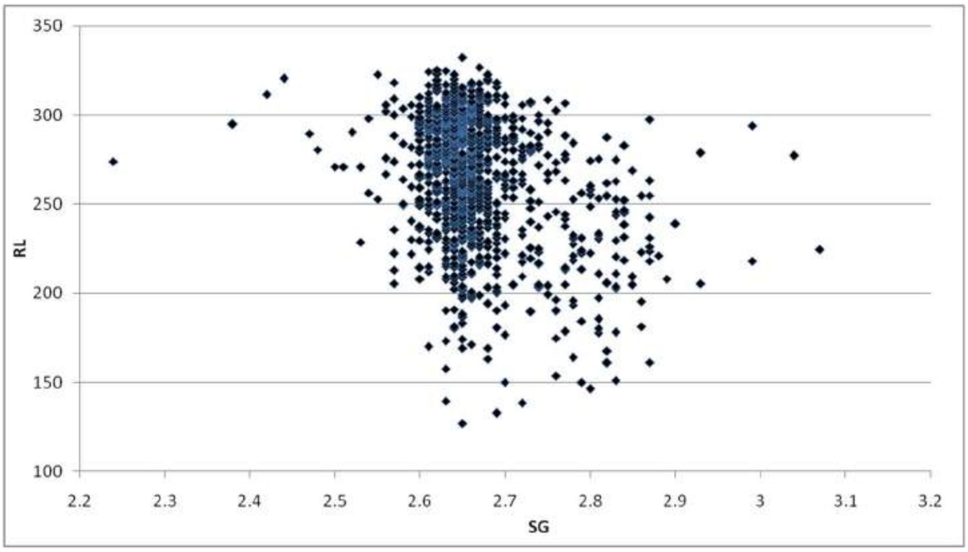


Figure 35 - Variation of Bulk Density with Depth

12. DATA VERIFICATION

12.1 Independent Qualified Person Review and Verification

Mr Anthony Keers of Auralia Mining Consulting Pty Ltd, visited the Segilola Gold Project from 5-7 May 2017. Steps undertaken to verify the integrity of data used in this report include:

- Field visits to the areas outlined in this report.
- Inspection of diamond drilling activities, sampling and logging procedures.

Mr Christopher Speedy of Auralia Mining Consulting Pty Ltd, undertook the following steps to verify the integrity of data used in this report:

- Review of the previous NI 43-101 report for the project titled "Updated Resource Estimate for the Segilola Gold Deposit, Osun State, Nigeria for Thor Explorations Ltd, February 2016"
- Validation of drilling, geology and assay database (including checks overlapping intervals, samples beyond hole depth and other data irregularities.

The Qualified Persons have reviewed and verified sections of this Report prepared by Thor.

Based on this review work, the Qualified Persons are of the opinion that the dataset provided by Thor is of an appropriate standard to use for resource estimation work.

12.2 Sampling QA/QC

Core recovery and sample weights are routinely collected during drillhole sample collection and logging. The recovery data is recorded into field logs that are retained at site then the data is manually entered into a Microsoft Excel spreadsheet.

12.3 Gold Assay QA/QC

Quality control procedures have been in place for the duration of the drilling program.

The QA/QC programs in place include the following:

- A total of 202 standards submitted on a routine basis in the sample stream
- A total of 199 blanks submitted on a routine basis in the sample stream
- Inter-laboratory checks of pulps

12.3.1 Standards

For controlling the quality of performance of the principal laboratories standard samples were added to each batch of samples. Standards were inserted into the sample stream at typically after every 10th (CGA) or 20th (Thor) sample.

The standards used by CGA were certified standards prepared by Geostats Pty Ltd of Western Australia. Thor utilised certified laboratory supplied standards (Table 16). Thor used standards supplied by both Geostats Pty Ltd and African Mineral Standards, South Africa.

There is a weighted variance of 0.06ppm that equates to an average negative bias of 4% towards the assay data. This bias is effected by standards in the 1-2ppm range. Outside this range the variance are negligible. Overall, the standard assay results indicate that acceptable accuracy was achieved and indicates a high level of accuracy for the analytical laboratories and the assay method given the arguably statistically low population of data.

Table 16 - List of Standards

Company	ID	Std Au (FA)ppm	Confidence Interval	Std Au (AR)	Number	Assay Auppm	Variance ppm	% Bias
CGA	G905-1	1.61	+/-0.12	1.14	27	1.2	-0.4	-25.5
CGA	G904-3	13.66	+/-0.135	13.73	26	14.6	0.9	6.9
CGA	G306-4	21.57	+/-0.164	21.73	24	21.8	0.2	1.1
CGA	G907-5	1.34	+/-0.009	1.32	27	1.3	0	-3.0
CGA	G905-6	5.96	+/-0.055	5.86	27	6.2	0.3	4.0
CGA	G399-9	6.27	+/-0.061	6.08	26	5.8	-0.4	-7.5
Thor	0174	2.12	+/-0.14	2.58 (ICP)	20	1.9	-0.22	-10.4
Thor	0175	0.5	+/-0.04	0.34(ICP)	29	0.49	-0.01	-2.0
Thor	G912-4	1.91	+/-0.014	1.95	10	1.94	0.03	1.6

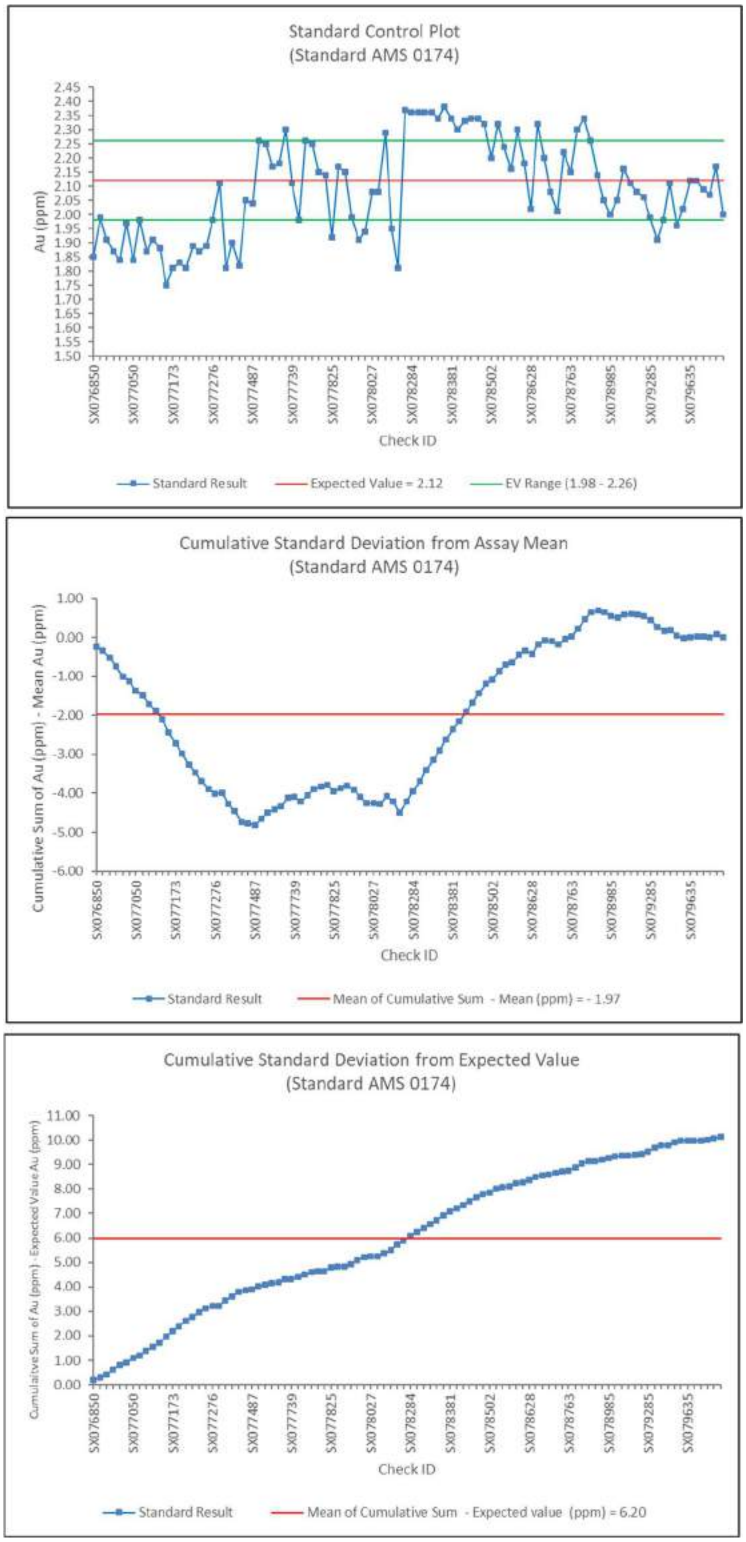


Figure 36 – QA/QC Plots AMS0174

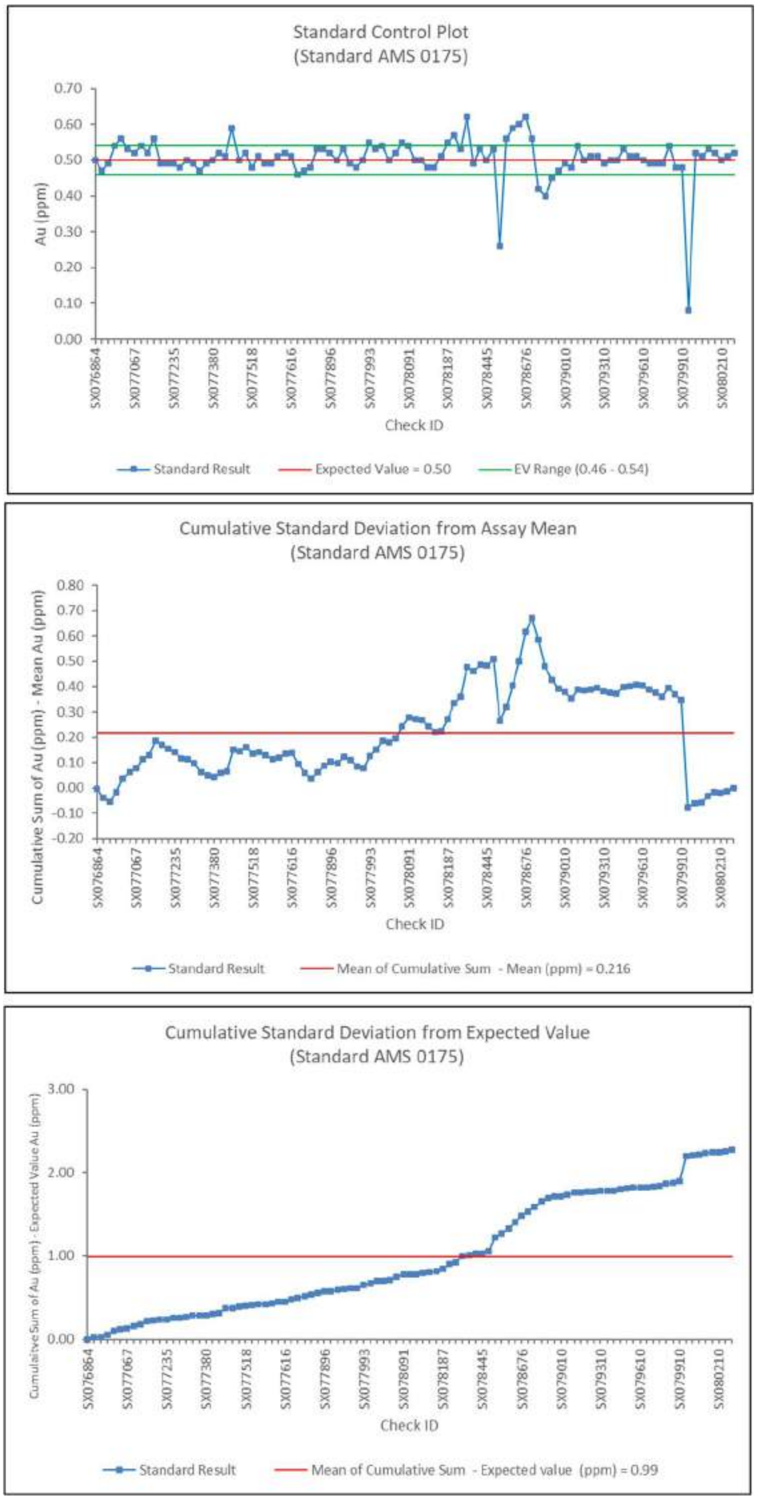


Figure 37 – QA/QC Plots AMS0175

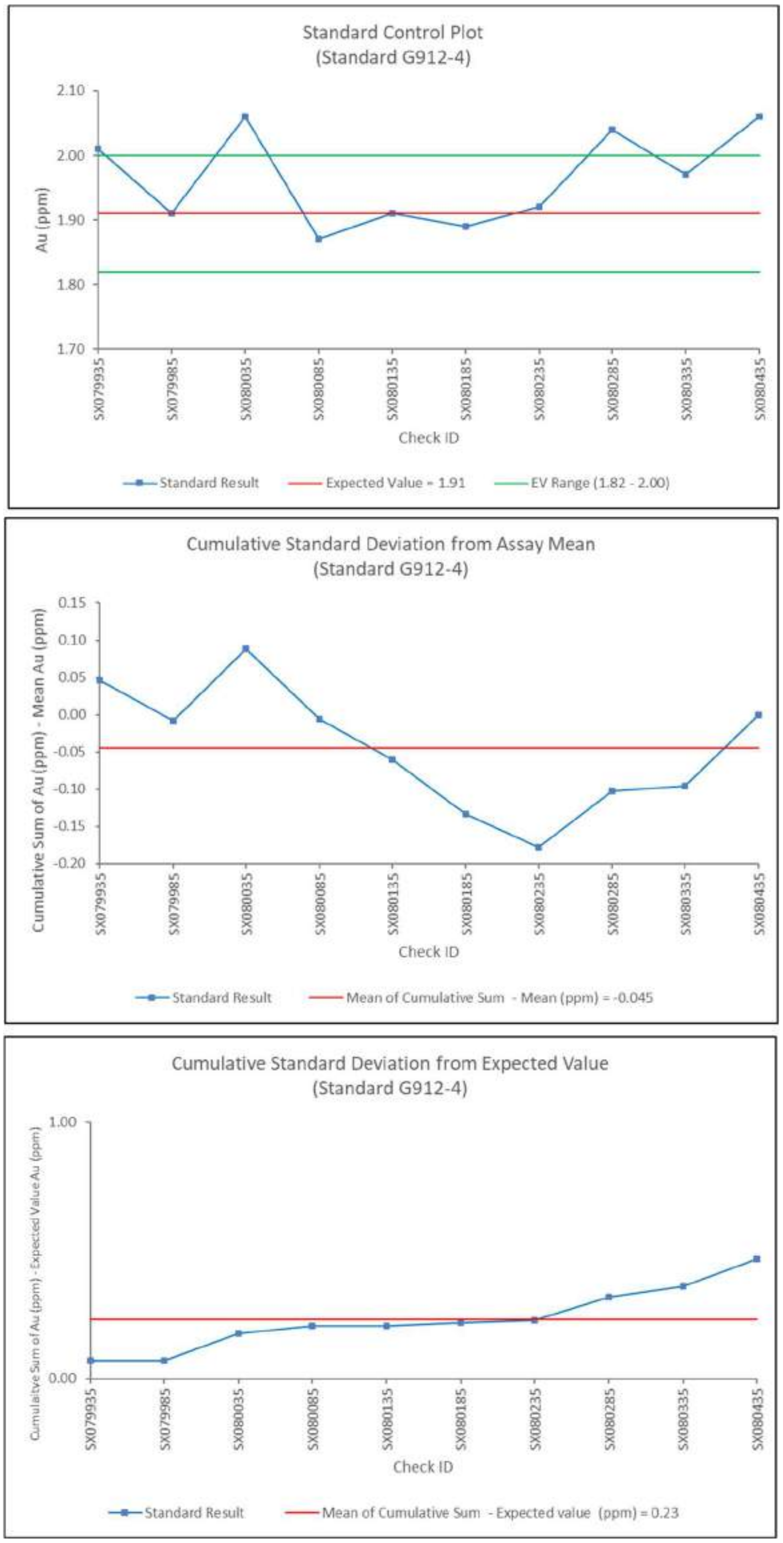


Figure 38 – QA/QC Plots G912-4

12.3.2 Blanks

The quality of the sample preparation process was controlled by adding blank samples to batches of samples. Blanks are used to test for contamination during the sample preparation process and are derived from typically silica (quartz) sand.

The blanks used by CGA were obtained from samples taken from a construction quarry located near Ibadan between Ilesha and Lagos. The rocks comprised mainly granites and gneissic granites and were assayed beforehand to confirm that they were free of any gold or base metal mineralisation. Blanks were inserted into the sample stream at between 4 and 20m intervals.

Of the 183 blanks submitted by CGA a total 134 were at or below detection, 43 ranged between 0.01 and 0.09ppm Au (detection limit 0.01ppmAu). Five blanks reported greater than the detection limit ranging from 0.15 to 0.93ppmAu.

The average of 15 of the 16 blanks submitted by Thor, nine were below detection and six averaged 0.024ppmAu. One blank, the first in the program, returned an assay of 0.72ppmAu. However, it is thought that a standard sachet was inadvertently inserted instead of a blank. QAQC plots for these together with 45 blanks used in the laetr part of the CGA program are shown in Figure 39.

Thor used certified laboratory-supplied blanks made from homogenised silica quartz (0166) with Au content of <0.001ppm.

The laboratory QA/QC report for MS Analytical (Vancouver) is provided in Appendix 1.

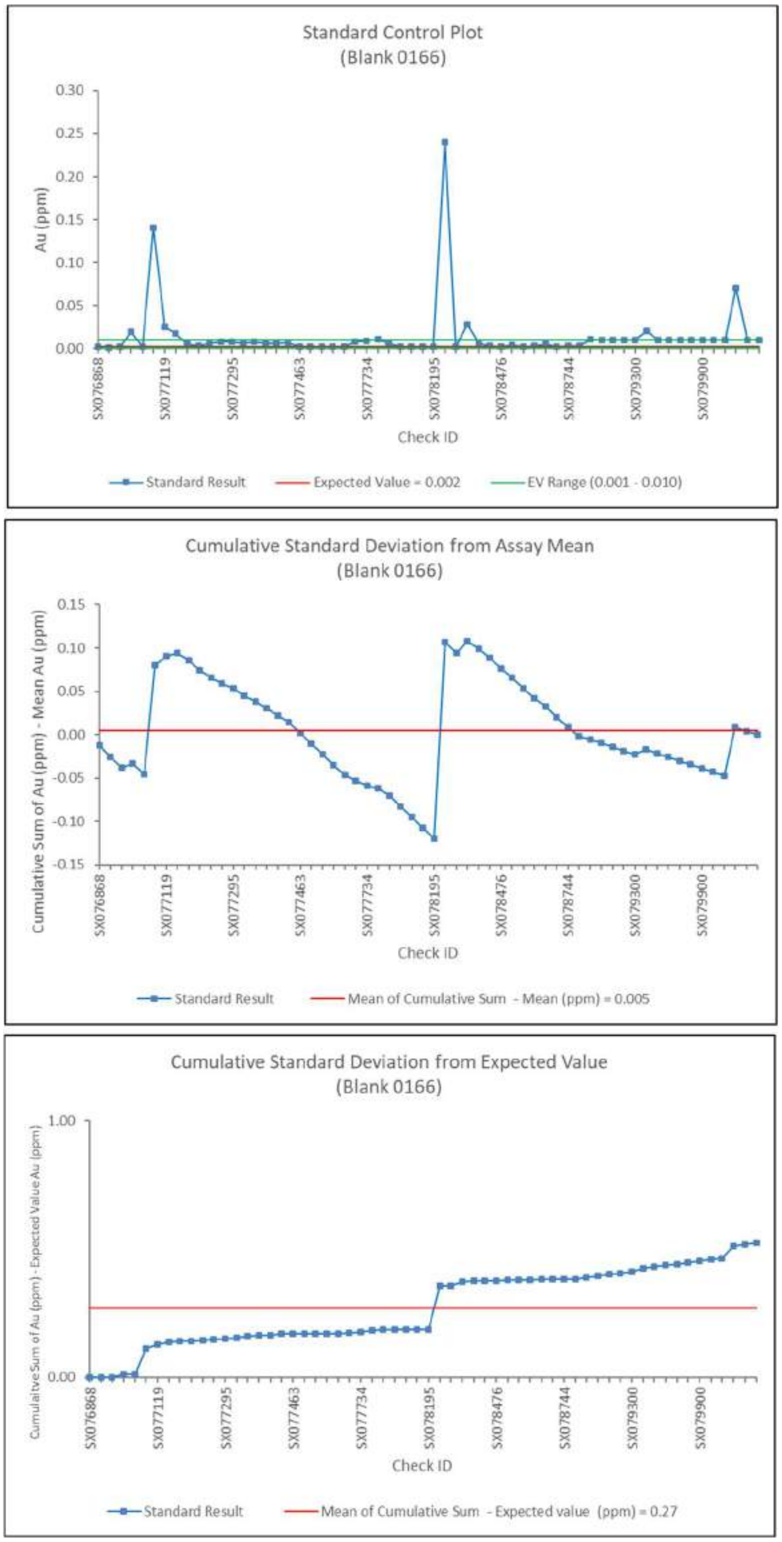


Figure 39 – Field Blank QA/QC Plots

12.3.3 Field Duplicates

Field duplicates are used to determine sampling error and also give an indication of the precision of the data pairs (original vs. duplicate). Diamond core duplicates comprised a second quarter core (HQ) or second half core (NQ) sample over the exact interval as the original sample. A total of 70 duplicates were available for comparison. Correlation plots (Figure 40) show some variability which is consistent the nuggety nature of the mineralisation. In general, the data indicates reasonable precision for the fire assay method.

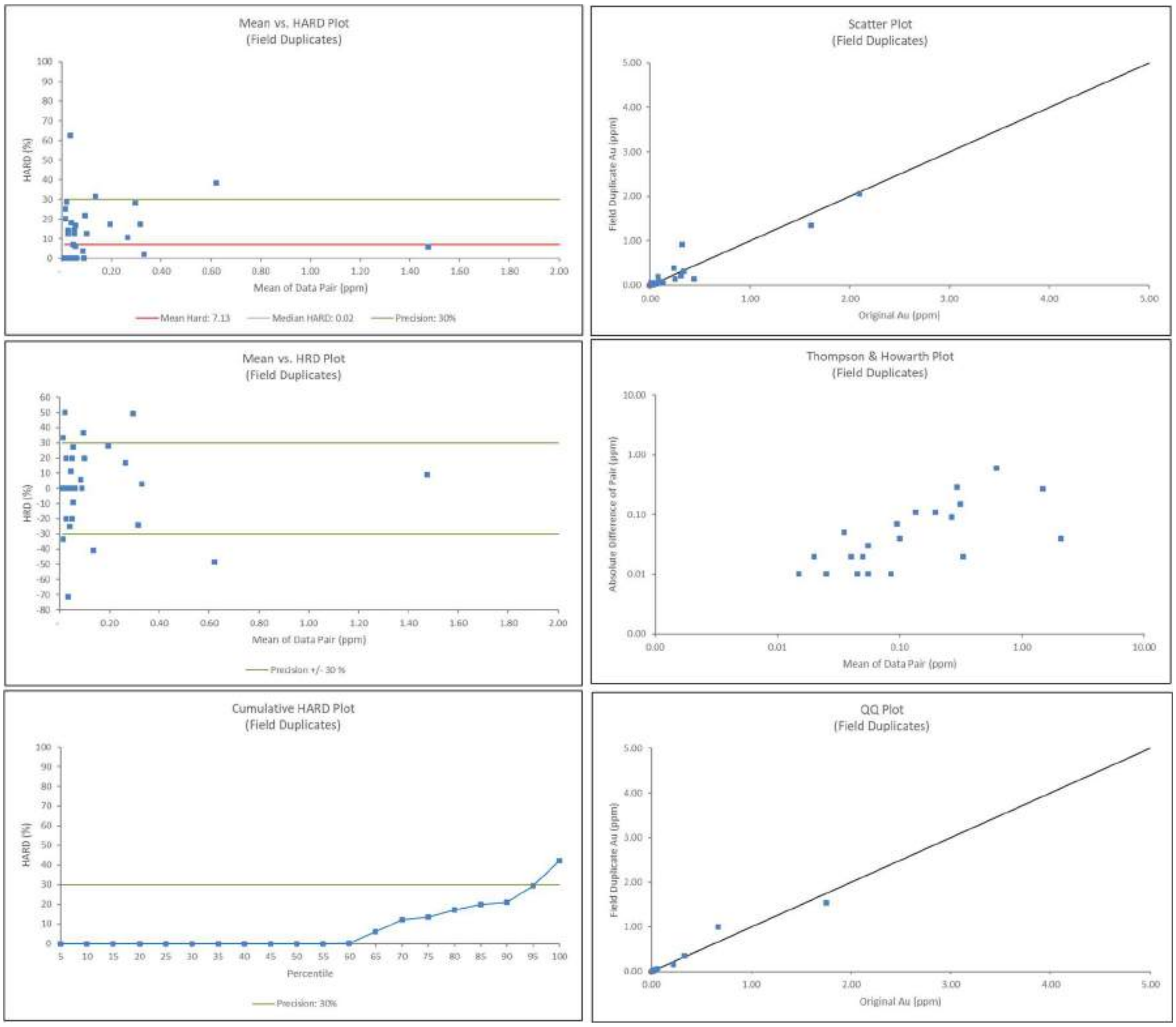


Figure 40 – QA/QC Plots of Field Duplicates

12.3.4 MS Analytical Laboratory QAQC

A total of 1,373 samples were submitted to MS Analytical including core and client inserted standards and blanks between May and July 2017.

All samples were weighed upon receipt (method code PWE-100). Core was dried, crushed to 70% passing 2mm, split (250g sub-sample) and pulverized to 85% passing 75µm (method code PRP-910). Two preparation blanks were inserted in each job as well as preparation duplicates at a rate of 1 per 30 samples.

12.3.4.1 Preparation Blanks

Preparation blanks are used to monitor contamination in the sample preparation process. All results are within the acceptable limit of 0.03 ppm for gold.

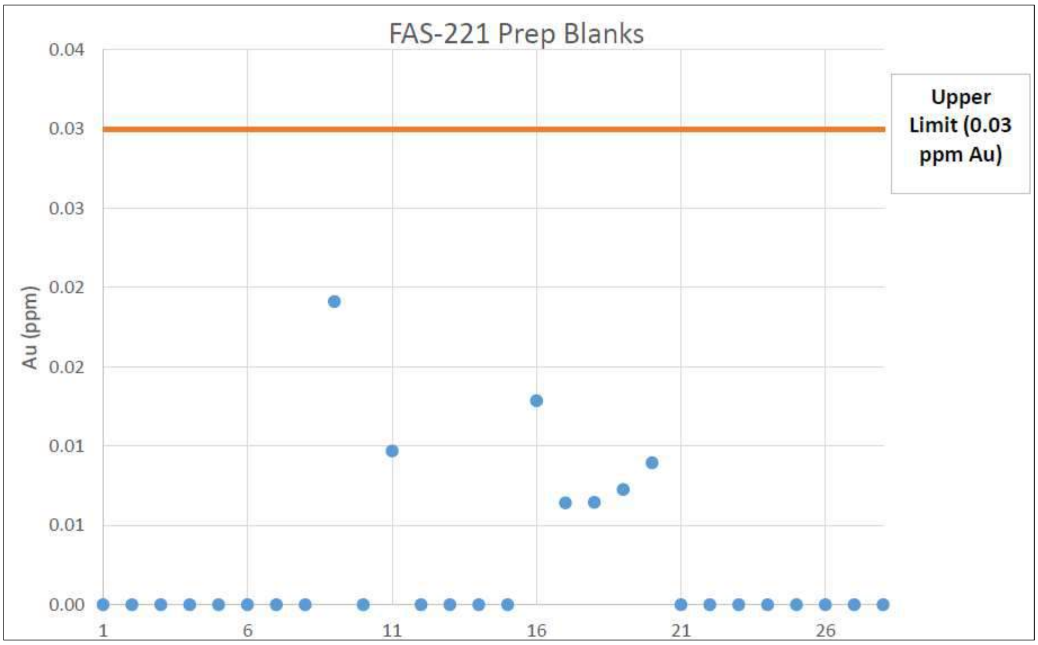


Figure 41 - Control Chart for Preparation Blanks

12.3.4.2 Preparation Duplicates

Preparation duplicates are split after the crushing stage. The results of the preparation duplicates are plotted against the original results in Figure 42. The calculated R2 value is 0.9952.

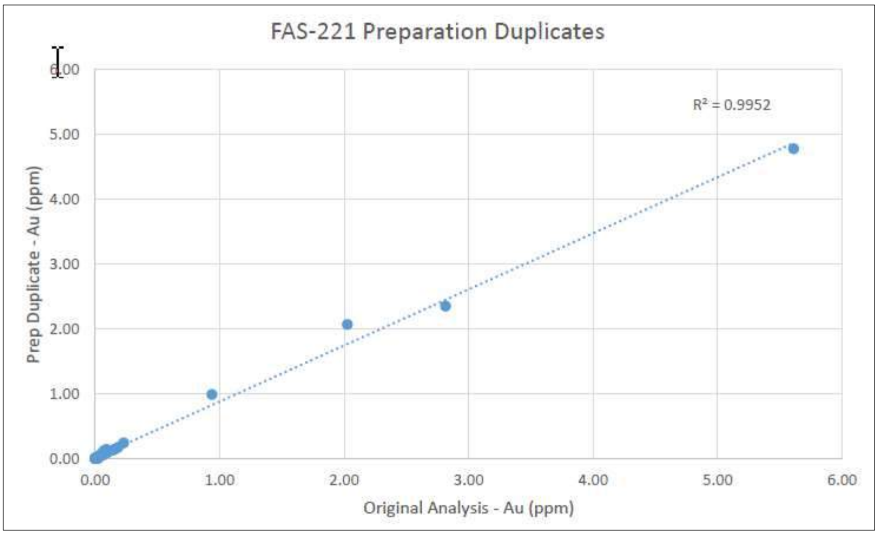


Figure 42 - Preparation Duplicates vs. Original

12.3.4.3 Method Blanks

Method blanks were inserted in every batch at a rate of 2 per 36 samples and are used to monitor contamination in the process. Results are within the acceptable limit of 0.03 ppm for gold by method FAS-221.

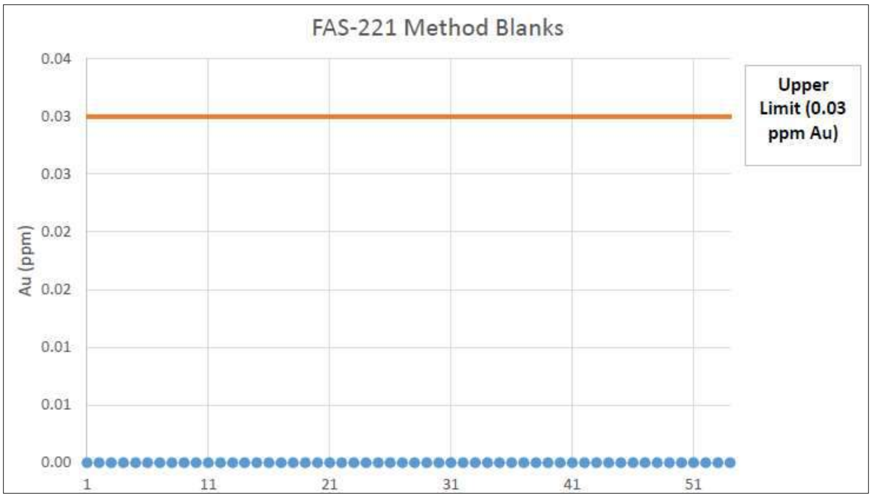


Figure 43 - Control Chart for Method Blanks

12.3.4.4 Analytical Duplicates

Analytical duplicates are taken at the weighing stage and are used to evaluate precision. The results of the analytical duplicates are plotted against the original results in Figure 44. The calculated R2 value is 0.9999.

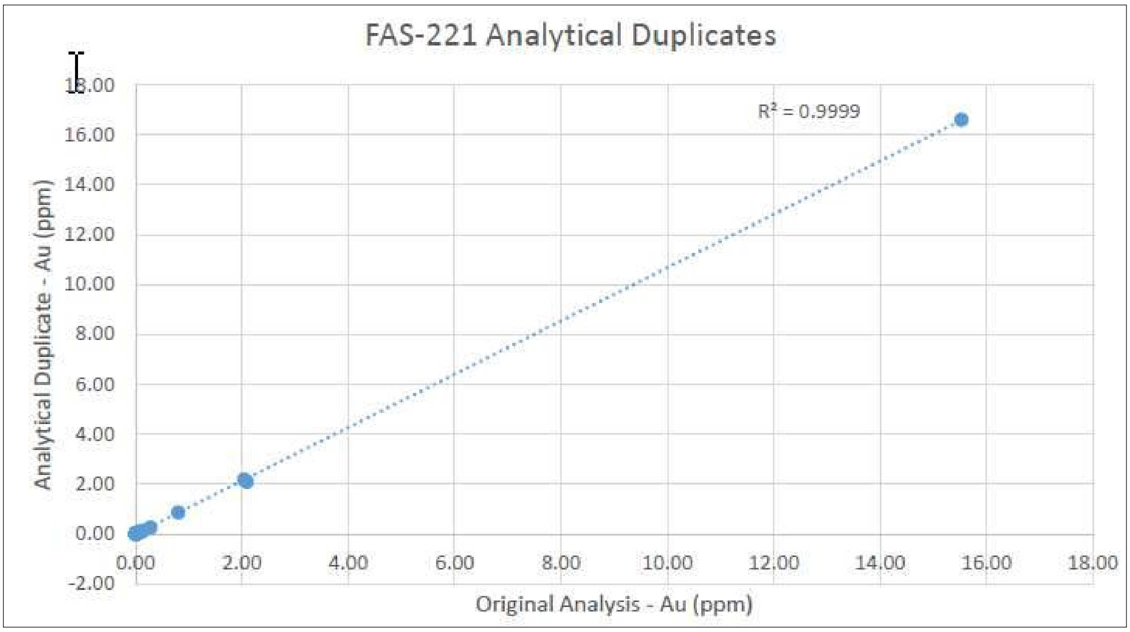


Figure 44 - Analytical Duplicates vs. Original

12.3.5 Metallic Screen Fire Assay

Samples were analysed for gold by lead collection fire assay followed by AAS finish (method code FAS-221). Any Au assays greater than 10ppm were re-analysed by 1000g metallic screening (method code MSC-150). Despite some high variability in several samples the overall variation is negligible as shown in Table 17 and Figure 45.

Table 17 - Comparison between Fire Assay and Metallic Screen Fire Assay

HOLE ID	From	To	FAS-221	MSC-150	Variance (ppm)	Variance %
SGD155	197.6	198.0	15.95	16.55	0.60	4%
SGD155	198.0	199.0	12.30	15.30	3.00	24%
SGD155	199.0	199.6	15.53	18.54	3.01	19%
SGD155	207.0	208.0	25.66	24.14	-1.52	-6%
SGD155	208.0	208.8	17.99	15.27	-2.72	-15%
SGD156	199.7	201.0	40.43	43.36	2.93	7%
SGD156	201.0	201.9	25.72	39.53	13.81	54%
SGD157	237.5	238.0	48.90	39.19	-9.71	-20%
SGD157	238.0	238.9	20.48	20.72	0.24	1%
SGD157	238.9	240.0	20.23	8.89	-11.34	-56%
SGD157	248.9	249.5	10.99	8.68	-2.31	-21%
Average			23.11	22.74	-0.36	-0.01

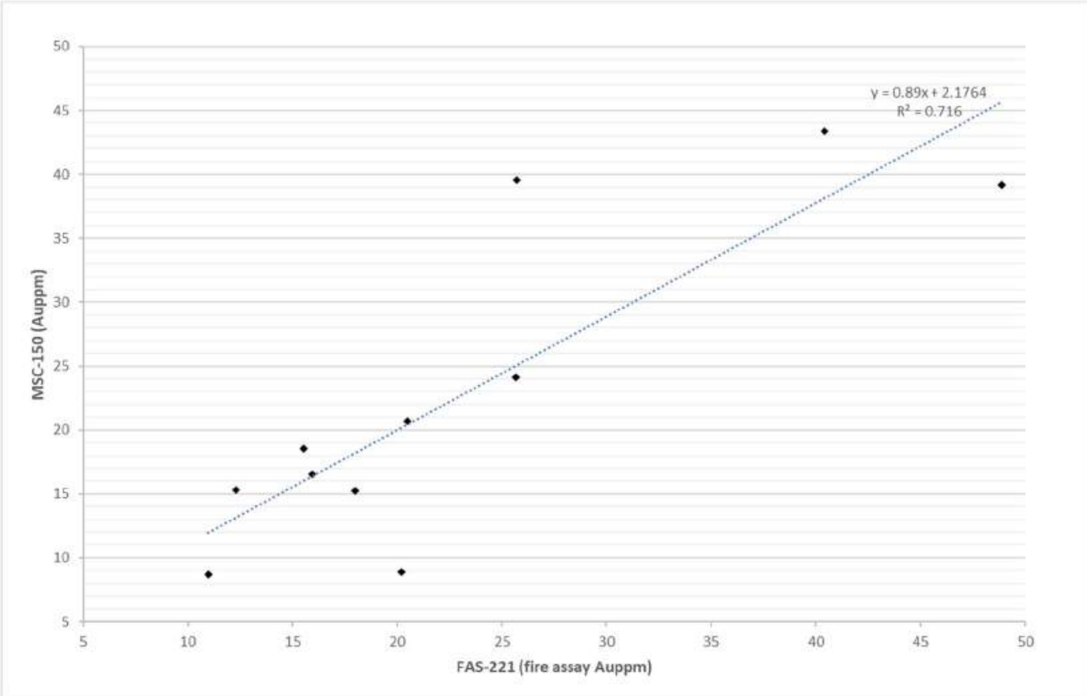


Figure 45 - Correlation between Fire Assay and Metallic Screen Fire Assay

12.4 Inter-laboratory Checks

The inter-laboratory checks on SGS Ghana, comprising 31 check assays, were carried out at Genalysis Laboratories located in Tarkwa, Ghana (Table 18). This data indicates no systematic bias in the SGS assays (Figure 46).

Table 18 - Inter-laboratory Assay Data

Hole-Id	Sample No	SGS		Genalysis		
		Au	Au(r)	Au	AuDR	AuR
SGD001A	SX070802	32.00		29.35	23.2	
SGD001A	SX070805	0.69		0.65	0.89	
SGD007	SX070127	1.31	-	1.75	1.91	
SGD013	SX070465	0.14	0.16	0.13	0.15	
SGD018	SX070681	9.3	10.1	10.2	12.12	9.7
SGD019	SX070713	12.3	8.5	7.04	7.57	
SGD020	SX070777	0.13		0.17	0.16	0.22
SGD026	SX071104	22.5	-	14.06	14.47	
SGD029	SX071242	3.74	4.51	1.94	2.19	2.04
SGD029	SX071243	0.67		0.71	0.8	
SGD030	SX071285	6.93	5.8	6.85	7.25	
SGD031	SX071361	3.42		3.18	3.16	
SGD033	SX071512	22.5	19	14.94	20.54	16.52
SGD039	SX071878	0.96		0.99	1.08	
SGD042	SX071984	1.62		1.48	1.52	
SGD043	SX072018	28.6	30.5	43.7	30.12	
SGD045	SX072110	18.4	20	26.88	22.39	
SGD046	SX072153	30.1	33.3	44.1	38.12	
SGD052	SX072406	0.21		0.21	0.3	0.2
SGD052	SX072409	1.04	0.88	1.16	0.9	0.9
SGD053	SX072449	2.3		2.15	2.75	1.82
SGD061	SX072769	27.7	25	26.72	23.29	24.78
SGD062	SX072842	33.4	34.9	30.89	34.02	58.12
SGD078	SX073657	27.2	30.4	22.55	25.17	25.79
SGD083	SX074007	24.4	25.2	27.41	26.01	25.99
SGD084	SX074099	11.8	12.9	15.54	13.68	
SGD088	SX074610	28.1	23.7	20.73	16.88	18.74
SGD091	SX074879	2.95	2.82	2.54	2.58	
SGD092	SX074899	0.73	-	1.33	1.2	
SGD093	SX074706	10.1	9.11	10.26	11.25	
SGD101	SX075108	0.75	-	0.5	0.43	

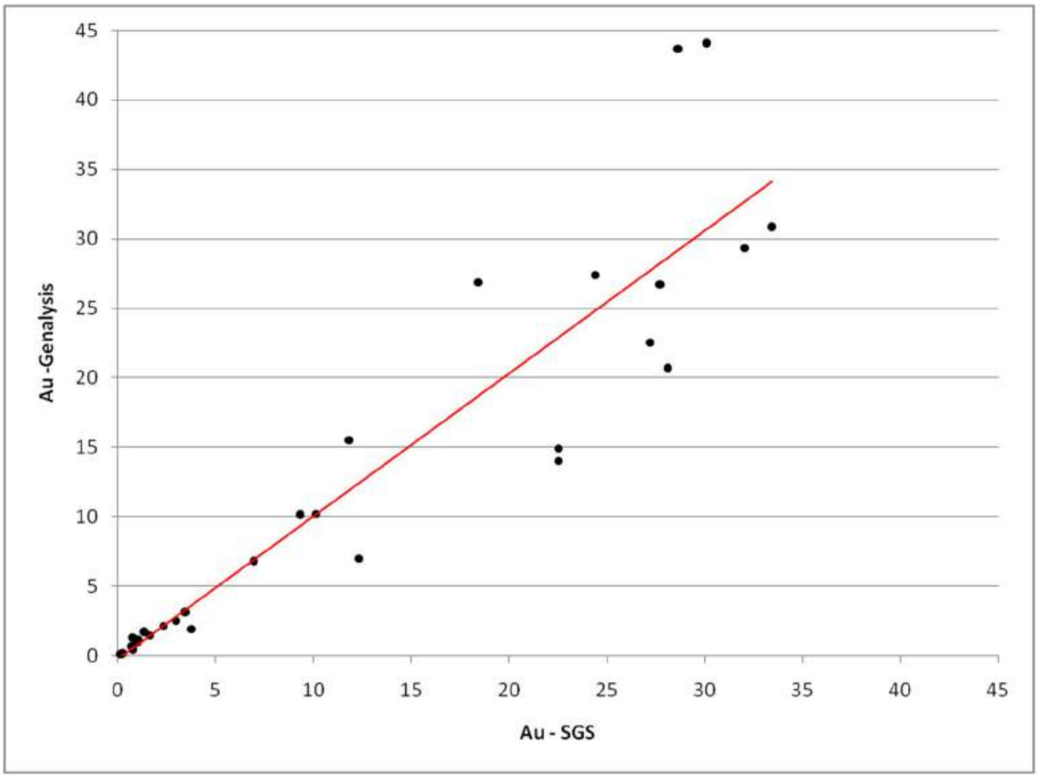


Figure 46 - Scatter Plot of Inter-laboratory Check Assays

12.5 Density QA/QC

Density data was collected with measurements carried out systematically either in the field or laboratory for the duration of the program and hence there is a substantial density database from which to obtain a representative resource bulk density. Rock type was also recorded at the time of the measurement. Data on duplicate density determinations was not provided.

12.6 Database Validation

The analytical database was provided by Thor in Excel spreadsheet format. Data verification was undertaken independently by Auralia.

During the importation of these tables into Surpac, the software's built-in validation tools were used to detect any errors.

To check the integrity of the electronic drilling database, a series of holes were selected for validation against original records. The holes were selected to cover the extent of the deposit. For the drillholes reviewed, all data was considered valid.

12.6.1 Collar Coordinate Validation

All drillhole collars were surveyed after each drilling campaign. A topographic surface was generated from the collar positions. The surface was reviewed visually to identify any possible outliers i.e. collars that deviated significantly in height between two adjacent points. No such deviations were noticed.

Several collar positions from the CGA-era drilling were resurveyed as checks. No significant positional shifts were noted.

12.6.2 Down-Hole Survey Validation

The down-hole survey data were validated by searching for large discrepancies between the dip and azimuth reading against the previous reading. No significant discrepancies were found.

12.6.3 Assay Verification

All the collars, surveys, geology and assays were imported into Surpac. The error checking function on import verified that the maximum depth of samples assay did not exceed hole depth and that no negative values were imported.

12.6.4 Geological Data Verification and Interpretation

Geology data verification comprised determining that the lithology designation was correct in each sample interval. The geological model is reasonable and adequate for use in the 2017 resource estimate. For each hole, all relevant data was compiled into graphic logs to check and review the various down hole datasets. An example of such a log is shown in Figure 47.

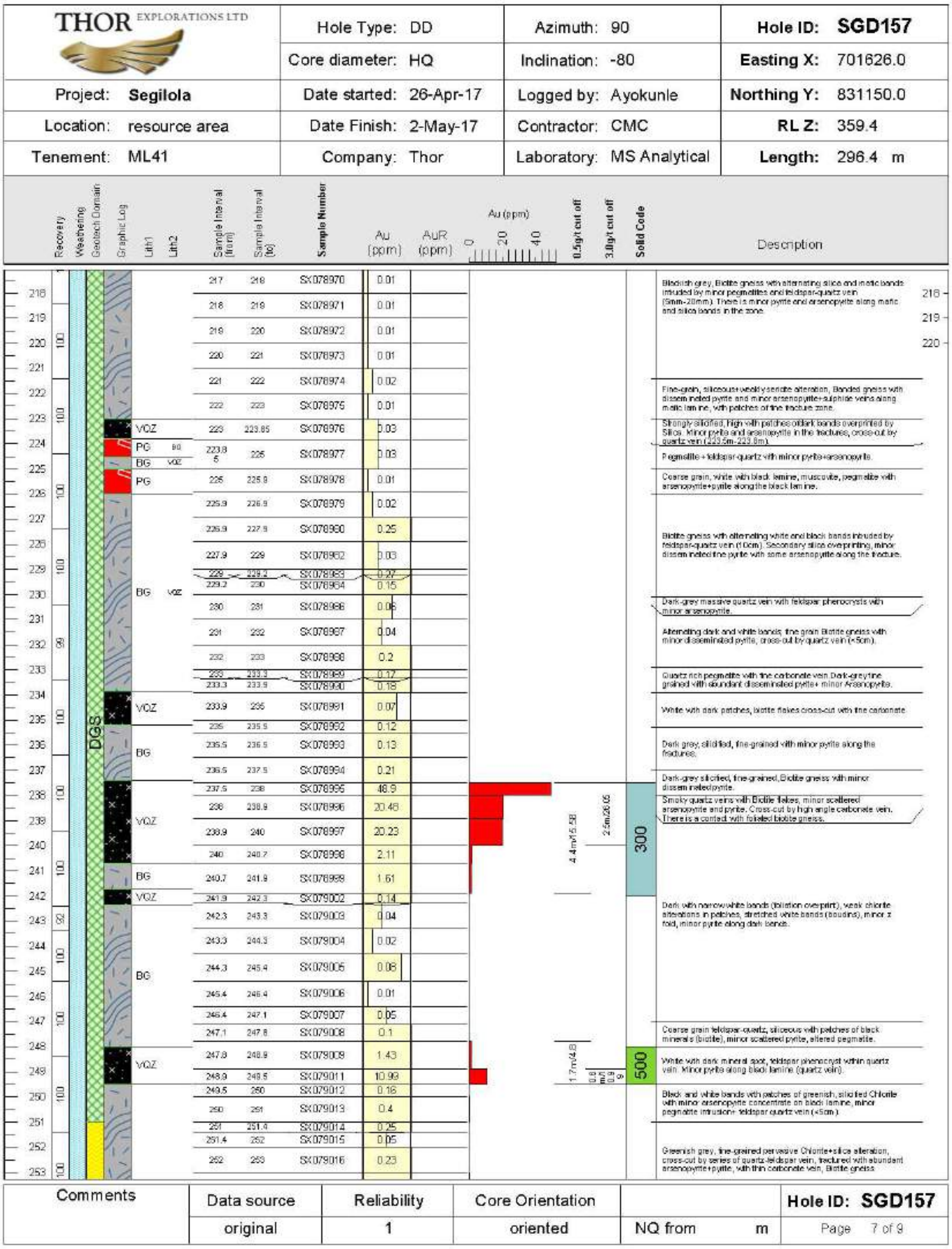


Figure 47 - Segilola Gold Project Example Drill Log

12.7 Conclusion

All sample preparation and analyses were carried out at independent laboratories in Tarkwa, Ghana and Vancouver, Canada. No aspect of laboratory sample preparation or analysis was conducted by an employee, officer, director or associate of Thor.

Thor and its predecessors have used a combination of duplicates, checks, blanks and standards to ensure suitable quality control of sampling methods and assay testing. The procedures and QA/QC management are consistent with good industry practice and are deemed fit for purpose. Results of recent sampling have not identified any issues which materially affect the accuracy, reliability or representativeness of the results.

Analytical results are considered to be reliable as the analyses were produced from a reputable laboratory. No material sample bias was identified during the review of the drill data and assays. Observation of the drill core during the site visits and inspection and validation of the data collected satisfied the QP that the drill data is adequate for the estimation of Inferred and Indicated resources.

12.8 The Relationship of the Laboratory to the Issuer

There is no relationship between the laboratory and the issuer.

Statement on the Adequacy of Sample Preparation, Security and Analytical Procedures

The author is satisfied that all sample preparation, security, and analytical procedures have been conducted to modern industry standards.

On the basis of the various data and from direct observations, the author concludes that CGA has adopted a thorough QA/QC sampling procedure that ensures assay and density data quality. The chain of custody, as currently set up, does not allow for any unwarranted handling and contamination of the samples.

Any potential errors in the bulk density determination procedure that may arise from rock porosity are considered by the author to be minimal due to the high crystallinity of the rock and hence would not materially affect the tonnage calculation. However, independent laboratory density check-determinations are recommended in any future work.

The author considers that sample preparation and analytical procedures comply with the required industry standards and that the data received from the laboratory provides a reliable and accurate basis for the resource estimation.

Results of the analysis of the internal laboratory repeats show a high correlation between the repeats with no bias evident. This indicates good precision for the analysis of gold.

Results of the analysis of the certified reference material and blanks show that the laboratories producing mostly accurate assays with no evidence of significant and systemic contamination.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

The information in this section of the report is summarised/replicated from the Revised BFS completed by Ratel Group Limited, a previous owner of the project, in 2012. There have been no material changes to this information since that time.

Metallurgical testing has been completed by AMMTEC Ltd (AMMTEC) for the project's previous owners. Test work was performed using drill core from the identified intersection zones within the orebody. Testing undertaken involved:

- Head assay.
- Grinding parameter determination.
- Gravity testing. Leach testing.
- Detox testing.
- Thickening tests – Outotec

The following sub-sections summarise the results of the testing.

13.1 Head Assay

The samples used for the testing were drill core intercepts of the orebody. A total of three composites were created, named North Composite, Central Composite and South Composite, representing the three main areas of the orebody. The respective core sections are listed in Table 19 to Table 21.

Table 19: North Composite Details

Section	Metreage		Interval Length (m)	Grade Au (g/t)	Sample ID	Mass (kg)
	From	To				
83200N SGD042	25.8	28.3	2.50	4.00	SM0001	2.40
	33.1	36.0	2.90	5.43	SM0002	2.70
832030N SGD044	20.0	23.0	3.00	6.86	SM0003	1.65
	25.8	26.8	1.00	1.30	SM0004	0.95
	27.5	29.0	1.50	3.40	SM0005	1.25
832030N SGD043	48.2	55.3	7.10	13.39	SM0006	4.05
832055N SGD045	33.5	34.5	1.00	18.40	SM0007	0.70
	34.5	35.05	0.55	16.30	SM0008	0.65
	37.0	40.0	3.00	9.45	SM0009	2.85
	42.0	43.0	1.00	1.48	SM0010	1.00
832055N SGD046	55.9	61.1	5.20	9.95	SM0011	5.20
832000N SGD048	88.0	93.0	5.00	8.40	SM0012	4.90
831960N SGD049	82.9	87.0	4.10	5.13	SM0013	4.10
831935N SGD050	93.0	97.5	4.50	4.34	SM0014	4.30
831905N SGD051	65.4	70.0	4.60	29.71	SM0015	4.20

Table 20: Central Composite Details

Section	Metreage		Interval Length (m)	Grade Au (g/t)	Sample ID	Mass (kg)
	From	To				
831690N SGD016	22.0	25.0	3.00	21.27	SM0016	2.55
831655N SGD037	27.0	29.8	2.80	19.98	SM0017	2.70
831630N SGD010	22.0	25.0	3.00	29.60	SM0018	4.50
831615N SGD059	20.0	22.0	2.00	1.24	SM0019	3.25
	24.2	28.0	3.80	14.42	SM0020	6.80
831440N SGD007	17.0	19.0	2.00	3.44	SM0021	0.45
831410N SGD008	22.0	30.0	8.00	3.42	SM0022	5.15
	34.0	39.0	5.00	4.36	SM0023	3.50
831390N SGD014	27.0	33.0	6.00	6.35	SM0024	5.20
831360N SGD011	35.0	40.0	5.00	5.96	SM0025	5.25

Table 21: South Composite Details

Section	Metreage		Interval Length (m)	Grade Au (g/t)	Sample ID	Mass (kg)
	From	To				
831330N SGD025	62.0	66.0	4.00	6.10	SM0026	3.90
831300N SGD033	73.0	84.0	11.00	4.84	SM0027	10.50
831235N SGD080	106.7	115.0	8.30	8.42	SM0028	5.60
831160N SGD078	64.0	74.7	10.70	6.55	SM0029	9.4
831160N SGD109	87.9	95.6	7.70	3.12	SM0030	6.25
831135N SGD108	119.0	122.6	3.60	15.43	SM0031	3.45
831085N SGD110	139.3	143.5	4.20	4.53	SM0032	3.45
830960N SGD107	118.5	126.0	7.50	15.98	SM0033	6.20
830935N SGD112	131.5	132.9	1.40	6.00	SM0034	1.40
	141.5	142.8	1.25	12.94	SM0035	1.45
	146.4	147.8	1.40	1.00	SM0036	1.25
830910N SGD113	101.4	106.0	4.60	5.09	SM0037	5.20

A summary of the head analysis results is provided in Table 22.

Table 22: Summary of Head Assay Analysis

Element	Unit	North Composite	Central Composite	South Composite
Ag	ppm	< 2	3	< 2
As	ppm	< 10	< 10	19
Au	ppm	7.28	9.29	9.67
Au repeat	ppm	6.25	6.90	8.78
Cu	ppm	122	197	199
Fe	%	0.86	1.31	0.86
Hg	ppm	< 0.1	< 0.1	< 0.1
S	%	0.37	0.24	0.24
SiO_2	%	87.4	84.7	87.0
SG		2.69	2.67	2.67

From the data in Table 22, the following was deduced:

- The ore has high SiO2 and low sulphur levels, indicating that a straight grind and leach process would be applicable.
- Low silver content appears to be present within the ore.

- Arsenic levels are not significant and will not require special processing techniques to be employed.
- Gold content is quite high, variability on the repeat indicates that the gold is potentially coarse (or 'spotty') which could be amenable to gravity recovery techniques.
- Copper content is low indicating that negligible problems with copper dissolution would be expected.
- Mercury levels are low (below detection limit) and as such the processing plant will not require mercury scrubbing equipment.

Use of drill core intercept sections has the potential to overstate the actual ore feed grade that a processing plant will see in practice due to the lack of mining dilution in the sample. In consultation with RML, the design feed grade to the processing plant has been set at 5.00 g/t Au and 0.75 g/t Ag.

Following Ore reserves and mining evaluations a second set of samples were tested (Composites 1 –4) with the inclusion of low grade and waste material to better represent the predicted mill feed grade of 4 – 5 g/t.

Specific details of sample location are included in the AMMTEC Report A12173 which is available upon request. The head assays of the samples tested are shown in Table 23 and Table 24.

Table 23: Head Assay

Sample Identity	Head Assays (ppm)				
	Gold	Gold (rpt)	Silver	Copper	Sulphur
North Composite	7.28	6.25	1	122	0.37
Central Composite	9.29	6.90	3	197	0.24
South Composite	9.67	8.78	1	199	0.24
Blend Composite	4.93	8.36	1.5	N/A	N/A
Composite 1 + 2	2.47	N/A	1.3	94	0.39
Composite 3 + 4	3.32	2.54	4.1	119	0.41

Table 24: Gold Assay

Sample Identity	Gold Grade (ppm)				
	Head Assay (1)	Head Assay (2)	Leach Test (1)	Leach Test (2)	Leach Test (3)
North Composite	7.28	6.25	10.7	9.17	30.8
Central Composite	9.29	6.90	12.9	11.0	11.3
South Composite	9.67	8.78	8.83	6.80	6.25
Blend Composite	4.93	8.36	13.7	9.30	9.42
Composite 1 + 2	2.47	N/A	5.44	3.78	N/A
Composite 3 + 4	3.32	2.54	3.12	3.74	N/A

13.2 Grinding Parameters

The following grinding and crushing parameters have been determined on the North, Central and South Composites:

- Unconfined Compressive Strength (UCS).
- Bond Ball Mill Work Index (BBWi) at 106µm closing screen size.
- Bond Abrasion Index (Ai).

Results of the testing are shown in Table 25.

Table 25: Results of Grinding Parameter Determination Tests

Parameter	North Composite	Central Composite	South Composite	Average
UCS (MPa)	12.0 – 72.2	28.0 – 105.0	26.4 – 86.7	51.3
BBWi (kWh/t)	18.0	18.1	18.8	18.3
Ai (g)	0.3865	0.3588	0.3319	0.3591

UCS testing was undertaken with ¼ NQ core. Results of the testing showed that a fair degree of variation in the UCS results, however, overall the ore would be relatively soft at the coarse sizes expected to be processed by the crushing circuit. In contrast, the results of the BBWi testing showed relatively consistent numbers across the three composites tested, suggesting little variation would be experienced by the grinding circuit. The ore was characterised as exhibiting hard properties for a ball mill grinding circuit. The abrasion index appeared to be relatively consistent across the three composites tested, and would be classified as moderate.

In addition, SMC testing was undertaken to determine the SAG milling parameters for circuit modelling. Testing was undertaken by AMMTEC with the analysis of the results being performed by JKTech Pty Ltd (JKTech). Results of the SMC testing are shown in Table 26.

Table 26: SMC Testing Results

Sample	DWi (kWh/m	DWi (%)	Mia (kWh/t	Mih (kWh/t	Mic (kWh/t	A	b	SG	ta
North Comp	4.59	35	14.8	10.1	5.2	75.8	0.76	2.63	0.56
Central Comp	3.90	26	12.9	8.6	4.4	77.3	0.87	2.63	0.66
South Comp	3.74	24	12.3	8.1	4.2	70.3	1.02	2.67	0.69

By comparing the results of the SMC testing to the JKTech database of 450 ore types, it was determined that the Segilola ore would be considered as being soft with moderate competency for SAG milling purposes. Following discussions between Sedgman and RML / RGL, a single stage SAG milling circuit was selected as being the preferred grinding circuit configuration.

A determination into appropriate mill sizing was undertaken by Orway Mineral Consultants (OMC), targeting a grind of P80 106 µm as determined from the leaching results. The design criteria used in the mill sizing is shown in Table 27.

Table 27: OMC Design Criteria

Criteria	Units	Value
Throughput	Mtpa	0.5
	tph	65.0
Availability	%	87.8
Fresh Ore Feed Size F80	Mm	90.0
Target Product Size P80	µm	106
Ore Characteristics		
BBWi	kWh/t	18.6
Ai	g	0.3782
Appearance Functions		
A		74.5
B		0.87
A x b		65.1
ta		0.64

Power modelling by OMC showed that the specific power requirements for the Segilola ore in a single stage SAG circuit would range between 17.5 – 18.6 kWh/t. This translated to a required mill pinion power of between 1,140 – 1,210 kW. OMC recommended the use of a square aspect SAG mill for the Segilola application citing that a square aspect mill would provide good grate open area, reasonable grinding length and opportunity for later circuit expansion by reconfiguration to SAB. Table 28 lists the OMC recommended mill sizing.

Table 28: OMC Recommended Mill Sizing

Criteria	Units	Mill Parameter
Inside Shell Diameter	m	4.88
EGL	m	5.05
L:D ratio		1.03
Discharge Arrangement		Grate
Speed	% Nc	65 – 78
Liner Type		Composite Rubber
Liner Thickness	mm	100
Operating Ball Charge	%	10
Maximum Ball Charge	%	15
Operating Total Charge	%	25
Maximum Total Charge	%	35
Recommended Installed Power	kW	1,550

OMC noted the following in their analysis:

- The ore is reasonably soft, which will make maintaining a suitable mill load difficult at times, this will require the installation of a variable speed drive; this was subsequently rejected by RML/RGL.
- The moderate ore competency displayed, combined with the fine grind size suggests that a recycle crusher would not be necessary in the grinding circuit flow sheet.

13.3 Composites 1-4 Low Grade Samples

Comminution parameters for the lower grade composites are indicated in Table 29.

Table 29: Comminution Parameters, Composites 1-4

Parameter	Composite 1 - 2	Composite 3 - 4
UCS (MPa)	65 – 121	28 – 91.4
BBWi (kWh/t)	17.5	14.1
Ai (g)	0.35	0.30
SMC (ta)	64.9	62.6

The Bond Work Indices are lower than the earlier results (average 18.3 kWh/t) but will not change the SAG mill sizing or power draw.

13.4 Gravity Testing

A limited amount of gravity testing has been undertaken by AMMTEC under the A12173 programme. Samples of the North, Central and South Composites were ground to a P80 of 75 µm and passed through a laboratory Knelson concentrator. The obtained Knelson concentrate was then panned to obtain less than 0.2 % mass recovery to final concentrate to represent recoveries similar to that which would be expected in a full scale operation.

The panning concentrate, panning tail and Knelson tail were assayed for gold content. The results of the gravity testing are presented in Table 30.

Table 30: Gravity Testing Results

Sample	Product	Mass (g)	Mass Distribution (%)	Au (ppm)	Au Distribution (%)
North Composite	Pan Concentrate	1.88	0.19	3,166	35.4
	Pan Tail	52.00	5.20	21.2	6.6
	Knelson Tail	947.10	94.6	10.3	58.0
	Total	1,000.9	100.0	16.8	100.0
Central Composite	Pan Concentrate	0.80	0.08	9,586	69.5
	Pan Tail	47.8	4.77	34.1	14.8
	Knelson Tail	955.0	95.2	1.82	15.7
	Total	1,003.6	100.0	11.0	100.0
South Composite	Pan Concentrate	1.13	0.11	759	33.1
	Pan Tail	44.7	4.49	1.78	3.07
	Knelson Tail	950.6	95.4	1.74	63.8
	Total	996.5	100.0	2.60	100.0

Results of the testing indicated significant levels of gravity recoverable gold were present within the Segilola ore composites with at least 30 % of the gold contained within the composites being recovered to concentrate. This confirmed the 'spotty' observations made during the head analysis.

Subsequent to this testwork SGL/RGL decided not to include a gravity circuit at Segilola because of increased security risks at this location.

13.5 Leach Testing

A total of three leach tests were performed on whole of ore samples for each of the three composites (North, Central and South) for grind size determination. Testing was performed at grind sizes of 150 µm, 106 µm and 75µm. Standardised conditions were:

- Slurry at 40 % w/w solids.
- 48 hour leach.
- pH 10.5.
- 500 mg/L NaCN initially and allowed to drift.
- Oxygen addition to saturation.

Reagent additions were monitored and solution grades were analysed for Au and Ag at 1, 2, 4, 8, 24 and 48 hours after the commencement of leaching. The results of the leach testing are presented in Table 31.

Table 31: Segilola Leach Testing Results

Sample	Grind (µm)	Calculated Head		NaCN Use (kg/t)	Lime Use (kg/t)	Au Extraction (%)			Ag Extraction (%)			Tail Au (ppm)
		Au (ppm)	Ag (ppm)			8 hr	24 hr	48 hr	8 hr	24 hr	48 hr	
North Composite	150	10.7	0.79	0.15	0.43	68.6	95.6	98.0	74.1	83.5	87.3	0.210
	106	9.2	0.90	0.18	0.34	80.3	97.7	98.1	75.4	83.8	88.8	0.170
	75	30.8	1.15	0.26	0.37	78.8	98.2	99.7	79.6	90.0	91.3	0.090
Central Composite	150	12.9	1.00	0.33	0.34	50.9	82.6	94.9	67.5	84.0	90.0	0.660
	106	11.0	0.90	0.45	0.32	57.4	86.4	98.4	70.4	83.8	88.8	0.180
	75	11.3	1.06	0.30	0.37	61.9	90.7	98.8	73.6	86.3	90.6	0.140
South Composite	150	8.8	1.74	0.18	0.27	56.1	84.5	97.7	77.8	89.9	94.2	0.200
	106	6.8	1.78	0.11	0.31	66.9	91.4	98.2	79.2	89.3	94.4	0.120
	75	6.4	2.01	0.41	0.31	64.7	94.4	99.1	76.3	90.5	95.0	0.060

** Note: Use values reported are consumption values*

Analysis of the leaching test results concluded the following:

- Cyanide consumption in all of the composites was low, averaging 0.26 kg/t. Overall addition for each trial was 0.75 kg/t.
- Lime consumption was equally low, averaging 0.34 kg/t on a 60% CaO lime.
- Gold and silver extraction was high after 48 hours of leaching with in excess of 97 % of the Au and 87 % of the Ag being leached.
- Reasonable solid tail grades of less than 0.2 g/t Au could be reliably achieved when the grind size was 106 µm and finer.

Analysis of the leach profile curves generated from the results (Figure 48 to Figure 50) showed that while the majority of the gold was rapidly leaching, there was a component present which was still leaching after 48 hours of cyanidation. This is generally an indication of coarse, leachable gold being present within the ore which would usually be amenable to gravity recovery techniques or the presence of electrum within the ore.

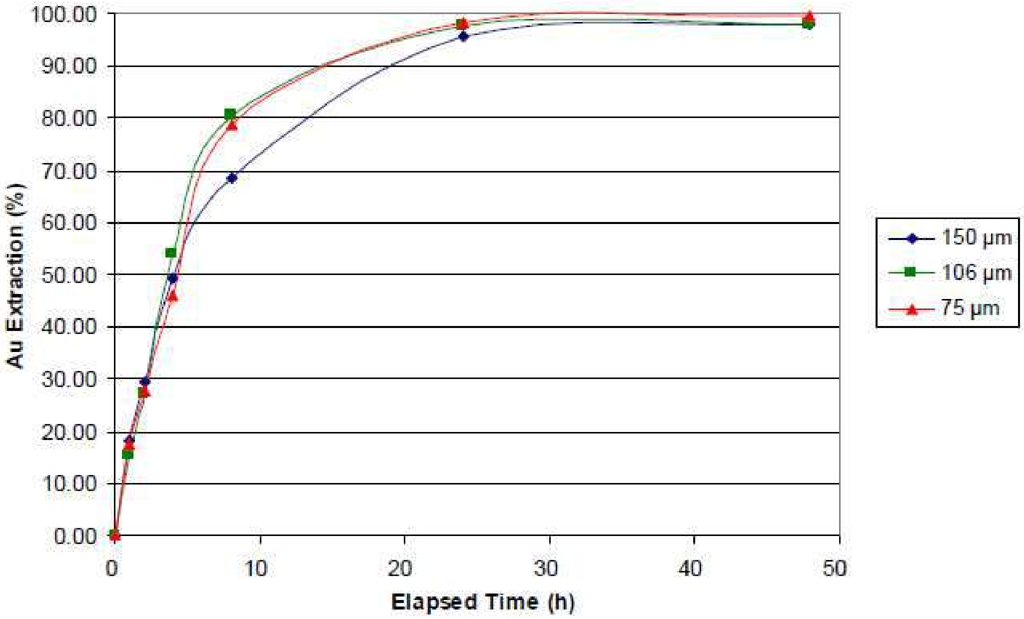


Figure 48: North Composite Gold Leach Curves

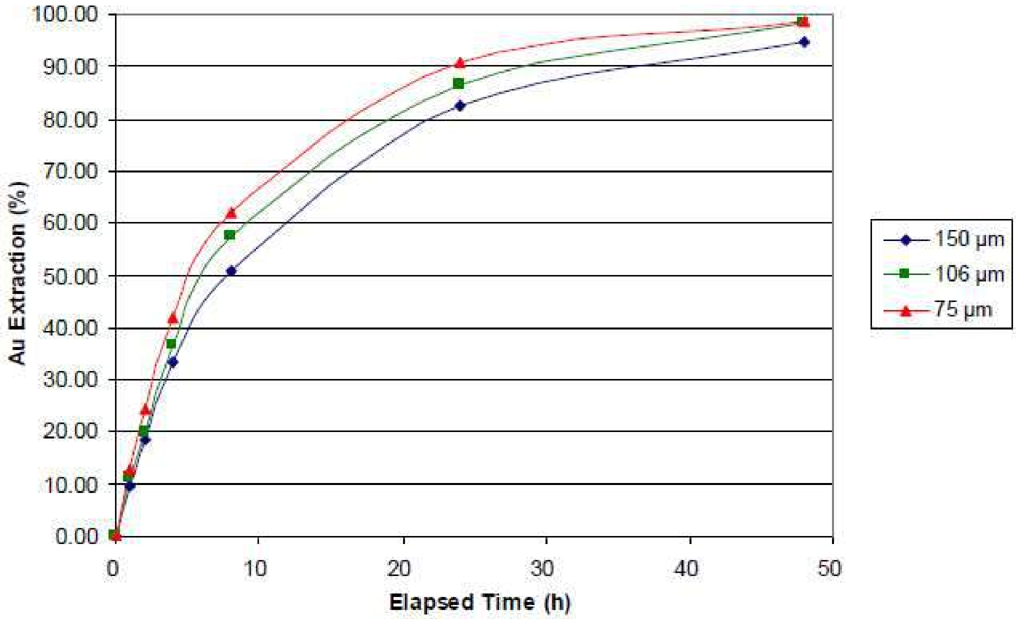


Figure 49: Central Composite Gold Leach Curves

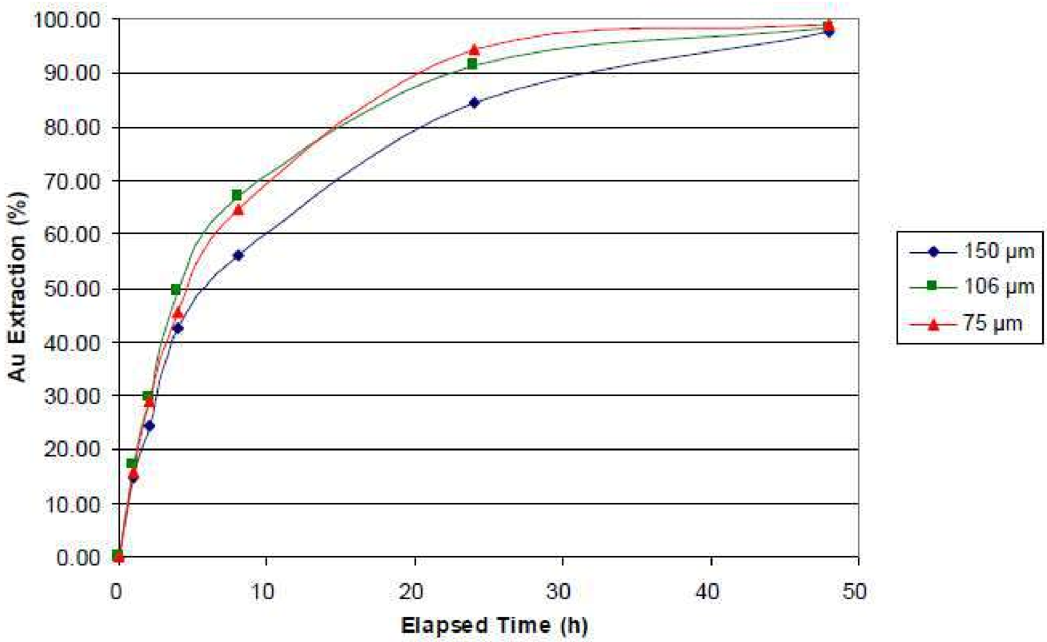


Figure 50: South Composite Gold Leach Curves

A further single test was performed on the South Composite ore to study the influence of oxygen on leaching by repeating the 106 µm test using air. The results of this test are presented in Table 32.

Table 32: Segilola Air Leach Test Results

Sample	Grind (µm)	Calculated Head		NaCN Use (kg/t)	Lime Use (kg/t)	Au Extraction (%)			Ag Extraction (%)			Tail Au (ppm)
		Au (ppm)	Ag (ppm)			8 hr	24 hr	48 hr	8 hr	24 hr	48 hr	
South Composite	106	6.3	1.71	0.15	0.38	39.4	73.4	97.9	60.7	87.1	94.1	0.13

Results of the air leach test showed that, while the leach rate was slower, the ultimate leach extraction achieved after 48 hours was effectively the same when compared to the results achieved for the corresponding oxygen sparged slurry test. Reagent addition requirements, with exception of oxygen, were seen to be unchanged.

A summary of the leach test results carried out on all composites is shown in Table 33, with comparative curves being presented in Figure 51.

Results of the air leach test showed that, while the leach rate was slower, the ultimate leach extraction achieved after 48 hours was effectively the same when compared to the results achieved for the corresponding oxygen sparged slurry test. Reagent addition requirements, with exception of oxygen, were seen to be unchanged.

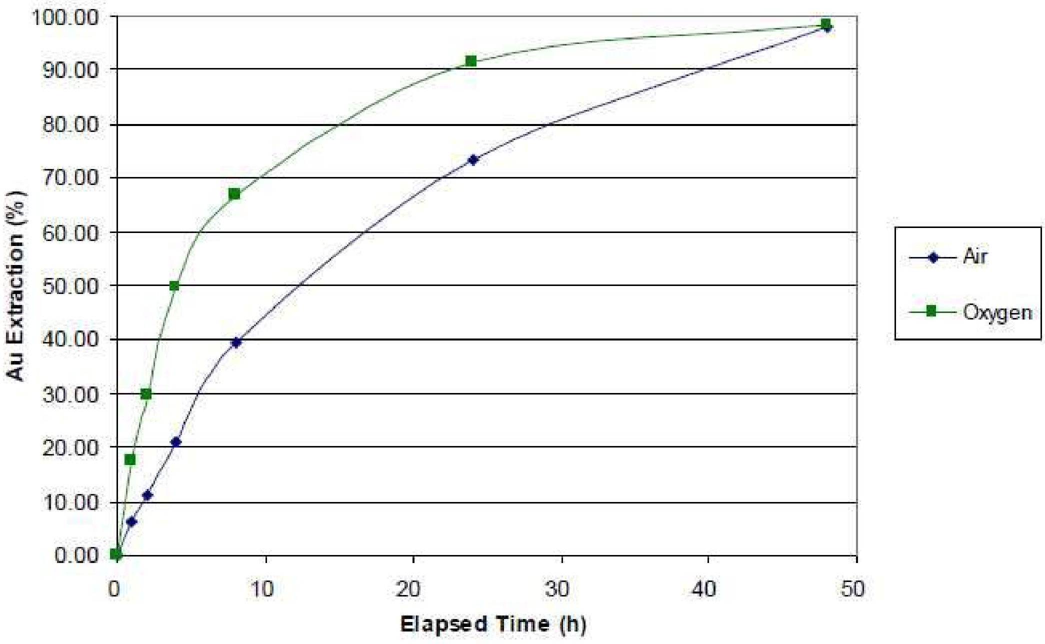


Figure 51: Gold Leach Profile Comparison – Oxygen vs Air

Table 33: Leach Testing

Test No	Sample Identity	Grind Size (µm)	Density % Solids	Lime Add'n (kg/t)	O2/Air	NaCN (kg/t)		Gold Extraction (%)			Residue Au (ppm)
						Add'n	48 hrs	8 hrs	24 hrs	48 hrs	
GSG428	North	150	40	0.43	O2	0.75	0.15	68.6	95.6	98.0	0.210
GSG429		106	40	0.34		0.75	0.18	80.3	97.7	98.2	0.170
GSG425		75	40	0.37		0.75	0.26	78.8	98.3	99.7	0.090
GSG430	Central	150	40	0.34	O2	0.75	0.33	50.9	82.6	94.9	0.660
GSG431		106	40	0.32		0.75	0.45	57.4	86.4	98.4	0.180
GSG426		75	40	0.37		0.75	0.30	61.9	90.7	98.8	0.140
GSG432	South	106	40	0.27	O2	0.75	0.18	56.1	84.5	97.7	0.200
GSG433		106	40	0.31		0.75	0.11	66.9	91.4	98.2	0.120
GSG427		106	40	0.31		0.75	0.41	64.7	94.4	99.1	0.060
GSG436		106	40	0.38	Air	0.45	0.15	39.4	73.4	97.9	0.130
GSG474	Blend	106	40	0.32	Air	0.45	0.23	41.8	77.2	95.3	0.640
GSG475		106	50	0.27		0.30	0.17	44.0	84.1	92.1	0.710
GSG476		106	55	0.29		0.25	0.14	42.0	78.3	93.2	0.630
GSG477		106	40	0.33		0.48	0.26	35.1	68.3	94.4	0.580
GSG478		106	55	0.23	O2	0.45	0.29	51.9	89.5	99.0	0.090
GSG555	Comp 1 + 2	106	40	0.27	O2	0.45	0.18	63.5	92.7	99.1	0.050
GSG557		106	40	0.26		0.75	0.23	81.9	94.2	98.5	0.055
GSG556	Comp 3 + 4	106	40	0.26	O2	0.45	0.18	81.2	89.9	97.1	0.090
GSG558		106	40	0.27		0.75	0.26	77.9	95.2	97.6	0.090

A summary of the results is as follows:

- Gold extraction – all gold extractions are high 92 – 99.7%
- Grind – a finer grind to P80 of 75 micron marginally reduces residue value (0.04 – 0.08 g/t) over P80 of 106 micron. Based on these results all subsequent test work was conducted at a grind size P80 of 106 microns.
- Oxygen – the use of oxygen instead of air during the leach increases leach rate and marginally increases gold extraction after 48 hours.
- Leach time – leach times of up to 48 hours are justified based on additional gold extraction.
- Cyanide concentrate – increased cyanide levels (up to 500 ppm CN) may be justified to reduce leach times.
- Reagent consumptions – lime and cyanide consumption are low at approximately 0.3 kg/t.
- Density – leach density up to 55% solids did not impact on leach rate or gold extraction.
- Head grade – significant variation in the calculated head grade of a given sample confirms the "spotty" nature of the gold occurrence.

From this leach work the grind and leach parameters were determined for plant design as indicated in Table 34.

Table 34: Design Grind and Leach Parameters

Grind	P80 of 106
Leach time	48 hours
Cyanide levels	300 ppm
Ph	10.0
Oxygen addition	required
Density	40% w/w solids
Gold extraction	+95%

13.6 Cyanide Detox Testwork

Cyanide destruction tests were carried out on leach tailings at SO_2/CN levels of 0.84 – 1.7 of theoretical requirements. The results are summarised in Table 35.

Table 35: SO_2/Air Tests

Test No	Test Conditions					Feed Effluent CNp (mg/l)	Treated Effluent CNp (mg/L)
	pH	Retention Time (min)	Reagents used				
			SO2 (g/g CN[wa	Cu^{2+} (mg/l)	Lime (g/g SO2)		
D1	8.54	56	4.20	22	0.93	135.5	0.60
D2	8.57	58	3.08	21	0.08	135.5	N/A
D3	8.62	58	2.10	21	0.47	135.5	15.02
D4	8.60	57	2.97	20	0.69	135.5	0.91

The results showed that SO_2/CN additions above 1.2 times theoretical produce an effluent of <1.0 ppm cyanide in solution.

13.7 Thickening Tests

Leach data had indicated that a 48 hour leach time would be required for optimal gold extraction. To reduce the tank volume required consideration was given to the use of a pre-leach thickener to increase the pulp density.

Tests were carried out by Outotec on leach residue samples to determine the thickener area required. Following these tests Outotec would not recommend the installation of a thickener because of the poor flocculant formation and the difficult rheology of the thickened underflow. Details of the testwork and conclusions are included in the AMMTEC report which is available upon request. The installation of a leach feed thickener was subsequently rejected by RML.

14. MINERAL RESOURCE ESTIMATE

Auralia Mining Consulting Pty Ltd (Auralia), were engaged by Thor Explorations Ltd (Thor) to undertake an NI 43-101 compliant Mineral Resource update on the Segilola Gold Project resource model (from July 2017), incorporating the results of the most recent drilling.

The following section details the Mineral Resource estimation work carried out by Auralia. The work includes:

- Data validation and QA/QC review
- Validation of the provided Geological and mineralisation interpretation
- Validation of the provided sectional interpretations
- Validation of the provided 3D wireframe modelling of the interpretations
- Spatial statistics, data preparation, wireframe assigning, data compositing
- Top cut assessment for the composites by domain
- Variography analysis
- Quantitative kriging neighbourhood analysis (QKNA)
- Building and interpolating the mineralisation and waste block model
- Block model validation
- Mineral Resource classification and reporting.

14.1 Summary

Auralia Mining Consulting Pty Ltd (Auralia), were engaged by Thor Explorations Ltd (Thor) to undertake an NI 43-101 compliant Mineral Resource update on the Segilola Gold Project resource model (from July 2017), incorporating the results of the most recent drilling.

Mineralisation wireframes are modelled based on geological interpretation provided to Auralia by Thor. The mineralisation within them has been delineated using lithology and a gold grade of 0.5 g/t Au. A 1 m composite data set for individual lodes was used for variography analysis and grade estimation. For continuity purposes, adjacent drillholes and sections were used to refine the geological model and to reduce the “saw-tooth” effect when modelling.

A block model was created using 20 m E by 20 m N by 20 m RL parent blocks. Ordinary Kriging (OK) was used to estimate block grades. Quantitative Kriging Neighbourhood Analysis (QKNA) was used to optimise parameters for the kriging search strategies.

The Segilola Mineral Resource has been classified and reported in accordance with the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI4-3101). The Mineral Resource classification is based on confidence in the geological interpretation, drill spacing, and geostatistical measures. A summary of the Mineral Resource estimate at 0.5g/t Au cut-off grade is shown in Table 36.

Table 36: Segilola Resource Model - Reported at 0.64g/t Au Cut-off Grade

Type	Indicated			Inferred			Total		
	Tonnes (Mt)	Au g/t	Ounces (koz)	Tonnes (Mt)	Au g/t	Ounces (koz)	Tonnes (Mt)	Au g/t	Ounces (koz)
Trans	0.16	3.21	16	0.02	1.66	1	0.18	3.04	17
Fresh	4.14	4.16	554	3.32	3.55	379	7.46	3.89	933
Total	4.30	4.13	570	3.34	3.54	380	7.63	3.87	950

Fixed density values were assigned into the block model for each weathering unit. The density values were based on physical measurements, and were 2.65 t/m3 for transitional material and 2.67 t/m3 for fresh material.

Statistical and visual assessment of the block model was undertaken to assess the successful application of the various estimation passes, to ensure that as far as the data allowed, all blocks within domains were correctly estimated. Each domain was checked against the composited data used in the estimation process.

On-screen validation involved comparing block estimates and composite grades in cross section. Swath plots were also created and showed good agreement between the sample data and block model mean grades for easting, northing and RL slices. The block model validation process shows that the block model estimates follow the trend of the 1 m composite grades across the deposit. Refer to Appendix 1 for swath plots.

14.1.1 Recommendations

Auralia recommends that further work programs should address the following:

1. The Mineral Resource Estimate shows a substantial volume of material classified as Inferred or Unclassified. This material is an immediate target for resource classification upgrade.

2. It is recommended that optimised pit shells are used as a guide to creating drilling progams that maximise the conversion from lower to higher resource classification and reduce mining risk attributed to data density and quality.

3. Maintain the current quality assurance procedures to ensure high quality data is available for subsequent resource estimates.

4. Collection of additional structural measurements of mineralised vein orientations, obtained from future diamond core drilling programs, is essential for fine tuning the mineralisation domain boundaries for any future resource model updates. Continued improvement in geological understanding and lithology unit interpretation.

5. Undertake programs to establish the suitability and reliability of drilling techniques, i.e. by collecting twinned samples from the same hole (field duplicates) and from paired holes of either the same method or of different methods.

14.1.2 Conclusions

The validity of the database used for the Mineral Resource estimation has been confirmed with checks for internal consistency and accuracy. As a result of these checks, Auralia considers that the drillhole data has been adequately validated with satisfactory quality control analysis. The Competent Person deems that the data is suitable for use in the estimation of Indicated and Inferred Mineral Resources, which are the subject of this technical report.

14.2 Database Validation

During preparation for grade estimation, the following validation steps were undertaken:

- The data was checked for missing collar co-ordinates, missing hole depths, missing down hole surveys; miss matched collar, survey or assay depths; and over lapping intervals;
- The data was checked for missing or overlapping intervals for geology and assay interval data.
- A check for Negative and null assays

14.3 Geology and Mineralisation Interpretation

The resource wireframes represent multiple 010° trending mineralized structures that extends over a continuous strike length of 2km and dips at between 80° to 70° towards the west.

A hangingwall lode that extends over a strike length of 360m is developed in the southern portion of the resource.

14.4 Preparation of Wireframes

The resource wireframe is defined by a nominal lower grade cut-off of 0.5g/t Au. However, in several places a slightly lower cut-off was allowed to maintain geological continuity. Generally, there a sharp transition between background or below detection levels to >0.5g/t Au. Due to the varying hole inclinations, the allowed amount of internal dilution was set nominally at 2.5m true width or less.

The following techniques were employed while interpreting the mineralisation:

- Each cross section or plan was displayed on screen with a clipping window equal to a half distance from the adjacent sections of levels,
- All interpreted polylines (strings) were snapped to the corresponding drillhole intervals
- Internal waste within the mineralised envelopes was included in the interpreted envelopes.
- The interpretation was extended perpendicular to the corresponding first and last interpreted cross section to the distance equal to a half distance between the adjacent exploration lines;
- If a mineralised envelope did not extend to the adjacent drillhole section, it was projected half way to the next section and terminated. The general direction and dip of the envelopes was maintained, and
- If mineralised lode was at the topographic surface, it was extended above the surface and then later clipped.
- A surface digital terrain model (dtm) was generated from the surveyed drill collars.

14.5 Statistical Analysis

14.5.1 Selection of Composite Length

The wireframe of the mineralised zone was used to code the database to allow identification of the resource intersections. Separate intersection files were generated for each resource object. Surpac software was then used to extract downhole composites within the intervals coded as resource intersections. Figure 52 shows a summary of all the sample lengths in the project area.

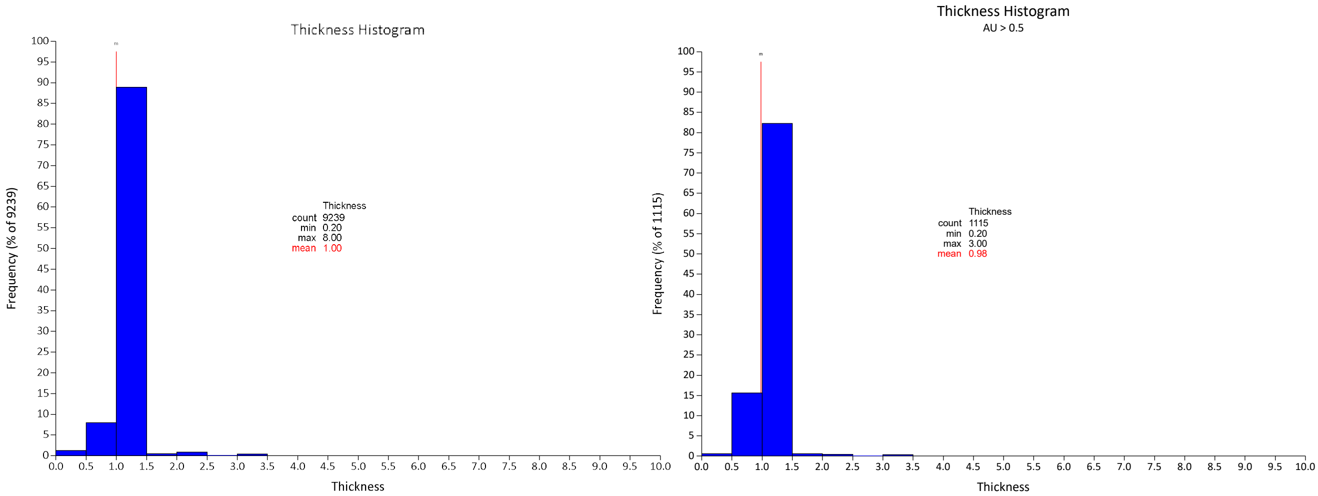


Figure 52 - Histogram of sample thicknesses (left) and Histogram of sample length where Au>0.5g/t (right)

All holes were composited to 1.0m as the large majority of the raw data was sampled at 1.0m intervals. Table 37 and Figure 53 to Figure 59 provide histograms showing the minimum and maximum thicknesses of the drillhole intersections of the mineralised lodes. The composites were checked for spatial correlation with the objects, the location of the rejected composites and zero composite values. Individual composite files were created for each object in the wireframe models.

Table 37 - Summary of Lode Intersections

Lode	# of Samples	Min	Max	Mean
100	42	0.11	10.00	3.99
101	1	4.70	4.70	4.70
200	96	0.90	35.35	10.15
300	36	0.46	13.00	2.65
400	23	0.90	13.00	2.35
500	1	1.70	1.70	1.70
600	1	1.70	1.70	1.70

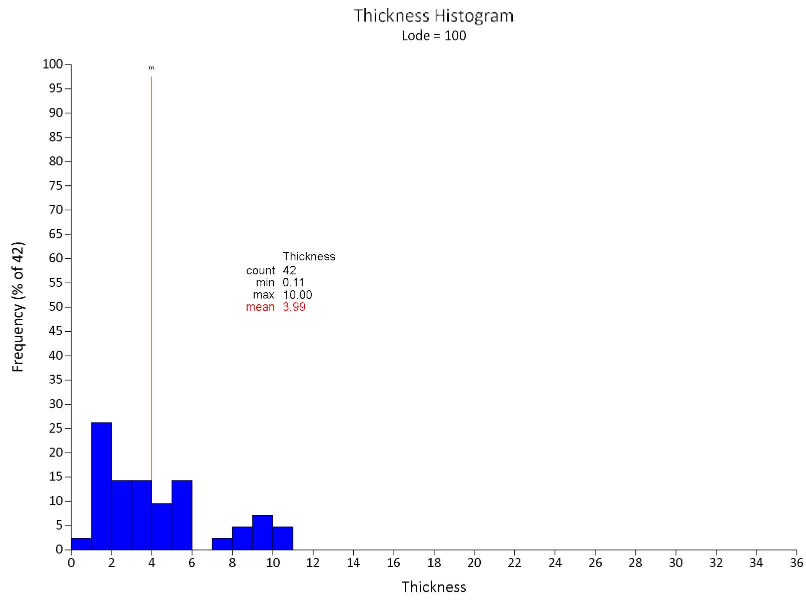


Figure 53 - Minimum and maximum drillhole intersections for Lode 100

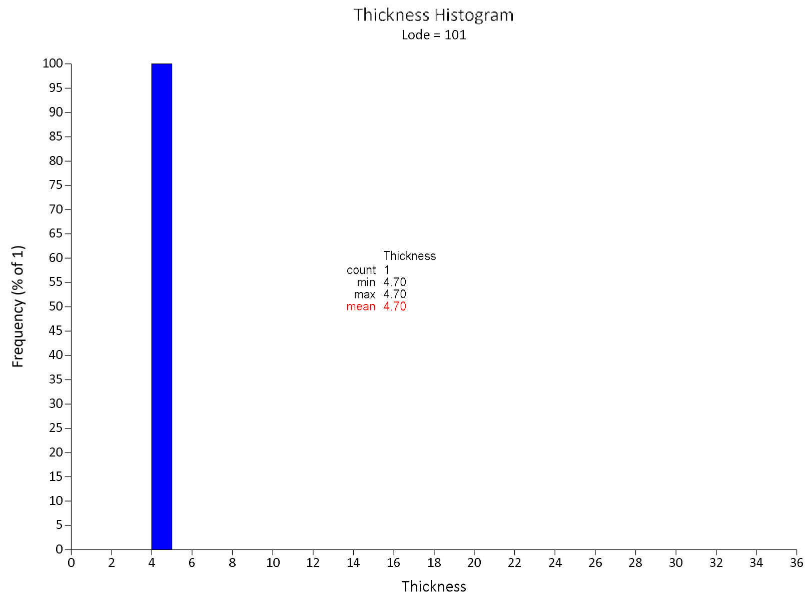


Figure 54 - Minimum and maximum drillhole Intersections for Lode 101

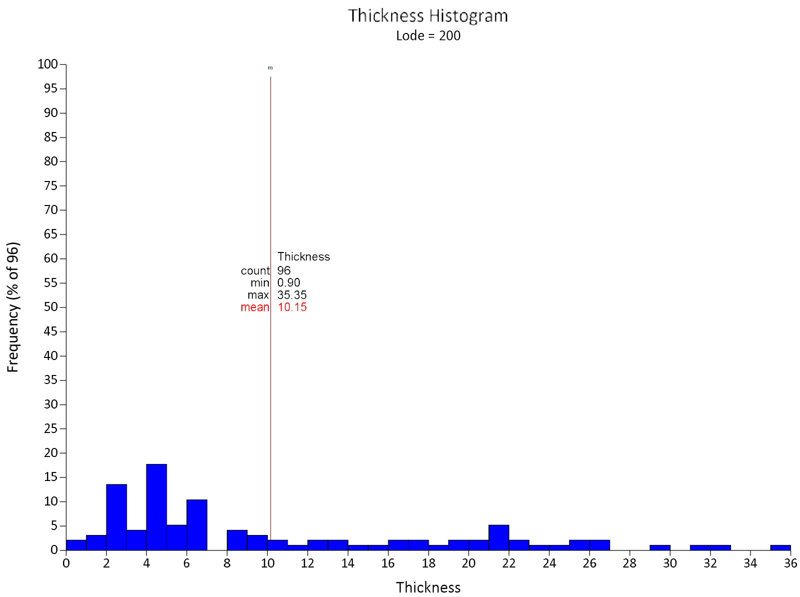


Figure 55 - Minimum and maximum drillhole intersections for Lode 200

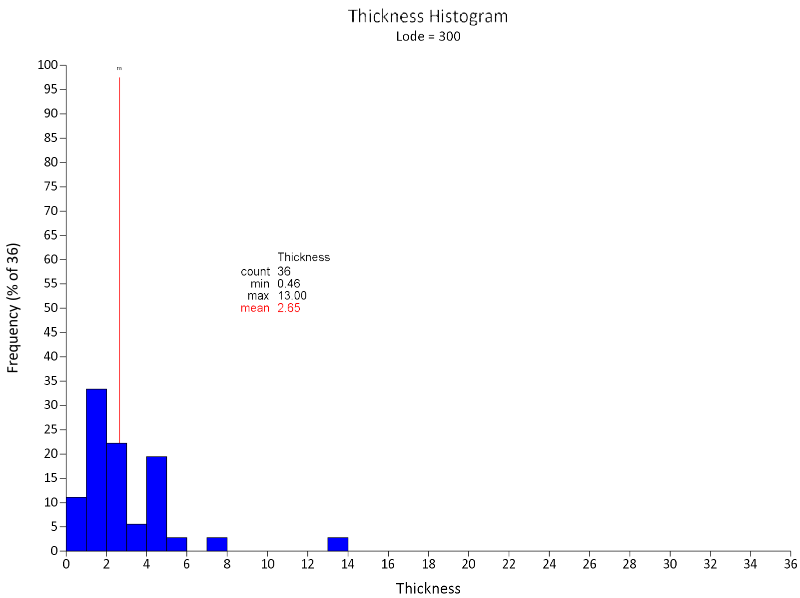


Figure 56 - Minimum and maximum drillhole intersections for Lode 300

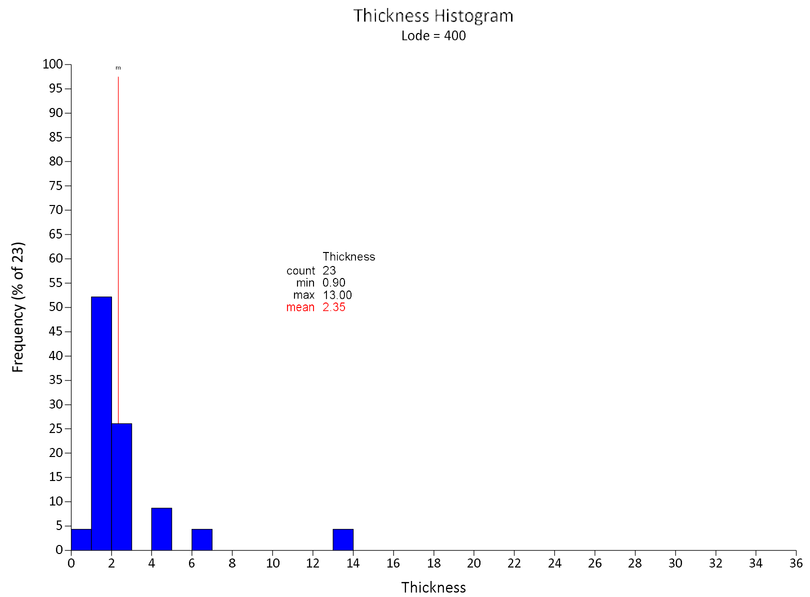


Figure 57 - Minimum and maximum drillhole intersections for Lode 400

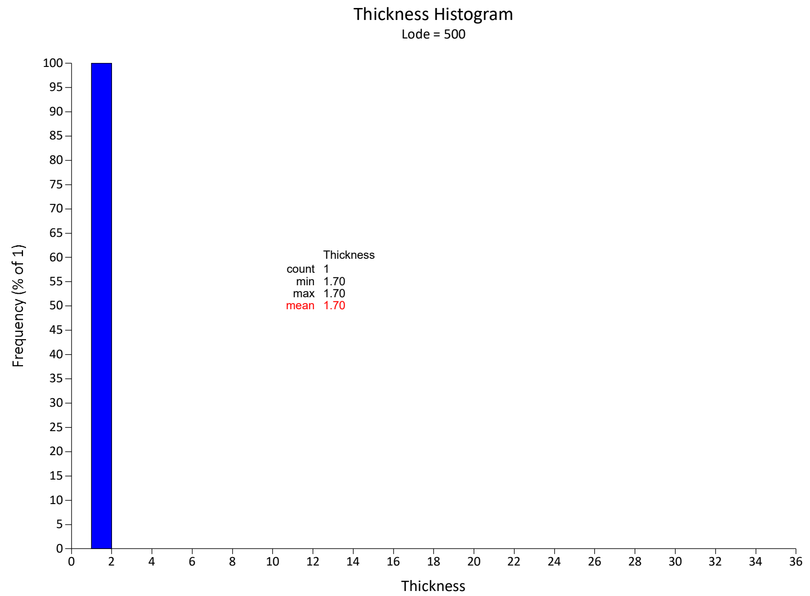


Figure 58 - Minimum and maximum drillhole intersections for Lode 500

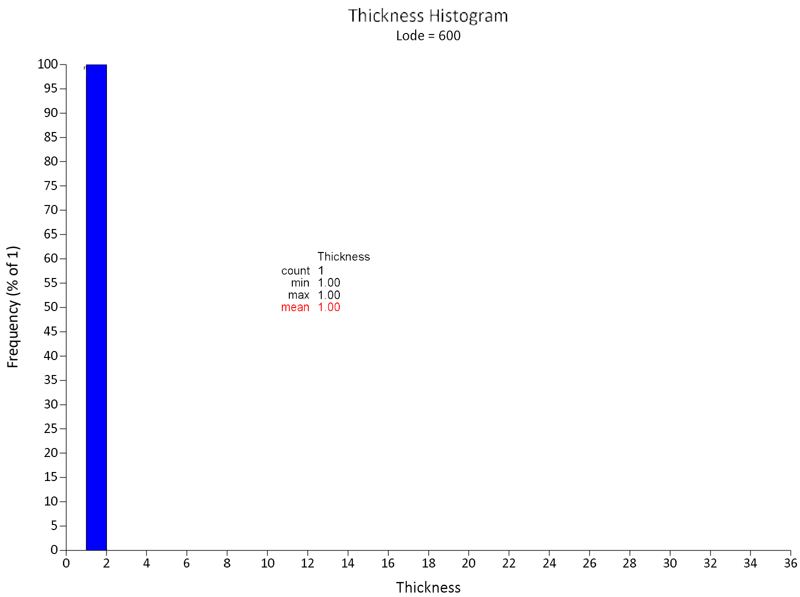


Figure 59 - Minimum and maximum drillhole intersections for Lode 600

14.5.2 Statistical Analysis of 1m Composites

The wireframes of the mineralised zones were used to code the database to allow identification of the resource intersections. Separate intersection files were generated for each resource solid. Surpac software was then used to extract downhole composites within the intervals coded as resource intersections. All holes were composited to 1.0m

The composites were checked for spatial correlation with the solids, the location of the rejected composites and zero composite values. Individual composite files were created for each object in the wireframe models.

Statistical analysis was carried out on each domain. Uncut versus cut statistics by domain are tabulated below in Table 38 and in Figure 60 to Figure 65.

Table 38 - Uncut vs. cut statistics by domain

Lode	# of Samples	Min	Max	Top Cut	Mean	Mean – Cut	Median	Std Dev	Std Dev – Cut	Coefficient of Variation	CV Cut
100	183	0.03	86.37	60.00	7.93	7.77	2.30	12.80	11.96	1.61	1.54
101	5	0.54	6.80		3.36		2.82	2.41		0.72	
200	977	0.02	56.20		3.55		1.23	6.22		1.75	
300	100	0.03	48.53		3.63		0.99	7.88		2.17	
400	55	0.02	135.00	60.00	14.21	7.92	1.08	37.1	17.35	2.61	2.19
500	2	1.43	9.62		5.53		5.53	5.79		1.05	
600	1	12.60	12.60		12.60						

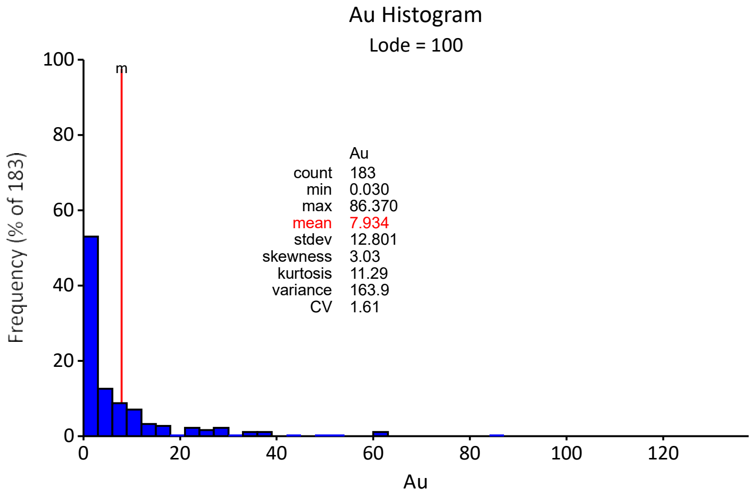


Figure 60 - Lode 100 Au Histogram

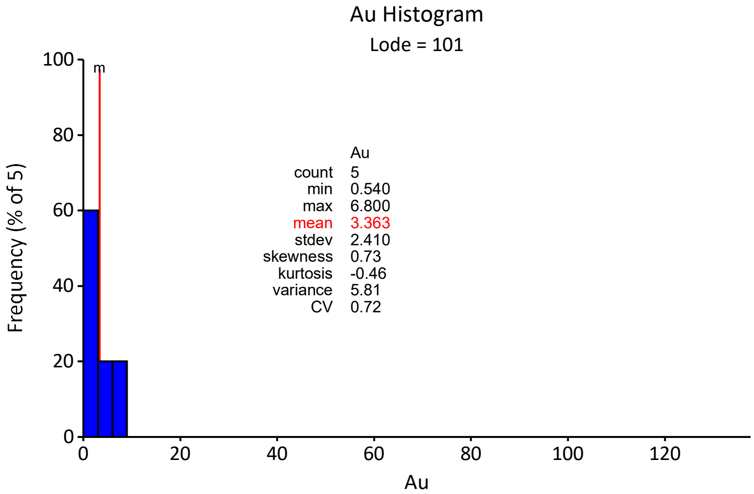


Figure 61 - Lode 101 Au Histogram

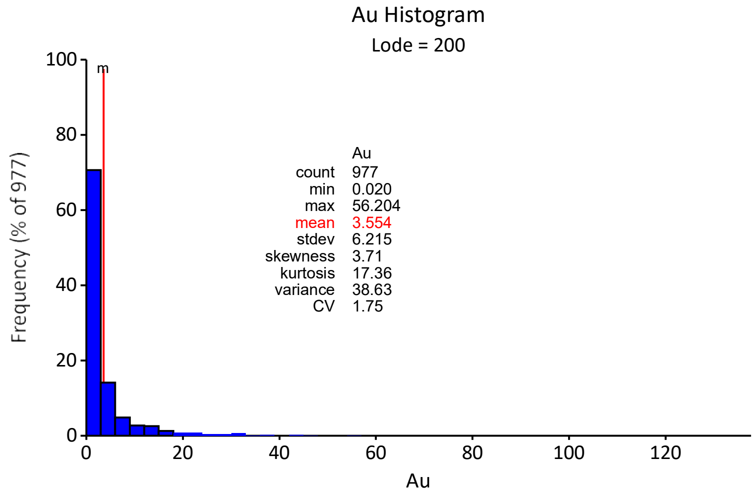


Figure 62 - Lode 200 Au Histogram

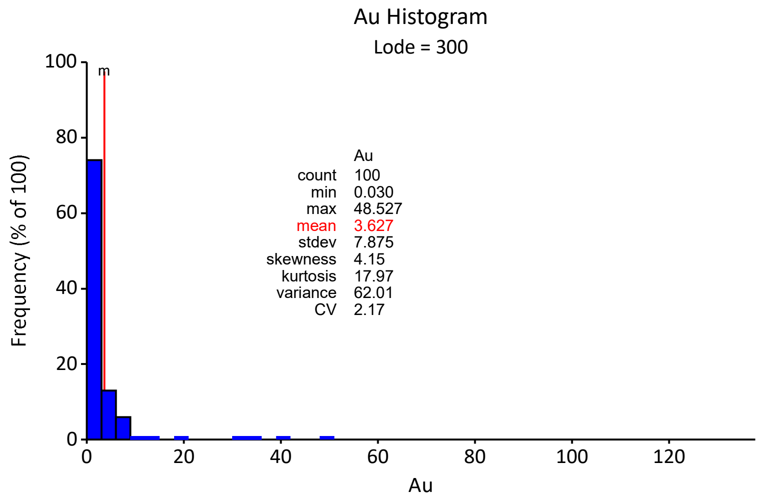


Figure 63 - Lode 300 Histogram

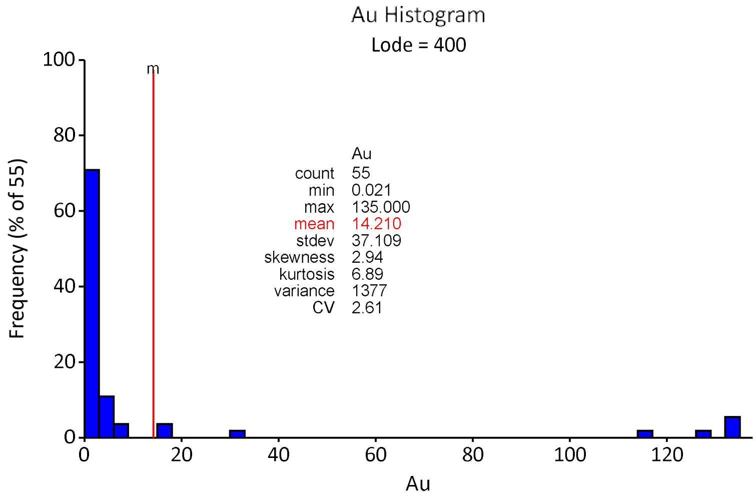


Figure 64 - Lode 400 Au Histogram

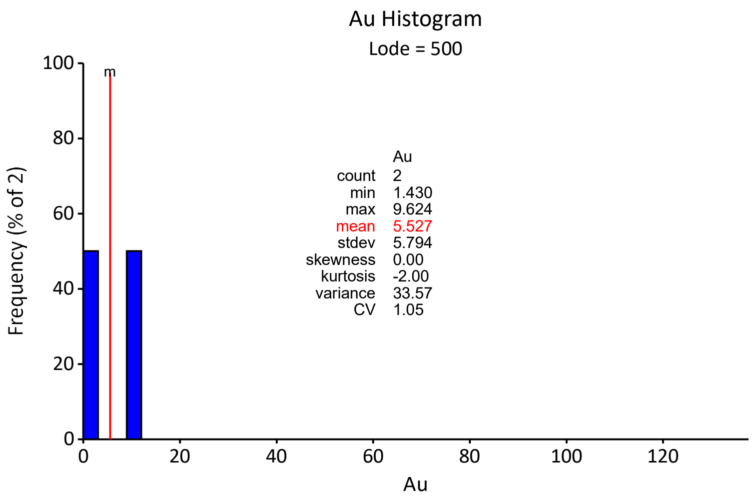


Figure 65 - Lode 500 Au Histogram

A normal probability plot (composited, no top cut) is shown in Figure 66.

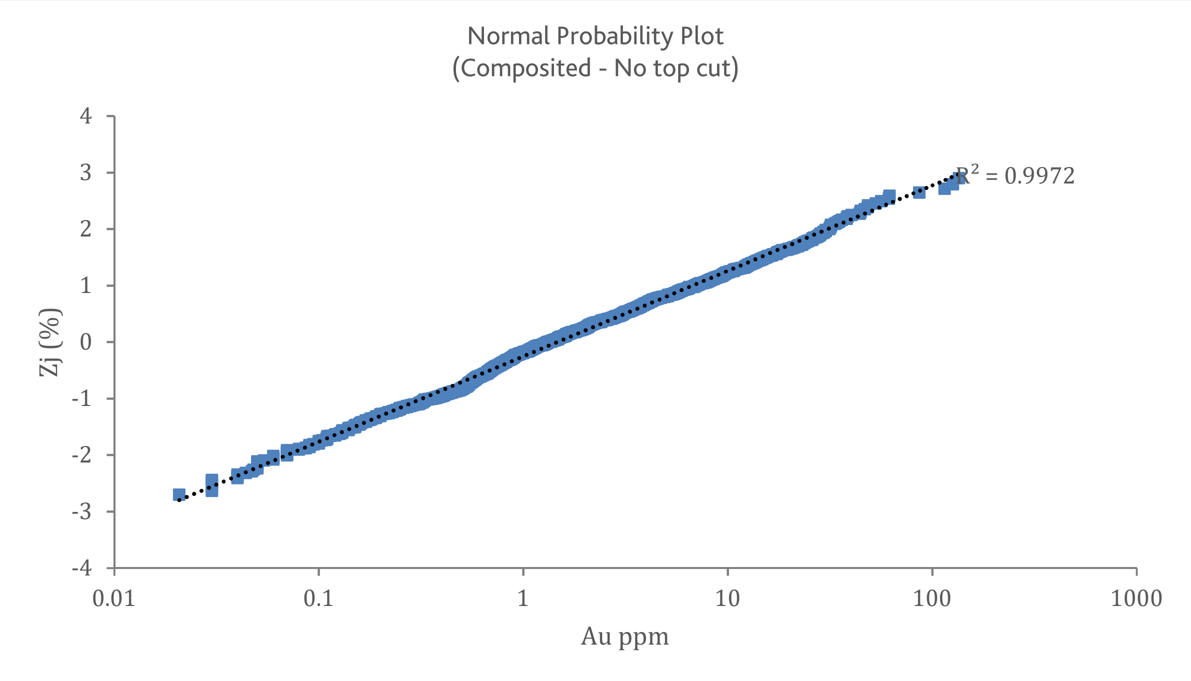


Figure 66 – Normal Probability Plot of Composite Data

14.5.3 High-Grade Assays

To assist in the selection of appropriate high-grade cuts, the composite data was loaded, and log-probability plots were generated for the various lodes. Several methods can be used to help select an appropriate high-grade cut value. The method chosen involves investigation of log probability plots looking for breaks and inflexions in the distribution (Figure 67); this approach was used for each lode and by grouping all the lodes together for all the composites. A top cut of 60 g/t has been selected for all lodes. This top cut value was also the same as suggested by SRK in 2012 as part of their Mineral Experts Report (SRK, 2012). Six samples were affected by the application of the top cut.

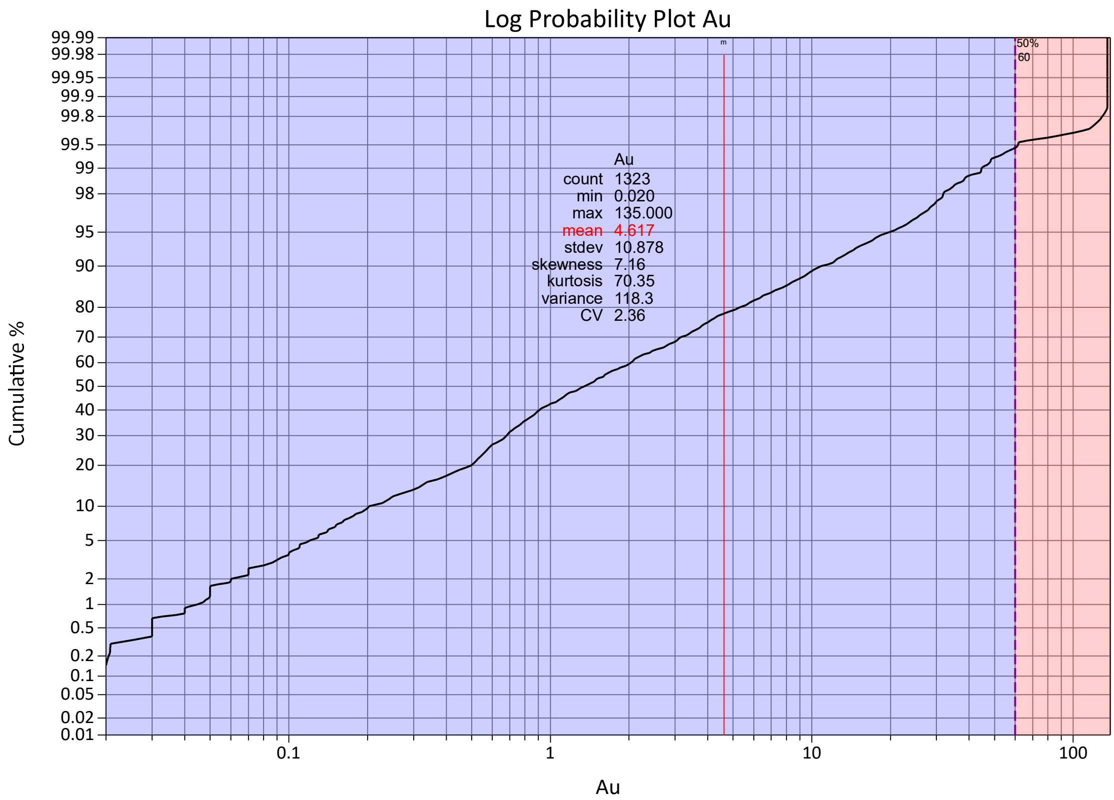


Figure 67 - Log probability plot all lodes

14.5.4 Variography Analysis

Variography was conducted to produce semi-variograms which describe the orientation and range of grade continuity. The variogram parameters are used in Kriging equations to determine sample weights and orientations of search ellipses during the grade estimation process.

Downhole variograms were produced to assess the range of grade variability over short distances and to determine the nugget effect.

Experimental semi-variograms were then generated to investigate the direction of maximum grade continuity in three directions (major, semi-major and minor), the fitted variogram for the strike direction is shown in Figure 68. Variography was conducted using Snowden – Supervisor variography tools. This proved successful for the larger lodes (domains) but not for the smaller lodes. To overcome this, domains were assigned parameters from Lode 200 (Table 39).

Table 39 - Variogram Parameters

Lode	Type	Azimuth	Plunge	Dip (to the west)	Ratio Semi	Ratio Minor	Nugget	Sill	Range
All	Spherical	020	0	-65	1.42	2.7	0.52	0.48	84

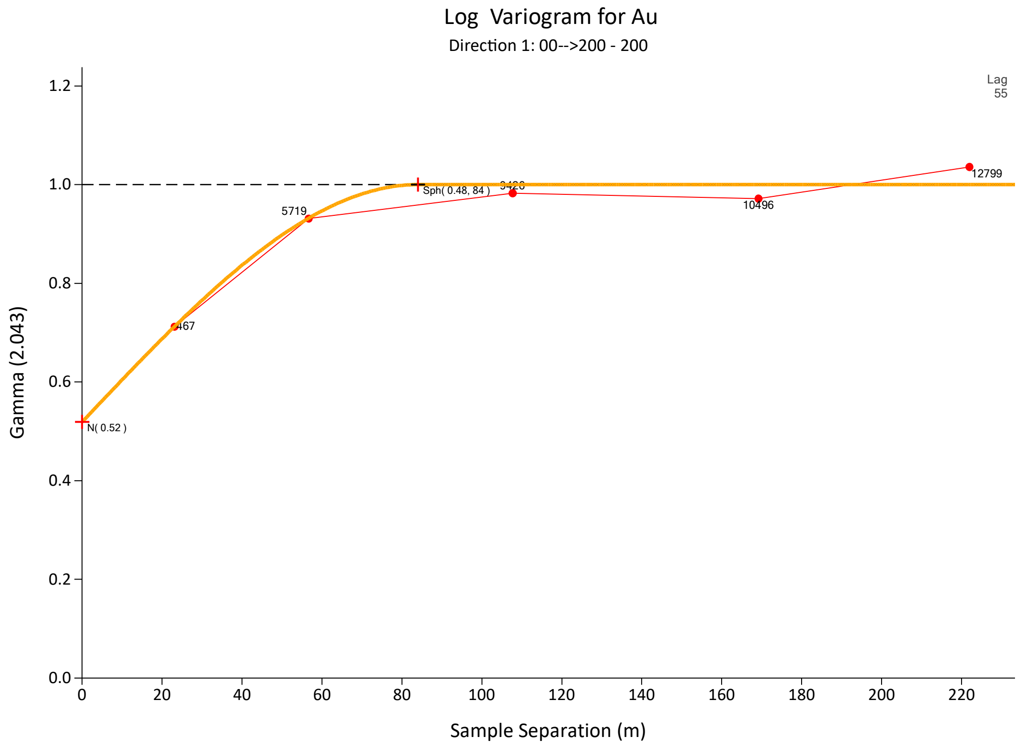


Figure 68 - Fitted Variogram – Strike

14.5.5 Kriging Neighbourhood Analysis

Quantitative Kriging Neighbourhood Analysis (QKNA) was undertaken on multiple blocks in the lode 200 domain to establish optimum search and minimum/maximum composite parameters. Goodness-of-fit statistics were generated to assess the efficiency of the various parameters. The primary statistics used are the kriging efficiency (KE) and the slope of regression.

A general summary of the main steps is provided:

- Complete QKNA for a range of potential kriging neighbourhoods
- Produce summary graphs for QKNA criteria (slope of regression, sum of negative kriging weights, kriging efficiency)

- Select kriging neighbourhoods using QKNA optimisation theory.

The Kriging Efficiency (KE) is calculated as (block variance-kriging variance)/ (block variance), where block variance is the total sill less the variance contained within a block.

The Slope of Regression is calculated as (block variance – kriging variance + μ)/ (block variance – kriging variance + 2μ).

KE calculates the overlap expected between the estimated block grade histogram and the 'true' block grade histogram. A high efficiency indicates a good match between estimated and 'true' grades, while as parameters become less optimal, KE drops. The Slope of Regression estimates the correlation between estimated and 'true' grades; a value close to 1.0 indicates a good fit. In addition, other statistics, such as the percentage of negative weights generated in a kriging plan can be considered.

A number of key input parameters can be tested in this way, including:

- Block size
- Number of discretisation points
- Search ellipse dimensions
- Minimum and maximum sample numbers in a search plan

A summary of the results is provided in Table 40.

Table 40 - QKNA Summary

Parameter	Results
Block Size	20m (X) x 20m (Y) x 20m (Z)
Minimum and Maximum Samples	12 and 24
Block discretisation	5 x 5 x 5
Search Ellipse Dimensions	42 x 30 x 10

14.6 Block Model Construction

A three-dimensional block model was generated to enable grade estimation. The selected block size was based on the geometry of the domain interpretation and the data configuration. A parent block size of 20 m E by 20 m N by 20 m RL was selected with sub-blocking to a 1.25 m E by 2.50 m N by 1.25 m RL cell size to improve volume representation of the interpreted wireframe models.

The block model construction parameters are displayed in Table 41. Block Model Attributes are displayed in Table 42.

Table 41 - Block Model Coordinates and Block Size

Model Name	Segilola_20170814		
	X	Y	Z
Minimum Coordinates	700,950	829,640	-180
Maximum Coordinates	702,340	832,510	370
Maximum Block Size	20	20	20
Minimum Block Size	1.25	2.50	1.25
Rotation	None		

Table 42 - Block Model Attribute Summary

Attribute Name	Type	Decimals	Background	Description
Au_ok_cut	Float	2	0	Estimated Grade Ordinary Kriging (OK) Top cut
Au_id2_cut	Float	2	0	Estimated Grade Inverse Distance Squared Top Cut
Au_id3_cut	Float	2	0	Estimated Grade Inverse Distance Cubed Top Cut
Au_nn_cut	Float	2	0	Estimated Grade Nearest Neighbour Top Cut
Avg_ani_dtns	Float	2	0	Average Anisotropic distance to samples
Bv_au_ok	Float	2	0	Block Variance
Cbs_au_ok	Float	2	0	Conditional Bias Slope
kani_dtns	Float	2	0	Anisotropic distance to nearest sample
Ke_au_ok	Float	2	0	Kriging Efficiency
Kvar_au_ok	Float	2	0	Kriging Variance
Lgm_au_ok	Float	2	0	Lagrange Multiplier
Negwts_au_ok	Integer	-	0	Negative Weights
Nums_au_ok	Integer	-	0	Number of Informing Samples
Sg	Float	2	2.70	Specific Gravity
Weath	Integer	-	2	Weathering (air =0, Oxide = 1, Transitional =2, Fresh =3)
Wfm_code	Integer	-	0	Coding Blocks with wireframe number
OreWaste	Integer	-	Waste	Ore or Waste
Pass	Integer	-		Pass 1 -4
Resclass	Integer	-		Resource Category (1 – Measured, 2 – Indicated, 3 – Inferred, 4 – Mineral Potential)

14.7 Grade Interpolation

Grade estimation was carried out using the Ordinary Kriging interpolation method. This uses estimation parameters defined by the variography. 1 m composites using the high grade cut dataset were used for the grade interpolation. Estimation of the resource was completed using Surpac software.

14.7.1 Estimation Technique and Parameters

OK interpolation methods were used for the current resource estimation. Sample search neighbourhoods have been optimised based on kriging statistics. Several data configurations (block locations and accompanying data spacing) were considered in this optimisation process. A minimum number of samples, numbers of drillholes, and search distances are determined by drill pattern spacing and the geometry of the mineralised lodes.

The kriging plan parameters used for grade interpolation are summarised in Table 43. Specific search ellipsoid rotations were used for each domain reflecting the domain variography orientations. A 4-pass search plan was used to estimate blocks that did not receive a grade estimate in a previous pass. Search ellipsoid dimensions were selected in relation to the nominal drillhole data spacing and identified variogram ranges. To minimize the effect of data clustering, a limit of 8 composites per drillhole was implemented on Pass 1. Block discretization of 5m E by 5m N by 5m Z RL points was adopted.

Table 43 - Search Parameters

Domain	All lodes			
Pass	1	2	3	4
Y Search Radius	42	84	168	252
X Search Radius	30	60	120	178
Max Vertical Distance	20	40	60	100
Min Samples	12	8	4	2
Max Samples	24	24	24	24
Search Azimuth	020	020	020	020
Search Dip	65	65	65	65
Search Plunge	0	0	0	0

14.8 Block Model Validation

Statistical and visual assessment of the block model was undertaken to validate block grade estimation.

14.8.1 Visual Validation

Each lode was visually checked against the composited data used in the estimation process. The onscreen validation process involved comparing block estimates and composite grades in cross section.

The onscreen validation sections showed a strong correlation between the block and composite drillhole grade and there no un-estimated block present within the mineralisation boundaries.

14.8.2 Comparison of Wireframe and Model Volume Checks

No major volume discrepancies have been found in the model comparing the solid volume (wireframe) to the volume contained in the model. A full listing of the volume checks is contained in Table 44.

Table 44 - Comparison of Wireframe v Model Volume

Lode	Wireframe Volume	Model Volume	Difference
100	253,422	253,563	+141
101	5,119	5,121	+2
200	2,622,568	2,622,539	-29
300	330,387	330,289	-98
400	98,051	97,508	-543
500	5,255	5,191	-64
600	404	395	-9
GRAND TOTAL	3,315,206	3,308,606	-600

14.8.3 Plots Validation of Interpolated Grades

The validation plots (Figure 69) show a reasonable correlation between the composite grades and the block model grades. The easting swathe plot is somewhat misleading, as it crosses multiple lodes at an oblique angle to the orientation of the block model. The broad trends demonstrated by the raw data are honoured by the block model, and the interpolated grades are generally lower than the composite values, where the raw cut grade exceeds 10g/t. The comparison illustrates the effect of the interpolation, which results in smoothing of the block grades compared to the raw grades.

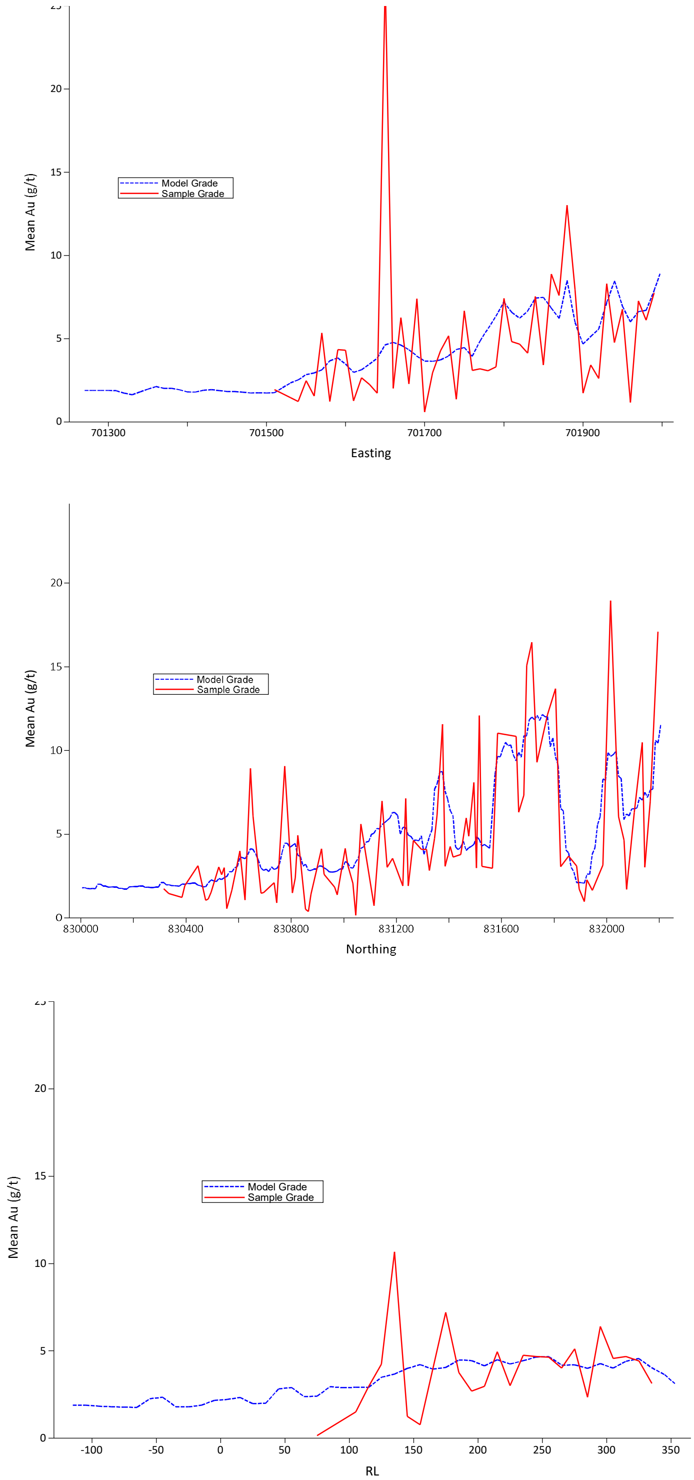


Figure 69 - Validation plots Raw Cut Grade (Red) v Block Model Grade (Blue)

14.8.4 Grade Tonnage Curve

A simple check that involves stating the global amount of ore that is available at a certain cut-off grade. A high cut-off grade would respond to a lower amount of ore tonnes available (Figure 70). For detailed grade-tonnage graphs of the classified resource refer to Figure 116,Figure 121,Figure 123 and Figure 126 in Appendix 1.

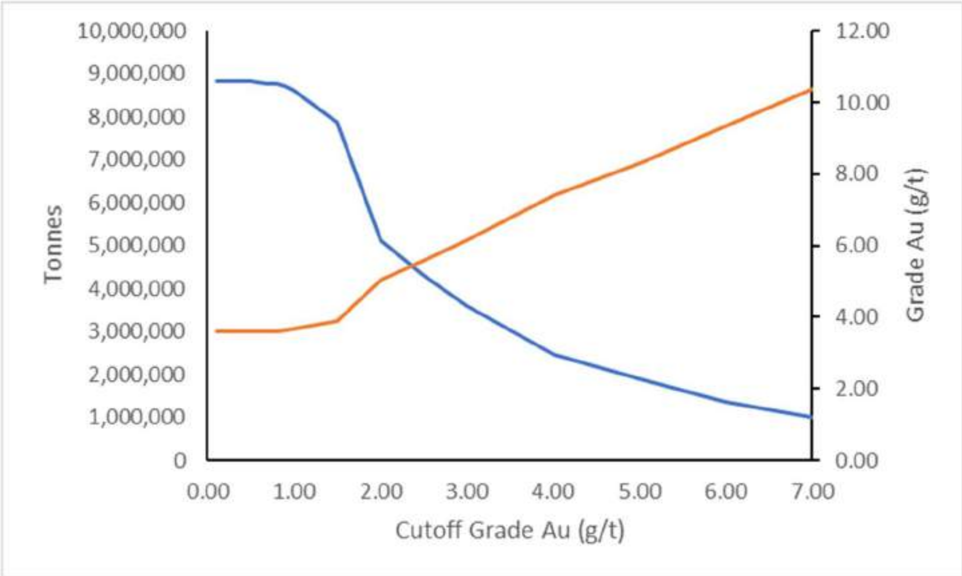


Figure 70 - Grade Tonnage Curve

14.8.5 Other Estimation Checks

An Inverse Distance Squared (ID2), Inverse Distance Cubed (ID3) and Nearest Neighbour (NN) multi pass estimation run (Table 45) were used as another validation check on the Ordinary Kriging Estimation.

Comparing the results of the passes against one another (Table 45) passes 3-4 produce slightly higher grades compared to the OK estimation. In Inverse Distance estimation, each sample is weighted according to the inverse of their separation. Therefore samples closer get a higher weighting than samples further away. Weight is directly related to distance, irrespective of the ranges of influence. Compared to Ordinary Kriging which is an inverse distance weighting technique where weights are selected via the variogram according to the samples distance and direction. Swath plots are shown in Figure 71 and show overall a very close correlation.

Table 45 - Comparison of ID2 v OK Estimation

Pass	ID2 Au Grade	ID3 Au Grade	NN Au Grade	OK Grade
Pass 1	4.32	4.38	4.16	4.44
Pass 2	3.99	4.01	3.64	4.12
Pass 3	3.95	3.90	3.67	3.86
Pass 4	2.67	2.57	2.43	2.41

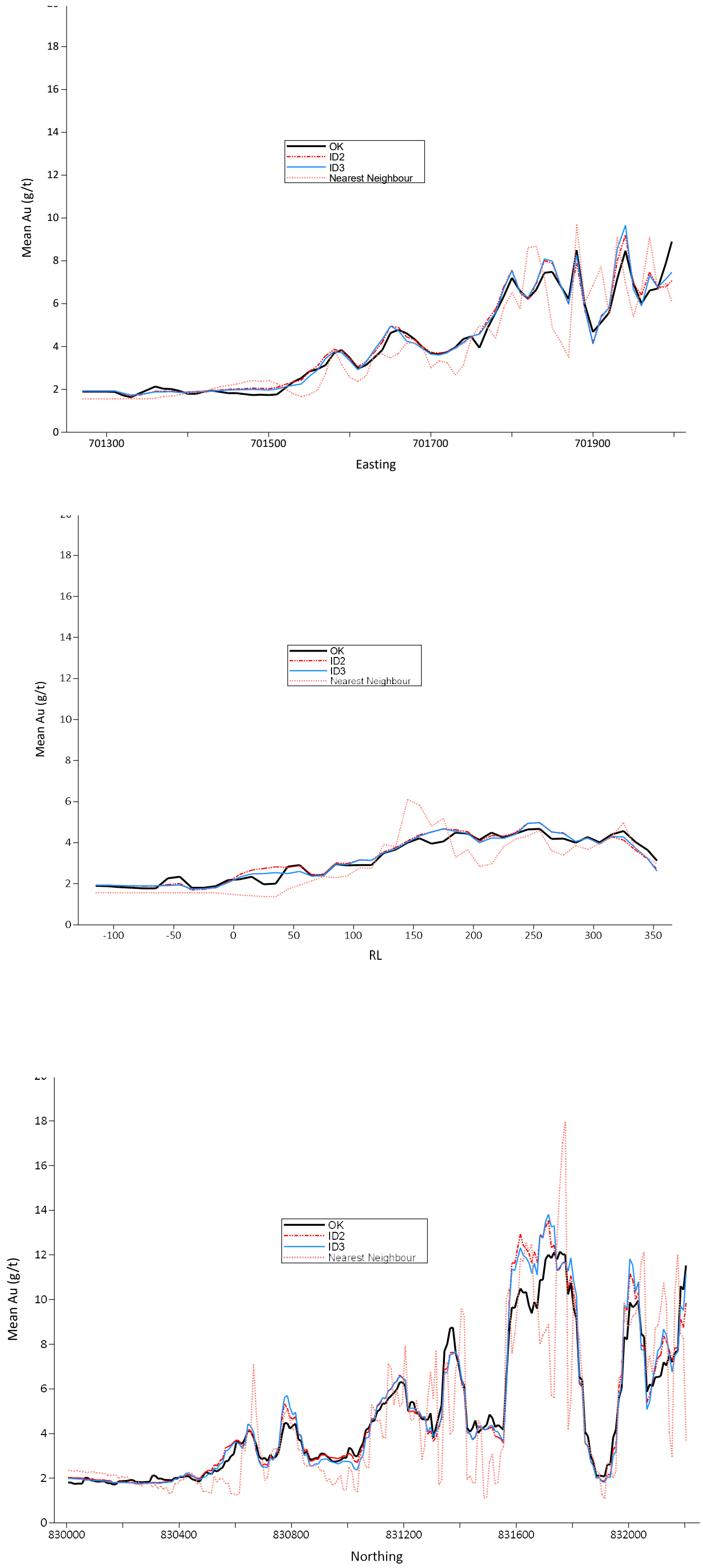


Figure 71 - Comparison of the various check estimation methods

The grade estimation check between ID2, ID3 and OK show very similar patterns in all directions, and confirms that the multiple search pass strategy for the Segilola Gold Deposit is suitable. The nearest neighbour estimation represents an attempt to adapt the amount of smoothing to the 'local' density of data.

14.9 Resource Reporting

14.9.1 Topography

A topography surface was supplied by Thor. Validation checks comparing drillhole against Topography showed no discernible errors.

14.10 Oxidation Surfaces

Constraining the block model to oxidation surfaces is important where oxidation is present. Mineralised material in oxidised zones can behave in a very different manner to that which is hosted within unoxidised or fresh material zones. In such cases, it is necessary to flag the blocks which occur within the oxidised zone to differentiate them from blocks which occur in the unoxidised material. Oxidation surfaces were digitized from weathering profiles logged in the geological database.

14.10.1 Specific Gravity

The Segilola Gold Project contains transitional (weathered) and fresh material. These materials have been assigned specific gravities of 2.65 t/m^3 and 2.67 t/m^3 respectively.

14.10.2 Model Depletion

No depletion of the model has occurred. Lode 101 wireframe was clipped against the topography and a small trench.

14.11 Resource Reporting and Classification

The Segilola Gold Mineral Resource has been classified and reported in accordance with the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI4-3101). Resource classification is based on confidence in the geological domaining, drill spacing and geostatistical measures. The initial classification process was based on an interpolation distance and minimum samples within the search ellipse.

The initial classification was reviewed visually. Based on the initial classification, changes were made to define Indicated and Inferred Mineral Resources. This defined resource categories are based on a combination of data density and geological confidence.

The resource classification codes in the model are as follows:

- Indicated Resource (resclass – 2)

- Inferred Resource (resclass – 3)
- Unclassified (resclass – 4)

A range of criteria has been considered in determining the classification including:

- Geological continuity
- Geology sections plan and structural data
- Previous resource estimates and assumptions used in the modelling and estimation process
- Interpolation criteria and estimate reliability based on sample density, search and interpolation parameters, not limited to kriging efficiency, kriging variance and conditional bias
- Drillhole spacing

14.12 Mineral Resource Estimate

Based on the 'reasonable prospects of economic extraction' test as required by the CIM, the Mineral Resources are reported within an optimised pit shell, as defined by the parameters shown in Table 46.

Table 46 –Optimisation Parameters applied for Mineral Resource reporting

Overall Pit Slope	45	degrees
Surface Mining Cost (Waste)	$2.46	US$/t
Processing Cost	$17.16	US$/t ore
Processing Recovery (Au)	96%	%
G&A Cost	$3.00	US$/t ore
Grade Control	$0.33	US$/t ore
Rehandle	$0.77	US$/t ore
Refining	$0.62	US$/t ore
Discount Rate	8%	%
Metal Price Gold	$1,600.00	US$/oz
Selling Cost/Royalties	0	-

An "Open Pit Resource" included all Indicated and Inferred material inside the RF1 pit shell resulting from the Whittle optimisation with the parameters shown in Table 47. A cut-off grade of 0.64g/t was applied to the

"Open Pit Resource" for reporting purposes to reflect the economic cut-off grade of the project as applied to the Mineral Reserve.

The "Underground Resource" was reported at a cut-off grade of 2.5g/t and includes Indicated and Inferred material external to the Whittle pit shell used to report the "Open Pit Resource".

The Mineral Resource released publicly on 11 September 2017 was reported as shown in Table 47.

Table 47 – Mineral Resource Statement using optimised pit shell

Type	Within Whittle Shell (Open Pit Resource) (0.64g/t Au cut-off)			External to Whittle Shell (Underground Resource) (2.5g/t Au cut-off)			Total		
	Tonnes	Au g/t	Ounces	Tonnes	Au g/t	Ounces	Tonnes	Au g/t	Ounces
Indicated	3,926,000	4.3	539,000	111,000	4.7	17,000	4,037,000	4.3	556,000
Inferred	835,000	5.1	137,000	1,195,000	4.4	169,000	2,030,000	4.7	306,000

Diagrams of Mineral Resource classification per lode along with tonnes and grade plots at two (2) cut-offs (0.5 and 2.5 g/t) are provided in Appendix 1.

14.12.1 Comparison with Previous Resource Estimates

The previous resource estimates are presented in Table 13 below. Slight differences exist in the classification of the resources between the 2016 and 2017 model.

Table 48 - Previous Resource Estimates

Year	Company	Tonnes (Mt)	Au g/t	Ounces (koz)	Comment
1992	Pineridge (Nig) Ltd	1.1	10.1	347	Unpublished, unclassified resource
1999	Hansa GCG	1.4	6.00	270	Unpublished unclassified resource
2009	Odessa	4.5	4.35	628	
2016	Odessa	4.6	3.80	555	Reported at 1g/t Cut-off, classified as Indicated Resource
2017	Auralia – July	5.3	4.22	725	Reported at 1 g/t Cut off Indicated & Inferred Material
2017	Auralia – August	7.4	3.96	944	Reported at 1 g/t Cut off Indicated & Inferred Material

These differences can be explained by;

- The 2016 model used a minimum number of samples (2) to classify indicated resources. The 2017 model based on KNA analysis uses a minimum of 8 samples.

- The increase in the tonnage particularly in the Inferred category, in the 2017 model is from the recently completed drilling by Thor, which extends the wireframe and estimation to the south and to deeper depths.

- A slight change in the top cut grade from 50 Au g/t (2016) to 60 Au g/t (2017)

14.13 Discussion

The current Mineral Resource model represents a robust global estimate of the in situ remaining gold mineralisation for the Segilola gold deposit.

With respect to Mineral Resources estimated at the Segilola deposit, Auralia has concluded that the geological interpretations for geology, weathering and mineralisation domains at Segilola Gold deposit are adequate for the estimation of Indicated and Inferred Mineral Resources.

The following work programs are recommended or are in progress for the Segilola Gold Project:

1. The Mineral Resource Estimate shows a substantial volume of material classified as Inferred or Unclassified. This material is an immediate target for resource classification upgrade.

2. It is recommended that optimised pit shells are used as a guide to creating drilling progams that maximise the conversion from lower to higher resource classification and reduce mining risk attributed to data density and quality.

3. Maintain the current quality assurance procedures to ensure high quality data is available for subsequent resource estimates.

4. Collection of additional structural measurements of mineralised vein orientations, obtained from future diamond core drilling programs, is essential for fine tuning the mineralisation domain boundaries for any future resource model updates. Continued improvement in geological understanding and lithology unit interpretation.

5. Undertake programs to establish the suitability and reliability of drilling techniques, i.e. by collecting twinned samples from the same hole (field duplicates) and from paired holes of either the same method or of different methods.

15. MINERAL RESERVE ESTIMATES

15.1 Mineral Reserve

The Mineral Reserve (shown in Table 49) constitutes the portion of the Mineral Resource which is economically and practically mineable under the specified project parameters. The Mineral Reserve has been estimated in accordance with NI 43-101 guidelines, which excludes Inferred Resources.

Table 49 – Mineral Reserve

Classification	Cut off	Tonnes	Grade	Ounces
	g/tAu	(t)	(g/tAu)	(Au)
Probable	0.64	3,345,000	4.2	448,000

15.2 Pit Optimisation

The following information outlines the steps taken to estimate the potential Mineral Resource contained within an economically viable pit using the industry benchmark Geovia Whittle open pit optimisation software.

The block model used for the pit optimisation was created by Auralia using drillhole data and mineralisation wireframes provided by Thor. Refer to Section 14 for full details on the processes and parameters used to create the block model. The final geological model had the following parameters:

Table 50 - Geological Model Parameters

Segilola Resource Model	Block Dimensions (XYZ)	Framework Dimensions (XYZ)	X Origin	Y Origin	Z Origin
Segilola_20170814	20.0x20.0x20.0	70x144x28	700,950	829,640	-180.0

Engineering processing, QA/QC and cross-checks were carried out before the model was scripted for export into Whittle optimisation software. The geological model was already cut to the topographical surface file provided by Thor ("topo_20170814"), and included waste rock within the framework.

The following processing and QA/QC was performed on the model:

- Rock code field created for Whittle. For the purposes of the engineering model, no differentiation was made between the two density values present in the model (2.65 and 2.67t/m^3) due to the difference being less than 1%.
- Mining operating costs written in to the block model

- Exported from Surpac with block dimensions 10.0m x 10.0m x 10.0m. This step was performed in order to ensure that Whittle was able to process the blocks without running out of capacity for ore parcels, and also to provide a block size which is more practical for pit design purposes.

The parameters of the final engineering model uploaded into Whittle are shown in Table 51. A 3D view of the model is shown in Figure 72.

Table 51 - Whittle Model Parameters

Segilola Whittle Model	Block Dimensions (XYZ)	Framework Dimensions (XYZ)	X Origin	Y Origin	Z Origin
Segilola_20170814_whit_pfs	10.0x10.0x10.0	140x288x56	700,950	829,640	-180.0

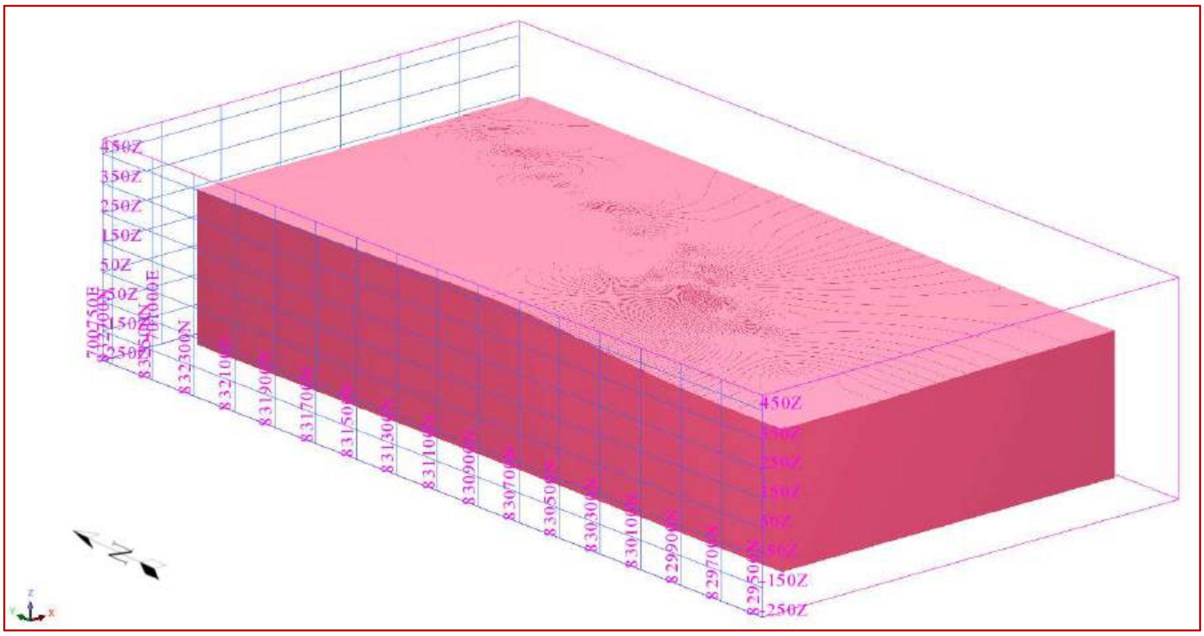


Figure 72 - Whittle Full Model 3D View

15.2.1 Whittle Optimiser Overview

The evaluation of alternative mining scenarios and selection of the final optimal pit shell was carried out using the Geovia Whittle™ optimiser package. Whittle is currently considered as the industry-standard open pit optimiser and is used as a benchmark for mining studies throughout the world.

Essentially, Whittle answers the question "how big should my pit be to maximise the net present value of the project?" In brief, it does so by utilising the Lerchs-Grossman 3D algorithm to produce a set of nested pits for each pit shell node. In doing so it evaluates block values and uses arc relationships to set block mining precedence. Each block has an economic value applied to it, and each linked block in the arc must 'pay' for each previous block in the arc in order to be economically extracted. An example of Lerchs-Grossman algorithm (shown as a very simplified 2D block out) is displayed in Figure 73. In the example below a 45 degree wall angle is used, and an assumption is made that all other non-value blocks are "waste" with an economic value of -1.

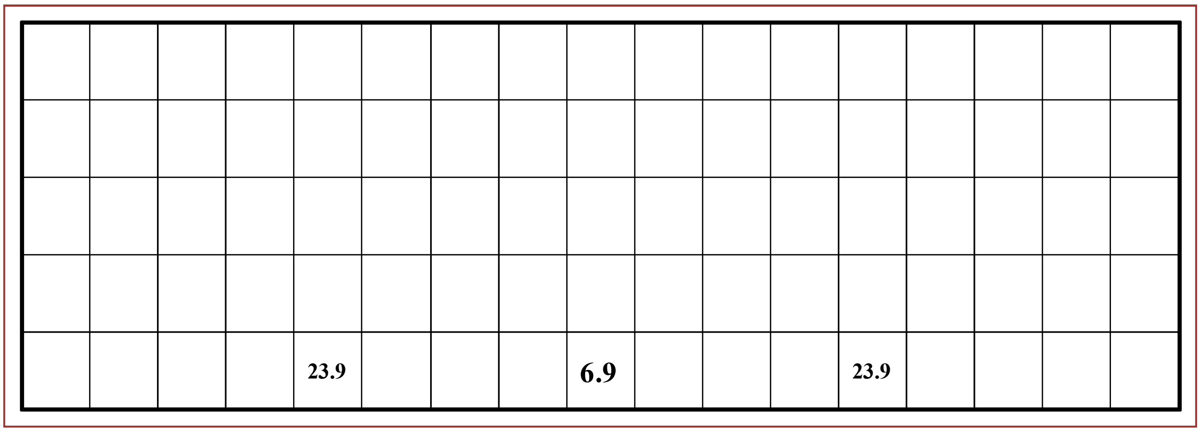


Figure 73 - LG 2D Example Part 1

The algorithm calculation for each block progresses upwards towards the surface, with each block adding to the cost of extracting the value blocks below. As the middle block value of 6.9 cannot pay for the cost of removing the 7 waste blocks, it drops out of the optimisation. The final optimisation produces a 'W' shaped pit shell as shown in Figure 74.

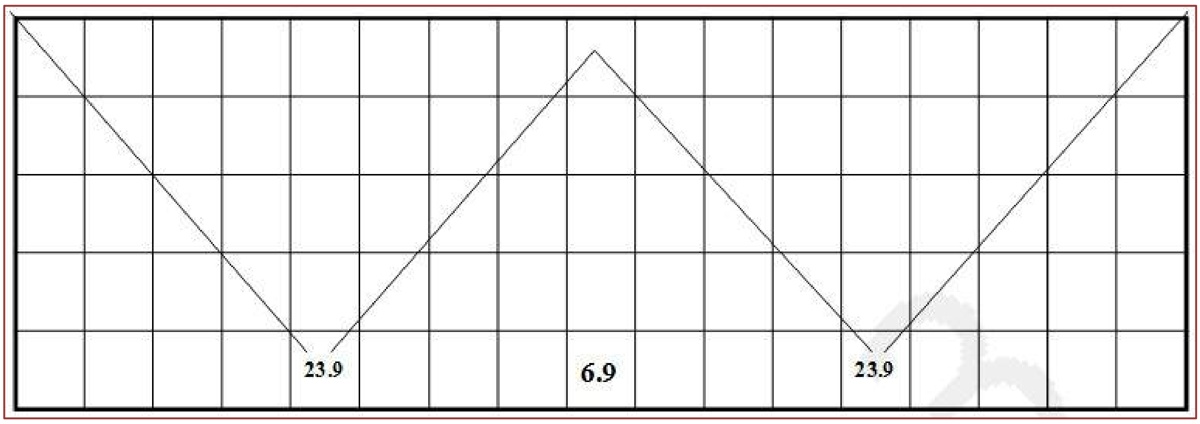
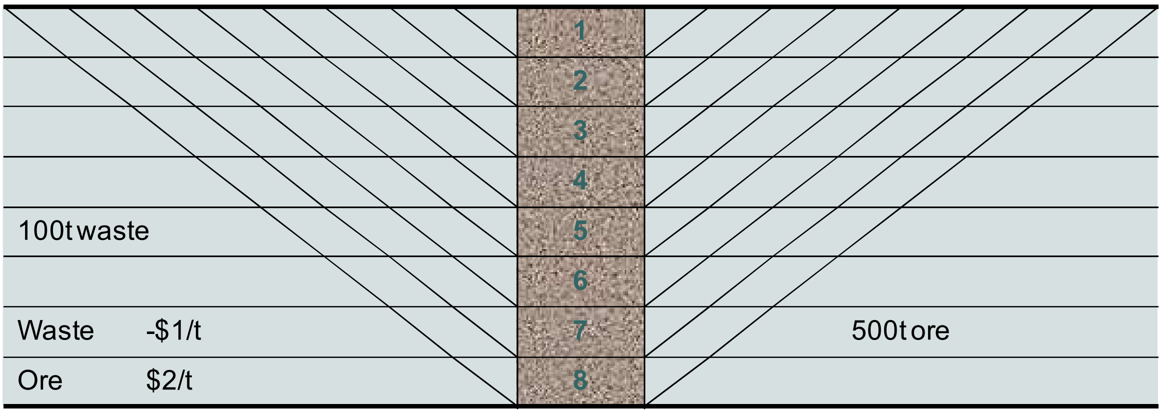


Figure 74 - LG 2D Example Part 2

As previously outlined, the Whittle optimiser produces a set of these shells (nested pits) by varying the revenue factor applied to provide shells and subsequent economic data on which to base a final design. This process is necessary, as in some cases the shell of highest economic value is not always chosen. The reasons for doing so can be varied, but can include strategic considerations such as maximising feed or life-of mine or expediting highest grade material to bring forward maximum cash flows.



Pit	1	2	3	4	5	6	7	8
Ore	500	1,000	1,500	2,000	2,500	3,000	3,500	4,000
Waste	100	400	900	1,600	2,500	3,600	4,900	6,400
Total	600	1,400	2,400	3,600	5,000	6,600	8,400	10,400
Value	$900	$1,600	$2,100	$2,400	$2,500	$2,400	$2,100	$1,600

Figure 75 - Whittle Sell Selection Example (Nested Pits)

For example, Figure 75 illustrates that although pit shell five has the highest theoretical value, all pit shells are economically viable.

Selecting pit shell four will see a loss of $100 when compared to the maximum value of pit shell five's $2,500, but this value is achieved more quickly, as only 3,600 tonnes of material are required to be excavated. Alternatively, pit shell eight is also economically viable, and provides the longest mine life.

15.2.2 Base Case Optimisation Parameters

Optimisation runs were conducted in Whittle using tailored parameters appropriate to the deposit. The parameters listed in the following sub-sections refer to the base case run. Parameters were altered for the sensitivity analysis runs, as detailed in section 15.2.5.

The base case optimisation parameters are summarised in Table 52, and detailed in the following sub-sections.

All selling prices and costs applied were in US dollars.

Table 52 - Base Case Whittle Optimisation Parameters

Overall Pit Slope	45	degrees
Surface Mining Cost (Waste)	$2.46	US$/t
Mining Dilution	10%	%
Mining Recovery	95%	%
Processing Cost	$17.16	US$/t ore
Processing Recovery (Au)	96%	%
G&A Cost	$3.00	US$/t ore
Grade Control	$0.33	US$/t ore
Rehandle	$0.77	US$/t ore
Refining	$0.62	US$/t ore
Discount Rate	8%	%
Metal Price Gold	$1,250.00	US$/oz
Selling Cost/Royalties*	0	-

**Selling Costs/Royalties were not applied in Whittle but were applied in the Economic Anaylsis – refer to section 15.2.2.10 for details.*

15.2.2.1 Base Date

Whittle optimisation work for this study was carried out over the period of August to September 2017. Parameters applied and outputs resulting from this study should be referenced to this period.

15.2.2.2 Materials Processed

Optimisations were run based on processing only “Indicated” classified material.

15.2.2.3 Slope Sets

An overall wall angle of 45 degrees was applied. This is based on a Pre-Feasibility Geotechnical Assessment report completed by Peter O’Bryan & Associates which was provided by Thor. The overall wall angles from this report are 45 degrees for the west wall, and 52 degrees for all other walls (not inclusive of ramps). In order to account for ramps in the pit design on the eastern side of the pit, a 45 degree wall angle was applied for all walls in the pit optimisation. Further detail on geotechnical studies is provided in 15.3.1.1, and the report is provided in Appendix 2.

15.2.2.4 Mining Operating Costs

Requests for Quotation (RFQ) were sought from a number of mining contractors operating in Africa. Several pricing submissions were received, and an appropriate submission was selected to use as a basis for the mining operating costs to be applied in the PFS.

The contractor pricing was supplied based on an assumed diesel fuel price of US$0.63/L, which was the best information available at the time of seeking the pricing submissions. An adjustment was subsequently applied to the load and haul contract mining rates based on a quote for bulk diesel supply obtained by Thor at US$0.47/L.

The unit mining costs ($/t) were written directly to the Surpac block model based on the block being considered ore or waste (using an arbitrary 0.5g/t cut-off), the weathering profile and depth from surface. The mining cost of waste at surface is US$2.46/t and is inclusive of drill and blast, load and haul, fuel, all labour and equipment maintenance as well as site overheads (owner's costs), geotechnical, rehabilitation and dayworks. Further detail including the full set of mining rates by depth can be found in section 21.4.1.

15.2.2.5 Mining Dilution and Recovery Factors

A 10% mining dilution factor was applied to the project within Whittle to account for dilution that would be expected to occur during the course of mining due to the mixing of ore and waste material during blasting and excavation processes.

A 95% mining recovery factor was applied within Whittle to account for the amount of ore that is lost due to spillage and/or rehandling and also accounts for any unforeseen additional ore losses (ore trammed to the waste dump, etc).

These factors are considered appropriate for the nature of the deposit and the dimensions of the ore lodes.

15.2.2.6 Processing Operating Costs

The processing all-inclusive operating cost was set at US$19.06/t of ore delivered to the processing plant. This is comprised of the following:

- US$17.16/t processing cost derived from preliminary metallurgical test work conducted for the RBFS, with adjustment factors applied to account for current diesel fuel costs for power generation
- US$3.00/t G&A overheads cost as directed by Thor, based on an assumption of US$1.5M per annum spending in this area on items such as site salaries and overheads, insurance, accounting, legals and consulting fees relating to this project
- US$0.33/t grade control cost - assumed value based on RBFS information and experience with similar operations

- US$0.77/t re-handle – estimated value based on 80% of material dumped on the ROM pad requiring re-handling, at ~US$0.96/t (contractor quoted rate)
- US$0.62/t refining cost – assumed value based on RBFS information

Further detail can be found in section 21.4.2.

15.2.2.7 Processing Recovery

A flat processing recovery of 96% for gold was applied to all ore based on preliminary metallurgical test work.

15.2.2.8 Process Constraint

Preliminary metallurgical test work has confirmed that conventional processing using Carbon-in-Leach ("CIL") would be suitable for the Segilola mineralisation. A process constraint of 500,000 t per annum of ore was applied to the optimisation.

15.2.2.9 Commodity Price

The commodity price used for the base case optimisation was US$1250/oz for gold.

15.2.2.10 Selling Costs

No selling costs or royalties were applied to the optimisations. These were applied in the Economic Analysis/cash flow modelling due to the fact that the royalties have maximum values which is not conducive to accurate application in Whittle. The applicable royalties are as follows:

- 1.5% Net Smelter Return royalty payable to Tropical Mines Limited, to a maximum value of US$4M
- 1.5% Net Smelter Return royalty payable to Ratel Group Limited, to a maximum value of US$3.5M

Refer to Section 22 for details on the application of these royalties as part of the Economic Analysis/cash flow modelling.

15.2.2.11 Economic Cut-Off

No cut-off grades were forced within the optimisations; Whittle applied the appropriate economic cut-off grade as required per run. The final economic cut-off determines what is classified as ore and waste. It varies depending upon the parameters input to the formula. The formula for economic cut-off grade is as follows:

$$ECOG = (Mining\ Dilution\ x\ Processing\ Cost)/Processing\ Recovery\ x\ (Sell\ Price - Sell\ Costs)$$

The cut-off grade for the parameters used in the base case optimisation was calculated as ~0.62g/t (not inclusive of royalties – refer to section 15.2.2.10); this was manually checked externally from Whittle with no error discovered.

15.2.2.12 Discount Rate

A discount rate of 8% was used; this is considered the rate that reflects the level of risk and cost of capital for this project.

15.2.2.13 Capital Costs

Pre-production capital and mine closure costs were not included in the optimisation as they have no impact on the selected pit shell, sustaining capital was included in the optimisation.

15.2.3 Whittle Outputs

Using the previously described optimisation input parameters, Whittle produced a set of nested pit shells for the base case optimisations. It must be noted that no capital costs or taxes were applied to the optimisations; these costs are applied during financial modelling. All discounted cash flow ("DCF") figures shown in section 15.2 of this report do not take the capital costs or taxes into account.

The following gives a brief breakdown describing each case scenario type (*ref: Geovia Whittle*):

Best: The best-case scenario consists of mining out pit 1, the smallest pit, and then mining out each subsequent pit shell from the top down, before starting the next pit shell. In other words, there are as many intermediate mining pushbacks as there are pit outlines. This schedule is seldom feasible because the pushbacks are usually much too narrow. Its usefulness lies in setting an upper limit to the best achievable DCF.

Worst: The worst-case scenario consists of mining each bench completely before starting on the next bench. This schedule is usually feasible and is used for most baseline and sensitivity runs, as the mining style sets a lower limit to the DCF (unless you mine waste to the exclusion of ore)

Scheduled (Specified): If, as is the case with longer term mining life pits, the difference between Worst and Best case is significant, you can approximate a more realistic mining schedule between the two extremes, by specifying the sequence of pit outlines to optimise. Ideally, you will want to choose pushbacks that satisfy your mining constraints and produce a DCF curve that is as close as possible to the best-case curve.

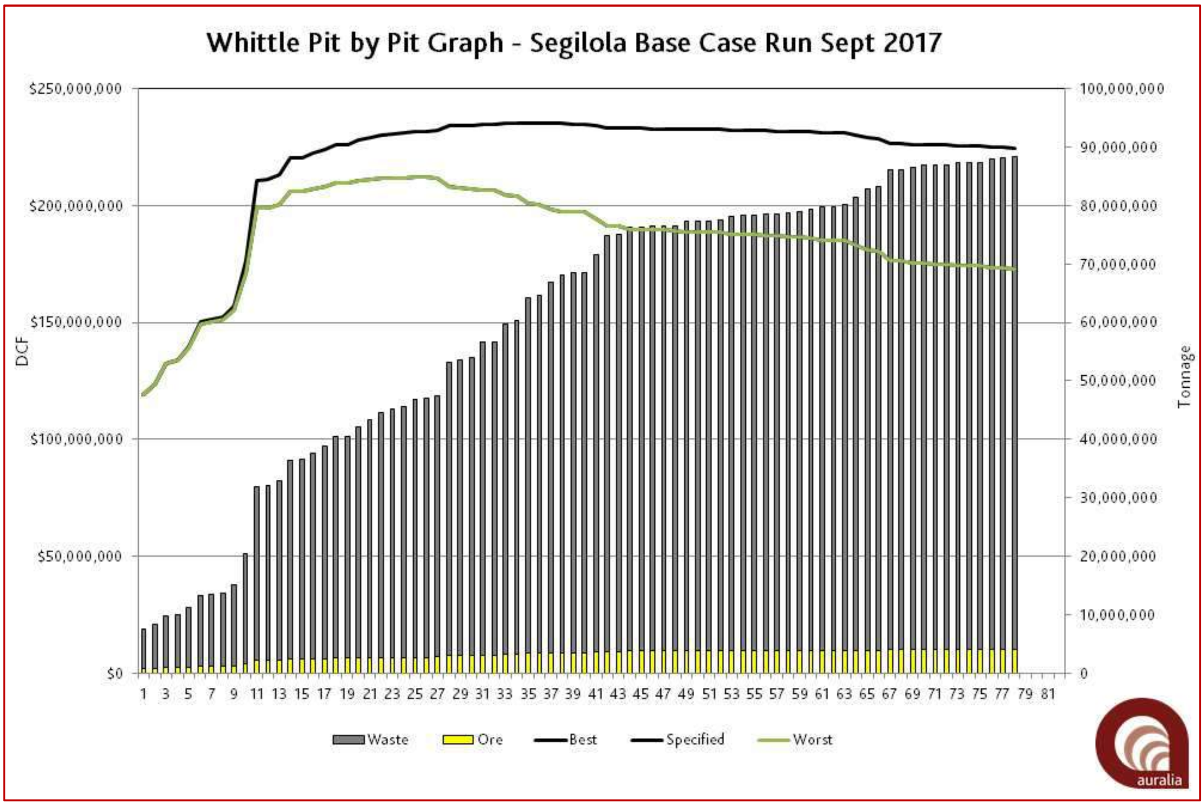


Figure 76 - Whittle Output: Pit by Pit Graph

15.2.4 Final Pit Shell Selection

Thor advised that final pit shell selection should be on the basis of maximising the NPV of the project. For this reason, the RF1 shell number 35 was chosen as the final pit shell and the basis for the sensitivity analyses and pit designs.

Due to the shape of the scheduled case DCF curve on the pit by pit graph (Figure 76), any pit selected between 27 and 48 would have negligible impact on project DCF. These other shells would reflect a change in philosophy by targeting more ounces (larger pit shell) or a higher margin on gold produced (smaller pit shell). As current work by Thor is targeting potential underground mineral resources, a shift to a smaller pit still could also be warranted; further work would be required to determine the pit size and potential point of transition to an underground operation.

The resulting Revenue Factor 1 (RF1) Whittle shell is shown in Figure 77 and Figure 78, with ore displayed at greater than or equal to the 0.64g/t economic cut-off grade (inclusive of royalties).

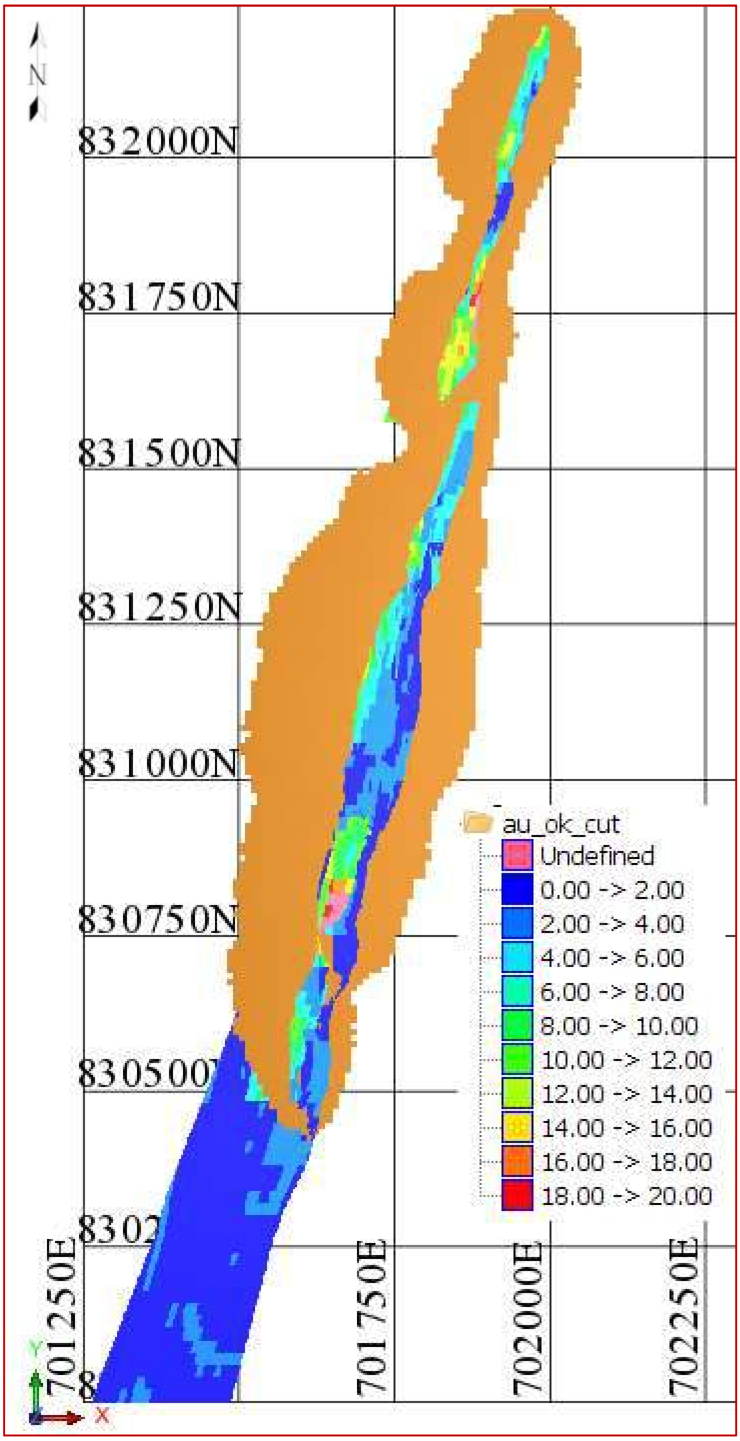


Figure 77 - Whittle RF1 Shell, Au >=0.64g/t Displayed (Plan)

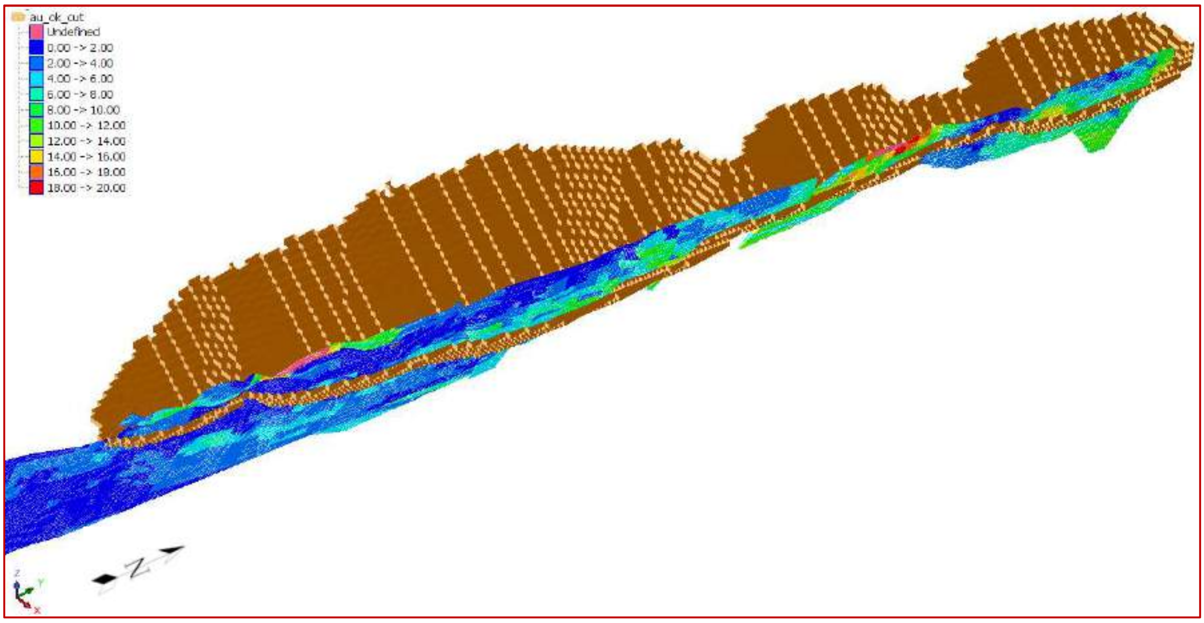


Figure 78 - Whittle RF1 Shell, Au >=0.64g/t Displayed (3D View)

Outputs for the base case run are shown in Table 53. Note that the discounted cashflow outputs are for the worst-case scenario in Whittle, the physical parameters of a given pit shell are the same regardless of whether the best, worst or scheduled case scenario is applied.

Table 53 - RF1 Base Case Whittle Optimisation Shell Outputs

$200.8M	3.3Mt	4.2	18.2

15.2.5 Open Pit Sensitivity Analysis

A sensitivity analysis was performed on the Segilola block model using Whittle. To compare like results, the RF1 pit shell was selected for each sensitivity, reported DCF values are from the worst-case scenario of each, again, no capital costs or taxes were included in this analysis. The sensitivity studies (or range runs) were carried out on the base case Whittle parameters; the analysis displayed standard expected linear shifts in economics based upon variation.

The following sensitivity runs were performed on the base case scenario:

- Sell price variations at -20%, 10%, +10% and +20%
- Processing cost variations at -20%, 10%, +10% and +20%

- Mining cost variations at -20%, 10%, +10% and +20%
- Overall slope angle variation at +5 degrees and -5 degrees (base case 45 degrees)

While the observed effects on the cash flow via sell price variation is as generally expected, the comparative project physicals have illustrated the robustness of the project in relation to changes in the other variables. This can be seen a positive indicator for the project.

As illustrated in Table 54, in terms of cash flow the project appears to be more sensitive to mining cost than processing cost due to the relatively high strip ratio, while moderately sensitive to a reduction in pit wall angle.

Table 54 - Sensitivity Outputs Summary

	-20%	-10%	Base Case	+10%	+20%
Metal Sell Price	$138.6	$168.9	$200.8	$237.8	$273.9
Processing Cost	$208.9	$205.2	$200.8	$196.7	$194.4
Mining Cost	$224.6	$213.0	$200.8	$193.3	$182.4
			Variance		
Metal Sell Price	-31%	-16%	0%	18%	36%
Processing Cost	4%	2%	0%	-2%	-3%
Mining Cost	12%	6%	0%	-4%	-9%
Parameter	Discounted Cashflow $M				
	40 Deg	Base Case	50 Deg		
Overall Slope	$184.4	$200.8	$220.4		
	Variance				
Overall Slope	-8%	0%	10%		
Parameter	**Discounted Cashflow $M**				
	Base Case	Base Case +5%	Base Case +10%		
Mining Dilution	$200.8	$195.3	$191.1		
	Variance				
Mining Dilution	0%	-3%	-6%		

The graphs in Figure 79 to Figure 83 the effects of the sensitivity variations upon ore tonnes, waste tonnes, recovered ounces and DCF.

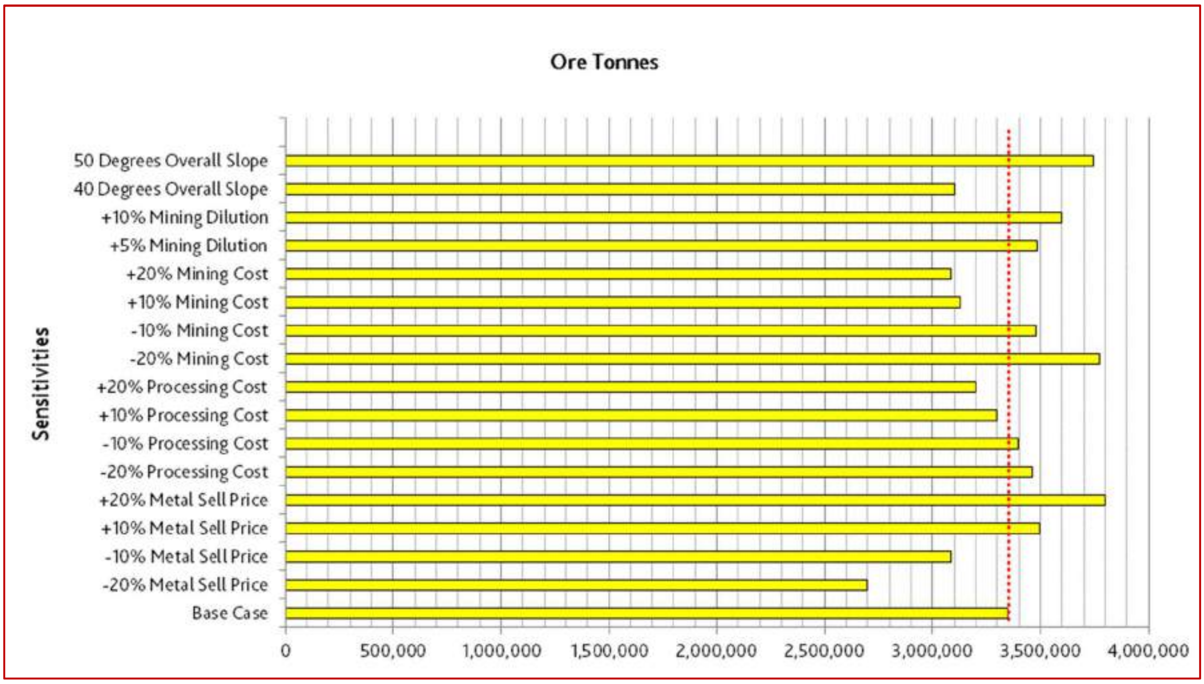


Figure 79 - Sensitivity Ore Tonnes Graph

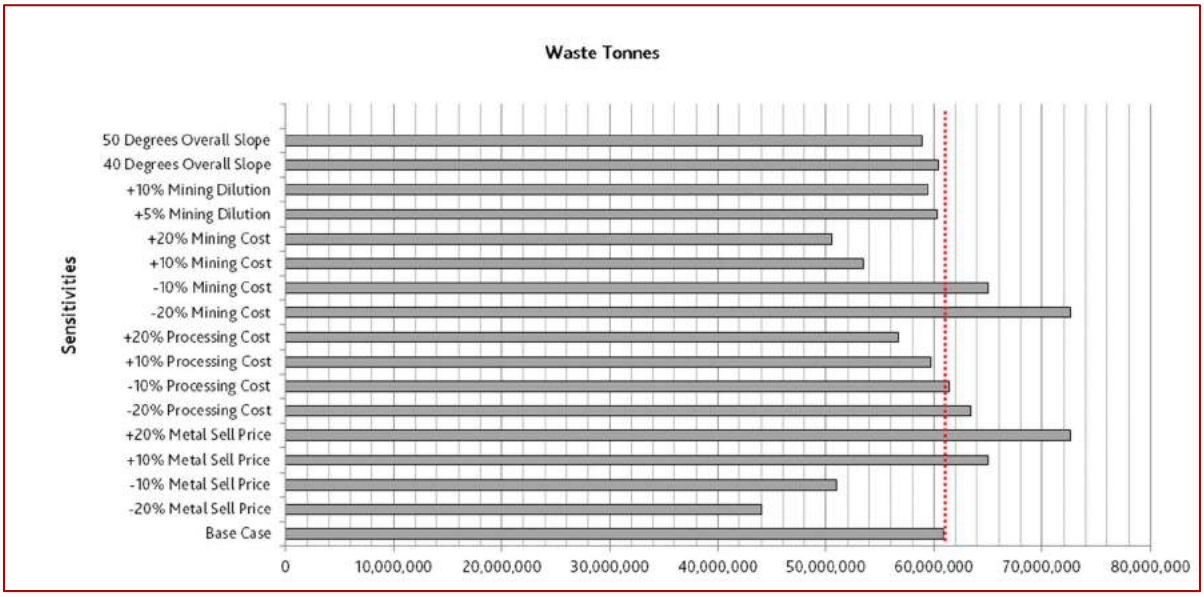


Figure 80 - Sensitivity Waste Tonnes Graph

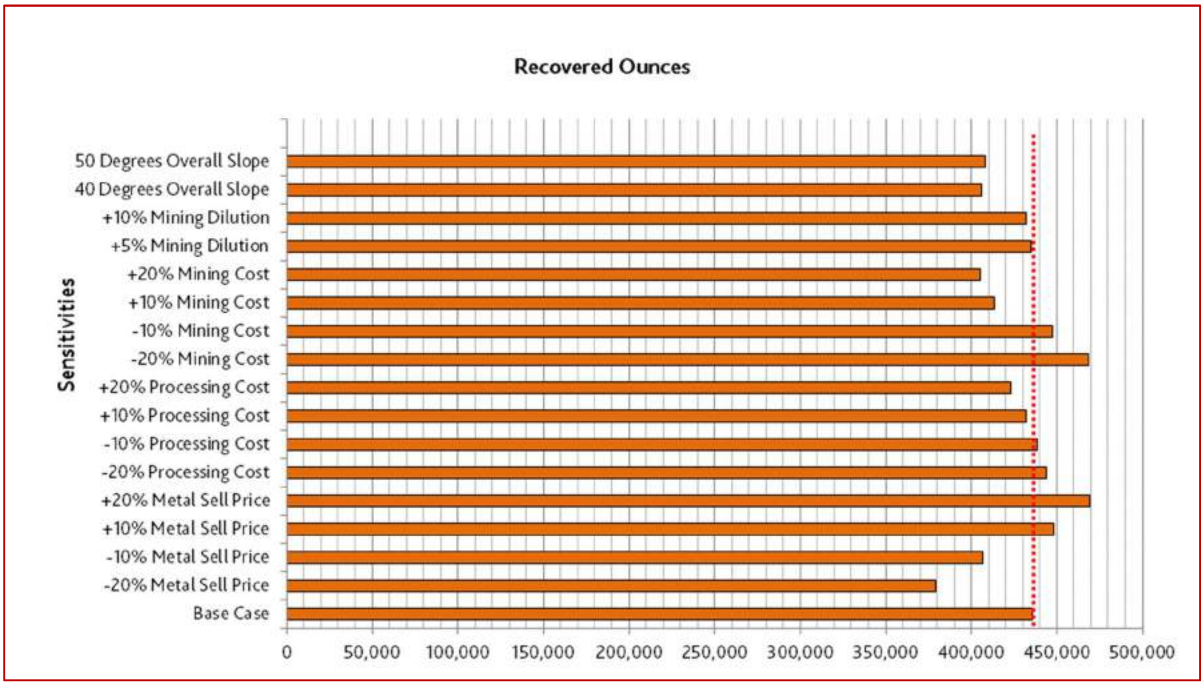


Figure 81 - Sensitivity Recovered Ounces Graph

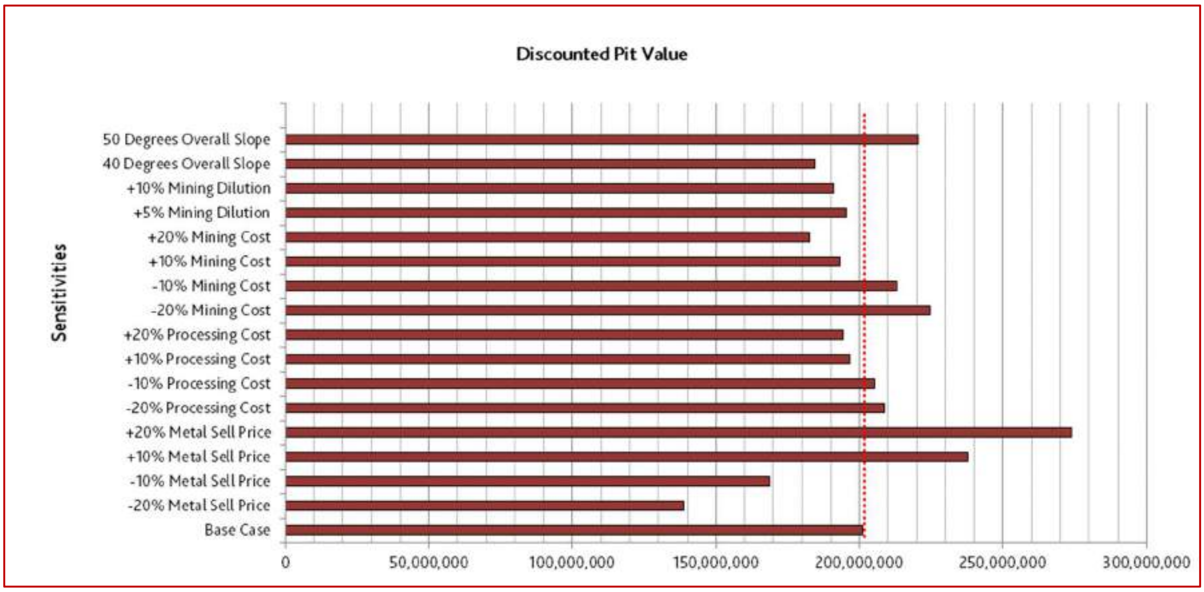


Figure 82 - Sensitivity DCF Graph

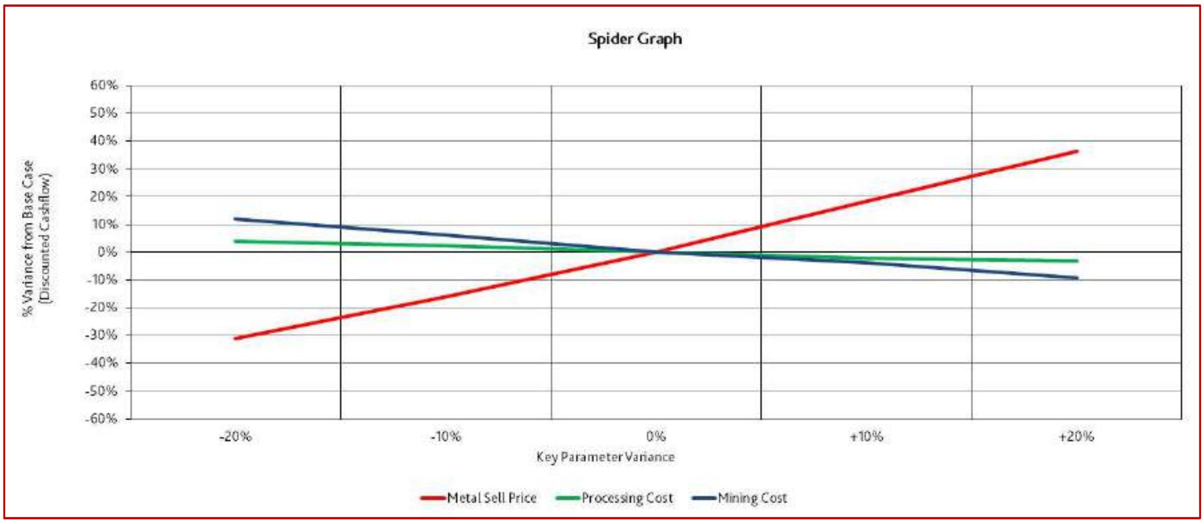


Figure 83 - Sensitivity Spider Graph

15.3 Pit Design

A final pit design has been completed based on the selected pit shell (refer to section 15.2.4). The material classified as "Indicated" contained within this final pit design constitutes the Mineral Reserve after the application of mining recovery and dilution factors.

The final pit design, shown in Figure 84 to Figure 86, has the following approximate dimensions:

- Length ~1,900m
- Width ~ 230 to 380m
- Depth ~90 to 200m
- Area ~ 45ha

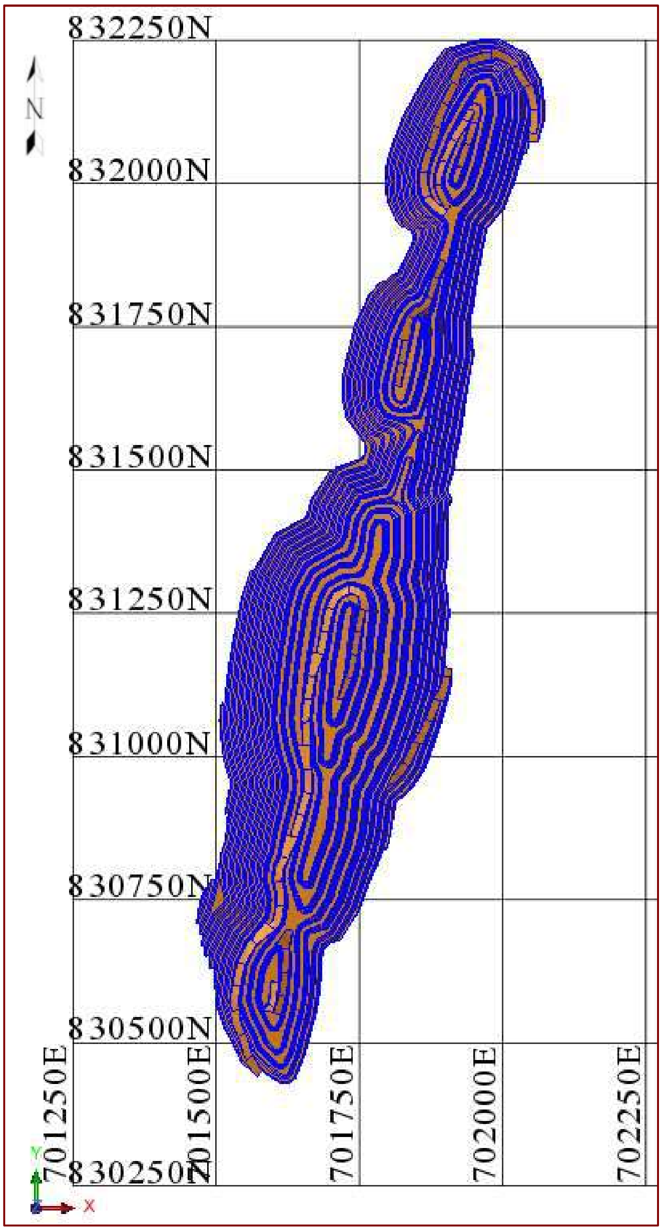


Figure 84 – PFS Pit Design, Plan View

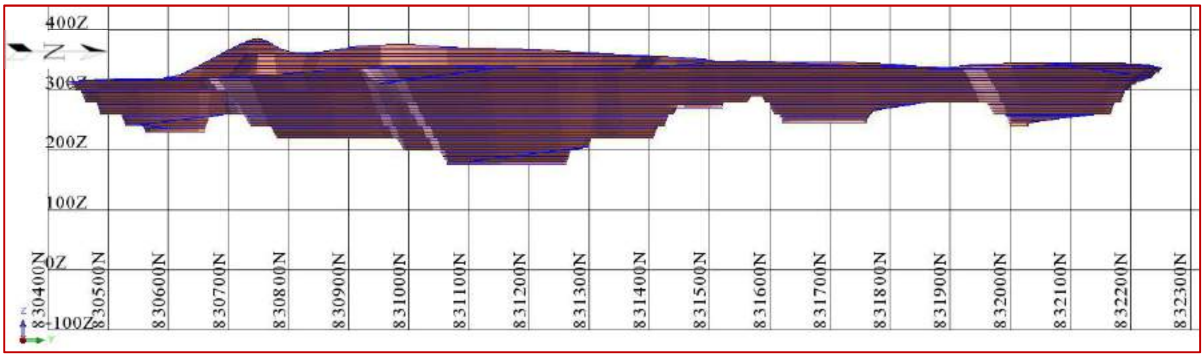


Figure 85 – PFS Pit Design, Long Section

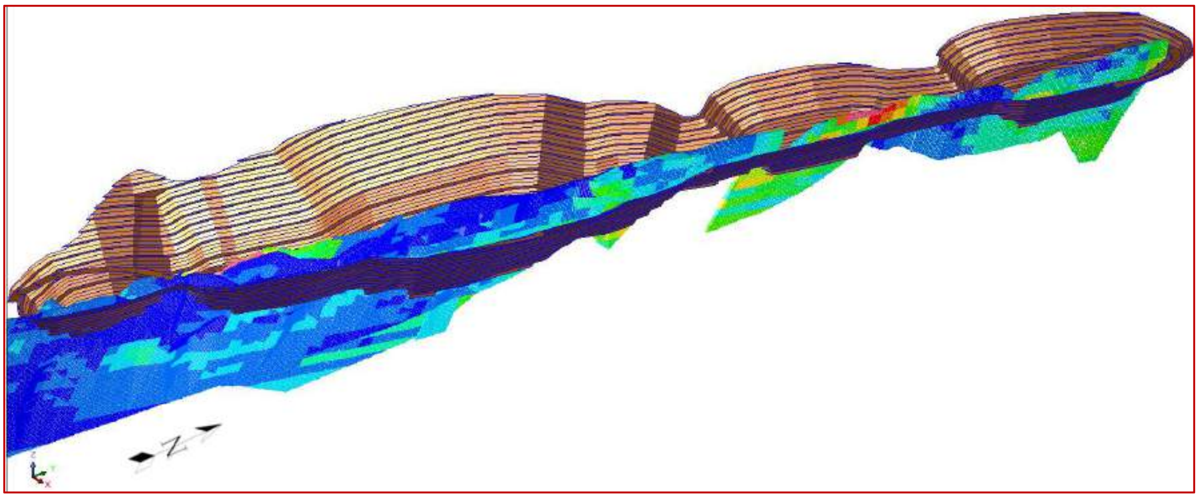


Figure 86 – PFS Pit Design, 3D View showing Ore >=0.64g/t Au

15.3.1 Pit Design Parameters

The following parameters were used to ensure that the final pit design could be achieved safely and efficiently with the specified mining equipment:

- Ramps 14m wide (single lane for 65t rear dump trucks with passing bays at berms levels) at 1 in 8 gradient
- Minimum mining width (pit floor) ~10m
- Minimum cutback width ~50m
- Walls designed as per recommendations from the Geotechnical Assessment report (refer to section 15.3.1.1) with some minor changes:
    - Weathered rock batter angles were applied to all benches above 300mRL

- Some berms were reduced from 12m to 8m or 8m to 5m when close to the final "local" pit floor

Ramp entry/exit points were designed primarily to limit the haulage distance to the waste dump and as such are towards the eastern side of the pit, closest to the waste dump. Where possible, the haulage distance to the ROM pad has been minimised by designing the ramp entry/exit point to sit at the northern or southern end of each stage/cutback.

15.3.1.1 Geotechnical

A Pre-Feasibility Geotechnical Assessment report was completed by Peter O'Bryan & Associates, who were commissioned by Thor to establish the geotechnical design to PFS level. This report incorporated data from structural and geotechnical logging of three boreholes which were drilled as part of the 2016-17 drilling campaign. These holes were located within a preliminary Whittle pit shell provided by Auralia, and were oriented and logged for geotechnical and structural data. Figure 87 shows the detail of the logged geotechnical holes within the pit shell, while Figure 88 shows the location of structural orientation data sources (highlighted in yellow).

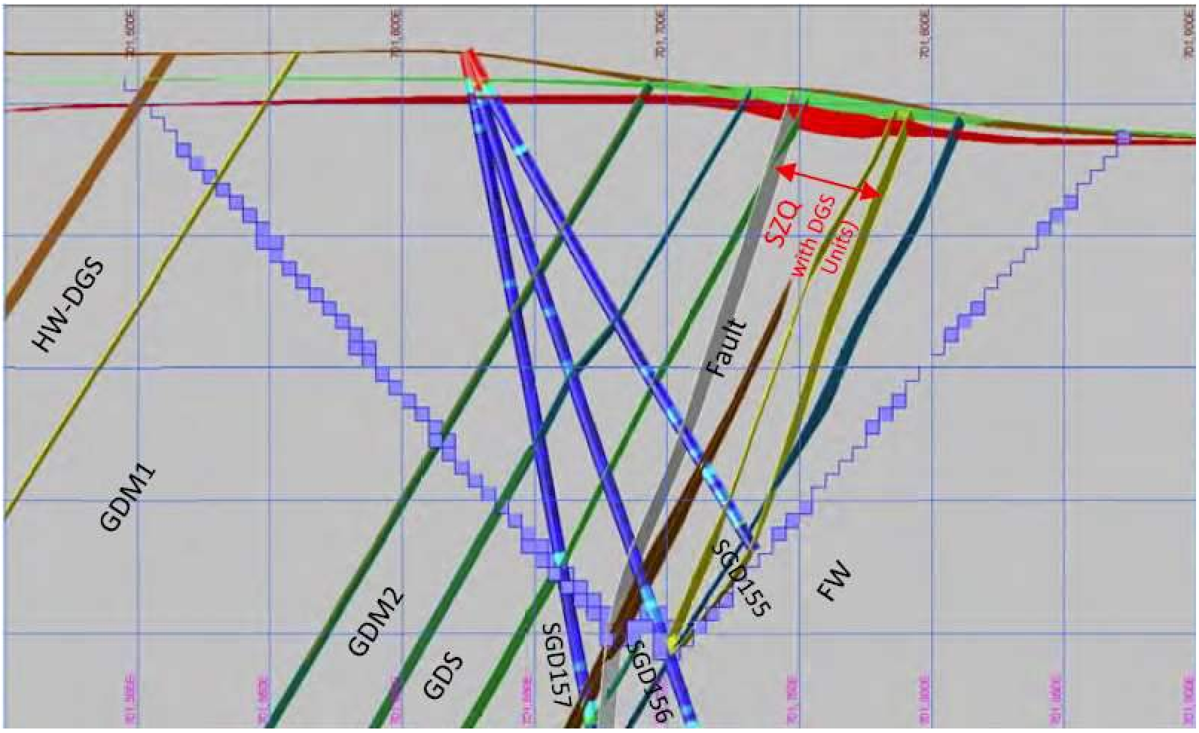


Figure 87 – Section showing detail of logged geotechnical holes within the pit shell

Source: "Thor Report POB17053C-Final", Peter O'Bryan & Associates

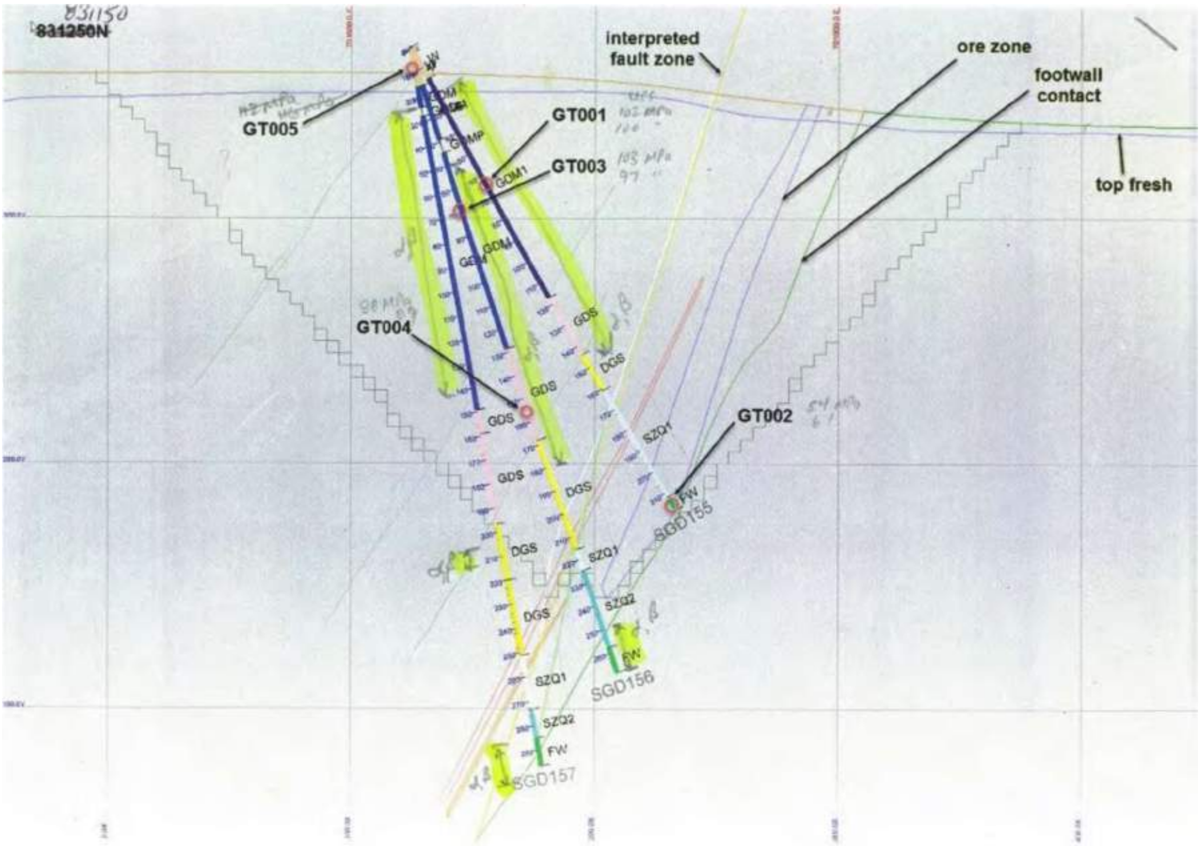


Figure 88 – Section showing geotechnical holes with structural orientation data sources

Source: "Thor Report POB17053C-Final", Peter O'Bryan & Associates

Samples were taken for Uniaxial Compressive Strength (UCS) testing and sent to Thurber Engineering. The results of this testing is shown in Table 55.

Table 55 – Summary of UCS Testing Results

Lithology	Sample Nos.	Borehole	Average UCS MPa	Comments
GDM1	GT001, 3, 6, 9, 10	SGD155, 156, 158, 159, 167	105.9	Samples failed axially
GDM2	GT007	SGD158	100.5	Samples failed axially
GDS	GT004, 8	SGD156, 158	99.0	Samples failed axially
MS	GT002	SGD155	57.5	Only sample to fail along structure
Extremely Weathered BG	GT005	SGD157	-	Sample arrived in degraded state. Testing not possible

The previous geotechnical study on the project was a Geotechnical Appraisal report, which was prepared by George Orr and Associates in 2010. Data and information used in the 2010 report included inspection of cores and core photography from exploration boreholes, as well as interpretations made from structural geological information collected during core logging. Observations made in old underground exploration workings were also used in the evaluation. The 2010 report was assessed by Peter O'Bryan & Associates in conjunction with the recent drilling data and UCS testing results, with the conclusion that the pit wall designs presented were considered appropriate for a PFS level study, with adequate design justification for the proposed pit.

The recommendations for base case wall design parameters for the final pit were as follows:

In "weathered" material (surface to 10-20m below topography, local variations in topography and weathered zone thickness exist, inferred pit crest for geotechnical work was 320m RL with weathered material down to ~300m RL)

- 10m vertical height benches
- Bench face angles of 55 degrees
- 5m berms

For the remainder of the pit (~300m RL to ~180m RL)

- 20m vertical height benches
- Bench face angles of 60 degrees on the footwall side, and 70 degrees for the hangingwall and endwall sides
- 8m berms, with 12m berms at 240m RL and 200m RL to mitigate the chance of restriction of mining activities in the case of rockfalls.

These parameters result in overall wall angles of 45 degrees for the footwall, and 52 degrees for the other walls (exclusive of ramps).

Peter O'Bryan & Associates have made a number of recommendations for a more comprehensive geotechnical study which will be required to improve the understanding of the rock mass to the appropriate level of confidence for a full (Definitive) Feasibility Study. This will include further geotechnical drilling, hydrological and hydrogeological studies, rock property testing and a site visit. The full report is provided in Appendix 2.

16. MINING METHODS

16.1 Open Pit Mining Operations

For the purposes of this Preliminary Feasibility Study, it has been assumed that the Segilola project will be operated primarily as a contract mining operation with all mobile equipment to be provided, operated and maintained by a mining contractor(s) over the life of the project. The mine owner would provide key professional staff for management functions including geology, grade control, survey, mine engineering and mine planning. Any contractor(s) would establish, construct and operate the open pits; they may also be singularly responsible for supplying all required infrastructure, water, dust suppression, lighting towers, pumps, etc.

The mining operating cost applied in Whittle is based on quoted contract mining costs obtained from a West African mining contractor. This mining cost is inclusive of drill and blast, load and haul, fuel, all labour and equipment maintenance.

Standard open pit mining methods using excavator and truck fleets are planned to be employed, with equipment numbers adjusted as required to meet ore delivery requirements whilst ensuring stockpiles do not exceed acceptable size.

16.1.1 Mobile Plant Selection

As previously outlined, it is planned that standard open pit mining methods be applied in production operations at the Segilola project, with the assumption that a contract miner is to apply a rear dump mining truck fleet for ore and waste mining, with payload capacity of 65t per truck.

These fleets are to be supported by a standard excavator (backhoe configuration) of around 120t in size, along with auxiliary equipment such as dozers, graders and water carts for dust suppression purposes. It is assumed that specific excavator selection will be undertaken by the mine owner in consultation with the mining contractor, but a general guide to selection (as shown in Figure 89) indicates that a backhoe-type excavator would be most suitable to the operation as it currently stands:

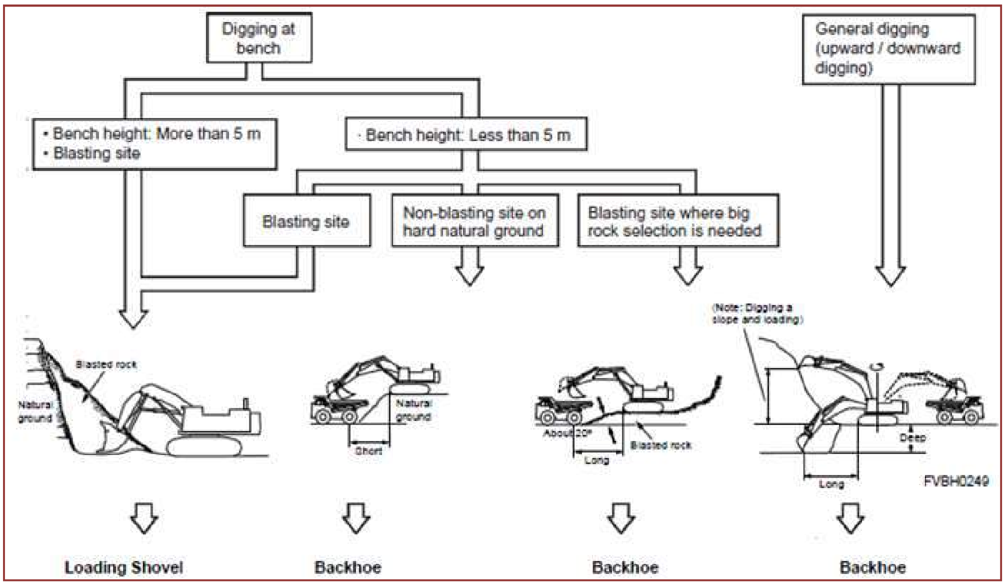


Figure 89 - Excavator Selection Guide (from Komatsu Handbook)

The general process of mining at the Segilola project would therefore be similar to any typical open pit mine utilising backhoe-type excavators; that is to say that once blasting has occurred the excavator would locate itself upon blasted rock while the haulage trucks are loaded on the level below. Standard auxiliary equipment such as dozers and graders will be used where necessary to achieve smooth graded floors, smooth heave and provide face clean-ups.

The Segilola project has been costed based on the application of 65t payload rigid body truck fleets. The size of the pit shells produced in this study suggests that there may be scope to increase the size of the mining equipment in order to reduce the mining operating cost. This should be investigated in future, more detailed studies.

The photograph in Figure 90 shows a typical rear dump truck fleet in operation.

Figure 90 - Typical rigid body truck fleet in Operation

An example of a rear dump truck mining fleet may consist of:

- Excavator 120t Example: Hitachi EX1200 Excavator
- Haul Trucks 65t Example: Caterpillar 775 Rigid Body Truck
- Dozer Example: Caterpillar D10 Dozer
- Grader Example: Caterpillar 16 Grader

The number of trucks required for both fleet types will vary given specific cycle time needs.

16.1.2 Production Blasting

This Preliminary Feasibility Study assumes contractor-supplied production blasting for all material (very little oxidised rock is present). Drill and blast costs have been included in the mining operating cost (refer to 21.4.1 for more detail).

Production blasting costs have been included in the mining operating costs as quoted by the selected mining contractor, and are based on a pumped emulsion bulk product and non-electric initiating systems. It is anticipated that a bulk explosives plant will be established on site to reduce the cost and security concerns associated with transporting bulk explosives to and from site.

The production blasting pattern parameters shown in Table 56 were selected based on the information available on the properties of the rock mass. Further review will be required in the Definitive Feasibility Study based on better understanding of the rock mass and hydrogeology.

Table 56 – Drill and Blast Pattern Parameters

Material	Bench Height (m)	Hole Diameter (mm)	Burden (m)	Spacing (m)	Subdrill (m)	Stemming (m)	Explosive Density (g/cc)	Powder Factor (kg/BCM)
Fresh Ore	5	102	3.0	3.5	1.0	2.0	1.15	0.9
Fresh Waste	5	102	2.7	3.2	1.0	2.0	1.15	0.7

16.1.3 Waste Rock Dump

Preliminary waste rock dump designs have been completed, with sufficient capacity to store all of the waste material from the final pit (~23M BCM). The waste rock dumping strategy includes both in-pit and out-of-pit dumping in order to achieve the required storage capacity whilst minimising the disturbance footprint. A swell factor of ~25% has been taken into account in producing these designs, which are shown in Figure 91.

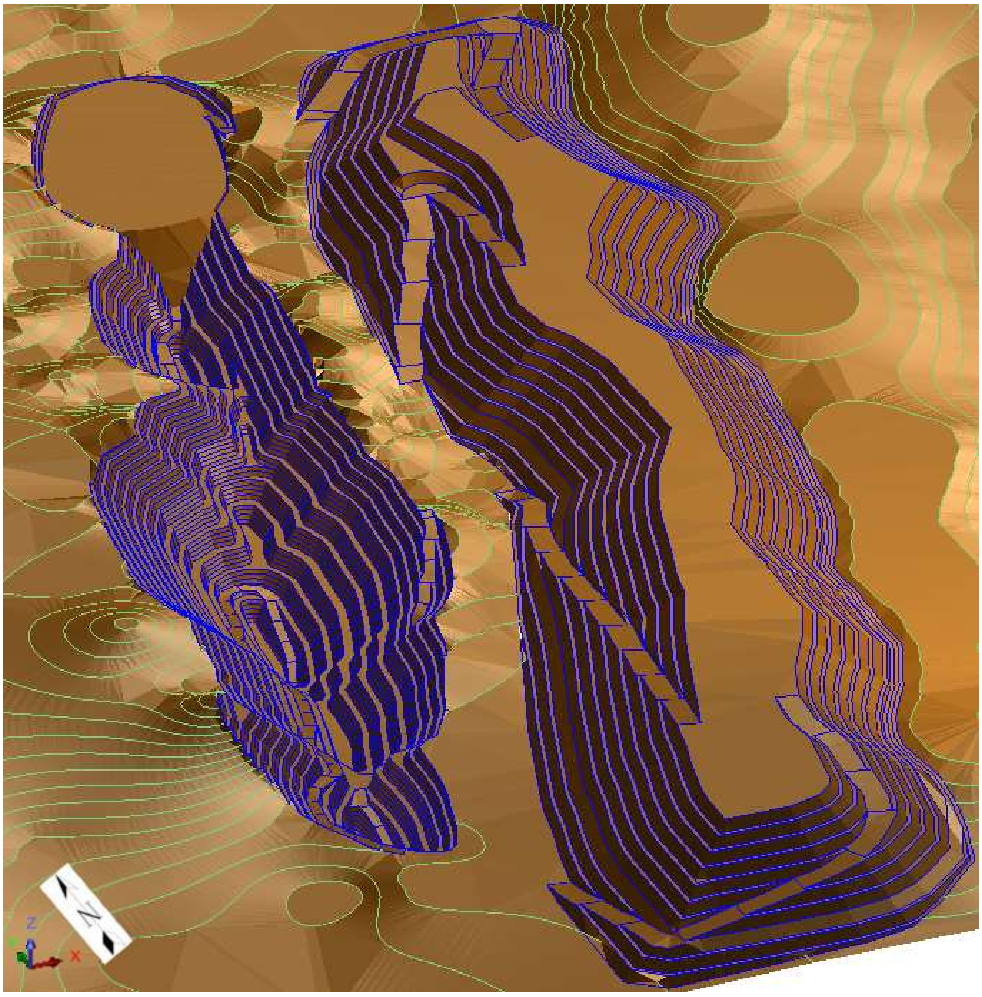


Figure 91 – Waste Rock Dump Designs

The parameters of the out-of-pit waste rock dump design are as follows:

- Total capacity ~27M m^3
- Batter angle 42 degrees
- 5m wide berms every 10m vertical
- Overall slope approximately 35 degrees
- Total height above topographical surface 60-100m (highest level at 415m RL does not exceed the highest point on the surrounding topography)
- Total area ~ 85ha

The in-pit waste dump capacity is approximately 2.3M m^3, which is achieved by filling in the northern section of the open pit.

Construction of the waste dumps will take into account any acid drainage potential, with acid generating rocks to be encapsulated within layers of non-acid generating rocks.

16.1.4 Hydrogeology and Pit Dewatering

A desktop review of Hydrogeology was completed at Thor's request by Peter Clifton & Associates in September 2017. The information presented in this section has been replicated and/or summarised from this report. The full report is available in Appendix 3.

The main objectives of this review were to:

- Provide an overview of the hydrogeology at the proposed mining operations from available information
- Identify aspects of the hydrogeology that could impact groundwater management at the proposed mining operations
- Identify knowledge gaps in the site hydrogeology where additional information is needed to improve confidence in possible methods for groundwater management prior to commencement of mining
- Provide a work scope for additional investigations to progress the project to a Definitive Feasibility Study (DFS)

16.1.4.1 Available Site Hydrogeological Data

Thor collected a set of groundwater levels from some of the exploration boreholes at the project in August 2017. Analysis of these boreholes indicated that groundwater levels in the northern half of the project are flat lying and generally in the range of 332m ASL to 340m ASL, with more variation in groundwater levels from boreholes in the southern half of the project, with groundwater levels in the range of 300m ASL to 330m ASL.

The vertical depth to water below the land surface measured in the boreholes is <5m in the northern half of the project, and up to 60m in the southern half. The cause of this contrast warrants further investigation as it is likely to influence groundwater management during mining operations.

16.1.4.2 Pumping Test Data

Thor conducted a pumping test at borehole SGD166 in August 2017. This borehole is located at the southern end of the conceptual pit crest. The total length of the borehole is 216.5m, and is thus exposed to a large section of the fresh rock sequence. Pumping was conducted for 3 hours at 0.3L/sec (26kL/day), with water levels measured at regular intervals. Recovery water levels were measured for 6 hours after the completion of pumping. The short duration pumping test indicated that the pumping removed a small volume of groundwater from aquifer storage. The magnitude of the drawdowns for the given pumping rate suggest a bulk formation transmissivity of order 1m^2/day, with an equivalent bulk hydraulic conductivity of ore 10^{-7} m/sec. Early recovery of drawdowns was observed, which can indicate that pumping caused displacement of fine materials from some of the fractures intersected by the borehole, or rapid aquifer recharge from nearby surface water for example. Refer to Appendix 3 for graphs of water drawdown and recovery.

16.1.4.3 Hydrogeological Issues for Mining

The hydrogeology of the project site is not well understood and has not been investigated sufficiently to confidently develop groundwater management strategies for the proposed mining operations. At this stage however, the available hydrogeological data are sufficient for the purpose of contributing to a Preliminary Feasibility Study for the proposed mining operations.

Some aspects of the site hydrogeology which are apparent from available information are as follows:

- Groundwater levels are close to the land surface over a large portion of the proposed open pit, and thus dewatering and management of groundwater will be required from the beginning of mining
- Groundwater and surface water across the site are probably closely related. Some groundwater recharge could occur either by means of direct infiltration or seepage from channels. During the dry season, some of the stream flow could be sustained from groundwater seepage into the channels. These processes are not unusual in tropical settings.
- Site surface water management will be important during mining, and surface water will need to the diverted from the crest of the open pit.

- Descriptions of the larger-scale hydrogeology from the "Basement Complex" in south western Nigeria indicate that, in general, water wells in this setting are not noted to be high yielding. While that would be true as a generalisation, the local setting at the project is different because of the presence of NNE and cross cutting structures. The groundwater-yielding characteristics of these structures is not known, and investigations are required to address this deficiency.

- The hydrochemistry of groundwater from the deeper unweathered sequence at the project is unknown. It will be important to develop an understanding of this hydrochemistry as it could affect the approach to groundwater management.

- Shallow groundwater from hand dug wells in nearby villages is relatively fresh, and there is clearly reliance on these shallow groundwater resources as a water supply. It will be important for the mining operations to be aware of all nearby groundwater users prior to site clearing and the start of mining.

Site-scale investigations will be required to improve the hydrogeological knowledge base and progress the project to Definitive Feasibility Study Level. The results of these investigations should be used to develop a mine groundwater management plan. A full list of recommendations is provided in section 26.

Preliminary groundwater management strategies as outlined in the 2012 RBFS completed by previous project owners Ratel Group limited are as follows:

It is anticipated that dewatering will be undertaken via in-pit sumps and horizontal drain holes. Sumps will be established in the hanging wall at the lowest elevation on the bench and all water flows from any aquifer sources will be channelled to the sump. Pumps mounted on pontoons or floaters will be used to pump water out of the pit. Horizontal drain holes will be typically 50 or 100 mm in diameter and may be lined with 25 or 50 mm perforated/slotted PVC pipe to maintain open drillholes and free draining conditions. Mine water will be pumped to settling ponds to be located close to the open pits and will be used for dust suppression in the pit, haul roads, waste dump and plant. Excess mine water will be pumped to the storm water dam for storage and for process plant requirements.

For initial planning purposes, the rainfall data available was used in conjunction with estimated ground water inflows. At this stage, it is estimated that the pit dewatering requirements will vary throughout the year between 3.6 m^3 and 18.6 m^3 per day. A conservative run-off factor of 95% was used and requires confirmation along with the estimated groundwater flow. Both of these will be covered under separate studies prior to commencement, but are not considered to be a major issue.

16.1.5 Roads and Other Mining Infrastructure

Preliminary surface road alignment designs have been completed, linking the pit, waste rock dump, site operations centre and other infrastructure.

Preliminary footprint designs have been completed for the following mining-related site infrastructure:

- Site operations centre, including:
    - Run of Mine (ROM) ore pad,
    - Processing plant and associated facilities
    - Administration, accommodation, security, first aid, ablutions and kitchen buildings
    - Helipad
    - Power generation facility and fuel storage
- Mining contractor compound, including:
    - Administration and ablutions buildings
    - Workshops and stores
    - Equipment parking, lay down area and wash bay
- Explosives magazines compound and bulk explosives plant compound.

An overview of the roads and mining infrastructure preliminary design is shown in Figure 92. Note that these locations are not final and are subject to review in the Definitive Feasibility Study.

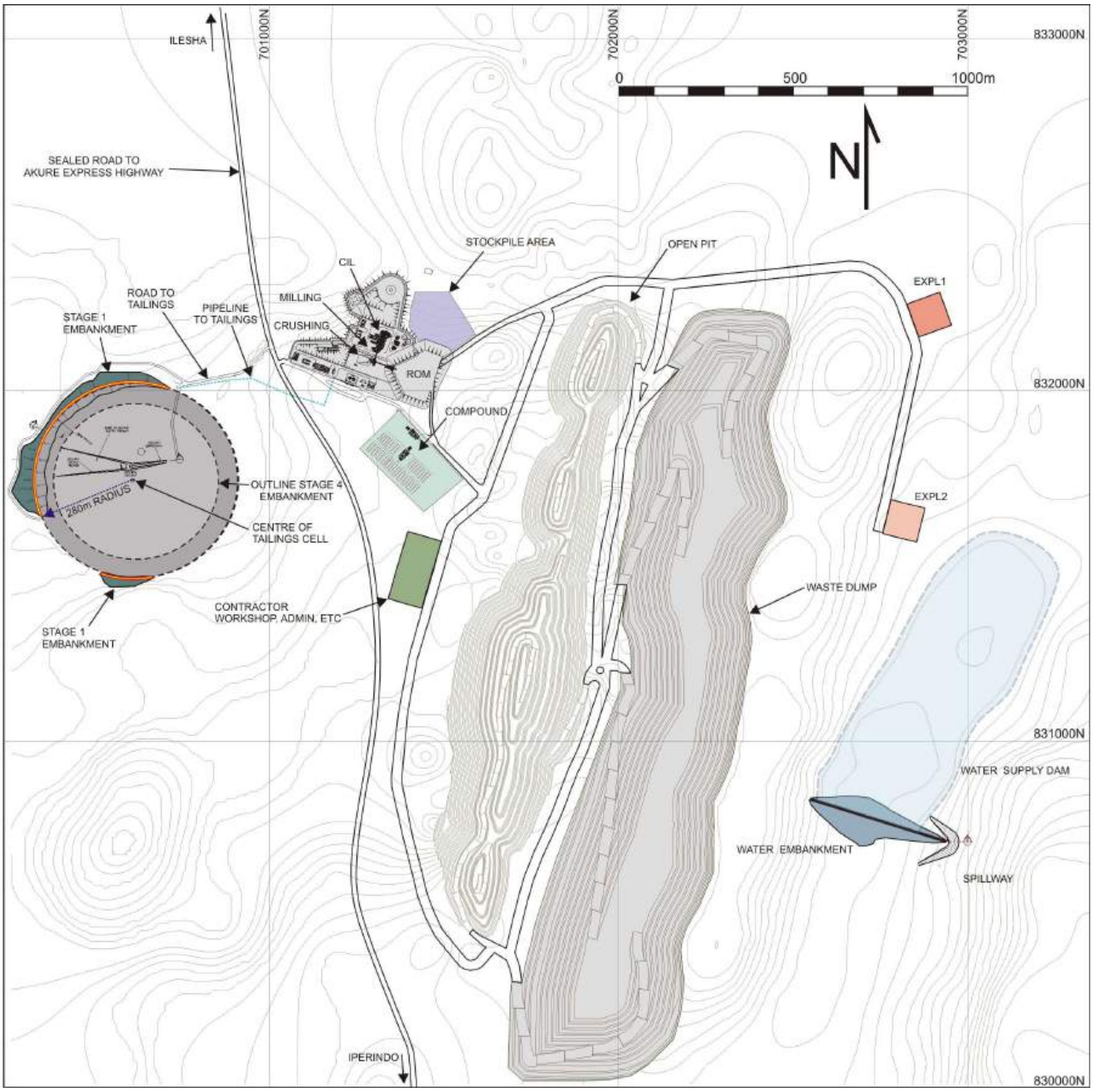


Figure 92 – Overview of Roads and Mining Infrastructure

16.2 Life of Mine Schedule

16.2.1 Interim Stage Pit Designs

To maximise the discounted cash flow (DCF) of the project, it is necessary to mine the pit in a series of stages. This brings forward cash flow as much as possible by preferentially developing the most profitable areas within the final pit shell while delaying mining of bulk waste.

This process needs to be balanced with practicality considerations – both in terms of minimum mining width and scheduling.

The interim stage pits to be developed were selected on this basis, using the order of development of shells in Whittle as a guide (highest return pit shells targeted first). This resulted in two interim pits (Stage 1 and Stage 2), which will be mined before developing the final pit. The Stage 1 and Stage 2 pits were designed according to the same parameters as the final pit, as listed in section 15.3.1. Ramp exit points have been designed on each stage pit to minimise the haulage distances to the waste rock dump and the ROM pad. Ramps exit on the eastern side of the pit, closest to the waste dump as most material removed from the pit is waste. Ore haulage distances have been minimised by designing the ramp exit points at either the northern or southern end of the Stage pits.

The stage and final pit designs are shown in Figure 93 to Figure 95.

- Dark Blue = Stage 1
- Light Blue = Stage 2
- Brown/yellow = Stage 3/Final Pit.

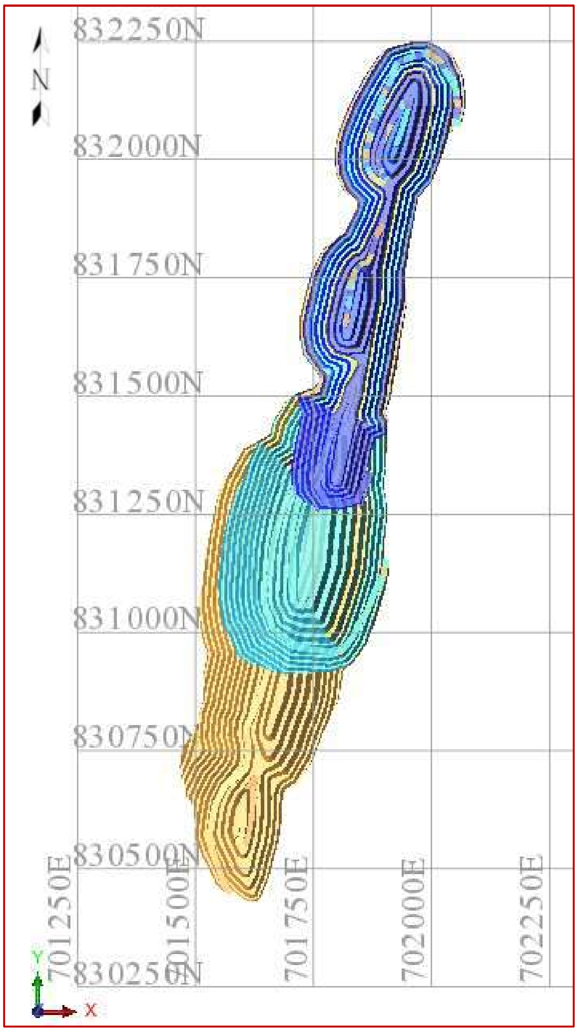


Figure 93 – Stage Pit Designs – Plan View

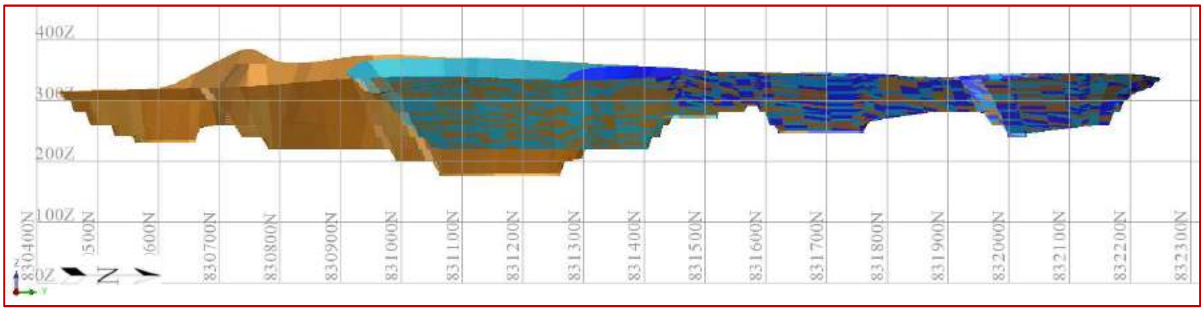


Figure 94 – Stage Pit Designs – Long Section

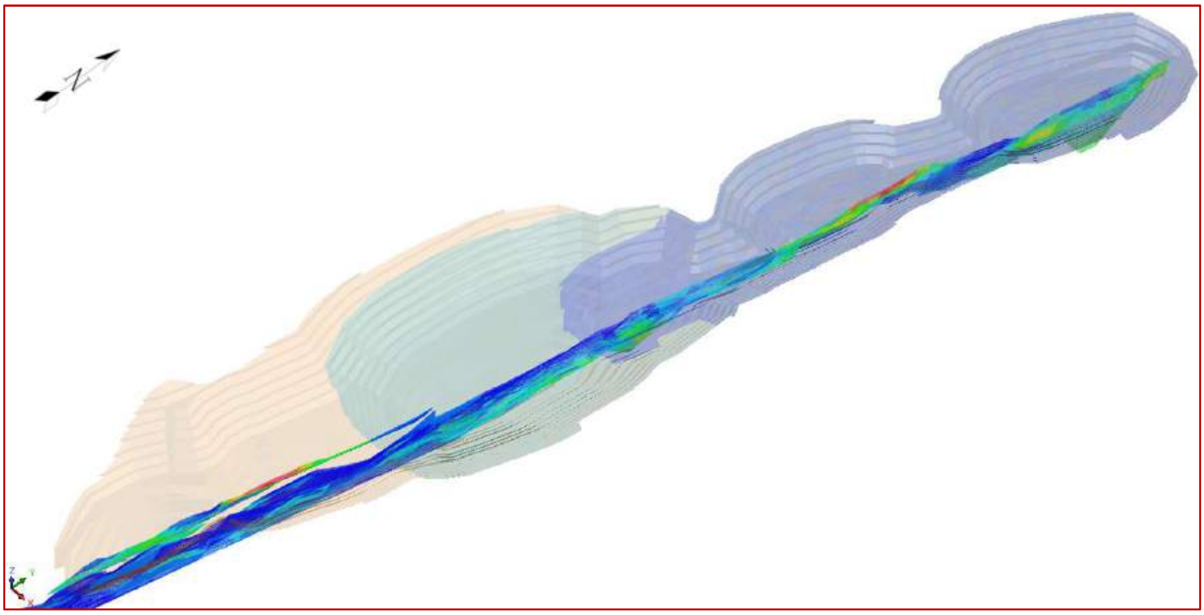


Figure 95 – Stage Pit Designs showing ore >=0.64g/t – 3D View

16.2.2 Mining Production Schedule

The production schedule physicals for mining, processing and ore stockpiles are shown in Table 57.

Note:

- Mining dilution and recovery factors have been applied
- The mining schedule includes all Inferred material contained within the final pit design, however Inferred material is sent to a separate stockpile and not processed or used to generate revenue. Inferred material makes up ~5% of the total ore and has not been used as an economic driver or included in the Mineral Reserve.

Table 57 – Mining Production Schedule Physicals

Mining Physicals		year	0	1	2	3	4	5	6	7	8
INDICATED ORE	kbcm	1,253.225	-	334.153	154.405	279.272	95.436	64.107	167.795	158.056	-
	kt	3,344.9	-	891.0	412.3	745.7	254.7	171.2	448.0	422.0	-
	g/t	4.17	-	4.78	3.75	4.33	4.55	2.91	2.78	4.73	-
	kg	13,936.3	-	4,262.8	1,544.9	3,230.7	1,160.2	497.3	1,245.3	1,995.2	-
INFERRED ORE	kbcm	70.1	-	1.3	9.0	5.7	6.0	17.6	15.9	14.5	-
	kt	187.1	-	3.4	24.1	15.3	15.9	47.0	42.5	38.8	-
	g/t	4.60	-	3.19	3.24	2.43	2.97	5.55	5.28	5.21	-
	kg	861.5	-	10.9	78.2	37.1	47.3	261.2	224.6	202.2	-
WASTE	kbcm	23,250.4	-	9,523.8	3,506.0	2,450.3	2,635.9	2,652.5	1,861.2	620.7	-
	kt	62,015.1	-	25,382.7	9,360.8	6,539.1	7,023.9	7,081.8	4,969.5	1,657.4	-
All Material (Ore+Waste)	kbcm	24,573.8	-	9,859.2	3,669.5	2,735.3	2,737.3	2,734.2	2,044.9	793.3	-
	kt	65,547.1	-	26,277.2	9,797.2	7,300.0	7,294.6	7,300.0	5,460.0	2,118.2	-
Stage 1	kbcm	6,723.8	-	6,412.0	311.8	-	-	-	-	-	-
	kt	17,929.6	-	17,097.2	832.4	-	-	-	-	-	-
Stage 2	kbcm	9,525.7	-	3,447.2	3,357.7	2,389.5	331.3	-	-	-	-
	kt	25,409.4	-	9,180.0	8,964.8	6,380.0	884.6	-	-	-	-
Stage 3	kbcm	8,324.2	-	-	-	345.8	2,406.0	2,734.2	2,044.9	793.3	-
	kt	22,208.2	-	-	-	920.0	6,410.0	7,300.0	5,460.0	2,118.2	-

The mining production profile is shown graphically in Figure 96. To minimise capital requirement there is no pre-strip mining period before the processing plant is commissioned. Production rates in the first nine months are very high to expedite access to high-grade ore, maximising project discounted cash flow and minimising the payback period on capital investment. Up to 8 mining fleets (a fleet consists of one excavator unit and its required number of trucks) will operate simultaneously during this period. Mining in the Stage 2 pit commences well before Stage 1 is complete to minimise the impact of waste mining in the cutback. Production rates step down in months 10 (5 fleets) and 16 (2 fleets) in order to restrict ore stockpile size to manageable levels. Only one mining fleet is required from month 67 until mining is complete after 79 months (~6.5 years).

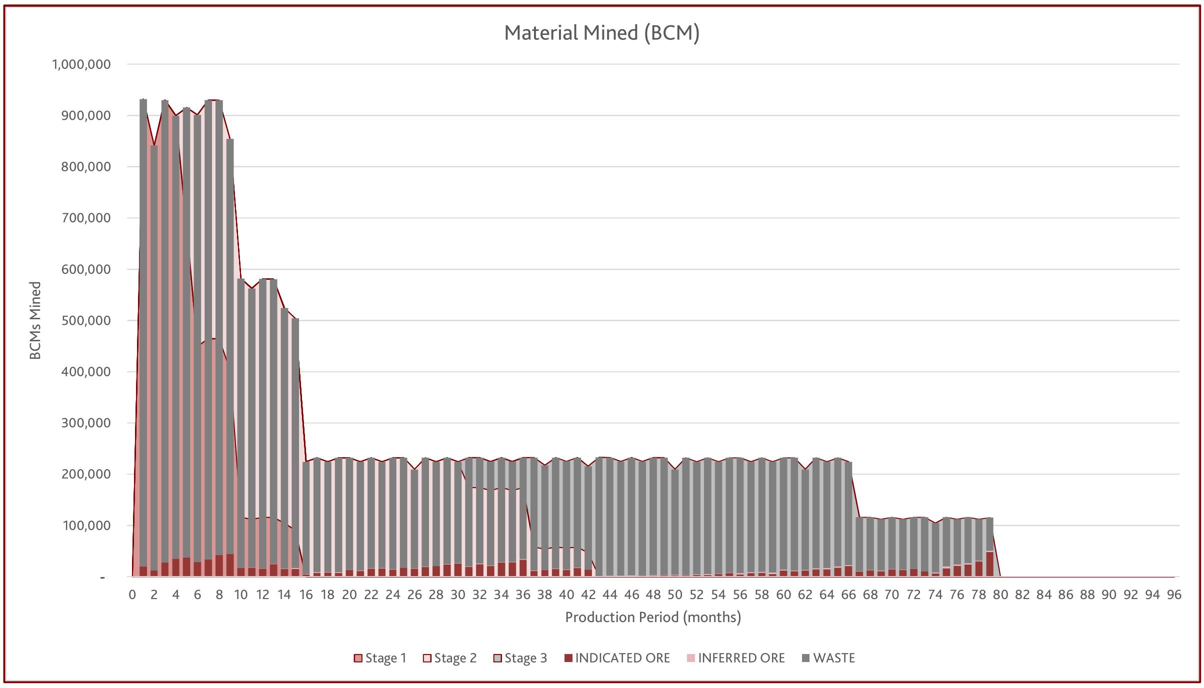


Figure 96 – Mining Production Profile Graph

16.2.3 Ore Stockpiles

Stockpile balance is shown in Table 58, and graphically in Figure 97. Four separate stockpiles have been implemented in the production schedule, with parameters as follows:

- High grade: Indicated material >5g/t
- Medium Grade: Indicated material 2.5g/t to 5g/t
- Low Grade: Indicated material 0.64g/t (economic cut-off grade) to 2.5g/t
- Inferred material (all material above economic cut-off grade)

The use of separate stockpiles allows the grade fed to the plant to be maximised by prioritising feed from each stockpile in order of grade, i.e. no medium grade ore is fed to the plant unless the high grade ore stockpile is empty, and no low grade ore is fed unless the medium and high grade stockpiles are both empty.

There is a negligible amount of mineralised material below the economic cut-off grade and as such there is currently no intention to stockpile "Mineralised Waste".

Table 58 – Stockpile Balance by Year

Stockpile Balance (EOY)		year	0	1	2	3	4	5	6	7	8
Total Stockpile	kt		-	423.3	359.7	620.5	389.8	107.9	148.3	187.1	187.1
Stockpile Grade	g/t		-	2.4	1.7	2.1	1.9	4.1	4.4	4.6	4.6
HG Stockpile	kt		-	-	-	13.3	-	-	-	-	-
	g/t		-	-	-	6.8	-	-	-	-	-
MG Stockpile	kt		-	183.1	-	94.2	-	-	-	-	-
	g/t		-	3.4	-	3.3	-	-	-	-	-
LG Stockpile	kt		-	236.8	332.1	470.2	331.0	2.1	-	-	-
	g/t		-	1.5	1.5	1.6	1.7	1.3	-	-	-
Inferred Stockpile	kt		-	3.4	27.6	42.8	58.8	105.8	148.3	187.1	187.1
	g/t		-	3.2	3.2	2.9	3.0	4.1	4.4	4.6	4.6

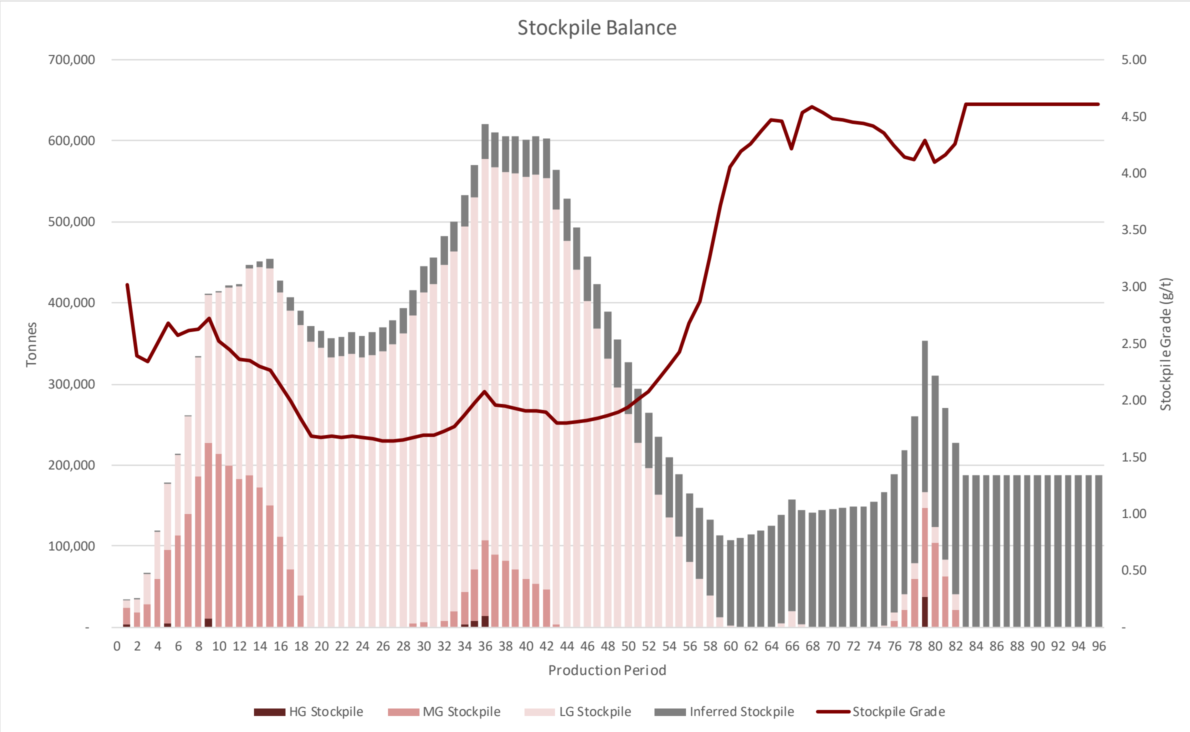


Figure 97 – Ore Stockpile Balance Graph

Maximum stockpile balance over the life of the project is ~620kt, which represents approximately 15 months of ore feed. This is generally higher than recommended, though in this case it is necessary due to the requirement to maximise the project DCF by bringing forward high grade ore feed as much as possible.

The area currently set aside for the ROM pad is approximately 3ha in size. A secondary stockpile area of similar size has been set aside to the north of this, as shown in Figure 92. Preliminary calculations indicate that these two areas combined should be sufficient for the maximum stockpile balance.

16.2.4 Processing Schedule

The ore processing schedule is shown in Table 59 and presented graphically in Figure 98. An initial ramp-up period of 2 months has been allowed for, with 50% production rate in the first month and 75% in the second, before the full production rate of 500ktpa is reached from the start of month 3. The processing schedule is a result of the mining schedule and stockpiling strategy which aim to maximise the project discounted cash flow by processing the highest grade material as early as possible within the bounds of mining practicality. The ore target is met in every month of the project for the first 5 years of the project. Inferred material is mined and stockpiled but not processed and therefore does not contribute to the economics of the project as per NI 43-101 requirements. If Inferred material can be upgraded to Indicated before or during production, this material would fill any shortfall in the current Indicated processing schedule.

Table 59 – Processing Schedule Physicals by Year

Processing Physicals		year	0	1	2	3	4	5	6	7	8
Total Ore Processed	kt	3,344.9	-	471.2	500.0	500.0	501.4	500.0	450.1	422.0	-
Head Grade	g/t	4.17	-	6.95	4.04	5.16	3.53	2.09	2.77	4.73	-
Recovered Ounces	koz	430.1	-	101.0	62.4	79.7	54.7	32.3	38.5	61.6	-
HG Ore Processed	kt	1,211	-	371.30	115.34	272.83	145.21	17.85	66.73	221.93	-
	g/t	7.33	-	7.93	8.21	7.24	6.58	13.11	7.08	6.10	-
	koz	274	-	90.87	29.21	61.01	29.50	7.22	14.57	41.76	-
MG Ore Processed	kt	991	-	99.86	257.59	165.69	155.48	38.06	119.40	154.94	-
	g/t	3.32	-	3.29	3.41	3.09	3.09	2.74	3.52	3.69	-
	koz	102	-	10.13	27.07	15.81	14.84	3.22	12.96	17.66	-
LG Ore Processed	kt	1,143	-	-	127.12	61.52	200.73	444.14	264.01	45.14	-
	g/t	1.54	-	-	1.55	1.52	1.67	1.59	1.35	1.55	-
	koz	54	-	-	6.10	2.90	10.32	21.84	10.99	2.16	-
Inferred Ore Processed	kt	-	-	-	-	-	-	-	-	-	-
	g/t	-	-	-	-	-	-	-	-	-	-
	koz	-	-	-	-	-	-	-	-	-	-

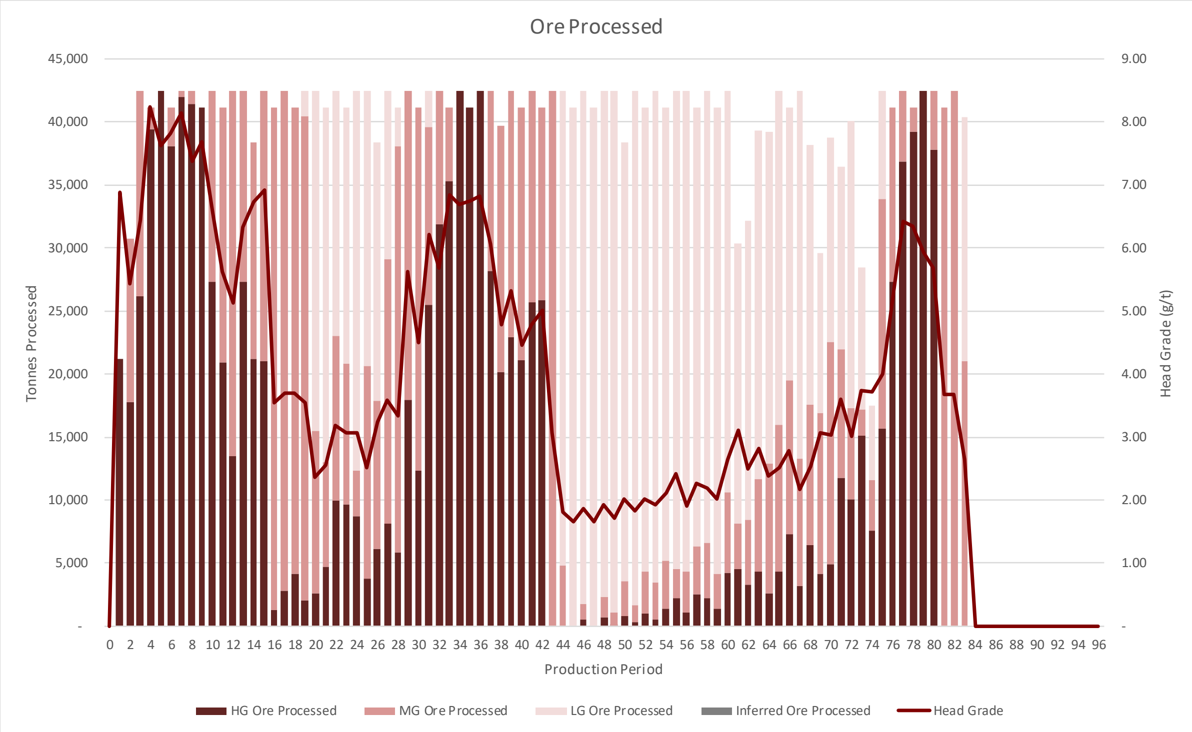


Figure 98 – Ore Processing Schedule Graph

16.2.5 Project Cashflow

Figure 99 shows the pre-tax, undiscounted cashflow generated from the mining and processing schedule. Refer to Section 22 for the detailed economic analysis.

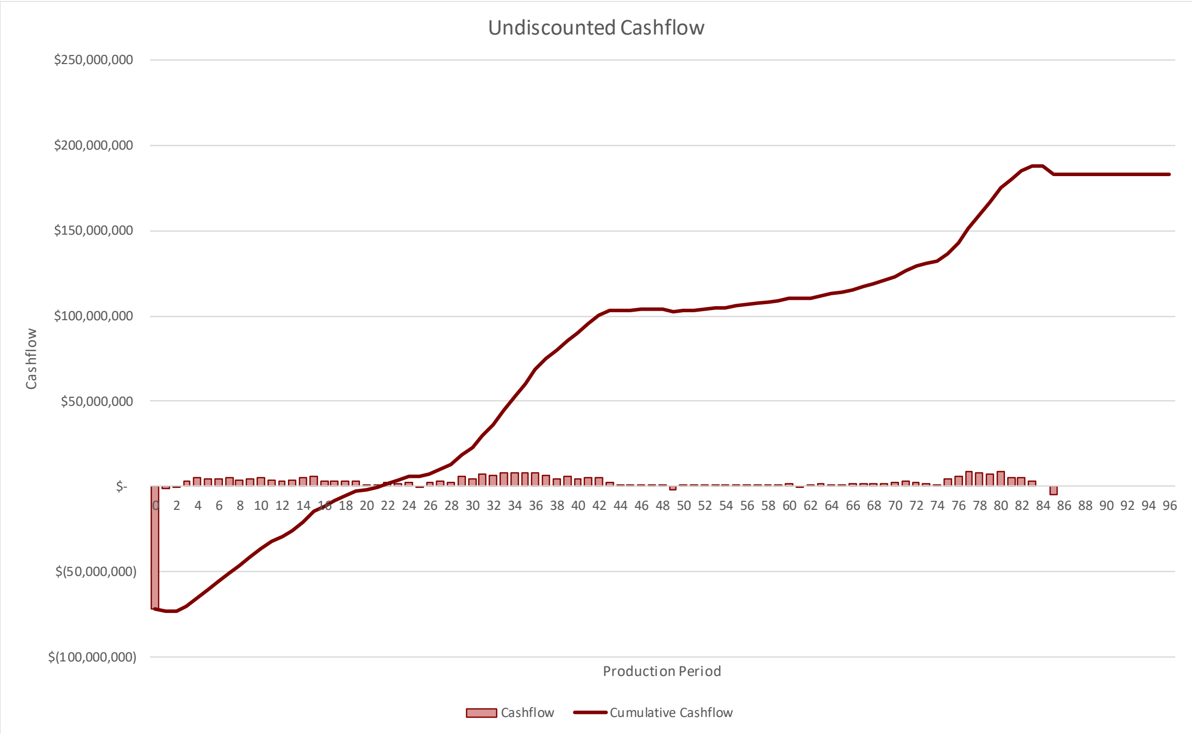


Figure 99 – Project Undiscounted Cash Flow – Pre-Tax

17. RECOVERY METHODS

17.1 Introduction

As part of a Revised Bankable Feasibility Study (RBFS) in 2012, previous project owners Ratel Group Limited conducted metallurgical testing on the Segilola ore, and a comprehensive plant design was completed by Sedgman for CGA Mining Limited. The proposed processing method based on this work is Carbon-in-Leach (CIL). This is a well proven and long-standing technology and comprises comminution, cyanide leaching, carbon adsorption, elution and electrowinning. The proposed plant would include a detoxification circuit using well proven technology to destroy toxic cyanide and its compounds.

Proposed plant throughput is 500,000 tonnes per annum, and test work suggests that a processing recovery of 96% should be achievable.

Sections, 17.2 to 17.10 are a direct reproduction/summary of the corresponding sections from the RBFS, being the most accurate information on recovery methods currently available.

Detailed processing plant design drawings and process flow sheets are available upon request.

17.2 Design Basis

17.2.1 General

The process plant design is based on well-proven Carbon-in-Leach (CIL) technology developed over the past four decades. The process comprises comminution, cyanide leaching, carbon adsorption, elution and electrowinning for the recovery of precious metals. The gold recovery technology is low risk and well understood from a design and materials handling perspective. The success of the process is based on the simplicity of the technology. Likewise, detoxification facility is also based on well-proven technology using sulphur dioxide (in the form of sodium metabisulphite) and air to destroy toxic cyanide and its compounds. The engineering philosophy is to provide a simple but robust design and plant layout, minimising the overall footprint where possible.

17.2.2 Equipment

The equipment required for the process is not complex, relatively inexpensive and readily sourced through a number of reputable suppliers worldwide, including South Africa and Australia.

17.2.3 Standards and Guides

It is intended that the plant will be designed to Nigerian, Australian or South African standards, and will be equivalent to or exceed world's best practice with regard to emission levels. It is intended that the design will comply with the Australian Cyanide Code with regard to the handling of cyanide. Nigerian statutory regulations will be complied with, or in the absence of any relevant regulations reference will be made to Australian and South African Standards.

17.2.4 Safety

Safety will be of paramount importance to both construction and operations personnel and regular HAZAN/HAZOP reviews will be undertaken during the design phase.

17.2.5 Hazardous Goods

There are a number of hazardous goods that are used as part of the process and these include:

- Sodium cyanide.
- Sodium hydroxide (caustic soda).
- Hydrochloric acid.
- Hydrated lime.
- Diesel.
- Sodium Metabisulphite (SMBS).

Particular care will be taken with regard to designs associated with handling of these substances to ensure adequate protection for construction and operating personnel. All designs will be in accordance with recognised international standards.

17.2.6 Future

The processing plant designs include consideration to the possible expansion of the plant by the installation of a ball mill and additional leach tanks if required.

17.2.7 Electrical and Instrumentation

Considering supply issues with the local grid power, electrical power for the facility will be provided by a diesel fired packaged generator plant comprising three generators. Power will be transmitted at 380 V with the exception of the SAG mill which will utilise 6.6 kV.

The plant will be controlled by a distributed control system; however, instrumentation will be limited to where it can be shown there is a real benefit to the plant operations.

17.3 Process Description

The Segilola processing plant will use conventional Carbon-in-Leach (CIL) technology to recover gold and silver from 0.5 Mtpa of ROM ore from an open pit. Design ore feed grade for the processing plant is 4.00 g/t Au and 0.75 g/t Ag.

The general plant design philosophy will be to provide a low cost processing facility whilst maintaining high levels of reliability, operability and maintainability. The following strategies will be employed:

- The plant will operate on a 365 day/year, 24 h/day operating cycle with a design plant availability of 91.3 % for a nominal ore throughput of 62.5 tph.
- The plant will have a moderate level of instrumentation and automation to minimise operator involvement without introducing undue complexity and expense.
- Adherence to well proven and conservative design practice appropriate to the gold industry.

The plant will be based upon the supply and installation of all new processing equipment and will comprise the following unit operations:

- Primary crushing.
- Single stage grinding.
- Leaching.
- Carbon Adsorption.
- Slurry cyanide destruction.
- Tailings pumps and pipeline.
- Acid wash and elution.
- Carbon regeneration.

Gold room refining.

Reagent mixing, storage and distribution.

Water supply, recovery and distribution and air supply.

Key design criteria are shown in Table 60.

Table 60: Processing Plant Key Design Criteria

Key Criteria	Unit	Value
Throughput	tpa	500,000
Availability	%	91.3
Operating hours per annum	h	8,000
Throughput	tph	62.5
Design head grade	g/t Au	4.00
Crusher feed top Size	mm	600
Crusher product size	mm	100
Grind product size (P80)	µm	106
Ball mill work index	kWh/t	18.3
Extraction (Au)	%	97%

17.3.1 Crushing

Fresh ore from the open pit, with a nominal top size of 600 mm, will be delivered to the ROM pad and dumped into stockpiles. A Front End Loader (FEL) will withdraw the ore from the stockpile and dump this into a 100 t live capacity feed bin. The design of the feed bin will not allow direct dumping of ore from the mining fleet. Oversize material will be prevented from entering the ROM Bin by a static grizzly with 600mm aperture.

The crushing circuit consists of a single stage of crushing. Ore will be withdrawn from the ROM bin by a variable speed apron feeder which will discharge directly to the jaw crusher (1,100 mm x 850 mm). This will operate with a closed side setting (CSS) of 90 mm for a nominal product size, P80 of 90 mm. Jaw crusher will discharge directly to the crusher discharge conveyor, which will in turn feed into a 30 t surge bin. The discharge rate from the crusher will be controlled manually by adjusting the speed of the apron feeder to maintain the surge bin at a constant level.

Ore will be withdrawn from the surge bin by the mill belt feeder and discharge to the mill feed conveyor, the discharge rate being measured by a weightometer located on the conveyor, which will control the feed rate to a specified set point by varying the belt feeder speed. An overflow chute will be installed on the surge bin to form an emergency stockpile to allow crusher operations to continue during periods of mill downtime. Overflowing material will report to a conveyor for stacking in the emergency stockpile by FEL.

During periods of crusher downtime, stockpiled material will be re-fed to the surge bin via a FEL using an adjacent constructed ramp.

17.3.2 *Grinding*

The Segilola grinding circuit will be based upon a Single Stage Semi Autogenous (SS SAG) design, having a nominal throughput of 62.5 dry tph and grinding to a nominal cyclone overflow P80 of 106µm. The mill dimensions are 4.88 m diameter by 5.05 m long powered by a 1550 kW motor. Ore discharging from the surge bin will be fed by a conveyor directly to the mill feed chute where it will be mixed with process water and returning cyclone underflow slurry.

SAG mill discharge will pass through a trommel screen to remove any entrained grinding balls and oversized rocks (scats). The screen underflow slurry will report to the cyclone feed hopper where it is mixed with both lime slurry, to modify the pH and sodium cyanide solution to dissolve the gold and silver. Slurry from this hopper will be sized by cyclones, with the design overflow of 43% w/w solids, being directed to the leach circuit, via the trash screen, cyclone underflow return to the SAG mill.

17.3.3 Leach and Adsorption

Gold and silver leaching will take place in a conventional CIL type circuit with a single leach tank (no carbon) and seven adsorption tanks containing activated carbon. Total residence time for the combined leach and adsorption circuit will be 48 hours.

Trash screen undersize will report directly to the leach tank. The leach circuit consists of a single 703 m^3 leach tank having 6 hours residence time. To aid in leaching, gaseous oxygen produced by the PSA plant will be added to the leach tank via the hollow agitator shaft.

The slurry overflowing exiting the leach tank will report to the adsorption stage, consisting of 7 tanks (703 m^3 nominal volume each) having 42 hours residence time. Dissolved gold and silver will be adsorbed by activated carbon contained within the adsorption tanks. Carbon will be transferred in a counter current direction to the main slurry flow, the flow being achieved by interstage recessed impeller pumps and intertank screens.

The intertank screens will be mechanically wiped, and will be of a pumped design, which lifts the discharging slurry above the bulk slurry level of the tank to allow gravity flow to the next adsorption tank in sequence operating at the same slurry level. Use of the pumped screens will allow:

- All CIL tanks to be identically sized, having identical agitators.
- Simpler, un-stepped foundations can be used, reducing capital cost.
- Top of tank steel work will be one level, improving operator safety, simplifying design and overall capital cost.

Flow through the CIL circuit will follow a ‘zig-zag’ pattern to allow the pumped screens to bypass out of service tanks through a ‘Y’ shaped launder.

An average of 2.5 t/d of activated carbon will be transferred to the elution area for gold stripping. Regenerated (or new) carbon will be added into Adsorber 7.

17.3.4 Elution

Slurry containing activated carbon will be pumped from the first adsorption tank, and pass over the loaded carbon screen. The screened slurry will be discharged back to the adsorption tank and the recovered carbon will gravitate to the elution column. The carbon will be initially acid washed with a 3% hydrochloric acid solution, rinsed and then eluted with a solution containing 3.0% sodium cyanide and 3.0% sodium hydroxide at 110°C to desorb the contained gold into the solution.

Seven, two and a half tonne strip cycles will be completed every week, for a total carbon movement of 17.5 tonnes. The elution circuit design is based on the AARL elution technology, using a combined acid wash/elution column. At the completion of the elution sequence, the pregnant solution will report to the electrowinning circuit, the carbon will be transferred to the regeneration kiln.

17.3.5 *Electrowinning*

Elution solution (eluate) containing dissolved precious metals will be passed batch wise through an electrowinning cell to plate out the contained gold. The cell will be a sludging cell, having cathodes consisting of woven stainless steel. The cell will be stripped on a weekly basis where the electroplated gold is washed off the cathodes by high pressure water and collected for smelting as ‘sludge’.

17.3.6 Gold Room Refining

Precious metal recovery and refining will be carried out in a secure building incorporating an access control system, security personnel and video surveillance.

The electrowinning sludge will be mixed with a range of fluxes and smelted at 1,100°C to produce doré. The refining process will convert the metal sludge from the electrowinning cells into doré bars. Doré bars will be stored in a gold room vault prior to transport and refining off site.

17.3.7 Carbon Regeneration

Eluted (stripped) carbon will be educted to the regeneration kiln for regeneration, i.e. the carbon will be heated to above 700˚C in an oxygen free environment for a period of at least 15 minutes to remove any entrained organic material. The carbon will then be quenched in water and sized to remove undersized particles. Regenerated carbon will be returned to the last adsorption tank.

17.3.8 Cyanide Destruction

The Segilola plant will employ a cyanide destruction (detoxification) circuit to treat CIL tailings prior to deposition in the tailings dam. The circuit is based on the use of SO2 (SMBS) / Air process.

The circuit will consist of two detoxification tanks operating in series, each 144 m^3 in volume giving a total residence time of two hours. Both tanks will be mechanically agitated and vigorously aerated. The SO2 supply will be by the addition of a 20 % sodium metabisulphite (Na2S2O5) solution. A 20 % copper sulphate solution will be added to catalyse the reaction. Lime slurry will be added to maintain the pH at 9.0. Low pressure compressed air will be supplied by a dedicated blower and be injected into the tank via sparge nozzles.

The feed slurry will gravitate from the first tank to the second tank, before overflowing to the tailings sump from here it will be pumped to the tailings storage facility.

Test work has confirmed that the detoxification circuit, as designed above has the capacity to reduce cyanide levels in the tailings slurry to < 1ppm.

17.3.9 Services

Raw water will primarily be provided to site from the plant feed water dam located to the east of the processing plant. A transfer pump will deliver the water from the dam to a raw water tank located at the processing plant.

The raw water tank will supply the plant raw water circuit, gland seal water system, potable water treatment plant and the fire water system. The tank will also overflow to the process water tank to maintain process water volume.

Water decanted from the TSF will be returned to the process water tank via a submersible tailings decant pump for re-use in the process water system. Similarly, water overflowing the leach feed thickener will be collected for re-use in the process water system.

Plant and instrument air will be delivered from the same compressor. Plant air will be delivered directly from the compressor and instrument air will be filtered and dried prior to being delivered to the instrument air circuits.

A plant spillage pond of 2,000 m^3 nominal volume will be located to the North of the processing plant. Major spillage event and site run off will be diverted to this dam by the use of site contouring and drainage channels. A submersible pump will be installed within the dam to return the captured material to the plant.

17.3.10 Layout

The processing plant will be located to the east of the Odo Ijesha – Iperindo Road and to the northwest of the open pit operations. A vegetation barrier of approximately 100 m will be maintained between the site operations and the road to minimise the visual impact of the operation.

The processing plant will be constructed over a number of terraced levels, with each terrace containing a number of processing areas. Preliminary terrace groupings will be:

- ROM Pad and Crushing.
- Grinding, CIL, Detox and Gold Room.
- Reagents, Warehouse and Workshop.

- Powerhouse.
- Administration Offices.
- Assay Laboratory.

The ROM pad and Crusher will be located at the southern end of the processing plant.

The main plant will be built on a single level. The Gold Room will be located adjacent to the mill to facilitate the installation of a gravity circuit at a later stage if required. Reagent mixing will be located to the east of the CIL circuit with an access road separating the two. The warehouse, workshop and reagent storage facility will be located to the west of the CIL circuit.

The Administration Offices and associated car park will be located to the southwest of the CIL circuit. These will consist of a number of offices being of cement block construction. Site security services will be located at these offices to control site access. The offices will be connected to the adjacent public road by the provision of a site access road. The access road will service the process plant, fuel storage and the mining area.

Additional site facilities including combined change room / ablution block, warehouse / stores and a kitchen will be located adjacent to the administration offices. Short term, 12 bed, demountable accommodation will also be provided adjacent to the administration facility for temporary accommodations within the work site.

An assay laboratory will be provided on the south-western side of the processing plant. This will be sourced as part of a laboratory assay contract and is anticipated to primarily use fire assay methodology. This laboratory will perform both mining grade control analysis and daily metallurgical performance analysis.

A helipad will be installed on a separate terrace to the north of the main processing facility. This will allow for good helicopter approach and departure routes via the natural formed valley bordering the northern end of the process facility.

Site water will be supplied from a fresh water storage facility established by damming of the creek to the east of the processing plant.

Tailings will be disposed in a purpose built Tailings Storage Facility (TSF) located to the west of the processing plant on the western side of the Odo Ijesha – Iperindo Road. Slurry delivery and decant water piping (buried) connecting the tailings dam and the processing plant will cross under the Odo Ijesha – Iperindo Road.

Fencing to control access to the processing site will vary according to the perceived security and safety risks present. The fencing philosophy will be:

- Illuminated, double row fencing surrounding the processing plant.
- Single stock fencing around the perimeter of the TSF.

- Single barb wire topped fencing encompassing the mining offices and workshop.

The mining pit itself is not planned to be fenced initially, however the pit will be bordered by an earthen bund which will be regularly patrolled by site security personnel.

17.4 Tailings Disposal

17.4.1 Geotechnical Characteristics

The ore feed to the plant will be competent rock with only minimal weathering. The tailings will be relatively coarse and will contain minimal fines. Settling testing on the tailings has shown that the solids will settle quickly and that a high initial insitu density will result. This will mean that the void ratio of the solids will be low and that a high percentage of the water with the tailings will be available for release to the decant pond and return to the process water tank.

The coarse nature of the tailings will also mean that the insitu strength of the material will be high. Upstream construction of the peripheral embankment, where the tailings beach provides part of the foundations for each lift, will be possible.

17.4.2 Geochemical Characteristics

The low sulphide content of the ore, and hence the tailings, will mean that there will be a low possibility of the tailings oxidising to produce low pH conditions.

The gold extraction process uses a variety of chemicals including cyanide. The detoxification circuit will reduce cyanide and metals to low levels (< 5 ppm) before the tailings are pumped to the TSF. Water balance calculations show a negative result with no requirement to discharge water from the TSF. Bird and animal life will not be endangered.

17.4.3 Project Environment

The site selected for the TSF lies to the west of the main public road, which provides access to the project.

The site is heavily timbered, the topography is steep in places however there is a shallow north facing valley which will provide the initial storage capacity behind a Stage 1 embankment.

17.4.4 Tailings Management

The tailings will be pumped to the TSF as a slurry. The tailings discharge will take place from the embankment using multiple spigots. The tailings solids will settle out of the slurry releasing water, which will flow to the centrally located, decant tower.

The majority of the tailings will be discharged above water using a technique known as sub aerial discharge. The active discharge spigots will be changed on a regular basis to ensure an even build-up of tailings around the whole area of the storage.

The water released from the tailings will flow over the beaches to the central decant for return to the plant.

17.4.5 Design of the Tailings Storage

The TSF will be circular with an area of around 25 ha. The initial storage will be provided by Stage 1 embankments, which will have a crest height of RL 330 with a length of approximately 700 m. The maximum embankment height will be approximately 10 m high on the northwestern side of the storage. There will also be a small length of embankment on the southern side of the storage.

The Stage 1 embankments will provide adequate capacity of approximately 1.5 Mt or three (3) years of plant production. At the end of that period, the rate of rise of the tailings in the storage will be less than 2 m/a.

The Stage 1 embankments will be constructed from a combination of material borrowed from within the storage basin and waste rock hauled from the open pit pre-strip. The total volume of fill required will be in the region of 275,000 m^3.

The Stage 1 embankment section will have a crest width of 6 m. The downstream face slope will be 1:3 (vertical:horizontal) which will allow rehabilitation with a layer of topsoil and local grasses. The upstream slope will be 1:1.5. A key trench will be excavated beneath the upstream zone of the embankment to limit seepage from the storage.

The decant causeway will be constructed from run-of-mine waste rock.

The ongoing construction of the peripheral embankment will be by a combination of excavated tailings and mine waste rock. Lifts of 2 m will be used to optimise the volume of fill required to increase the storage capacity by approximately 12 months production.

TSF designs completed by DE Cooper & Associates as part of the Revised Bankable Feasibility Study for previous project owners Ratel Group Limited are shown in Figure 100 and Figure 101.

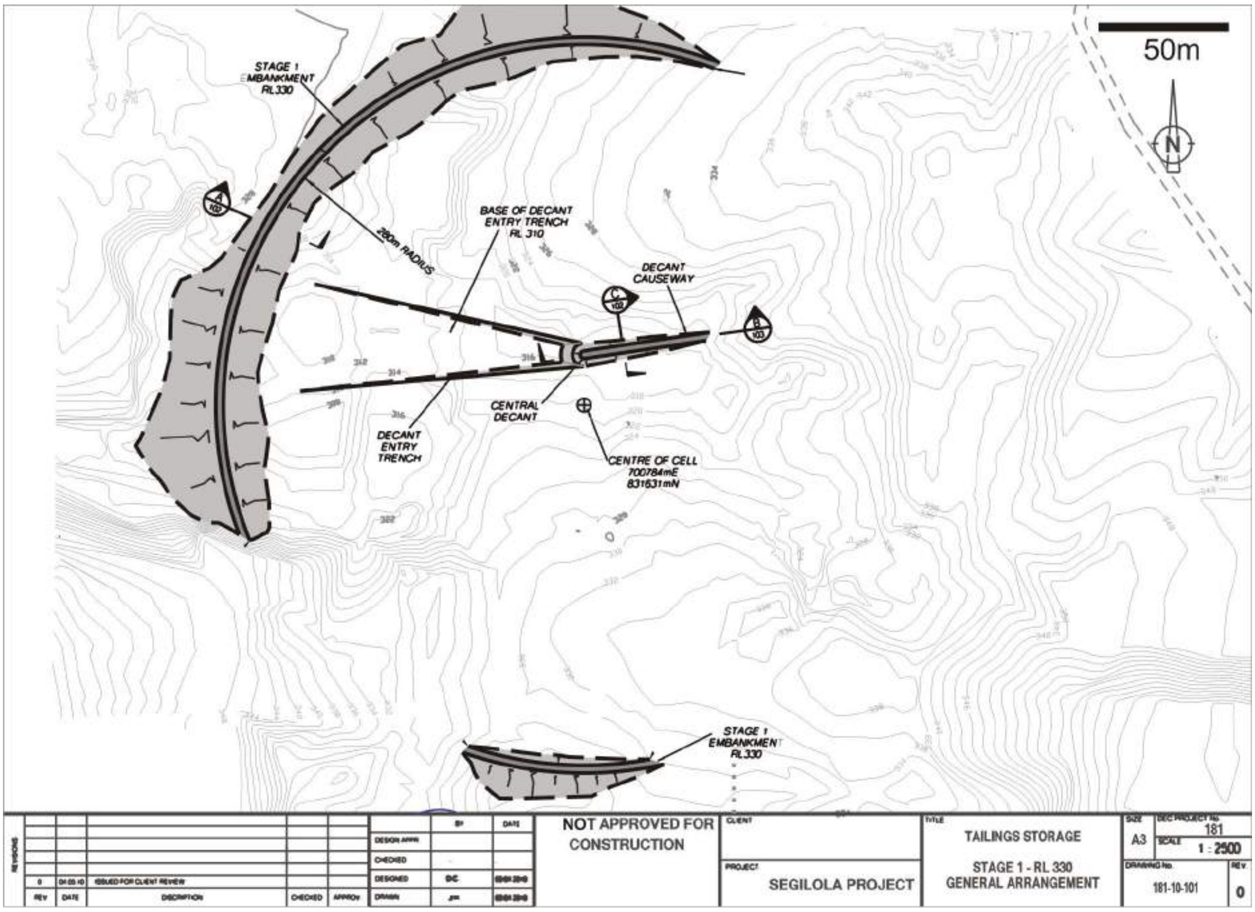


Figure 100 – Tailings Dam Facility Design Layout

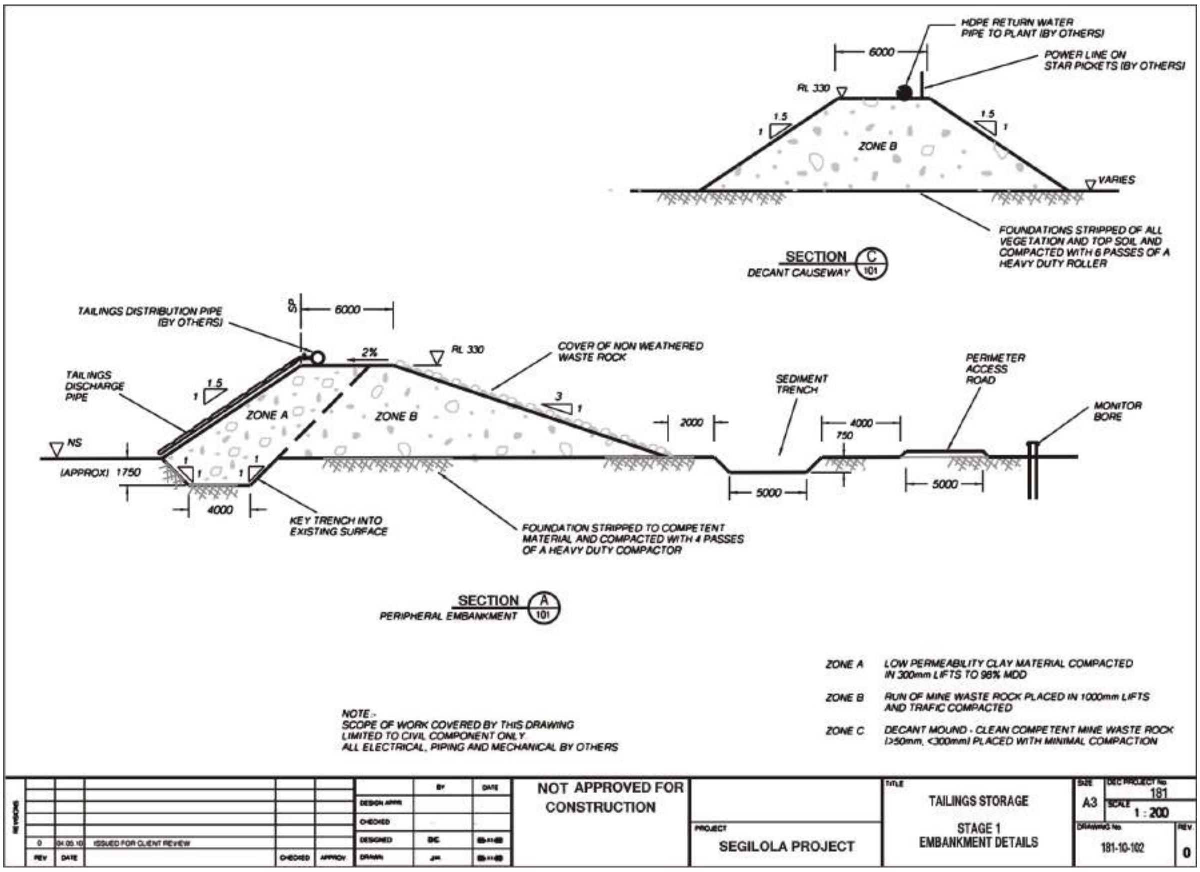


Figure 101 – Tailings Dam Facility Design Profile

17.4.6 Water Management

The water which is released from the settling tailings and incident rainfall will flow to the central decant. The decant will comprise a tower constructed from large diameter (1.8 m) slotted reinforced concrete pipe stacked vertically and surrounded by a large mound of clean waste rock. A submersible pump will be suspended in the tower and will pump the water back to the plant for re-use.

The size of the decant pond will be minimised to reduce evaporation and seepage losses. Return water from the TSF will at all times be used preferentially over make-up water taken from the water supply dam. In this way the freeboard conditions in the TSF will be maintained throughout the highest rainfall time of the year.

17.4.7 Environmental Considerations

The tailings storage facility (TSF) will have a low impact on the surrounding environment. The tailings solids, and the water which will be pumped to the storage as a slurry, will be treated in a detoxification circuit prior to leaving the plant. The tailings and the water will be benign and will not endanger bird life. The storage will be fenced to prevent wandering animals and local village people from gaining access.

The management of the storage, where the majority of the tailings in the storage are unsaturated and the decant pond maintained at minimum levels, will ensure that seepage losses from the storage into the foundations will be minor and will not impact on the scarce local groundwater.

The TSF will be closely monitored throughout the life of the project. There will be piezometers in the peripheral embankment to detect any movement of water. Bores around the outside of the TSF will be used to monitor groundwater levels and quality. Ongoing records will be maintained and open for inspection.

Annual inspections from an external consultant will be made to confirm that the storage is in compliance with the lease conditions and that the embankments are stable. Because of the importance of the storage to the overall project, the level of management will be as high as for any other section of the project.

17.5 Sampling

Samples will to be manually taken by the area operators with no automated systems being included in the design. Suitable access points will be included in the design of the processing plant at strategic locations to facilitate manual sampling.

17.6 Raw Water

Water will primarily be provided to site from the plant feed water dam located to the east of the processing plant. A raw water transfer pump will supply the water from the dam to a raw water tank located at the processing plant.

The raw water tank will supply the plant raw water circuit, gland seal water system, potable water treatment plant and the fire water system. A facility will also be provided to overflow the raw water tank to the process water tank to maintain process water volume.

17.7 Potable Water

Potable water will be used at Segilola primarily for human consumption and to supply the processing plant safety shower circuit. Water will be withdrawn from the raw water supply to feed the potable water treatment plant. The raw water will be upgraded to potable water standards within the treatment plant.

The potable water product will be stored in a dedicated 150 m^3 covered storage tank prior to use. The potable water will be distributed throughout the site by the potable water pump. To maintain limited supply to the safety shower circuit in the event of the potable water pump failure, the circuit shall be fed from a dedicated header tank that will supply the circuit by gravity.

17.8 Process Water

Primary sources for process water at Segilola will be decanted from the TSF and raw water make up the balance. Process water will be stored within a dedicated 500 m^3 process water tank and will be distributed around the processing plant by a dedicated process water pump.

Due to evaporation losses in the TSF, the process water system will be supplemented from time to time, this will be obtained from the raw water system.

17.9 Fire Water

Fire water will be distributed around the Segilola site via a closed circuit, buried ring main with take-off points for both hydrants and hose reels being strategically located within the processing plant.

A set of fire water pumps will be installed consisting of a duty electric fire pump, a diesel powered standby pump and multistage jockey pump. The jockey pump will operate on a continuous basis to maintain the ring main pressure during normal operations. In the event of a hydrant or hose being activated, a pressure switch will sense a drop in the circuit pressure and start the duty fire pump. In the event of a power failure the diesel pump will start automatically.

Fire water will be sourced from the raw water tank. Suctions for the fire water pumps will be positioned lower in the tank wall than either the gland water or raw water pump suctions such that 30 minutes of fire reserve will be maintained at all times.

17.10 Reagents

A range of reagents will be used at Segilola. The major reagents are:

- Hydrated Lime.
- Sodium Cyanide.
- Sodium Hydroxide.
- Hydrochloric Acid.
- Sodium Metabisulphite (SMBS).
- Copper Sulphate.
- Oxygen.
- Diesel Fuel.

Sodium Cyanide will be delivered to site in one tonne bulka bags and will be stored in a secure facility until their use. The sodium cyanide will be made into a 20 % solution in two tonne batches, by mixing with raw water in the cyanide mixing tank. The mixed solution will be transferred to the cyanide storage tank from where it will be circulated to the processing plant via a dedicated ring main.

Hydrated lime will be delivered to site in one tonne bulk bags. The bags will be hoisted batch wise on to the bag breaker and discharged into the lime mixing tank where the powder will be mixed with water to form a 20% w/w solids slurry. The mixed lime slurry batch will be transferred from the mixing tank to the storage tank and pumped to the process via a dedicated ring main.

Sodium hydroxide will be delivered to site in a pearl form in either 25 kg or 1 tonne bags. Sodium hydroxide solution will be mixed in 2 tonne batches to form a 50 % w/w solution. This will be transferred from the mixing tank to the storage tank by a dedicated pump and then metered to the elution and electrowinning circuits.

Hydrochloric acid will be delivered to site as a 32 % solution in 1,000 litre bulk containers. The acid will be transferred to the storage tank by decanting from the bulk containers. The reagent will be used neat within the acid washing process and will be metered using dedicated pumps.

Sodium metabisulphite will be delivered to site in bulka bags. This will be mixed with raw water in the mixing tank to form a 20 % w/w solution. Once mixed, it will be transferred to the storage tank from where it will be metered into the detoxification process.

Copper sulphate will be delivered as crystals in 1 tonne bulk bags. This will be added in 1 tonne batches to raw water in the mix tank to make a 20 % w/w solution. Mixed copper sulphate solution will be transferred to the storage tank where it will be metered to the detoxification process.

Oxygen will be produced on site by the installation of a PSA oxygen generation facility. This will have the capacity to generate 700 kg/day of oxygen at a purity of 92%. Capability will be provided to deliver the produced oxygen to the leach tank and the first three adsorption tanks via sparging down the agitator shafts.

Diesel fuel will be delivered to site in a road tanker. A single bulk storage facility will be provided on site containing 1,000 m^3 capacity which represents approximately 4 weeks' supply. Diesel will be reticulated to two day tanks (one mining and one processing) by service truck deliveries.

18. PROJECT INFRASTRUCTURE

18.1 Existing Infrastructure and Services

The proposed mining area contains only minimal service roads created to provide access around the site. Local infrastructure includes sealed roads and several towns and villages, including the main highway from Lagos and the towns of Ilesha and Iperindo. These roads and towns will provide non-mining specific services and staff for the project.

18.2 Site Development

Ground works will be undertaken using a staged approach by contractors specific to the area required. Initial work will be required to provide access, followed by the processing and administration facilities and ultimately mining areas (including waste dump areas). Capital costs for site establishment are shown in Section 21.1.

18.3 Buildings

Most buildings will be constructed within, or in close proximity to the security fenced processing area compound. Buildings within the compound will include the security post, main administration building, first aid room, short term accommodation, kitchen and dining hall, plant workshop and warehouse, laboratory, reagent store, diesel storage, powerhouse and change rooms. The mining office, mobile equipment workshop and further change rooms will be built in the mine compound.

18.4 Power Supply

Electrical power will be generated on site by the use of diesel powered generators. A total of three 1.6 MW generating sets will be installed and operated on a two duty, one standby basis.

18.5 Water Supply

Process water will primarily be recycled from the tailings storage facility, any shortfall in water supply will be supplemented by water sourced from a proposed dam to be constructed the east of the mining area.

Water from the dam will also be used for potable water (after treatment), dust suppression and firefighting.

18.6 Mining Contractors Infrastructure

Mining contractors will provide their own buildings and infrastructure. An area within the mine compound will be allocated to the mining contractor. Thor will provide power, water and communications to the mining contractors.

18.7 Communications

Primary communications on site will utilise the existing mobile telephone networks which currently service the area surrounding Segilola.

External data communications will be via a satellite link, downloading primarily to the head office. The satellite system will link into a site PABX which services the administration offices to provide limited voice capability.

18.8 Accommodation

Varying accommodation options will be used throughout the construction and operation of this Segilola project. Temporary and permanent on-site camps as well as units or houses in Ilesha will be used to house Expat or senior local staff. The remaining workforce will primarily be sourced from the local area and be required to arrange their own accommodation

18.9 Security

The main administration and processing area will be contained within a patrolled security fence. Security patrols will be conducted throughout the unfenced areas of the project site.

18.10 Roads

Access to the site will be from Odo Ijesha – Iperindo Road via an unsealed access road that will service both the processing plant and mining compounds.

The road will run along the southern border of the processing plant compound and terminate at the main entry gate to the mining compound. Site roads and haul roads will be designed and constructed as required. An overview of the proposed site layout showing preliminary road alignments is shown in Figure 102.

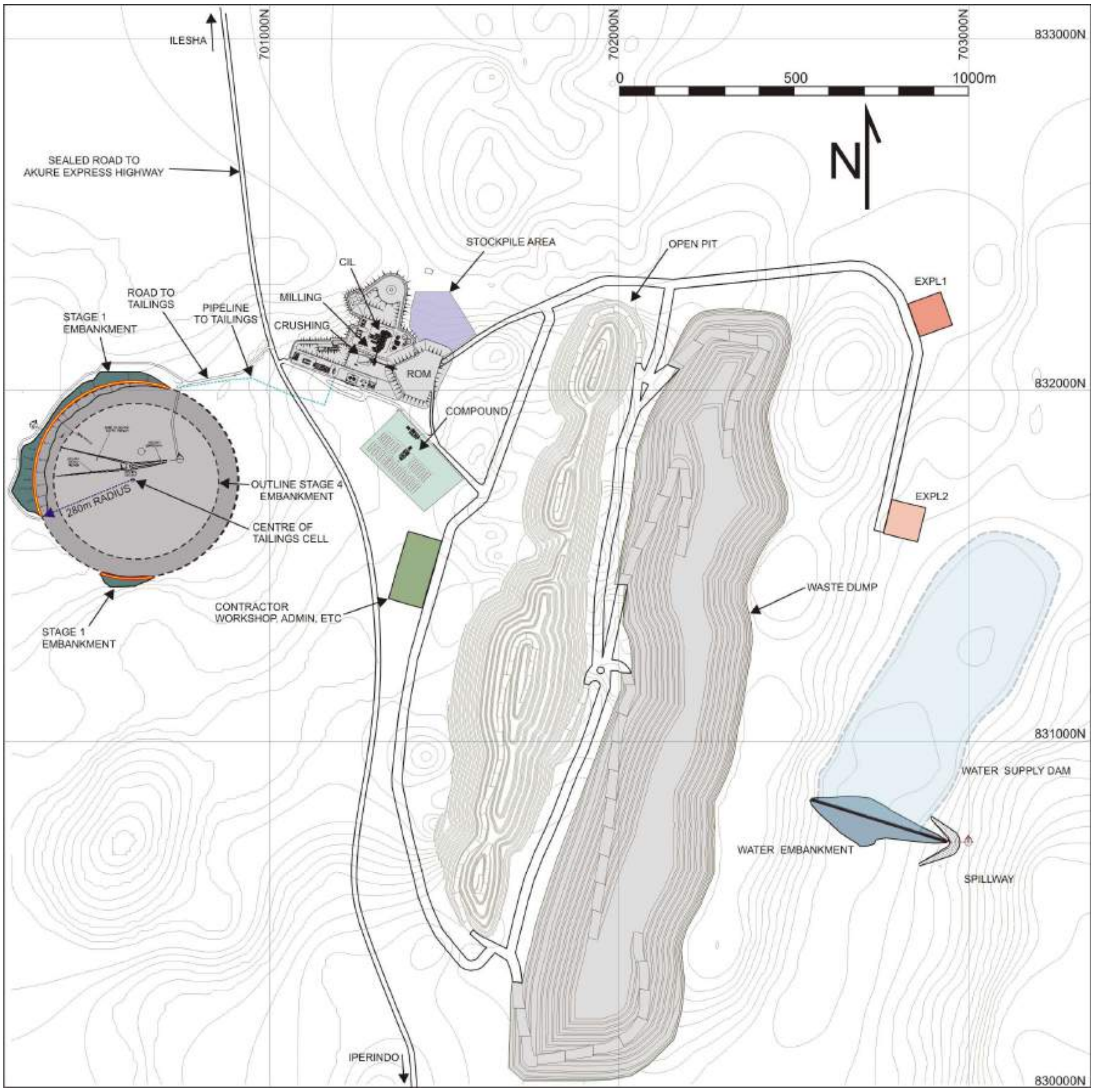


Figure 102 – Site Layout showing preliminary road alignments

19. MARKET STUDIES AND CONTRACTS

Gold Doré will be produced on site and transported to the nearest gold refining facility. All gold will be sold at spot price in US Dollars, there are no forward sales or hedges currently in place.

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL IMPACT

20.1 Environment Context

An EIA prepared between 2008 and 2012 and given approval in 2013 (Certificate issued 13 March 2013) included detailed environmental and social baselines for the mining license area (ML41). These surveys provide comprehensive environment and social data for the project site and for the 3 communities (Odo Ijesha and Imogbara villages and Iperindo town) surrounding the proposed mine development area. No development, except for drilling programs, have occurred on the proposed mine site area and recent surveys indicate that baseline information gathered for the EIA is still robust.

In light of the EIA consent the Ministry of Mine and Steel Development (MMSD) State Plan designates mining as the primary land use for the Project area.

The main impacts identified and analysed in the EIA are outlined in Figure 103.

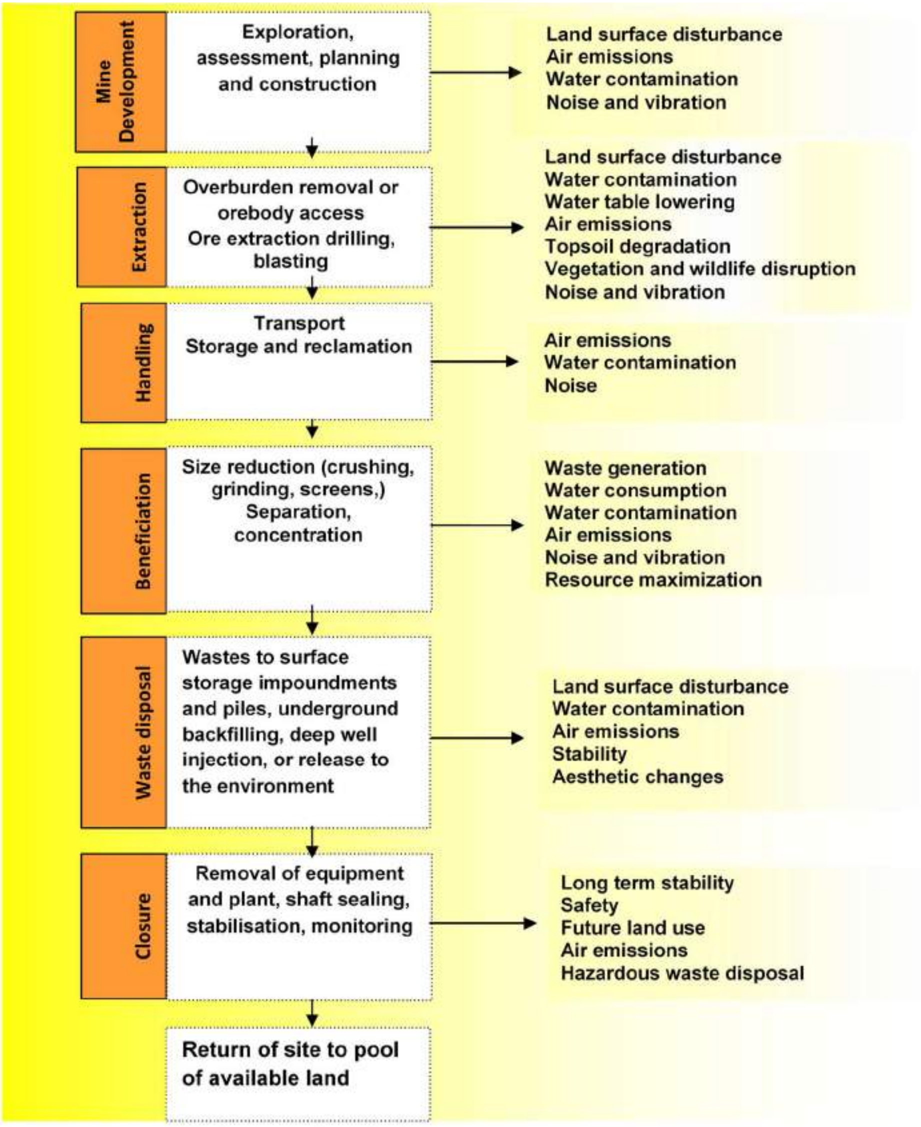


Figure 103 - Environment Impacts for each Phase of Mine Project

The EIA Certificate issued by the Federal Ministry of Environment approved the gold mine project within ML41 and included conditions to compile an Environmental Management Plan (EMP), an Environment Protection and Restoration Plan (EPRP) and Community Development Agreements – these are required to be completed prior to operations commencing on site. These documents (and others to manage environment and social impacts) are either completed or currently being prepared – refer to 20.2.

20.2 Baseline studies

As part of baseline studies undertaken for the preparation of the EIA, primary data was obtained on a comprehensive range of environmental parameters including flora, fauna, aquatic ecology, surface and ground water quality, air quality, soils, noise as well as socio-economic parameters. The natural environment data gathered within ML41 indicated that the area had been impacted by human development over many years. Vegetation surveys indicated that the area comprised mainly cash crop plantations together with secondary forests and bush fallows. Human impact has consequently reduced biodiversity but had not impacted adversely on water quality or air quality parameters.

Surface water quality testing completed in 2008 is outlined in Table 61. This is followed by surface water quality testing completed in August 2017 (Table 62).

Although not all surface water quality parameters from the 2008 EIA baseline survey are currently being tested a comparison of the two survey results indicate surface water quality remains consistent over a 10-year period. Only total iron exceeds in both instances and this is caused naturally by contact with rocks and minerals in the area.

It was concluded in the EIA that the physico-chemical and biological characteristics of surface water and sediment samples of the project area showed consistency with natural background levels for a freshwater environment. Aquatic surveys of zooplankton and phytoplankton species also undertaken for the EIA also concluded that species levels were low and typical of natural shallow freshwater system (streams) of the area.

Other baseline studies under taken for the EIA indicated the following:

- The physico-chemical parameters of groundwater from the project area had concentrations within WHO limits for groundwater while only microbial characteristics exceeded the WHO limit. This was generated by human development in villages surround the mine location.
- Air quality pollutants namely CO, SO2, NOx, Chi, H2S and SPM were within their natural background levels.
- Noise levels were between 32.2 – 46.2 dB(A) which reflect the rural nature of the mine site area.
- The physico-chemical characteristics of the soils showed consistency with natural background levels for agronomic soils.
- The vegetation type is characterised by rainforest on the hillsides and valleys. The predominant trees are umbrella trees, cam wood, Acassia sp and oil-palm trees. There are several cocoa and cola nut plantations interspaced with orange, mango, banana, plantain, pineapple and other trees.
- Wildlife of the project area was dominated with grass cutter, antelope, bush-buck, Sitatunga, civet cats, genets and bush pigs. Hunting for bush meat occurring over many years has also impacted low biodiversity within the mining lease area.

Table 61 - Summary of Surface Water Samples Physico-chemical Characteristics (EIA 2008)

Parameter		Range	Mean	*Natural Levels
pH		6.27 – 7.44	6.68	6.50 – 7.40
Temperature (°C)		26.2 – 28.8	27.4	25.0 – 29.0
Turbidity (NTU)		9.00 – 41.0	22.3	1 – 1000
Salinity (ppt)		<0.10	<0.10	–
Redox Potential (mV)		11.0 – 71.0	41.7	–
DO (mg/l)		4.15 – 5.89	5.41	5.0
TSS (mg/l)		5.00 – 180	66	25.0
TDS (g/l)		27.7 – 69.3	42.0	73.2 - 89
Chloride (mg/l)		3.00 – 13.0	5.73	2.0 – 100
Carbonate (mg/l)		<1.00	<1.00	5 – 25
Cyanide (mg/l)		<0.01	<0.01	–
Nutrients (mg/l)	Nitrate	1.84 – 3.48	2.61	0.05 – 3.00
	Nitrite	<0.02	<0.02	–
	Sulphate	0.93 – 4.76	2.46	2.00 – 150
	Phosphate	0.21 – 0.71	0.39	5 – 500
Cations (mg/l)	Sodium	2.31 – 6.95	3.79	2.00 – 100
	Calcium	3.91 – 12.0	6.02	0.1 – 100
	Potassium	1.02 – 4.08	2.45	–
	Magnesium	0.61 – 3.86	1.88	0.5 – 20.0
Heavy Metals (mg/l)	Cadmium	<0.002	<0.002	0.001 – 0.01
	Total Chromium	<0.10	<0.10	0.0001 – 0.0005
	Copper	<0.05	<0.05	0.002 – 0.05
	Total Iron	0.34 – 2.91	1.52	0.05 – 1.00
	Lead	<0.20	<0.20	0.001 – 0.02
	Zinc	<0.05	<0.05	0.005 – 0.05
	Mercury	<0.0002	<0.0002	0.0003 – 0.003
	Nickel	<0.10	<0.10	0.005 – 0.1
	Vanadium	<0.20	<0.20	0.0001 – 0.003
	Barium	<0.10	<0.10	0.005 – 0.1
	Arsenic	<0.001	<0.001	0.0002 – 0.001
	Manganese	<0.1	<0.1	0.001 – 0.08

RPI (1985), Hem (1986), *Allen (1989), GEMS (1992), UNESCO/WHO/UNEP (1992)

Table 62 - Results for Physico-chemical Analysis of Surface Water (2017)

Parameter	WHO Maximum Permissible Level	FMEnv Limit	Results			Remarks
			SROL 1	SROL 2	SROL 3	
pH	6.5-8.5	6.5 -8.5	7.32	7.33	7.47	Within reference limit
Temperature (°C)			23.7	23.9	24.8	Within reference limit
Turbidity, (NTU)	<5.0	<5.0	0.7	2.2	2.4	Within reference limit
TSS, (mg/l)	30.0	30	6.0	24.0	28.0	Within reference limit
Oil and Grease mg/l	0.3	10	<0.001	<0.001	<0.001	Within reference limit
Phenolic Compounds	0.001	0.2	<0.001	<0.001	<0.001	Within reference limit
Lead (Pb^{2+}), mg/l	0.03	0.05	<0.001	<0.001	<0.001	Within reference limit
Zinc (Zn^{2+}), mg/l	3.0	5.0	0.026	<0.001	0.010	Within reference limit
Cyanide, mg/l	0.07		<0.001	0.006	0.003	Within reference limit
Cyanide, mg/l (free)	0.07		<0.001	<0.001	<0.001	Within reference limit
Cyanide, mg/l (WAD)	0.07		N/A	N/A	N/A	Within reference limit
Arsenic, mg/l	0.01	0.05	<0.001	<0.001	<0.001	Within reference limit
Copper (Cu^{2+}), mg/l	2.0	1.0	<0.001	<0.001	<0.001	Within reference limit
Chromium (Cr^{6+}), mg/l	0.05	0.05	<0.001	<0.001	<0.001	Within reference limit
Total Iron (Fe^{2+}), mg/l	0.3	0.3	0.590	1.220	1.240	Above reference limit
Cadmium (Cd^{2+}), mg/l	0.3	0.01	<0.001	0.020	<0.001	Within reference limit
Nickel (Ni), mg/l	0.2		<0.001	0.030	0.108	Within reference limit
Mercury (Hg), mg/l	0.001		<0.001	<0.001	<0.001	Within reference limit
COD, mg/l	-	80	<1.0	<1.0	<1.0	Within reference limit
BOD, mg/l	-	50	5.90	0.20	0.70	Within reference limit

20.3 Environmental Management Plan

The EIA progressed the potential impacts expected to be generated through the mine project life cycle and environmental management parameters and monitoring were outlined to mitigate potentially adverse impacts. Table 63 summaries the impacts and mitigation measures proposed to redress potentially adverse impacts and to maximise potentially beneficial impacts of the mine development. These impacts and mitigation measures have been expanded in an Environmental Management Plan (EMP) prepared by SROL to manage the environment and social factors throughout the lifecycle of the development.

Table 63 - Summary of Potential Impacts and Mitigation

Project Activity	Associated and Potential Impact	Impact Category Before Mitigation	Mitigation	Impact Category After Mitigation
Land Acquisition	• Risks of communal clashes during paying of compensation • Restriction of land available for farming	Medium	SGL shall pay adequate compensation to land owners SGL shall carryout early identification of rightful land owners before acquisition • All neighbours shall be informed/notified of the intended mining project and the expected boundaries of operation and acquisition • SGL shall limit acquisition of land to the minimum specified by mining lease • Where compatible with the project, SGL shall allow crop farming within lease boundary (outside active mine site)	Negligible
Mobilisation of men and materials to site	Increased risks of road traffic disruption and accidents on local roads as result of large/bulky equipment as well as fuel	Medium	• SGL shall develop and maintain an effective journey management schedule • Prospective route for large/bulky loads associated with the proposed project shall be surveyed by qualified structural engineers. • SGL shall ensure its drivers observe road load and speed limits • SGL vehicle drivers and conductors shall undergo competency training and certification before engagement • SGL shall use road signs at strategic points, sirens Signallers and out-riders to be used as appropriate SGL shall ensure all its vehicles are certified roadworthy and in good maintenance state	Negligible

Project Activity	Associated and Potential Impact	Impact Category Before Mitigation	Mitigation	Impact Category After Mitigation
Land clearing / stripping	• Ecosystem fragmentation and exposure of wildlife to increased exploitation • Exposure of soil to erosion resulting from devegetation and soil movements • Loss of medicinal herbs and rare / endangered plant species	High Low	• Vegetation clearing shall be to the minimum area required for safe operations. • All vegetation not required to be removed shall be protected against damage. • Cleared vegetation debris will not be burnt on site but will be used as land covers and to re-vegetate already completed mine area • SGL shall reclaim unoccupied portions of the disturbed area by re-vegetating with appropriate native species. • SGL shall reclaim unoccupied portions of the disturbed area by re-vegetating with appropriate native species to restore the natural habitat.	Moderate
	Contamination of soils and surface/ground water from accidental fuel spills during fuel delivery, storage, and vehicle / equipment refuelling	High	• Certified road based tankers and water based fuel barges with qualified and certified operators shall be used • Personnel handling petroleum products shall be trained on Safe Handling of Chemical training. • Regular inspection/checks of fuel transport vehicles • Onsite/offsite emergency plan for spill and fire shall be developed and implemented. • All fuel storage tanks (temporary and permanent) shall be designed and installed in full compliance with the DPR requirements as well as relevant industry standards. • Areas where fuels and flammable materials are stored shall have appropriate and adequate fire protection.	Moderate

Project Activity	Associated and Potential Impact	Impact Category Before Mitigation	Mitigation	Impact Category After Mitigation
Land clearing / stripping	Contamination of soils and surface/ground water from accidental fuel spills during fuel delivery, storage, and vehicle / equipment refuelling	High	• Symbolic safety signs depicting "No Smoking", "No Naked Flame", etc shall be prominently displayed in and around the area. • Capacity of every fuel storage tank shall be clearly displayed and the content of each tank identified. • Tanks shall be situated within bund walls that are 110% of the tank volume. • The floor of the bunded area round storage tank shall be smooth and impermeable.	Negligible
	Sedimentation of stream by particulate transported from mine site by rain run-off	Medium	• SGL shall establish and implement a sediment control plan. • Appropriate channels for run-off water shall be constructed. • Drainage channels shall be lined with large rock pieces to prevent erosion and trap sediments. • Spoil dumps shall be located at least 10m away from natural drainage lines. • Progressive reclamation of land cleared for mining shall be adopted. • Sampling, measurement/testing of surface water body in the area to determine any and extent of riverbed sedimentation – resulting from mine sediment migration	Moderate
	Loss/disturbance of wildlife due to fragmentation and loss of natural habitat	Medium	Injuring/killing of any animal shall not be permitted No feeding of wildlife No hunting No domestic animals shall be brought into the site Project site shall be kept clean/tidy and free of rubbish which will attract animal pest species	Moderate
Land clearing / stripping	Employment of local labour Income generation to the villagers from sale of food and other consumable items	Beneficial	This benefit shall be enhanced by: • SGL and host communities defining a representative formula for employment, supplies and contract awards • SGL ensuring that contractors (Engineering and Construction) shall comply with any agreed community employment quotas • SGL shall carry out periodic review of jobs, supplies and contract awards	-Beneficial

Project Activity	Associated and Potential Impact	Impact Category Before Mitigation	Mitigation	Impact Category After Mitigation
Installation of equipment and structures (offices, health clinic, storage yards, etc)	Modification in fauna (wildlife) behaviour with regards to food sources from wastes or garbage produced by the personnel	High	• SGL shall adopt an effective waste management practice SGL shall conduct environmental awareness training in waste handling for personnel Waste management focal point shall be appointed Littering shall be prohibited • Waste bins and skips shall be provided at appropriate locations within the site • Illegal dumping of waste shall be prohibited	Negligible
Drilling and blasting	Noise nuisance to human and wildlife from drilling, blasting, crushing machines and vehicular movements	High	• Appropriate and effective mufflers/silencers shall be fitted to earth moving, vehicles and other site equipment. • Protective devices shall be provided to personnel and their use enforced. • Suitable buffer zone shall be retained around the plant site. • Planting green-belts and building earth bunds with waste rocks shall be incorporated in the overall site design to minimise noise effect and visual intrusion.	Moderate
	Risk of injury from fly-rock resulting from blasting	High	• Prior to rock blasting operations, barriers shall be erected, warning notices prominently displayed and every precaution taken to prevent unauthorised persons from the immediate area. • Personnel and villagers shall be made aware (through public enlightenment programme) of the meanings of the barriers, signs and warnings • Prior to rock blasting, an audible warning either by announcement over a public-address system or by whistle shall be made. • Adequate buffer/distance shall be maintained between site offices/workshops and the mine-site • Farming shall not be allowed in close proximity of the mine-sites	Negligible
	Change in hydrological pattern resulting from mine pit and disposal of overburden	High	• Progressive reclamation of mine pit with waste rock and overburden materials • Planting of tree/vegetation (with native species) on cleared areas	Moderate

Project Activity	Associated and Potential Impact	Impact Category Before Mitigation	Mitigation	Impact Category After Mitigation
Loading and hauling, crushing and grinding	Risks of chronic health diseases (asbestosis, silicosis etc.) to personnel and natives from inhalation of dust released from extraction, transport, beneficiation and waste handling activities crushers	High	• Dust generated from vehicle traffic shall be reduced through regular watering of heavily used access way. Sealing of heavily accessed way shall be considered. Dust from beneficiation shall be controlled by water spray and installation of scrubbers as practicable. • Personnel shall be provided appropriate PPE fit such as respirators and masks.	Moderate
Chemical treatment Water consumption Wastewater discharge Chemical handling / storage	Generation and disposal of mill tailings leading to surface water contamination	High	Mill tailing disposal site shall be properly engineered and its use controlled	Moderate
	Potential stream/river water diversion and water shortage downstream	High	Tailing disposal site shall be away from the natural drainage path	Moderate
	Depletion of water resources – 500,000tones/ annum (surface/ground) to meet treatment process requirement	High	• Wastewater produced from extraction stage shall be reused for downstream processing operations • There shall be a water management plan which shall be developed and implemented.	Moderate
	Potential contamination of stream water by discharged process effluent water	High	• Sampling, measurement/testing of surface water at regular interval shall be carried out for identification of mine process chemicals. • Retention pond design shall provide maximum free water surface possible to reduce depth so as to minimise cyanide concentrations (by natural degradation on exposure to air) in the wastewater • Waste water shall be treated to comply with effluent discharge limits in Nigeria before release	Moderate
	Animal mortality from contact/exposure to process solutions (containing cyanide)	High	Barriers shall be erected to prevent wildlife and domestic animals from getting into retention pond	Negligible

Project Activity	Associated and Potential Impact	Impact Category Before Mitigation	Mitigation	Impact Category After Mitigation
Water storage dam installation and operation	Groundwater contamination by leachate from the mine effluent in the dam Potential stream/river water diversion and water shortage downstream Risk of flooding of communities downstream in an event of dam embankment failure Conflicting demand for water use – shrink in stream/river catchment area	High	• Tailing dump design and construction shall be performed by experienced engineers • Impermeable layers shall be used to prevent leachates from dumpsite from percolating into the groundwater. • Water shall be recycled in the mine operation to minimise quantity of water drawn from both groundwater and surface water systems. • Routine inspection and maintenance shall be carried out on the dam facility to assure its continuing integrity.	Moderate
Camp site activities	Contamination of soil from disposal of domestic wastes, kitchen wastes etc	High	SGL shall adopt an effective waste management practice Sewage shall be collected in septic tanks. Evacuation, treatment and disposal of sewage shall be by government approved personnel / facilities	Negligible
Work environment and personnel health and safety	Health hazards from noise, dust and worksite accidents on humans in the project area	High	A health and safety plan will be developed in respect of construction worker safety Personnel shall be trained in basic site procedures A Health and Safety Officer shall be employed to monitor project activities and adherence to HSE requirements • Appropriate PPE shall be provided for site workers; and the use of the PPE shall be strictly enforced • Dust shall be suppressed on access roads, mine site and crushing areas during dry periods by regular application of water spray as practicable • Work equipment shall be maintained at optimal operating condition • All employees shall be provided ear protection devices (earmuff, earplugs, etc) as necessary • Use of protective gear provided to workers shall be enforced • Workers shall be appropriately trained/made aware of the hazards inherent in their jobs, protection devices and consequence of non-compliance with procedures • Audiometric test shall be carried out periodically on workers in high noise areas to determine: 1. Whether or not hearing loss exists,	Moderate

Project Activity	Associated and Potential Impact	Impact Category Before Mitigation	Mitigation	Impact Category After Mitigation
			2. Type of hearing loss if any, 3. Effectiveness of noise control programmes • Appropriate and effective mufflers/silencers shall be fitted to earth moving and other vehicles on site. First Aid facilities shall be on hand at all times in accordance with international practice.	
Equipment / machinery Maintenance	Generation of used oil / waste oil and spare parts that will contaminate the soil on disposal	High	Scrap yard shall be designated. Scrap metals shall be re-used or sold for recycling as appropriate Waste oil shall be collected in metal drums or cans with lid for reuse and sale. Maintenance technicians' induction and refresher training shall include proper handling of oil/used oil	Negligible
Abandonment, decommissioning and rehabilitation	Excavation and mining shall leave a surface opening or ditch that will serve as natural ponds for fishing and water supply Risk of flooding from overflowing of ditch and springs Risks of trip or fall into ditch Permanent destruction of landform and aesthetic beauty of the area Dramatic increase in unemployment	High	• Consultation shall be made with host communities for possible development of fish farm using abandoned flooded mine pit Signs shall be placed at strategic points Progressive rehabilitation (re-vegetation, backfilling, etc.) shall be performed throughout the mine life • Long term post closure monitoring programme shall be implemented	Moderate

20.4 Monitoring

A monitoring programme (see Table 64) was established in the EIA and included in the EMP. The only two environmental parameters required to be monitored in this phase of the project are surface water and underground water analysis.

Table 64 - Environmental Monitoring Programme

Environmental components	Phase / Activity	Impact Indicator	Sampling Location	Duration
		Biophysical		
Air Quality	Site preparation: Land clearing, Construction (earth movement. asphalt overlay) mine operation and decommissioning	CO, Sox, NOx, SPM, HC	Mine area	Monthly
Soil	Site preparation: Construction (Soil), mine operation and decommissioning	HC. Heavy metal, hydrocarbons, microbes	Mine area	Quarterly
Surface Water	Project life	Water quality parameters	Rivers and streams	Monthly
Underground Water	Project life	Water quality parameters	Monitoring boreholes and wells within the mine area communities	Monthly
Noise/Vibration	Site preparation: Construction	Noise Level (dBA)	Mine Area, Base Camp and Office area	Monthly
		Health		
Communicable Diseases	Mobilisation	Incidence of malaria. STLs. Cholera. Typhoid. TB, HIV	Hospitals and Clinics	Yearly
Stress-Related Diseases	All phases	Hypertension coronary heart diseases		Yearly
Malnutrition	All phases	BMI; MUAC	Hospitals and Communities	Yearly
Accident	All phases	Fatality injuries	Police records. company	Monthly
Hearing Impairment	Site preparation construction	Tolerance shift	Hospitals, Mine Area random communities	Monthly
		Social		
Demography	Project life	Demographic structure occupational distribution: ethnic mix	All communities	Annually
Social Infrastructure	Project life	Infrastructural development	Ditto	Monthly
Local Economy	Project life	Major economic indices	All Communities	Monthly
Natural Resources	Project life	Quality and quantities of exploited natural resources (timber, shell, fish etc.)	All communities	Monthly

20.5 Closure Plan and Post Closure Activities and Costs

A detailed Environmental Protection and Restoration Plan (EPRP) has been prepared recently and approved by the Ministry of Mines and Steel Development. (22 August 2017, Ref. No. MMSD/S/39/S.657).This plan detailed a wide range of activities to occur over the project lifecycle to progressively rehabilitate and restore the mine site during operation, mine closure and post closure phases. The EPRP was based on mine closure components and timing outlined in Table 65. The cost of mine closure outlined in the EPRP is $5 Million USD.

Table 65 - Mine Closure Components and Timing

Mine Component	Closure Objectives	Start	Finish
Open Pit	Dewatering will cease on closure of the mine and the open pit will be allowed to flood. The pit slopes will be stable and groundwater quality will not be compromised. Reassignment to local community for water storage/use. A pounded protection band will be formed around the pit to discourage entry.	Mine Closure	End of Project
Waste Rock Dump	Final slopes of the waste rock dump will be stable and any possible re-vegetation carried out.	End of Project	5 years post closure
Tailings Storage Facility	Profiling of the dam surface and layering with soil to encourage re-vegetation with sustainable crops. Closure of the decant and evaporation of supernatant.	End of Project	5 years post closure
ROM Pad and Process Plant	Decommissioning of the plant, dismantling of buildings and removal of foundations. Removal of scrap metal and used oils etc. Re-profiling and re-vegetation of the site.	End of Project	5 years post closure
Offices and Car Park	Dismantling of buildings and removal of foundations. Removal of office waste. Reprofiling and re-vegetation of the site.	End of Project	5 years post closure
Raw and Process Water Storage tanks	Evaporation of remnant water, removal of contaminated solids to a waste disposal location offsite.	End of Project	5 years post closure
Raw Water Dam	Reassignment of responsibility to local community	End of Project	End of Project

20.6 Permitting

20.6.1 Project permitting requirements and status of permit applications

Table 66 contains the project permitting requirements for this stage of the project.

Table 66 - Permitting Requirements and Permits Obtained

Permit	Relevant Regulatory Authority	Phase(s) that require the permit	Time Frame	Reference(s) in Applicable Regulations	Status	Notes
Exploration License 19066EL	MCO	Exploration	3 Years	Sec. 59,62 of Mining Act, 2007 & Reg.35-44 of Mining Regulations, 2011	Obtained	Renewable for two times each time covering a period of 2 years
Mining Lease 41ML	MCO	Mining	25 Years	Sec.65-69 of Mining Act,2007 & Reg.56-60 of Mining Regulations, 2011	Obtained	Renewable every 24th year throughout the life span of the mine
Environmental Impact Statement	Fed. Min. of Environment & MEC	Pre-Exploration & Mining	5 years	Sec.71 & 119 of Mining Act, 2007 & Reg. 157 - 160 of Mining Regulations, 2011	Obtained	
Environmental Protection & Rehabilitation Programme	MEC	Pre-Exploration & Mining	For the period of the lease	Sec. 120 of Mining Act, 2007 & Reg. 180- 183 of Mining Regulations, 2011	Obtained	
Community Development Agreement	MEC	Pre- Mining	5 years	Sec.71 & 116 of Mining Act, 2007 & Reg. 193 of Mining Regulations, 2011	CDA process ongoing	
Minimum Work Programme	MID	Pre-Exploration & Mining	For the period of the lease	Sec.71 of Mining Act, 2007 & Reg. 116 of Mining Regulations, 2011	Obtained	

20.7 Requirements for Environmental Protection and Rehabilitation Fund

In line with Section 184 of the Nigerian Minerals and Mining Regulations 2011, “every holder of a mineral title shall contribute to the Environmental Protection and Rehabilitation Fund created under the Act.” The amount to be contributed is determined by the Mines Environmental Compliance Department and the mineral titleholder, based on the estimate and the work plan contained in the Environmental Protection and Rehabilitation Programme (EPRP). The regulation also specifies that 5% of the total project cost be set aside for the Environmental Protection and Rehabilitation Fund.

20.8 Socio-Economic Conditions

The EIA socio-economic baseline incorporated household surveys between 2008 to 2010, focus group discussions, a review of built education and health infrastructure, and a review of relevant published data. The last census to occur in Nigeria was in 2006. Although the EIA baseline and census are now dated there are socio-economic aspects that are expected to have changed little over the past 10 years.

20.9 State Socio-economic Factors

The total estimated population of the study communities in 2008 was 8,726. The average household size obtained in the survey was 5.4 for the Iperindo, 5.8 for Odo Ijesha, 5.5 for Imogbara and 5.4 for Ipole Ijesha. The average household size from the collective sample was 5.6 persons. The affected communities were also largely youthful, given that over half of the members of the households that were polled were less than 20 years old. The proportion of population of people between 20 – 64 years which produce most of the workforce was 47.5% in Iperindo, 43.1% in Odo Ijesha and 42.7% in Imogbara. The elderly was about 2.50%, 2.75% and 4.55% in Iperindo, Odo Ijesha and Imogbara respectively.

The estimated age dependency ratios were 110.5% or 1.105:1 for Iperindo, 131.9% or1.319:1 for Odo Ijesha and 134.0% or 1.34:1 for Imogbara. This means that the economically active segment of the population (aged 20 – 64 years) in the affected communities bears a heavy burden to support the dependent segment (those aged 0 – 19 years and 65 years and above). The implications were that a high proportion of the community's resources were dedicated to providing the needs of the young and elderly. This type of age distribution and dependency is typical of developing economies.

The primary occupation in the study communities was farming and this engaged member of households across gender and age divides. Major crops were cocoa, kola nut, plantain, cassava, maize, yam and beans. Other occupations and economic activities identified in study communities included livestock rearing, collection of forest products, petty trading, food processing, hunting, lumbering, artisanship/handicraft, and civil/public service employment.

20.10 Local Socio-economic Conditions

20.10.1 Language, Marriage and Family

The original inhabitants of the study communities are Ijesas, a variant of the Yoruba race. There is however, a significant presence of people from other parts of Yoruba especially Oyo. These communities also have a few people from other parts of Nigeria such as Igbos, Urhobos, Igala and Igede. Although Yoruba (or the Ijesha variant) is the dominant language in the communities, English (in both its formal and pidgin forms) is also widely spoken.

Like in most African societies, the people of the study communities revere the marriage institution and consider it as a binding social contract. The extended family also exists in the communities. It is usually made up of family units descending from a common ancestry. Extended families are usually custodians of certain properties especially lands.

20.10.2 Culture and Religion

The study communities are multi religious, having adherents of the Christian, Muslim and the traditional African faiths. Although most respondents indicated that they were Christians, there is large scale acceptance and practice of some aspects or entire beliefs of traditional religious practices.

20.10.3 Land Management

The land is a very important and key asset in economies that are rural and agrarian like those of the study communities. The ownership and use of land are very significant issues. Lands in the affected communities are not owned and managed by individuals. They are owned by families (extended families) and assigned or leased to individuals.

Traditionally lands are used for economic, housing, development and cultural purposes. The most significant economic purpose is use as farmlands. Some lands have been used for residential purposes, siting of social amenities and some are the location of shrines.

When lands are leased, they are for a specific period or duration but most land on which residences are developed have been handed down by ancestors. Such are held on a freehold basis. Community lands are used for development of infrastructure and common facilities.

20.10.4 Community Governance

The affected communities have similar traditional systems of administration. Communities are headed by a traditional ruler known as Oba. The office of the Oba in the study communities is by appointment and it is held for a life time except if the occupant is deposed by the community. It is only occupied by indigenous males from the ruling families; they may not necessarily be literate, but must be of good conduct. The Oba is assisted by a traditional council of chiefs and an array of other figures in the administration of the communities. The maintenance of law and order in the communities falls on the shoulders of the Obas, their councils of chiefs and various other figures appointed for this purpose in the various communities.

20.10.5 Health and Education Facilities

In 2010 the public health facilities in the affected communities were either inadequate or non-existent. Because of the inadequacies of the existing health facilities or their total absence, residents of the study communities relied heavily on the services provided by drug stores, and traditional medicine practitioners such as traditional birth attendants (TBAs) and herbalists.

20.10.6 Community Development Agreements (CDAs)

Community Development Agreements are currently being prepared each of the affected communities – Iperindo, Odo Ijesha and Imogbara. Subject to the Nigerian Minerals and Mining Act (NMMA), section 116, the company shall, prior to the commencement of any development activity within the project area, conclude with the host communities an agreement: CDA that will ensure the transfer of social and economic benefits to the community. The CDAs usually include training and education assistance, assistance with maintaining public facilities and infrastructure (schools, clinics, civic buildings, grading roads) as well as preferential employment opportunities, support for agricultural enhancement and enterprise development. As part of the collaborative approach to devising the CDAs a 12-person committee is being established (one in each community) with representatives from different professions who will represent each faction. Also, a community liaison representative has been acquired by SROL to promote and aid in the facilitating of the CDA.

20.10.7 Employment Opportunities

Developing the Segilola Project into a mine would offer residents and families from the surrounding communities the opportunity of year-round stable wage paying jobs. Continuing local hire efforts by SROL will be a key focus of the Project. Training programs such as the Drill Helper Training Program conducted in other Mining projects, a partnership with the Nigerian Labour Congress (NLC) Osun State chapter, will be used to attract, train and retain state workforce for the various construction and operating jobs available.

21. CAPITAL AND OPERATING COST ESTIMATES

21.1 Initial Capital Costs

Capital costs for this Preliminary Feasibility Study (PFS) are based largely on the estimates detailed in the Revised Bankable Feasibility Study (RBFS) completed by previous project owners Ratel Group Ltd in 2012. These costs were quoted in detail by industry suppliers and despite the fact that as of the date of this study are five years old, a benchmarking exercise indicated that these capital costs are within the bounds of accuracy required for a PFS.

The RBFS included a capital allowance for pre-production mining. Based on the analysis conducted in this PFS, a pre-strip period will not be required as ore is accessible from the start of the project. This represents a reduction in capital expenditure of around US$17M, with all mining costs incurred as operating costs.

Capital cost estimates by area are summarised in Table 67. A 10% contingency factor has been added to the base cost estimates (numbers may not sum exactly due to rounding).

21.2 Sustaining Capital Costs

Sustaining capital costs have been included to cover miscellaneous upgrades to the processing facility and two lifts of the tailings storage facility. The sustaining capital shown in Table 67 was sourced from the 2012 RBFS (the best available data source on sustaining capital costs as at the date of this study), however with an additional year of production increasing the overall sustaining capital by US$0.5M.

21.3 Mine Closure Costs

The approved Environmental Protection and Restoration Plan (EPRP) outlined the cost of mine closure at $5 Million USD.

Capital costs (initial and sustaining) and mine closure costs are summarised in Table 67.

Table 67- Initial and Sustaining Capital and Mine Closure Costs

Area		0	1	2	3	4	5	6	7	8	9
Mining Contractor Capital Costs		$ 4.68	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Site Establishment, Clearing and Drainage	$USD M	$ 1.81									
Mining Contractor Mobilisation	$USD M	$ 2.88									
Processing Capital Costs		$ 51.99	$ -	$ -	$ 0.33	$ -	$ 0.76	$ -	$ -	$ -	$ -
Direct Plant Costs	$USD M	$ 35.27									
EPCM Contract	$USD M	$ 10.13									
First Fills and Spares	$USD M	$ 3.00									
Earthworks - Tailings and Water Storage Dams	$USD M	$ 3.60			$ 0.33		$ 0.76				
Other Project Capital Costs		$ 8.24	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Accomodation, Roads, Fencing	$USD M	$ 1.78									
Owner's Facilities and Equipment	$USD M	$ 5.61									
Owner's Vehicles/Mobile Plant	$USD M	$ 0.85									
Sustaining Capital	$USD M	$ -	$ -	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 0.50	$ -	$ -
TOTAL US$M, Rounded		$ 64.91	$ -	$ 1.00	$ 1.33	$ 1.00	$ 1.76	$ 1.00	$ 0.50	$ -	$ -
Contingency		$ 6.49									
Estimate + Contingency		$ 71.40	$ -	$ 1.00	$ 1.33	$ 1.00	$ 1.76	$ 1.00	$ 0.50	$ -	$ -
Pre-strip	$USD M	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Closure Costs	$USD M	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 5.00	$ -
Capital Reclaim	$USD M	$ -	$ -	$ -	$ -	$ -	$ -	$ 19.01	$ 48.82	$ -	$ -

21.4 Operating Costs

Processing operating costs for this Preliminary Feasibility Study (PFS) are based largely on the estimates detailed in the Revised Bankable Feasibility Study (RBFS) completed by previous project owners Ratel Group Ltd in 2012, with adjustments made to reflect current conditions.

Mining operating costs are based on quoted budget pricing from West African mining contractors obtained as part of this PFS. Adjustments have been applied to reflect the quoted bulk diesel supply price from Nipco PLC which was received after the contractor submissions were made (late August 2017).

21.4.1 Mining Costs

Requests for Quotation (RFQ) were sought from a number of mining contractors operating in Africa. Several pricing submissions were received, and an appropriate submission was selected to use as a basis for the mining operating costs to be applied in the PFS.

The contractor pricing was supplied based on an assumed diesel fuel price of US$0.63/L, which was the best information available at the time of seeking the pricing submissions. An adjustment was subsequently applied to the load and haul contract mining rates based on a quote for bulk diesel supply obtained by Thor from Nipco PLC at US$0.47/L. This adjustment was applied based on the total diesel usage quoted by the mining contractor of 41M L over the life of the contract.

The unit mining costs ($/t) were written directly to the Surpac block model based on the grade of the block using a nominal cut-off grade of 0.5g/t. The mining cost of waste at surface is US$2.46/t and is inclusive of drill and blast, load and haul, fuel, all labour and equipment maintenance as well as contractor costs, geotechnical, rehabilitation and dayworks.

The full list of mining unit costs by depth as coded into the block model are provided in Table 68 and Table 69.

Table 68 – Waste Mining Unit Costs by depth

WASTE						SG	2.67	
Bench Toe	Load & Haul Contract Rate	Drill and Blast/Ripping Fresh	Dayworks	Geotech (Fresh Only)	Site Mining OH	Rehab, Clear&Grub	Total Mining Cost Fresh ($/BCM)	Total Mining Cost Fresh ($/t)
370.00	$ 3.13	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 6.57**	**$ 2.46**
365.00	$ 3.13	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 6.57**	**$ 2.46**
360.00	$ 3.13	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 6.57**	**$ 2.46**
355.00	$ 3.13	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 6.57**	**$ 2.46**
350.00	$ 3.10	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 6.54**	**$ 2.45**
345.00	$ 3.09	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 6.53**	**$ 2.45**
340.00	$ 3.16	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 6.60**	**$ 2.47**
335.00	$ 3.23	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 6.67**	**$ 2.50**
330.00	$ 3.37	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 6.81**	**$ 2.55**
325.00	$ 3.43	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 6.87**	**$ 2.57**
320.00	$ 3.51	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 6.95**	**$ 2.60**
315.00	$ 3.65	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.09**	**$ 2.66**
310.00	$ 3.72	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.16**	**$ 2.68**
305.00	$ 3.79	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.23**	**$ 2.71**
300.00	$ 4.08	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.52**	**$ 2.82**
295.00	$ 4.15	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.59**	**$ 2.84**
290.00	$ 4.22	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.66**	**$ 2.87**
285.00	$ 4.29	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.73**	**$ 2.90**
280.00	$ 4.43	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.87**	**$ 2.95**
275.00	$ 4.50	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.94**	**$ 2.97**
270.00	$ 4.57	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.01**	**$ 3.00**
265.00	$ 4.64	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.08**	**$ 3.03**
260.00	$ 4.71	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.15**	**$ 3.05**
255.00	$ 4.85	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.29**	**$ 3.11**
250.00	$ 4.92	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.36**	**$ 3.13**
245.00	$ 5.03	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.47**	**$ 3.17**
240.00	$ 5.11	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.55**	**$ 3.20**
235.00	$ 5.17	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.61**	**$ 3.23**
230.00	$ 5.24	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.68**	**$ 3.25**
225.00	$ 5.31	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.75**	**$ 3.28**
220.00	$ 5.43	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.87**	**$ 3.32**
215.00	$ 5.50	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.94**	**$ 3.35**
210.00	$ 5.57	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.01**	**$ 3.38**
205.00	$ 5.64	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.08**	**$ 3.40**
200.00	$ 5.76	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.20**	**$ 3.45**
195.00	$ 5.83	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.27**	**$ 3.47**
190.00	$ 5.90	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.34**	**$ 3.50**
185.00	$ 6.09	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.53**	**$ 3.57**
180.00	$ 6.16	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.60**	**$ 3.60**
175.00	$ 6.23	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.67**	**$ 3.62**
170.00	$ 6.41	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.85**	**$ 3.69**
165.00	$ 6.48	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.92**	**$ 3.72**
160.00	$ 8.01	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 11.45**	**$ 4.29**
155.00	$ 8.07	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 11.51**	**$ 4.31**
150.00	$ 8.15	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 11.59**	**$ 4.34**
145.00	$ 8.21	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 11.65**	**$ 4.36**
140.00	$ 8.28	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 11.72**	**$ 4.39**
135.00	$ 8.35	$ 2.17	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 11.79**	**$ 4.42**

Table 69 – Ore Mining Unit Costs by depth

ORE					SG		2.67	
Bench Toe	Load & Haul Contract Rate	Drill and Blast/Ripping Fresh	Dayworks	Geotech (Fresh Only)	Site Mining OH	Rehab, Clear&Grub	Total Mining Cost Fresh ($/BCM)	Total Mining Cost Fresh ($/t)
370.00	$ 3.44	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.42**	**$ 2.78**
365.00	$ 3.44	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.42**	**$ 2.78**
360.00	$ 3.44	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.42**	**$ 2.78**
355.00	$ 3.44	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.42**	**$ 2.78**
350.00	$ 3.39	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.37**	**$ 2.76**
345.00	$ 3.34	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.32**	**$ 2.74**
340.00	$ 3.41	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.39**	**$ 2.77**
335.00	$ 3.48	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.46**	**$ 2.79**
330.00	$ 3.56	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.54**	**$ 2.82**
325.00	$ 3.62	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.60**	**$ 2.85**
320.00	$ 3.70	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.68**	**$ 2.88**
315.00	$ 3.76	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.74**	**$ 2.90**
310.00	$ 3.83	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.81**	**$ 2.93**
305.00	$ 3.90	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 7.88**	**$ 2.95**
300.00	$ 4.13	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.11**	**$ 3.04**
295.00	$ 4.20	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.18**	**$ 3.06**
290.00	$ 4.26	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.24**	**$ 3.09**
285.00	$ 4.34	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.32**	**$ 3.12**
280.00	$ 4.40	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.38**	**$ 3.14**
275.00	$ 4.48	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.46**	**$ 3.17**
270.00	$ 4.54	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.52**	**$ 3.19**
265.00	$ 4.62	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.60**	**$ 3.22**
260.00	$ 4.68	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.66**	**$ 3.24**
255.00	$ 4.76	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.74**	**$ 3.27**
250.00	$ 4.83	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.81**	**$ 3.30**
245.00	$ 4.95	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 8.93**	**$ 3.35**
240.00	$ 5.02	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.00**	**$ 3.37**
235.00	$ 5.09	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.07**	**$ 3.40**
230.00	$ 5.16	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.14**	**$ 3.42**
225.00	$ 5.23	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.21**	**$ 3.45**
220.00	$ 5.36	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.34**	**$ 3.50**
215.00	$ 5.43	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.41**	**$ 3.53**
210.00	$ 5.49	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.47**	**$ 3.55**
205.00	$ 5.57	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.55**	**$ 3.58**
200.00	$ 5.70	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.68**	**$ 3.63**
195.00	$ 5.77	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.75**	**$ 3.65**
190.00	$ 5.84	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.82**	**$ 3.68**
185.00	$ 5.97	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 9.95**	**$ 3.73**
180.00	$ 6.04	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 10.02**	**$ 3.75**
175.00	$ 6.11	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 10.09**	**$ 3.78**
170.00	$ 6.33	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 10.31**	**$ 3.86**
165.00	$ 6.40	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 10.38**	**$ 3.89**
160.00	$ 8.58	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 12.56**	**$ 4.71**
155.00	$ 8.65	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 12.63**	**$ 4.73**
150.00	$ 8.72	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 12.70**	**$ 4.76**
145.00	$ 8.79	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 12.77**	**$ 4.78**
140.00	$ 8.86	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 12.84**	**$ 4.81**
135.00	$ 8.93	$ 2.71	$ 0.15	$ 0.05	$ 1.00	$ 0.07	**$ 12.91**	**$ 4.84**

21.4.2 Processing Costs

Direct processing costs were derived from the preliminary metallurgical test work conducted for the RBFS, adjusted for current diesel fuel costs for power generation.

In 2012, the RBFS average direct processing cost was US$23.11/t using a fuel cost of US$0.97/L. Power cost per tonne of ore in the RBFS was assumed at US$11.20/t of ore.

The unit price of diesel quoted by Nipco PLC is US$0.47/L, based on a bulk supply proforma invoice obtained by Thor in late August 2017. Applying this revised fuel cost to the fuel component of the processing cost results in an updated processing cost of US$17.16/t.

21.4.3 General and Administration Costs

G&A costs were set at US$3.00/t as directed by Thor, based on an assumption of US$1.5M per annum spending in this area on items such as site salaries and overheads, insurance, accounting, legals and consulting fees relating to this project.

21.4.4 Other Operating Costs

- US$0.33/t grade control cost - assumed value based on RBFS information and experience with similar operations
- US$0.77/t re-handle – estimated value based on 80% of material dumped on the ROM pad requiring re-handling, at a contractor quoted rate of ~US$0.96/t
- US$0.62/t refining cost – assumed value based on RBFS information

21.4.5 Operating Cost Summary

Life of mine operating cost totals are shown in Table 70.

Table 70 – Life of Mine Operating Costs

LoM Project Operating Costs	Total $M	$/oz	$/t
Mining	$ 183.5	427	54.9
Processing	$ 60.0	139	17.9
G&A	$ 18.3	42	5.5
Refining	$ 2.1	5	0.6
Cash Operating Cost	$ 263.8	613	78.9
Royalties	$ 7.7	18	2.3
Total Cash Cost	$ 271.5	631	81.2
Sustaining Capital	$ 11.6	27	3.5
Corporate G&A	$ 10.3	24	3.1
All-in Sustaining Cost	$ 293.4	682	87.7

22. ECONOMIC ANALYSIS

22.1 Summary

A cash flow model was developed to evaluate the project economics based on the life of mine schedule detailed in Section 16.2. Economic parameters quoted are based on 100% ownership of the Project and the assumptions detailed in section 22.2. No Inferred material has been used to generate revenue in the economic analysis.

The key inputs of the economic analysis are detailed in Table 71. A summary of the outputs is shown in Table 72.

Table 71 – Economic Analysis Inputs

Economic Inputs	Units	Value
Mining Recovery	%	95%
Mining Dilution	%	110%
Processing Throughput	tpa	500,000
Processing Recovery	%	96%
Sell Price	oz	1,250.00
Royalty (TML)	% sell price	1.5%
TML Royalty Cap	$	$ 4,000,000
Royalty (Ratel)	% sell price	1.5%
Ratel Royalty Cap	$	$ 3,500,000
Nigerian Government Royalty	% sell price	0.0%
Industrial Levy	% payroll	1%
Discount Rate	%	8%
Corporate Income Tax (CIT)	% profit	30%
CIT Holiday Period	years	5.0
Education Tax (EDT)	% profit	2%
Capital Reclaim	%	95%
g:oz conversion	g/oz	31.1034768

Table 72 – Economic Analysis Outputs

Economic Outputs		Units	Value
NPV @ 8%	Pre tax	$M	$ 121
	Post Tax	$M	$ 119
IRR	Pre tax	%	53%
	Post Tax	%	53%
Payback		Years	1.8
Pre-production Capital		$M	$ 71.4
Gold Production	Years 1-3	koz	81
	Years 4-7	koz	47
Production Cost	LOM C1	$/oz	$ 613
	LOM AISC	$/oz	$ 682
Mine Life		Years	7.0
Probable Mineral Reserves	Tonnes	Mt	3.34
	Grade	g/t	4.2
	Ounces	koz	448
LOM Processing	Tonnes	Mt	3.34
	Grade	g/t	4.2
	Recovered Oz	koz	430
Processing Recovery		%	96%

Economic modelling on various gold prices was performed on both a pre-tax and post-tax basis, with the results summarised in Table 73.

Table 73 – Segilola PFS Project Economics

Gold Price	1,050	1,150	1,250	1,350	1,450
Pre Tax					
NPV 8%	55	88	121	155	188
NPV 5%	69	105	141	178	214
IRR	29%	41%	53%	65%	78%
Payback	2.8	2.4	1.8	1.3	1.2
After Tax					
NPV 8%	54	87	119	150	181
NPV 5%	68	104	138	172	206
IRR	29%	41%	53%	65%	77%
Payback	2.8	2.4	1.8	1.3	1.2

Notes: Economics have been centred on a base case using a 8% discount rate and a gold price of $1,250/oz. Economics based on 100% equity financing with contractor mining. Payback period calculated on an undiscounted basis starting from production start. West African peers commonly use 5% NPV and these figures are quoted for comparison.

22.2 Principal Assumptions

- Base case gold price of $1,250/oz as per the basis of the Mineral Reserves
- No Inferred material has been included in the economic analysis
- Base case discount rate of 8% (results at 5% also included for comparison with West African peers)

- Capital costs and mine closure costs as detailed in sections 21.1, 21.2 and 21.3
- Operating costs as detailed in section 21.4
- Taxes and Royalties as detailed in the following section 22.2.1

22.2.1 Taxes and Royalties

Thor has provided the following assumptions regarding taxes and royalties to be used in the economic analysis.

- Zero import duties (in accordance with the Nigerian Mining Act)
- Zero Value Added Tax (VAT)
- Zero Government Royalty (an exemption is expected but is yet to be confirmed)
- Education Tax – 2% on profit
- Industrial Levy – 1% of the payroll
- Royalties payable to previous project owners, based on Net Smelter Return (NSR):
    - 1.5% Net Smelter Return royalty payable to Tropical Mines Limited, to a maximum value of US$4M
    - 1.5% Net Smelter Return royalty payable to Ratel Group Limited, to a maximum value of US$3.5M
- Pioneer Status Tax Holiday:
    - A three year tax holiday followed by a further two year extension (pending confirmation of the extension requirements).
    - A return of 95% of capital invested (in accordance with the Nigerian Mining Act).
    - Following the five year tax holiday and return on capital invested, tax is payable at 30%

Thor has advised that certain approvals will be required in order to meet the requirements of the incentive regimes set out in Nigerian legislation, however at this stage there are no known reasons why the incentives would not be granted.

The life of mine cash flow summary is shown in Table 74.

Table 74 – Life of mine cash flow summary

Capital Costs													
Mining Contractor Capital	$M	$ 4.7		$ 4.7	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Processing Capital	$M	$ 53.1		$ 52.0	$ -	$ -	$ 0.3	$ -	$ 0.8	$ -	$ -	$ -	$ -
Project Capital	$M	$ 8.2		$ 8.2	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital	$M	$ 5.5		$ -	$ -	$ 1.0	$ 1.0	$ 1.0	$ 1.0	$ 1.0	$ 0.5	$ -	$ -
Contingency	$M	$ 6.5		$ 6.5	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Pre-strip	$M	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Closure Costs	$M	$ 5.0		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 5.0	$ -
Operating Costs													
Mining													
Total	$M	$ 182.3		$ -	$ 68.7	$ 27.3	$ 21.8	$ 19.6	$ 20.7	$ 16.9	$ 7.2	$ -	$ -
Thor Overheads	$M	$ 7.9		$ -	$ 1.2	$ 1.2	$ 1.2	$ 1.2	$ 1.2	$ 1.2	$ 0.7	$ -	$ -
Grade Control	$M	$ 1.2		$ -	$ 0.3	$ 0.1	$ 0.3	$ 0.1	$ 0.1	$ 0.2	$ 0.2	$ -	$ -
Processing													
Variable Processing Costs	$M	$ 57.4		$ -	$ 8.1	$ 8.6	$ 8.6	$ 8.6	$ 8.6	$ 7.7	$ 7.2	$ -	$ -
Rehandle	$M	$ 2.6		$ -	$ 0.4	$ 0.4	$ 0.4	$ 0.4	$ 0.4	$ 0.3	$ 0.3	$ -	$ -
Refining	$M	$ 2.1		$ -	$ 0.3	$ 0.3	$ 0.3	$ 0.3	$ 0.3	$ 0.3	$ 0.3	$ -	$ -
Thor Overheads	$M	$ 10.4		$ -	$ 1.5	$ 1.5	$ 1.5	$ 1.5	$ 1.5	$ 1.5	$ 1.4	$ -	$ -
Royalties & Levies													
TML Royalty	$M	$ 4.0		$ -	$ 1.9	$ 1.2	$ 0.9	$ -	$ -	$ -	$ -	$ -	$ -
Ratel Royalty	$M	$ 3.5		$ -	$ 1.9	$ 1.2	$ 0.4	$ -	$ -	$ -	$ -	$ -	$ -
Nigerian Government Royalty	$M	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Industrial Levy	$M	$ 0.2		$ -	$ 0.0	$ 0.0	$ 0.0	$ 0.0	$ 0.0	$ 0.0	$ 0.0	$ -	$ -
Revenue													
Gold	$M	$ 537.7		$ -	$ 126.3	$ 78.0	$ 99.6	$ 68.3	$ 40.3	$ 48.2	$ 77.0	$ -	$ -
Cashflow													
Cashflow	$M	$ 183.2		$ (71.4)	$ 42.0	$ 35.2	$ 62.9	$ 35.6	$ 5.8	$ 19.0	$ 59.2	$ (5.0)	$ -
Cumulative Cashflow	$M			$ (71.4)	$ (29.4)	$ 5.8	$ 68.6	$ 104.2	$ 110.0	$ 129.0	$ 188.2	$ 183.2	$ 183.2
Discounted Cashflow	$M	$ 121.4		$ (71.4)	$ 38.9	$ 30.2	$ 49.9	$ 26.2	$ 4.0	$ 12.0	$ 34.5	$ (2.7)	$ -
Cumulative Discounted Cashflow	$M			$ (71.4)	$ (32.5)	$ (2.4)	$ 47.5	$ 73.7	$ 77.6	$ 89.6	$ 124.1	$ 121.4	$ 121.4
Cost per Ounce													
C1	$/oz	$ 613		$ -	$ 797	$ 632	$ 427	$ 580	$ 1,015	$ 730	$ 281	$ -	$ -
AISC	$/oz	$ 824		$ -	$ 834	$ 686	$ 461	$ 599	$ 1,070	$ 757	$ 289	$ -	$ -
Taxation													
Capital Allowance o/b	$			$ 67.8	$ 67.8	$ 67.8	$ 67.8	$ 67.8	$ 67.8	$ 51.5	$ -	$ -	$ -
Capital Allowance Useage	$			$ -	$ -	$ -	$ -	$ -	$ -	$ (19.0)	$ (48.8)	$ -	$ -
Capital Allowance c/b	$			$ 67.8	$ 67.8	$ 67.8	$ 67.8	$ 67.8	$ 67.8	$ 48.8	$ -	$ -	$ -
Assessable Profit for CIT	$	$ 10.4		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 10.4	$ -	$ -
CIT	$	$ 3.1		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3.1	$ -	$ -
Assessable Profit for EDT	$	$ 78.2		$ -	$ -	$ -	$ -	$ -	$ -	$ 19.0	$ 59.2	$ -	$ -
EDT	$	$ 1.6		$ -	$ -	$ -	$ -	$ -	$ -	$ 0.4	$ 1.2	$ -	$ -
Total Taxes	$	$ 4.7		$ -	$ -	$ -	$ -	$ -	$ -	$ 0.4	$ 4.3	$ -	$ -
Cashflow (After tax)													
Cashflow (After Tax)	$	$ 178.5		$ (71.4)	$ 42.0	$ 35.2	$ 62.9	$ 35.6	$ 5.8	$ 18.6	$ 54.9	$ (5.0)	$ -
Cumulative Cashflow	$			$ (71.4)	$ (29.4)	$ 5.8	$ 68.6	$ 104.2	$ 110.0	$ 128.6	$ 183.5	$ 178.5	$ 178.5
Discounted Cashflow	$	$ 118.7		$ (71.4)	$ 38.9	$ 30.2	$ 49.9	$ 26.2	$ 4.0	$ 11.7	$ 32.0	$ (2.7)	$ -
Discounted Cumulative Cashflow	$			$ (71.4)	$ (32.5)	$ (2.4)	$ 47.5	$ 73.7	$ 77.6	$ 89.4	$ 121.4	$ 118.7	$ 118.7

Due to the advantageous tax regimes currently afforded to companies such as Thor from the Nigerian Government, there is limited tax liability for the Project.

22.3 Financial Sensitivity Analysis

The project's NPV (8% discount rate, pre-tax) sensitivity to variations in the main inputs was examined. Inputs parameters were varied using the base case input and minimum and maximum expected values. The probability of an input was estimated using a simple triangular distribution with the base case being the most likely individual input (c), while no input sits outside the range of the minimum (a) and maximum (b) expected values (Figure 104). To simplify the analysis, the minimum and maximum expected values were reduced to a factor/multiple of the base case value (Table 75) and plotted on a cumulative distribution curve (Figure 105).

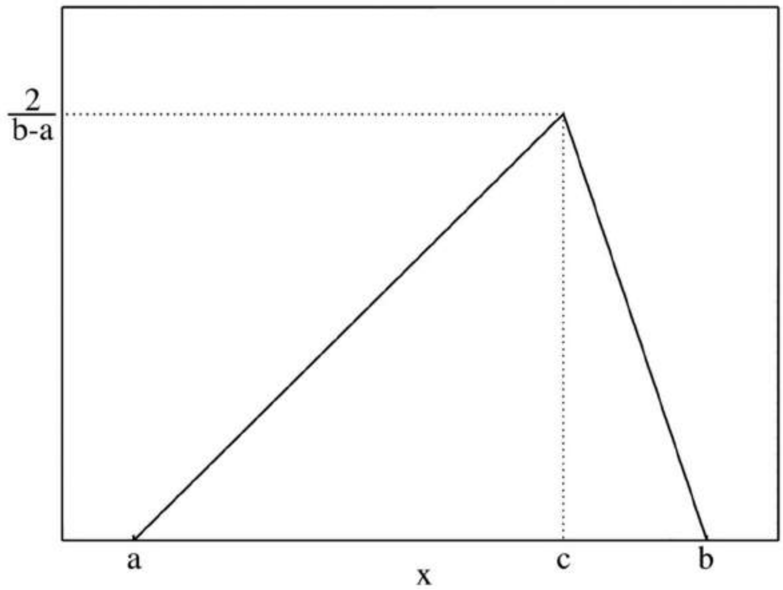


Figure 104- Triangular Distribution used for Sensitivity Analysis Inputs

Table 75- Sensitivity Analysis Inputs

Variable	Unit	Minimum Value	Base Case Value	Maximum Value	Minimum Factor	Base Case Factor	Maximum Factor
Gold Price	$/oz	1050	1250	1450	0.84	1	1.16
Mining Cost[1]	$/t	2.24	2.8	3.36	0.8	1	1.2
Processing Cost[2]	$/t	15.77	18.55	22.26	0.85	1	1.25
Fixed Costs	$k/mth	202	225	270	0.9	1	1.2
Capital Costs	$M	57.1	71.4	92.8	0.8	1	1.3
Discount Rate	%	5	8	10	0.625	1	1.25

1- Mining costs and grade control costs
2- Direct variable processing costs, rehandle and refining

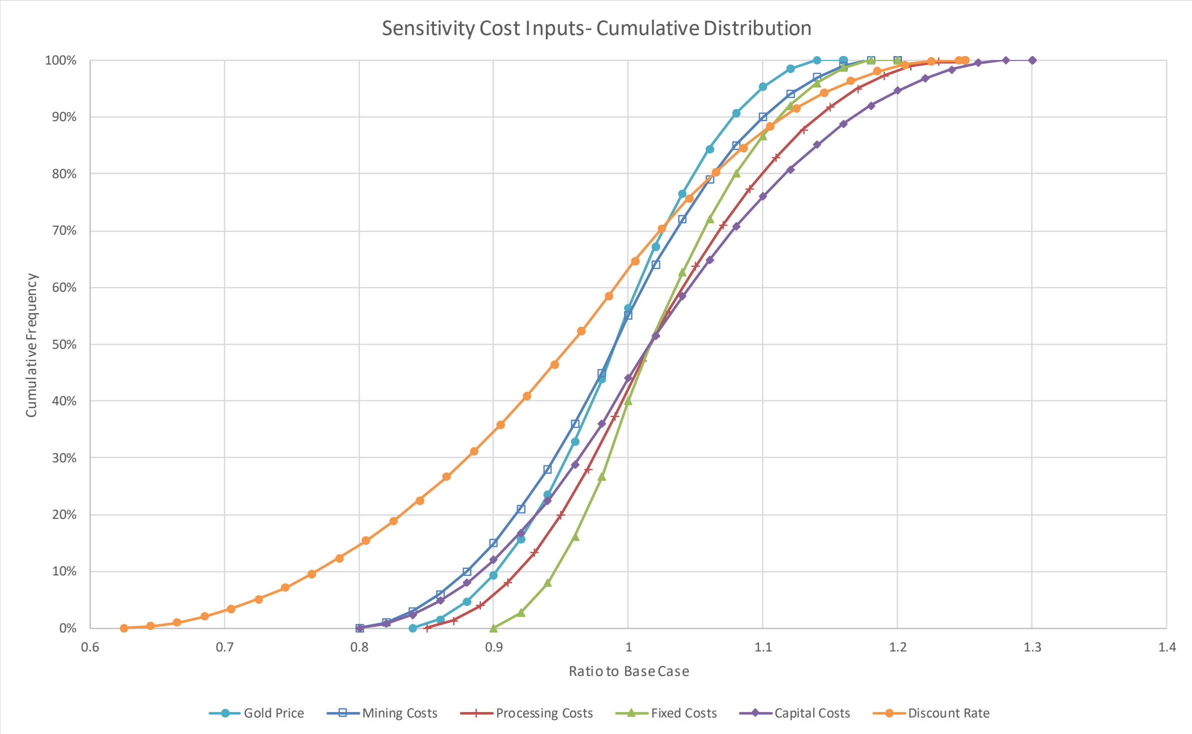


Figure 105- Cumulative Distribution of Sensitivity Analysis Inputs

The resulting analysis indicated that the project is fairly robust with the probability of exceeding the base case pre-tax NPV at 8% discount rate of $121M over 40%, while there is a 75% probability of the project producing a return in excess of $90M.

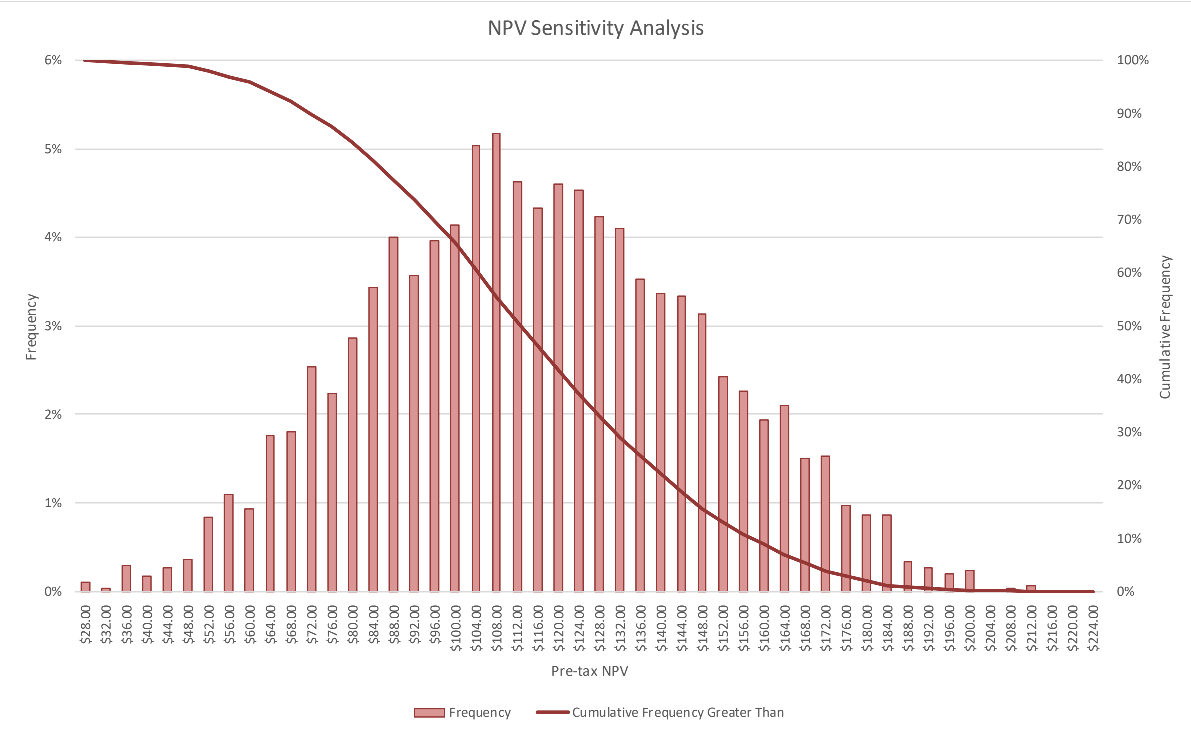


Figure 106- Sensitivity Analysis Outputs (pre-tax NPV @ 8% discount rate)

23. ADJACENT PROPERTIES

There are no mineral exploration or development projects in the near vicinity of the Segilola Project.

24. OTHER RELEVANT INFORMATION

The author and contributors believe that all relevant information has been included in the preceding sections of this report.

25. INTERPRETATION AND CONCLUSIONS

The inputs and ultimate outputs for this Preliminary Feasibility Study are considered to have an accuracy of ±25%.

The results of this Preliminary Feasibility Study indicate that work on the Segilola Project should progress to a more detailed study. The current Mineral Reserve and mining schedule show a project that is robust and will generate significant returns. The Mineral Resource and ongoing exploration also point to a potential expansion to the Project.

Significant work will be required in multiple areas to bring the level of detail up to the standard required of Definitive Feasibility Study. These areas include, but may not be limited to:

- Capital Cost Estimates- re-evaluating plant design and supplier options
- Operating Cost Estimates- potential processing plan changes and detailed discussions with potential mining contractors
- Geotechnical Drilling and Analysis
- Metallurgical Testwork- based on any processing plant changes
- Pit Designs- updated to suit final optimisation, equipment selection
- Tailings Storage Facility Design and Analysis
- Surface Infrastructure Design- updated, detailed processing/office compound
- Construction/Mining/Processing Applications

  Mine closure procesa and costs

  Environment and HSE management, monitoring and resourcing

  Community and social benefits, funding, monitoring and resourcing

Specific recommendations on requirements to progress the project to Definitive Feasibility Study level are provided in section 26.

26. RECOMMENDATIONS

26.1 Mineral Resources

Further work programs should address the following with regard to the Mineral Resources:

1. The Mineral Resource Estimate shows a substantial volume of material classified as Inferred or Unclassified. This material is an immediate target for resource classification upgrade.

2. It is recommended that optimised pit shells are used as a guide to creating drilling programs that maximise the conversion from lower to higher resource classification and reduce mining risk attributed to data density and quality.

3. Maintain the current quality assurance procedures to ensure high quality data is available for subsequent resource estimates.

4. Collection of additional structural measurements of mineralised vein orientations, obtained from future diamond core drilling programs, is essential for fine tuning the mineralisation domain boundaries for any future resource model updates. Continued improvement in geological understanding and lithology unit interpretation.

5. Undertake programs to establish the suitability and reliability of drilling techniques, i.e. by collecting twinned samples from the same hole (field duplicates) and from paired holes of either the same method or of different methods.

26.2 Mining

26.2.1 Underground Mining Potential

Existing exploration drilling and Resource modelling has indicated that there is a potential to undertake mining at the Project via underground mining methods upon completion of open cut mining. Upgrading Inferred Mineral Resources to Indicated and the extending the Resource through additional exploration drilling would be required before underground could be realistically considered. If or when underground mining is considered an option, the pit would need to be re-optimised to determine the most beneficial point to transition from open cut to underground mining, which could ultimately result in a smaller pit than has been presented here. In such a case, most tonnes and ounces eliminated from the pit would be mineable by underground methods.

26.2.2 Geotechnical

The following is reproduced from the Preliminary Feasibility Study Geotechnical Assessment report completed by Peter O'Bryan & Associates in September 2017. Further detail is provided in Appendix 2.

In order to bring the project to a full (Bankable) Feasibility Study stage, the following additional work would be required:

- Additional off-section geotechnical holes in the hangingwall (West Wall) and footwall (East Wall) are recommended to define the extent and frequency of E-W steeply dipping structures and rockmass and structure within the pit walls. A summary of required drilling is shown in Table 76. It is assumed that all drilling would be conducted using triple tube (HQ/NQ) drilling techniques as it is vital that as much of the natural fabric as practicably possible is preserved during the drilling process. Drill collar locations are approximate, dependent on accessibility, topography and other local operating factors and can be adjusted to suit local conditions, but should be reviewed by the authors should changes be required.

Table 76 – Proposed Geotechnical Drilling Program, Definitive Feasibility Study

HOLE-ID	Easting m	Northing m	RL, m	Azimuth	Plunge	Hole Length, m
GTFS17-001	701500	831150	370	090	-55	0
GTFS17-002	701500	831150	370	045	-45	280
GTFS17-003	701500	831150	370	140	-45	280
GTFS17-004	701725	831100	340	090	-55	250
GTFS17-005	701795	831100	340	035	-50	170
GTFS17-006	701795	831100	340	145	-50	170
GTFS17-007	701655	831475	355	140	-55	200
GTFS17-008	701900	831440	340	235	-45	175
GTFS17-009	701765	830750	330	330	-45	175
			Total			1670 m

- A hydrogeological and hydrology study to identify the potential and impact of pressurised groundwater occurring along faults/shears and fracture networks, and surface drainage conditions associated with seasonal watercourses. It is understood a consultant has been commissioned to conduct a preliminary desktop study. The availably of wall holes for future water level monitoring should be communicated as part of the overall FS.

- Rock property testing of critical structures (structures controlling sliding, wedge failures – direct shear testing of representative structures) and UCS tests of selected representative samples from the footwall calc-silicate rock unit (one sample tested as part of this review). Holes GTFS004, 5, 6, 8 and 9 will be located in the footwall rock unit along the length of the southern pit.

- A site visit to view local conditions, topography, water and specifically drill core, logging process and sample selection for direct shear testing

- Define the extent and frequency of E-W steeply dipping structures, aiming to confirm/ extend the database and, if present, identify and characterise additional structures not detected in the geotechnically logged holes within the pit shell. Ensuring best quality drilling and core recovery is vital – use of triple tube core barrels is considered to be essential.

- Due to the ~ 1,000m strike length of the southern pit, up to nine (9) additional holes are recommended and include holes into the east wall footwall and two additional holes angled at 120° to the perpendicular holes in the west wall, as well as holes covering the north and south wall of the preliminary southern pit.
- Additional holes were also considered to better define the designs for the central and northern pits, however, the current level of data is considered appropriate, given the relatively shallow pit depths and to manage the pit design through initial starter pits, using shallower temporary wall angles (Langille, et.al. 2017A; Orr, 2010) and using mapping data collected during Stage 1 to optimize and finalise the final slope designs.

26.2.3 Hydrogeology

The following is reproduced from the desktop Hydrogeology Review report completed by Peter Clifton & Associates in September 2017. The full report is included in Appendix 3.

Site-scale investigations are clearly needed to improve the hydrogeological knowledge base and progress the project to Definitive Feasibility Study level. The broad objective of these investigations is to gain sufficient understanding of the site hydrogeology to confidently define the approach or approaches to groundwater management both during mining and when the mine is abandoned.

A staged investigation program is recommended, and key components are:

- Review available drilling reports for any indications of difficult drilling conditions, groundwater production (from RC air hammer boreholes), or drilling fluid circulation problems (from diamond core holes).
- Conduct air-lit pumping tests in several of the existing open boreholes, noting the yield and whether the yield varies with time. These tests may provide an opportunity to collect samples of groundwater form the deeper rock sequence and structures at the project for determination of salinity.
- Depending on the results of the air lift tests, review the site geological and structural models and select at least two sites for the construction of trial dewatering wells. These wells would then be subjected to pumping test, and this may require nearby observation piezometers to be constructed. Samples of groundwater would be collected from these wells for comprehensive hydrochemical analysis.
- Assess all of the results, and develop a mine groundwater management plan. This plan will need to address any groundwater issues identified by the mining geotechnical consultant, such as the need to lower groundwater pressures in the pit walls. The plan will also need to address the management of

any seasonal variation in groundwater levels, and this may require some interaction with the surface water management strategy for the open pit.

- As part of the management plan, identify sites for piezometers, either open standpipes or multi-level vibrating wire piezometers.
- Prior to any site clearing and the start of mining, prepare an inventory of local groundwater and surface water users who may be impacted by the mining operations if possible, collect water samples from comprehensive hydrochemical analysis. Periodic monitoring of these water resources during mining should occur.

All of the site hydrogeological investigations will need to be conducted in accordance with the requirements of Nigerian government agencies and/or the equivalent agencies in Osun state.

26.3 Processing

As part of the DFS, detailed metallurgical test work will be required. This will also provide the opportunity to optimise recoveries and flow sheet inputs for design and construction.

A broad metallurgical test work program should provide further definition and confidence with respect to the variability of the ore across the deposit, and allow optimisation of the existing process flowsheet, prior to finalising the DFS design parameters. Specific opportunities may include:

- Reviewing the grind size – Thor management considers that there may be operational benefits from a reduction in grind size from P80 106microns to P80 75microns;
- Gravity recovery circuit (existing test work indicates 36-84% gravity recovery is achievable);
- Pre leach thickening; and
- The use of air or oxygen as an oxidant.

26.4 Environment and Social Parameters

- Mine closure costs. This will need to be linked to mine planning, progressive closure over the lifecycle of the mine, and capability accommodated other land uses in the future (agriculture, return of native species and re-establishing biodiversity to prior to construction);
- Environmental and HSE management and monitoring outlined in the environmental management plan and committed to in the certified project EIA; and
- Social parameters addressing activities and costings for community benefits outline in the CDAs, land acquisition and compensation for loss of crops, trees of economic value, other assets for those people

currently undertaking agricultural activities within the mine footprint area and ongoing corporate social responsibility funding.

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28. CERTIFICATES OF QUALIFIED PERSONS

I, Anthony Keers BEng(Hons) DipPM MAusIMM (CP), do hereby certify that:

1. I am an Independent Consultant of Auralia Mining Consulting Pty Ltd, 19-21 Outram Street, West Perth, WA, Australia, 6005

2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Pre-Feasibility Study, Segilola Gold Project, Osun State, Nigeria, Thor Explorations Limited" with an effective date of 16th October 2017

3. I graduated from the University of Queensland with a Bachelor of Engineering (Mining) in 2001 and have worked as a mining engineer for a total of 15 years since graduation. My relevant work experience for the purpose of the Technical Report is through 9 years of consulting, undertaking numerous open pit Mineral Reserve studies in Australia, South America and Asia.

4. I am a Member and Chartered Professional of the Australian Institute of Mining and Metallurgy, registration number 209571.

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 relevant past work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am a co-author author of the Technical Report. I am responsible for sections 15, 16, 21 and 22 and supervising of the preparation of the technical report.

7. I have visited the Project that is the subject of this Technical report from 4th to 6th May, 2017.

8. I, as a "qualified person", am independent of Thor Explorations Ltd, as defined in section 1.5 of National Instrument 43-101

9. I have read NI 43-101 and Form 43-101F1 and confirm that this Technical Report has been prepared in compliance herewith.

10. I have had no prior involvement with the Project that is the subject of this Technical Report. I certify that there is no circumstance that could interfere with my judgement regarding the preparation of this Technical Report

11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading

Dated this 16th day of October, 2017

Original Copy Signed by Anthony Keers

Anthony Keers, BEng(Hons) DipPM MAusIMM (CP)

I, Christopher Speedy BSc(Geology) MAIG, do hereby certify that:

1. I am an Independent Consultant of Auralia Mining Consulting Pty Ltd, 19-21 Outram Street, West Perth, WA, Australia, 6005

2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Pre-Feasibility Study, Segilola Gold Project, Osun State, Nigeria, Thor Explorations Limited" with an effective date of 16th October, 2017

3. I graduated from the University of Otago with a Bachelor of Science (Geology) in 2006 and have worked as a geologist for a total of 11 years since graduation. My relevant work experience for the purpose of the Technical Report is through 9 years of consulting, undertaking numerous open pit Mineral Resource studies in Australia and Asia.

4. I am a Member of the Australian Institute of Geoscientists registration number 5349.

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 relevant past work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am a co-author author of the Technical Report. I am responsible for sections 12 & 14.

7. I have not visited the Project that is the subject of this Technical report.

8. I, as a "qualified person", am independent of Thor Explorations Ltd, as defined in section 1.5 of National Instrument 43-101

9. I have read NI 43-101 and Form 43-101F1 and confirm that this Technical Report has been prepared in compliance herewith.

10. I have had no prior involvement with the Project that is the subject of this Technical Report. I certify that there is no circumstance that could interfere with my judgement regarding the preparation of this Technical Report

11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading

Dated this 16th day of October, 2017

Original Copy Signed by Christoper Speedy

Christopher Speedy BSc(Geology)

29. DISCLAIMER AND LIMITATIONS

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APPENDIX 1 – GEOLOGY

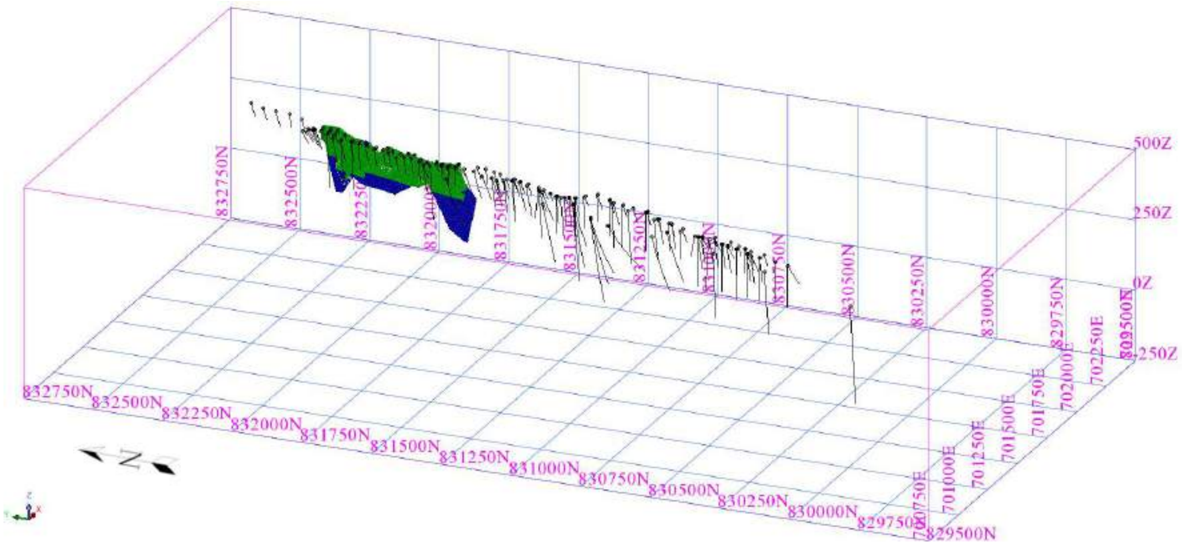

Figure 107 - Lode 100 Resource Classification (Green – Indicated, Blue – Inferred, Grey – Mineral Potential)

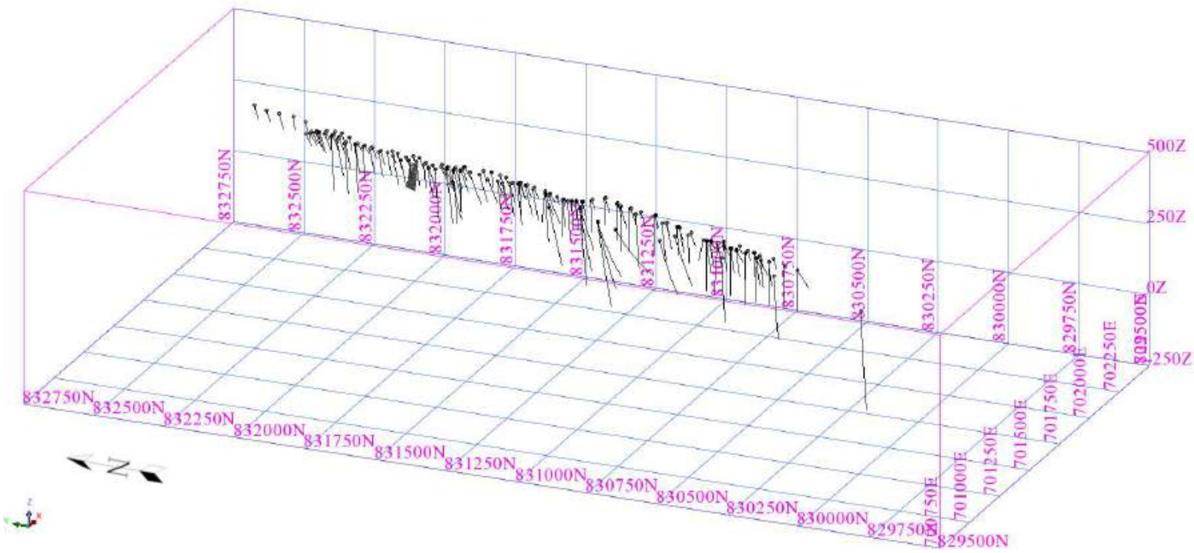

Figure 108 - Lode 101 Resource Classification (Green – Indicated, Blue – Inferred, Grey – Mineral Potential)

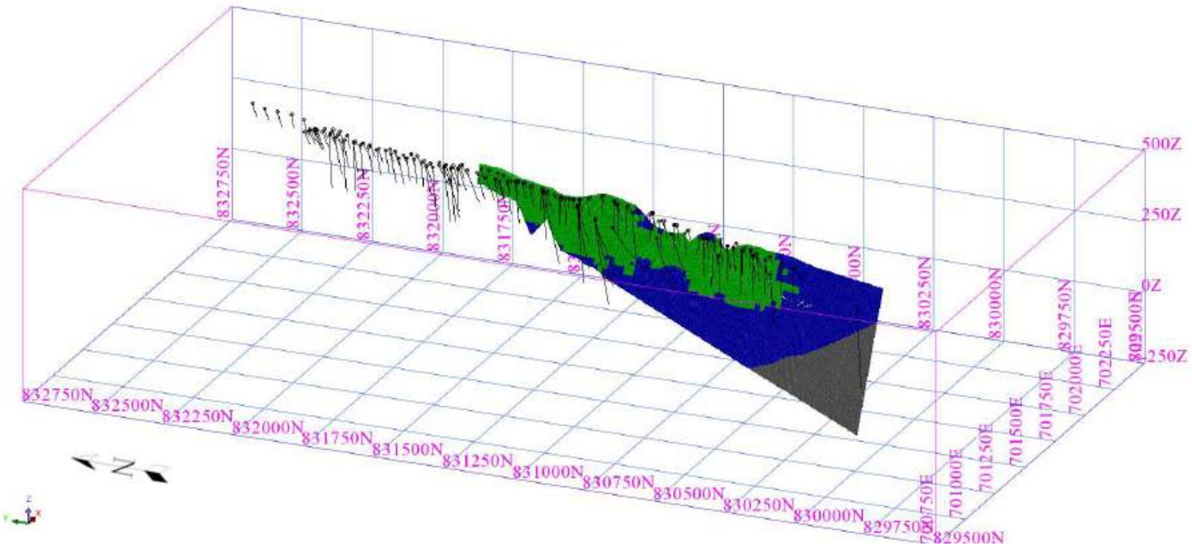

Figure 109 - Lode 200 Resource Classification (Green – Indicated, Blue – Inferred, Grey – Mineral Potential)

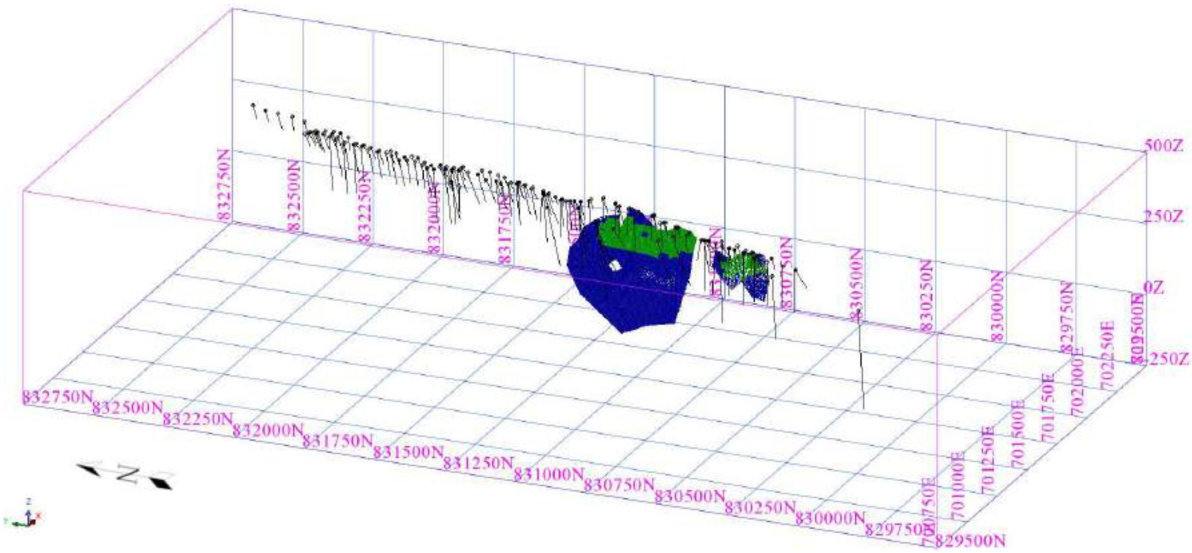

Figure 110 - Lode 300 Resource Classification (Green – Indicated, Blue – Inferred, Grey – Mineral Potential)

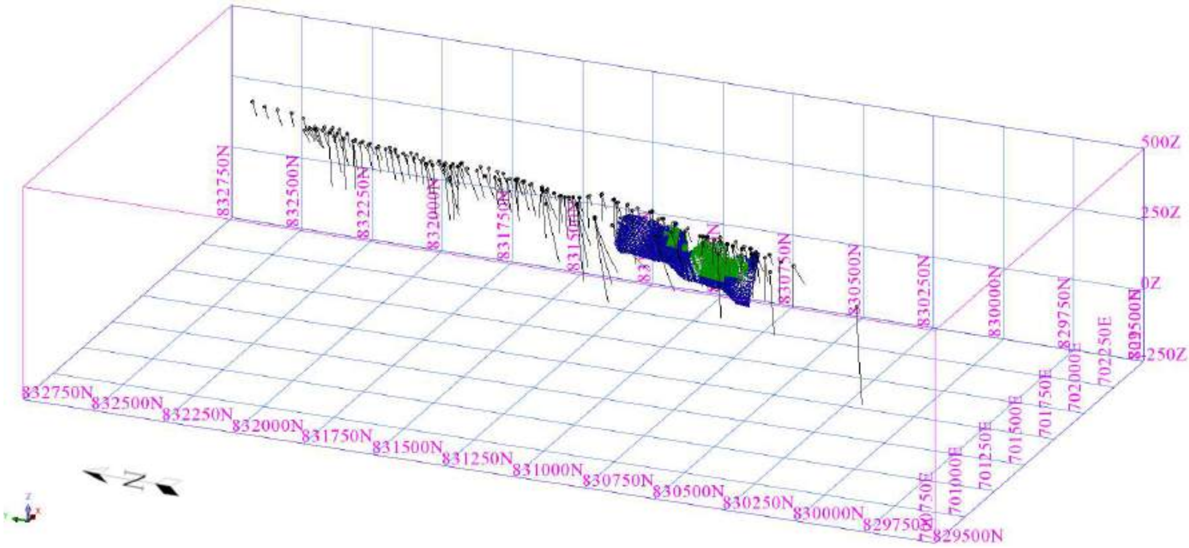

Figure 111 - Lode 400 Resource Classification (Green – Indicated, Blue – Inferred, Grey – Mineral Potential)

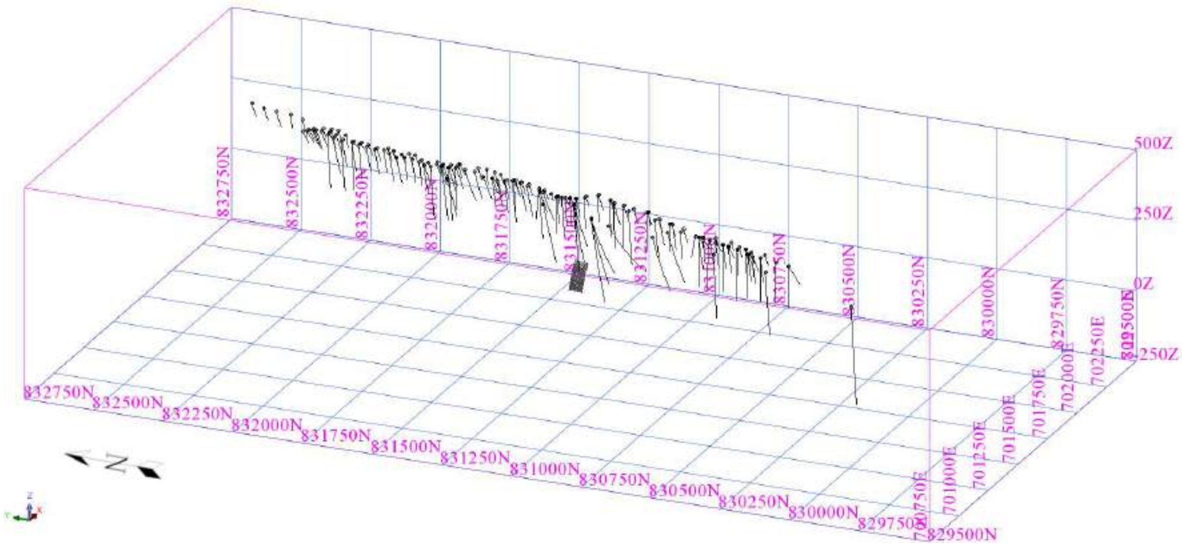

Figure 112 - Lode 500 Resource Classification (Green – Indicated, Blue – Inferred, Grey – Mineral Potential)

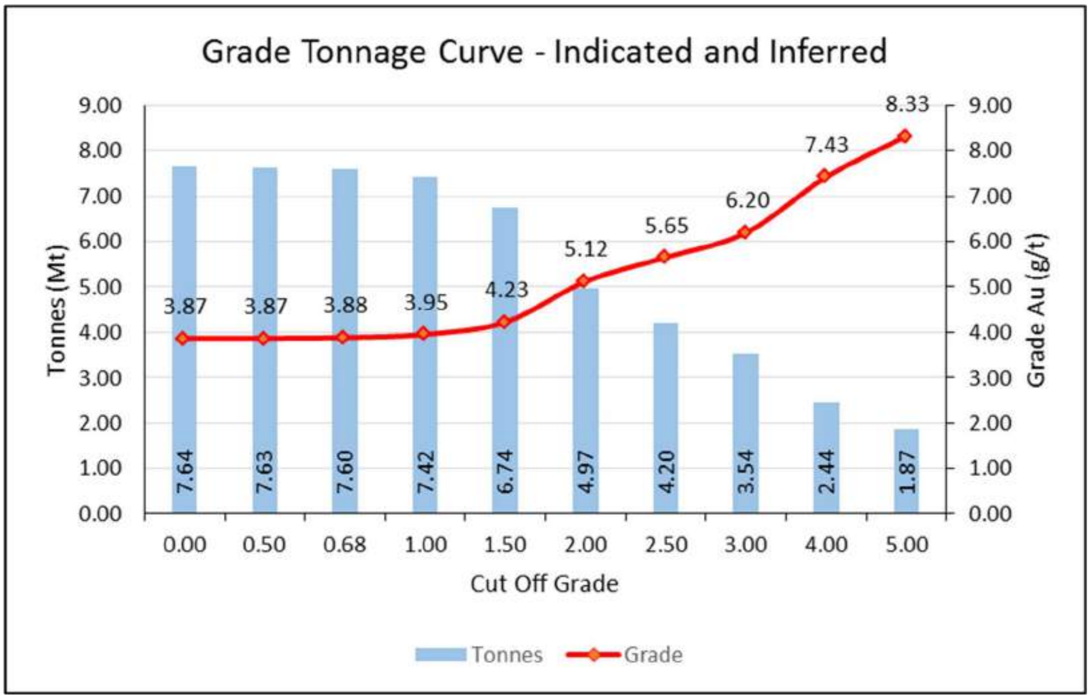

Figure 113 - Grade Tonnage Curve - Indicated & Inferred

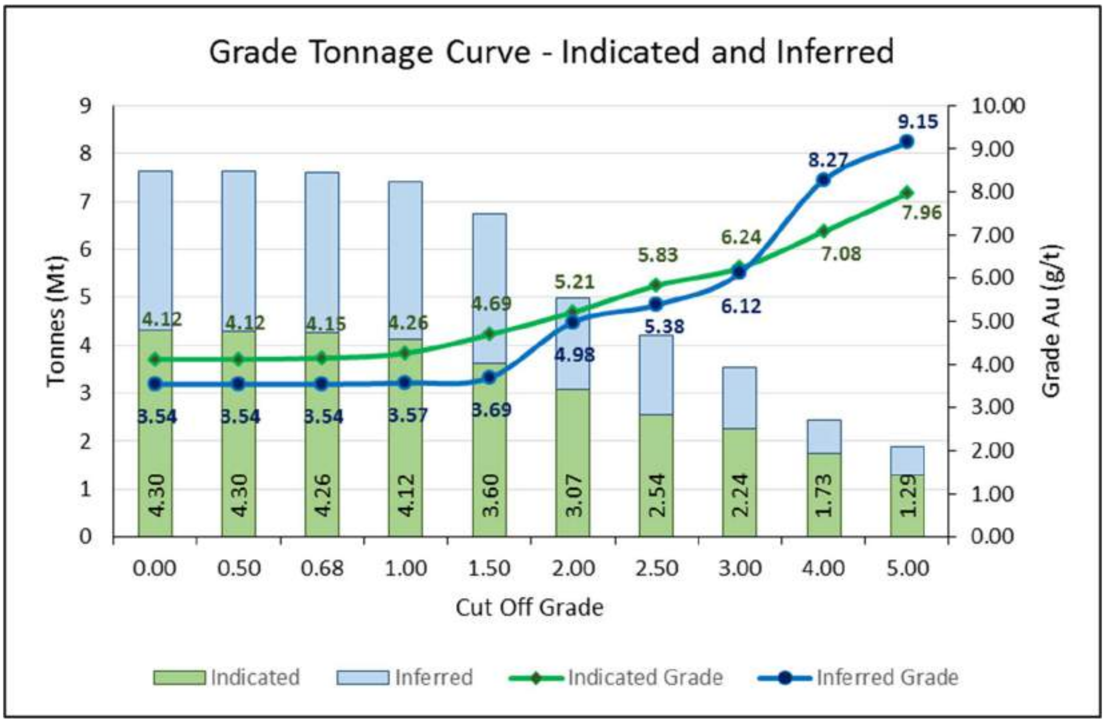

Figure 114 - Grade Tonnage Curve - Indicated & Inferred

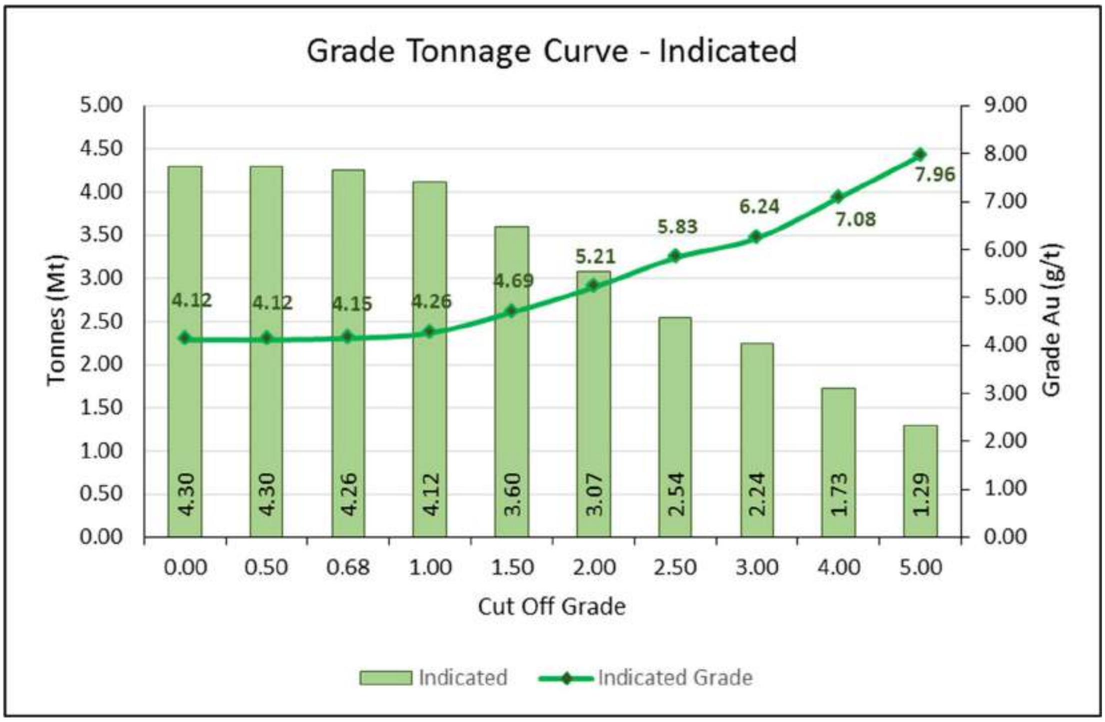

Figure 115 - Grade Tonnage Curve – Indicated

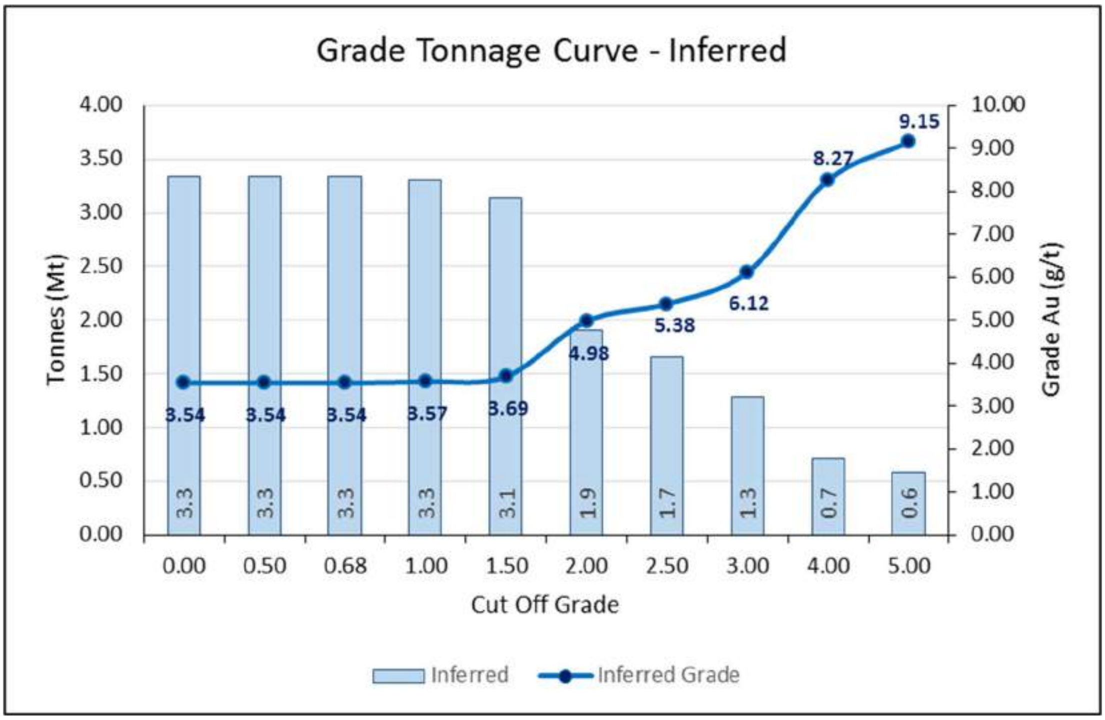

Figure 116 - Grade Tonnage Curve - Inferred

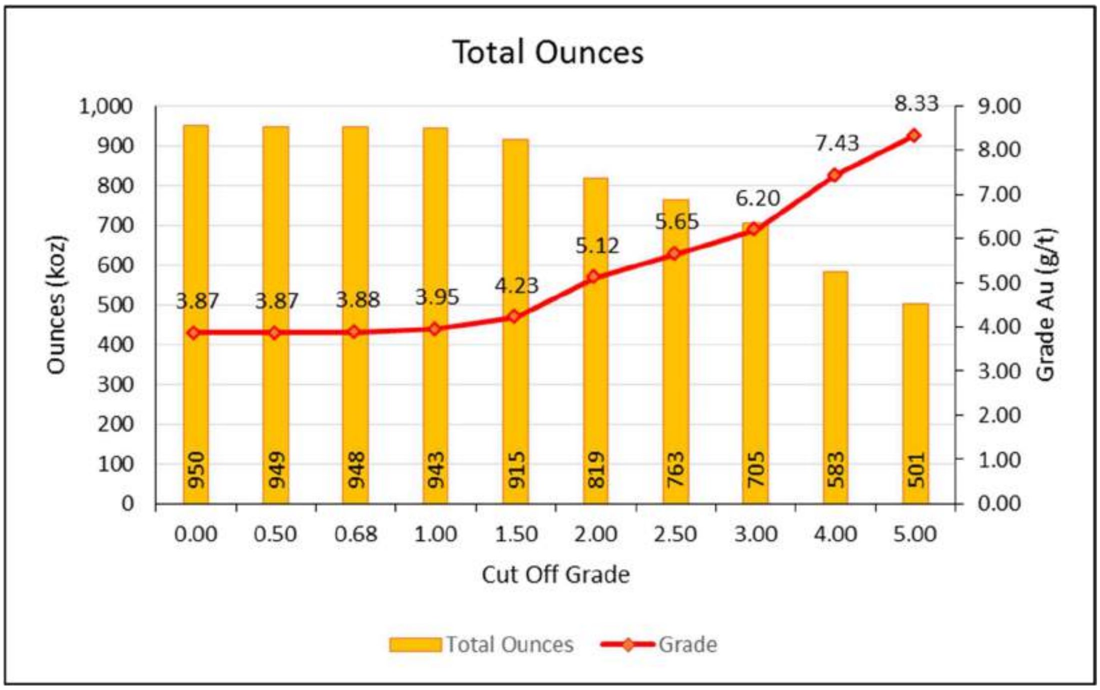

Figure 117 - Total Ounces - Indicated & Inferred

Total Ounces - Indicated & Inferred

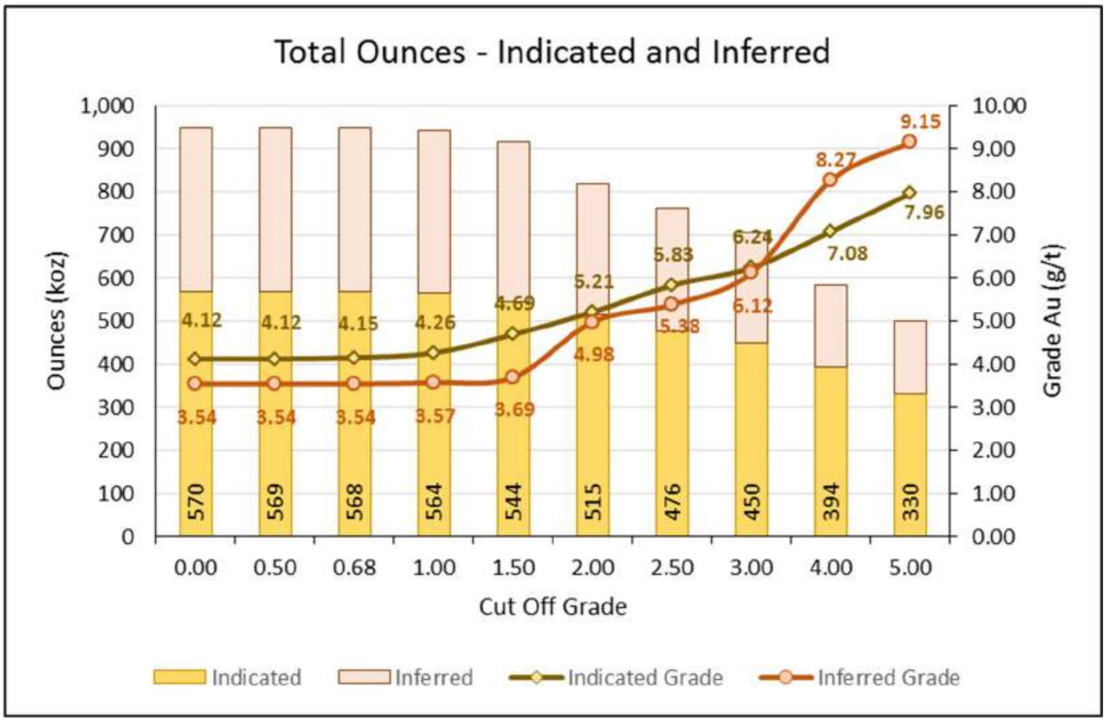

Figure 118 - Total Ounces - Indicated & Inferred

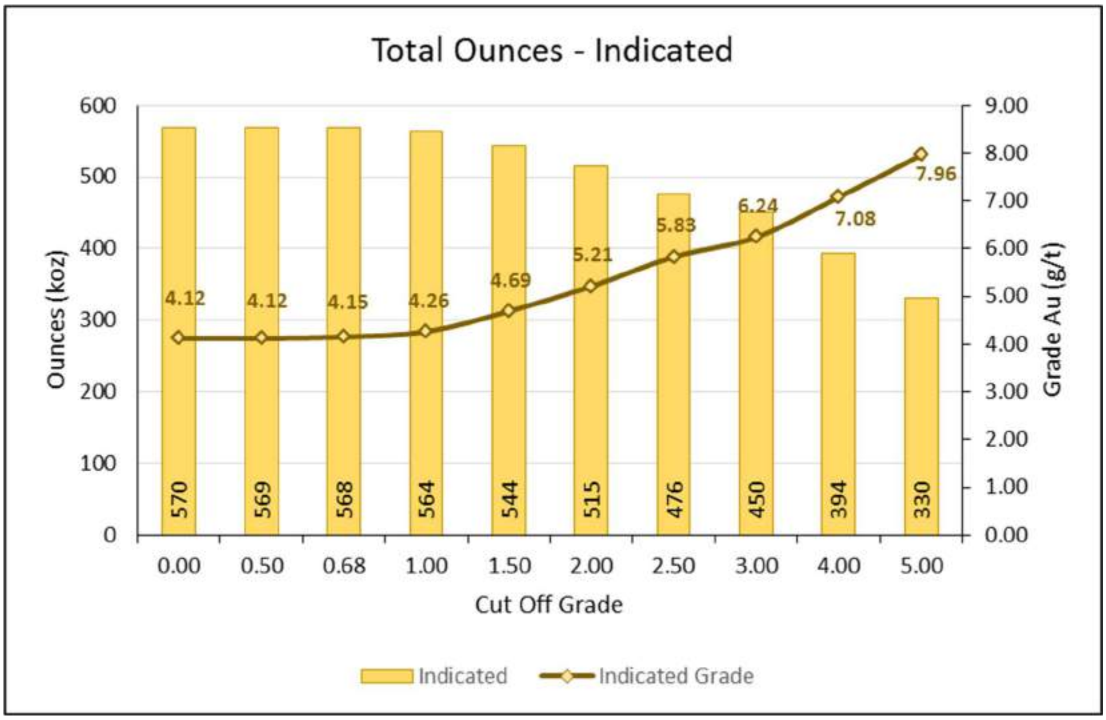

Figure 119 - Total Ounces – Indicated

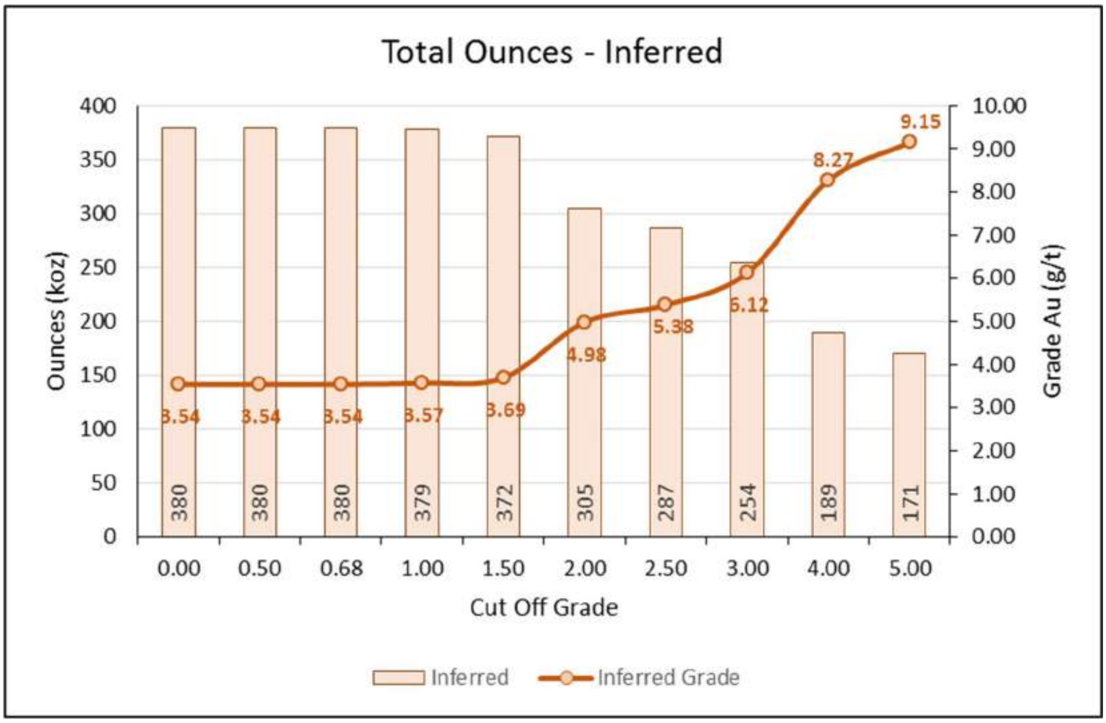

Figure 120 - Total Ounces – Inferred

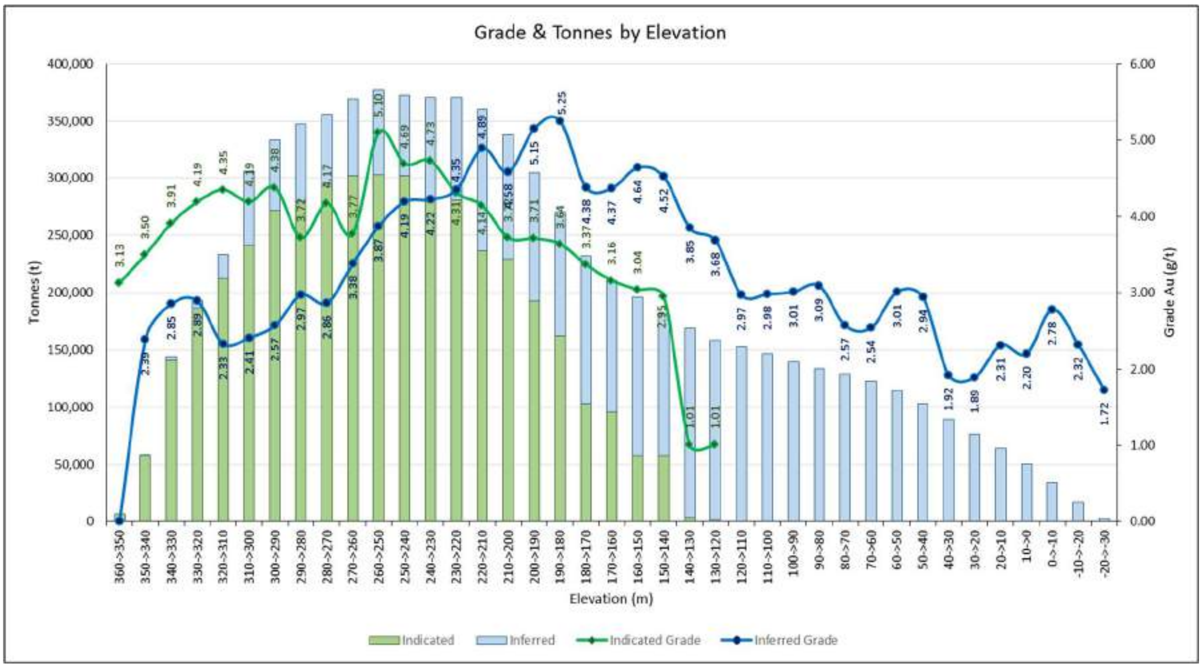

Figure 121 - Grade & Tonnes by Elevation. Reported at a 0.5g/t Au Cut off

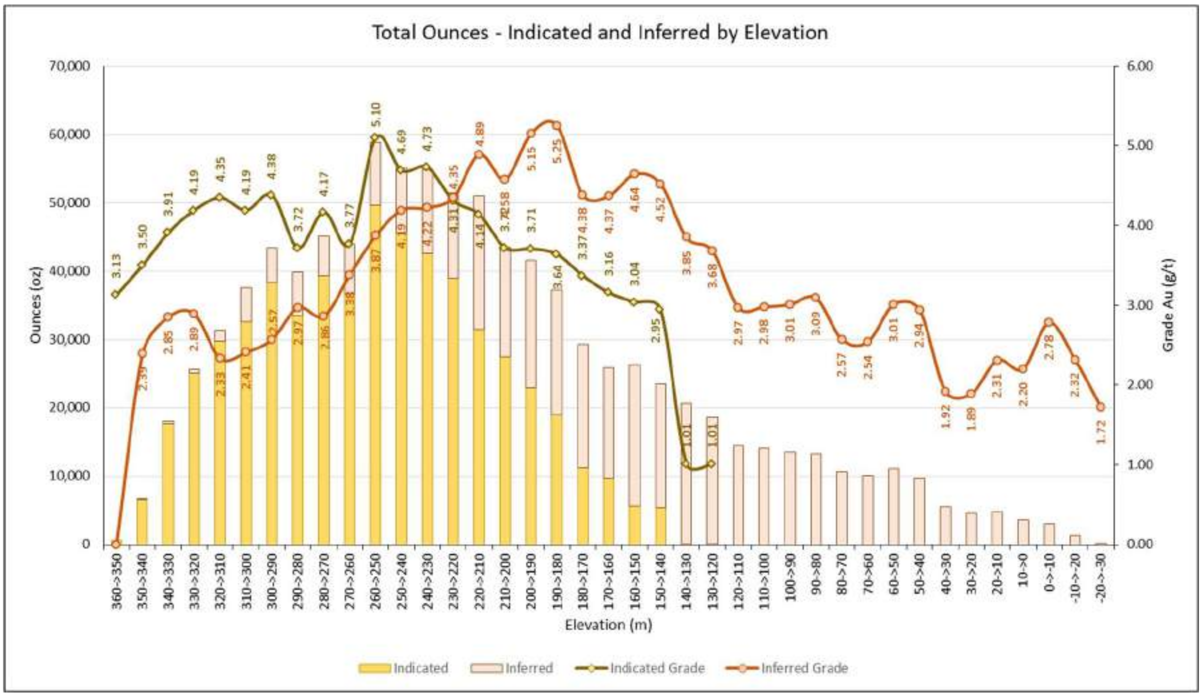

Figure 122 - Total Ounces by Elevation. Reported at a 0.5g/t Au Cut off

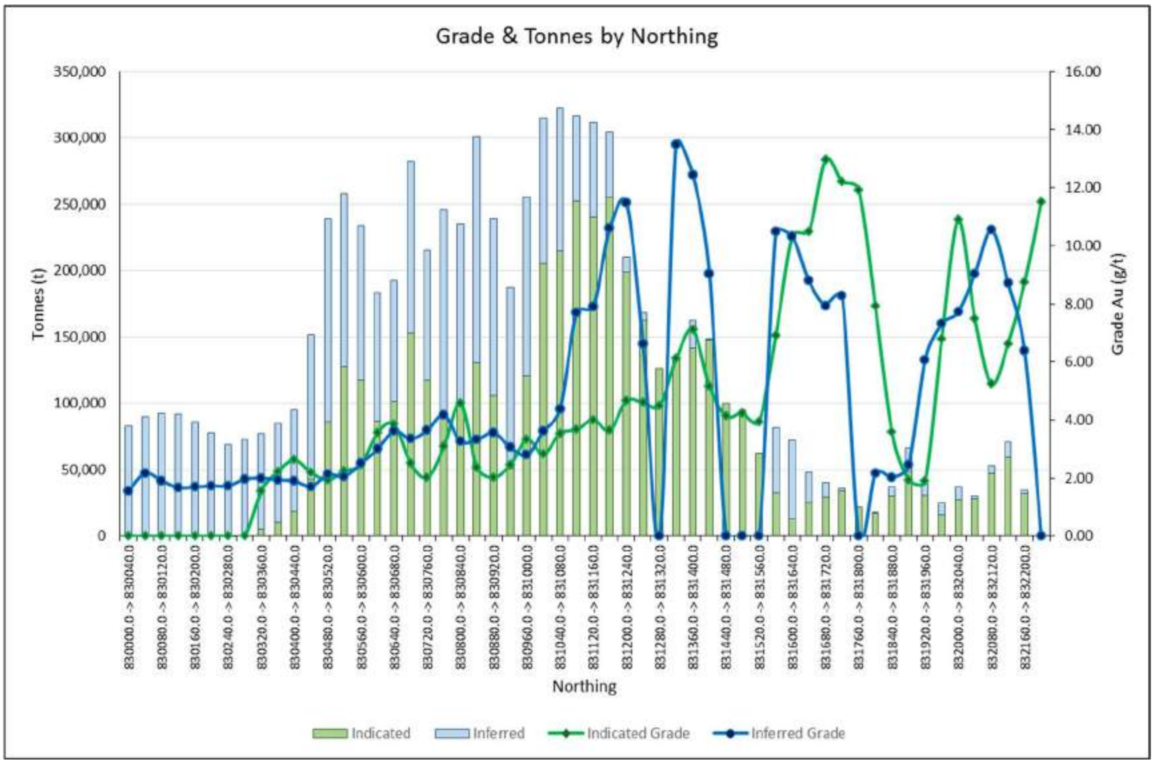

Figure 123 - Grade & Tonnes by Northing. Reported at a 0.5g/t Au Cut off

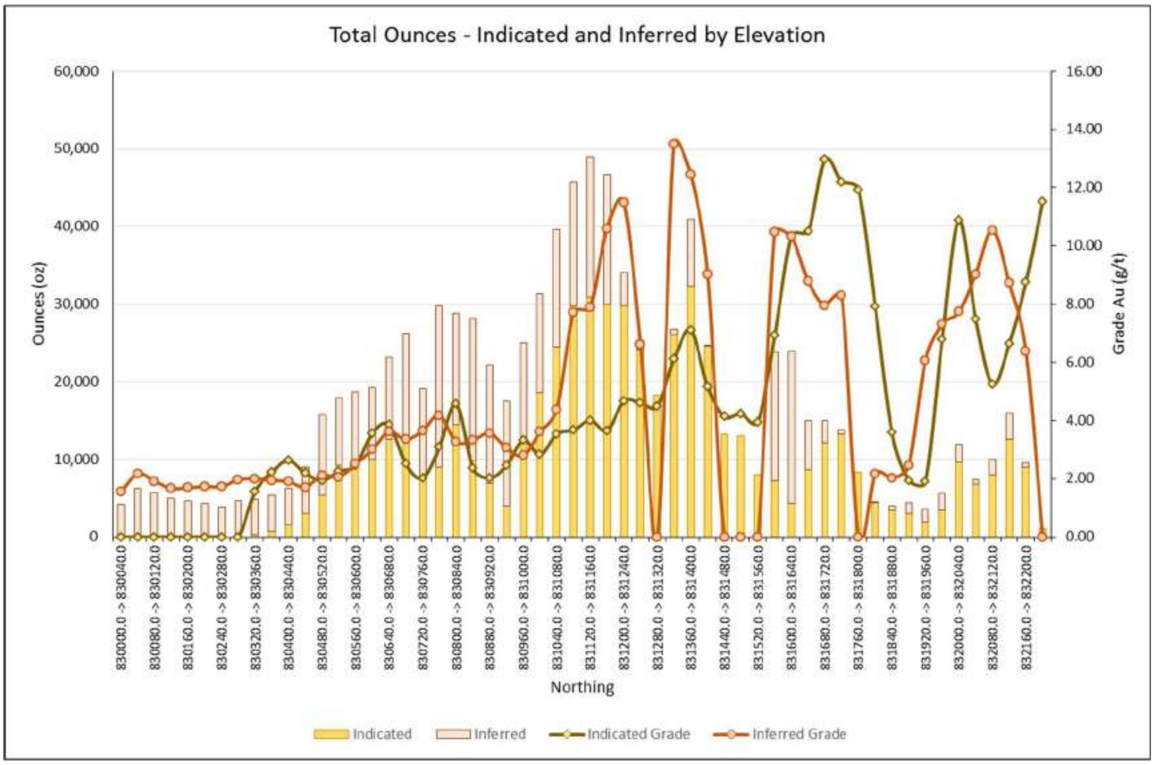

Figure 124 – Total Ounces by Elevation. Reported at 0.5g/t Au Cut-off

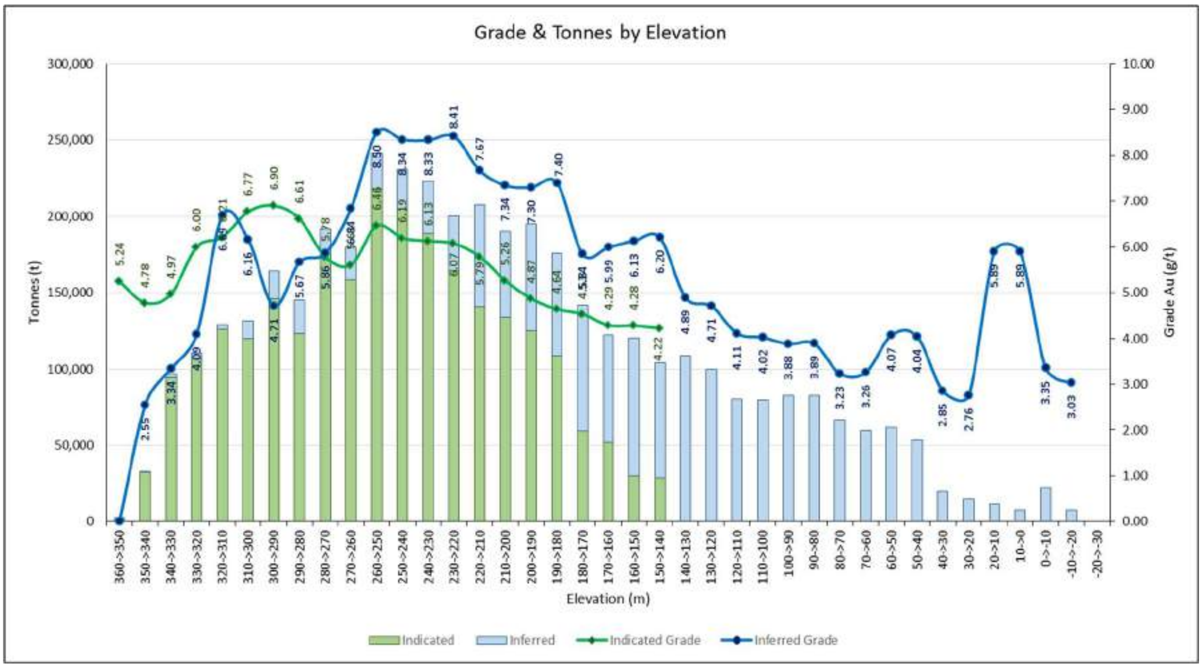

Figure 125 - Total Ounces by Elevation. Reported at a 2.50g/t Au Cut off

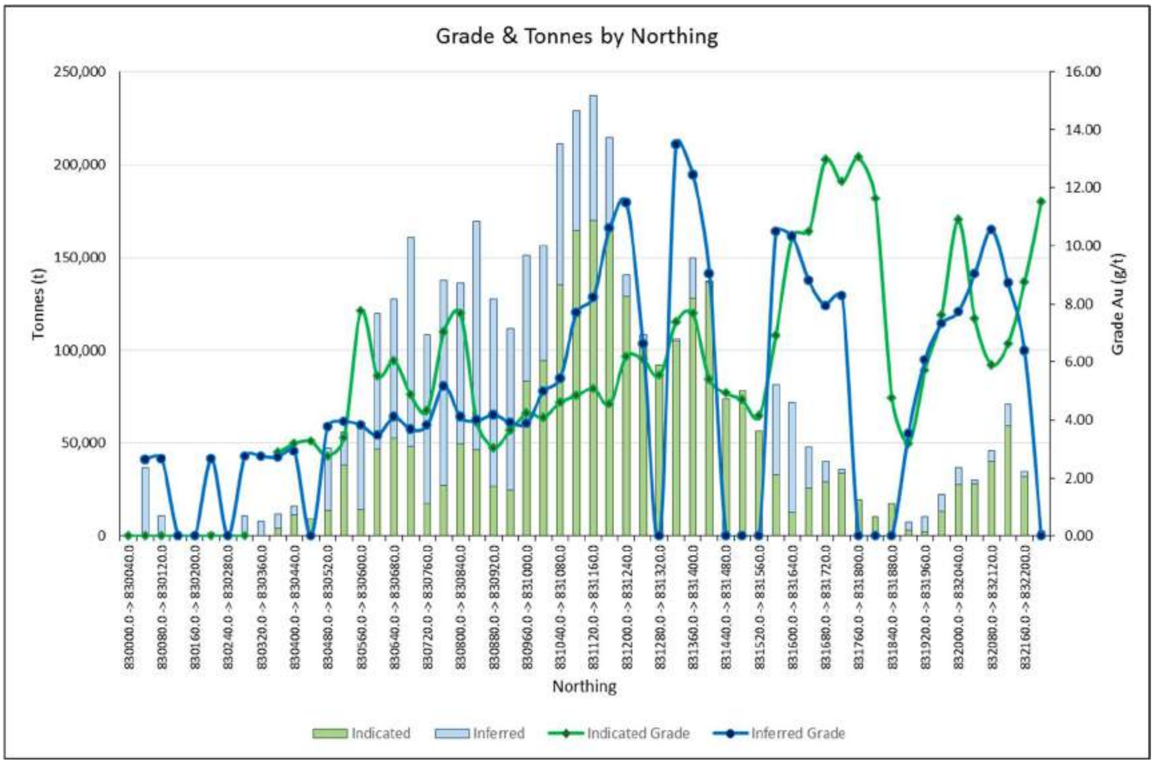

Figure 126 - Grade & Tonnes by Northing. Reported at a 2.50g/t Au Cut off

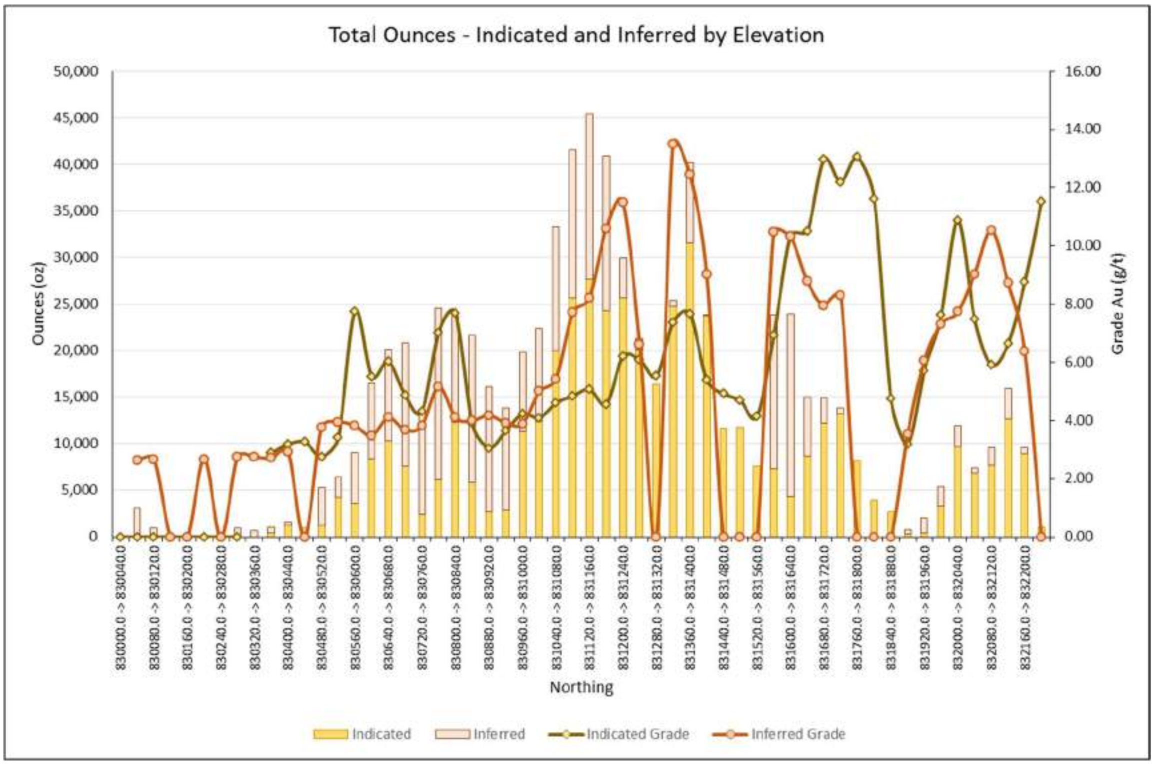

Figure 127 - Total Ounces by Northing. Reported at a 2.50g/t Au Cut off

MS Analytical
Unit 1, 20120 102nd Avenue,
Langley, BC
V1M 4B4

August 10, 2017

1. Introduction

A total of 1373 samples were submitted to MS Analytical including core and client inserted pulps between May and July 2017. All samples were assayed for gold by fire assay, aqua regia digestion and AAS finish and every 10th sample was analyzed for specific gravity.

Job #	# Assay	Received	Project	Prep Methods	Analytical Methods
YVR1710408	61	02/05/2017	Segilola Gold Project	PRP-910	FAS-221
YVR1710425	42	11/05/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410
YVR1710427	102	12/05/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410
YVR1710439	86	17/05/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410
YVR1710467	118	24/05/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410
YVR1710505	141	07/06/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410
YVR1710506	21	07/06/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410
YVR1710513	45	09/06/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410
YVR1710514	135	09/06/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410
YVR1710538	171	19/06/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410
YVR1710562	115	26/06/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410
YVR1710588	105	05/07/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410
YVR1710623	80	19/07/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410
YVR1710625	92	19/07/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410
YVR1710645	59	24/07/2017	Segilola Gold Project	PRP-910	FAS-221, SPG-410

2. Sample Preparation

All samples were weighed upon receipt (method code PWE-100). Core was dried, crushed to 70% passing 2mm, split (250g sub-sample) and pulverized to 85% passing 75µm (method code PRP-910). Two preparation blanks were inserted in each job as well as preparation duplicates at a rate of 1 per 30 samples.

2.1. Preparation Blanks

Preparation blanks are used to monitor contamination in the sample preparation process. All results are within the acceptable limit of 0.03 ppm for gold.

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Langley, BC
V1M 4B4

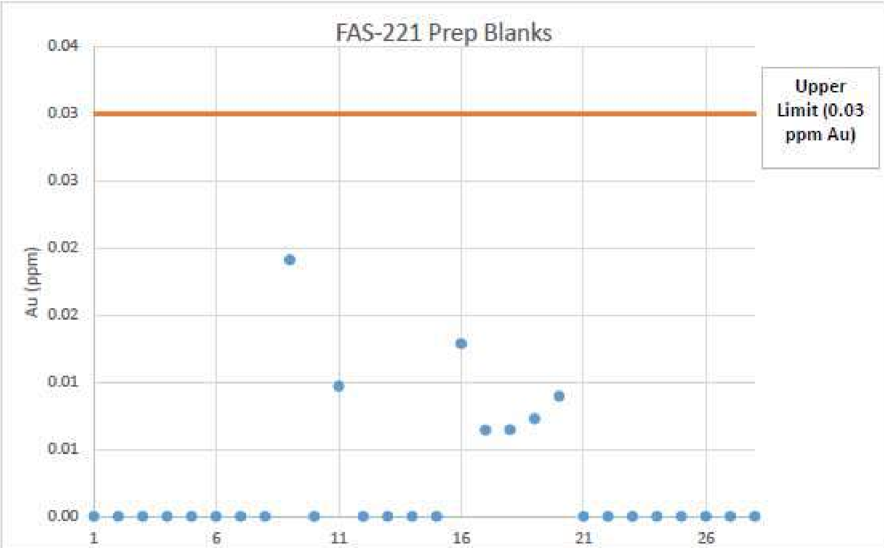

Figure 2-1 Control Chart for Preparation Blanks

2.2. Preparation Duplicates

Preparation duplicates are split after the crushing stage and are denoted by 'PD' following the sample name. The results of the preparation duplicates are plotted against the original results below. The calculated R^2 value is 0.9952.

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Langley, BC
V1M 4B4

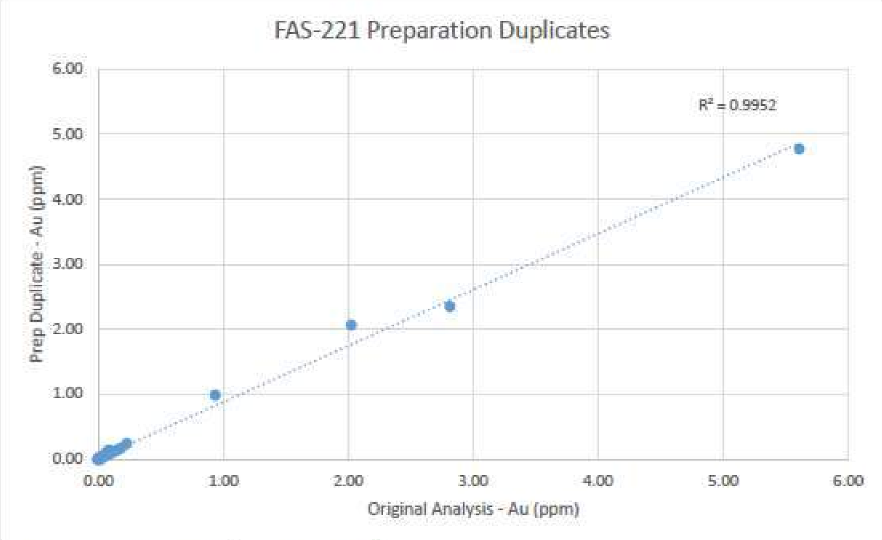

Figure 2-2 Preparation Duplicates vs. Original

3. Sample Analysis

Samples were analyzed for gold by lead collection fire assay followed by AAS finish (method code FAS-221). Any Au assays > 10ppm were re-analyzed by 1000g metallic screening (method code MSC-150). Every 10th core was tested for specific gravity by water displacement (method code SPG-410).

3.1. Method Blanks

Method blanks are inserted in every batch at a rate of 2 per 36 samples and are used to monitor contamination in the process. Results are within the acceptable limit of 0.03 ppm for gold by method FAS-221.

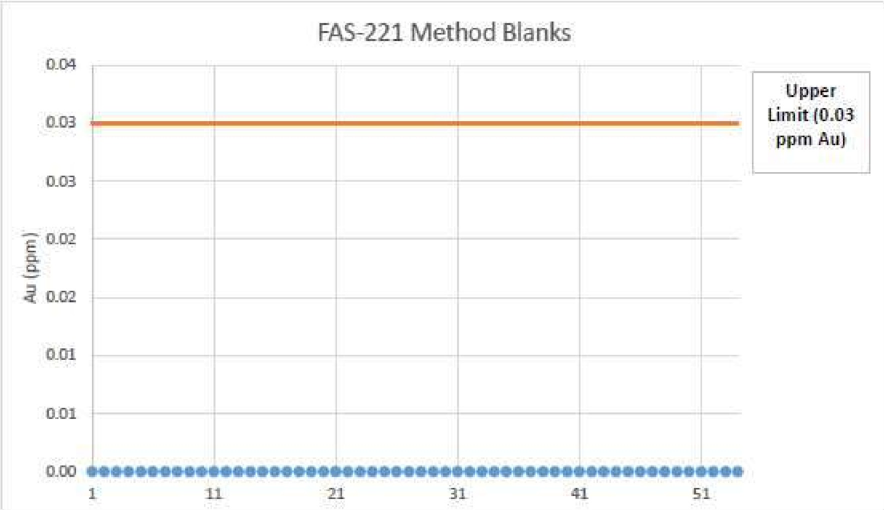

Figure 3-1 Control Chart for Method Blanks

3.2. Certified Reference Materials

Certified reference materials (CRMs) are inserted at random in every batch at a rate of 2 per 36 samples and are used to monitor the accuracy of the process. Control chart limits are set based on the certified value, precision and detection limit for the specified analyte and method. CRM results are plotted below and are all within the control limits.

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V1M 4B4

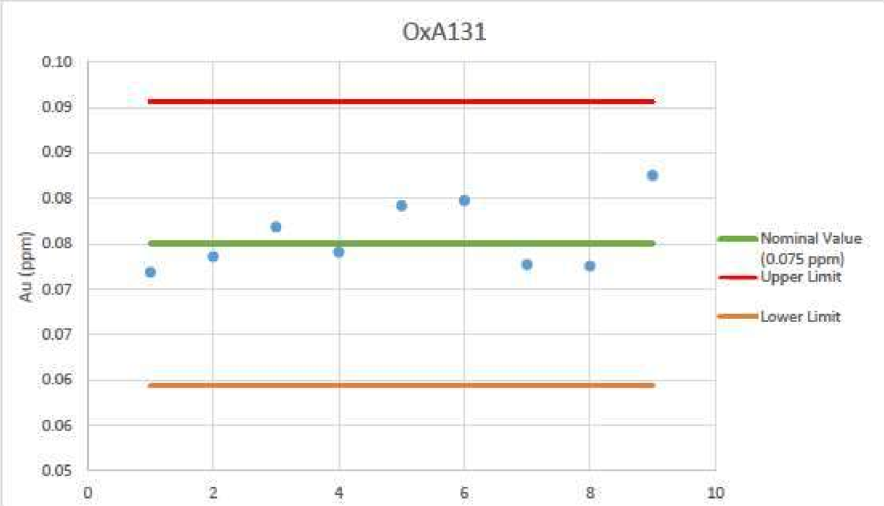

Figure 3-2 Control Chart for CRM OxA131

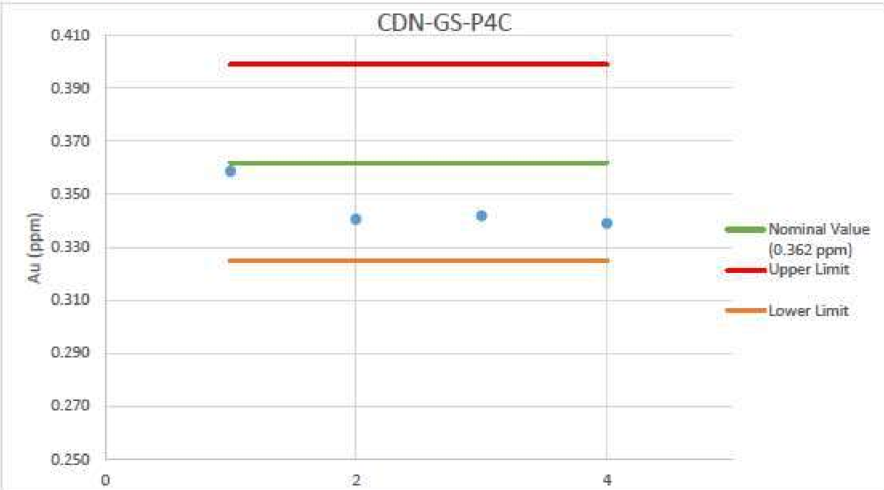

Figure 3-3 Control Chart for CRM CDN-GS-P4C

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Unit 1, 20120 102nd Avenue,
Langley, BC
V1M 4B4

Figure 3-4 Control Chart for CRM OxH122

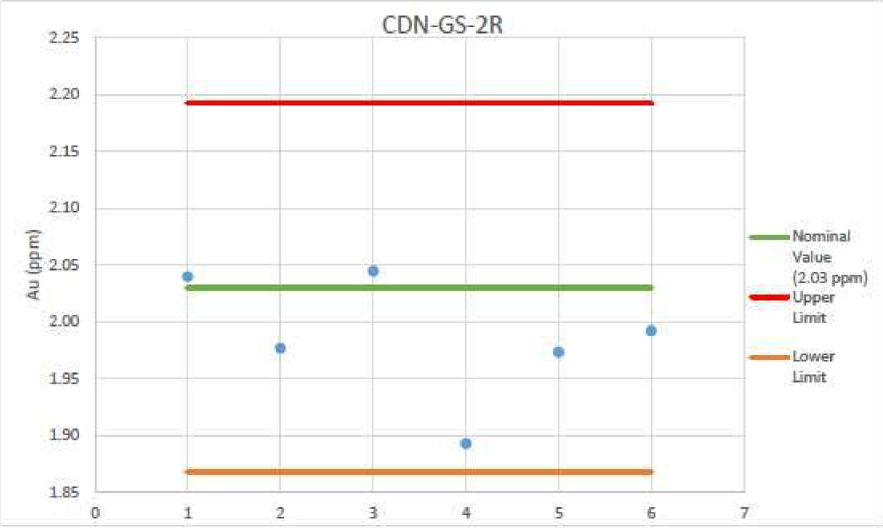

Figure 3-5 Control Chart for CRM CDN-GS-2R

MS Analytical

MS Analytical
Unit 1, 20120 102nd Avenue,
Langley, BC
V1M 4B4

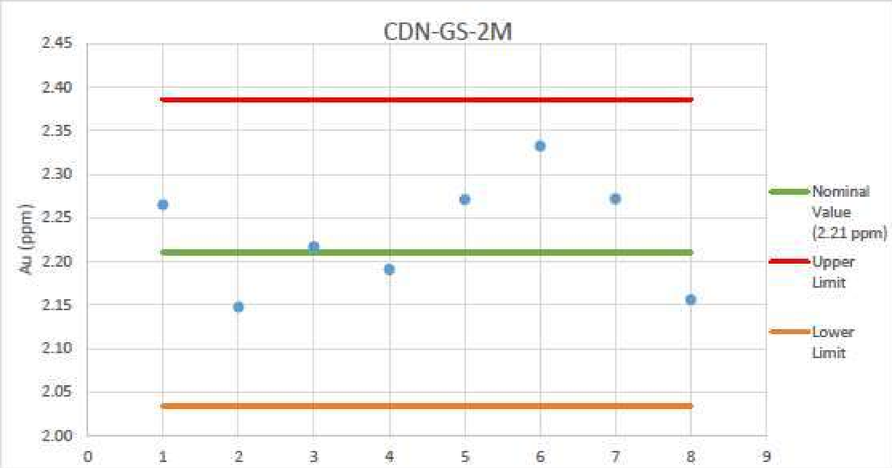

Figure 3-6 Control Chart for CRM CDN-GS-2M

Figure 3-7 Control Chart for CRM CDN-GS-5T

3.3. Analytical Duplicates

Analytical duplicates are taken at the weighing stage and are used to evaluate precision. The results of the analytical duplicates are plotted against the original results below. The calculated R^2 value is 0.9999.

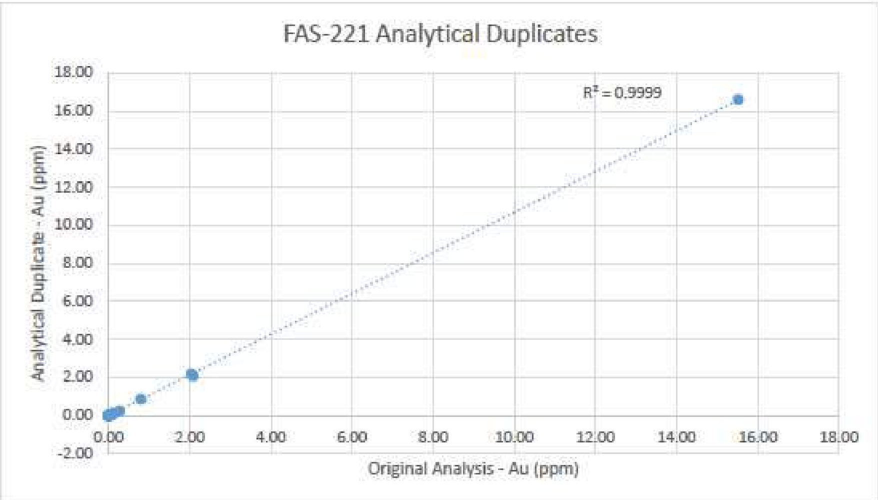

Figure 3-8 Analytical Duplicates vs. Original

APPENDIX 2 – GEOTECHNICAL

PETER O'BRYAN *& Associates*
consultants in mining geomechanics

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PO Box 1157 WEST LEEDERVILLE WA 6901
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A division of WBG *Pty Ltd*
ABN 94 082 091 236

THOR EXPLORATIONS LTD

SEGILOLA GOLD PROJECT

PRE-FEASIBILITY GEOTECHNICAL ASSESSMENT
SOUTHERN, CENTRAL & NORTHERN PITS

REPORT 17053C

Prepared for:	Prepared by:
Thor Explorations Ltd 1st Floor 167 Broadhurst Gardens London, UK, NW6 3AU	Chris Langille Peter O'Bryan **12 September 2017**

In association with:
Northwind Enterprises Pty Ltd Consulting Geotechnical Engineers
Peter Clifton & Associates Consulting Hydrogeologists

Executive Summary

This study provides a review of the preliminary analysis and recommendations provided by George Orr & Associates (GOA, 2010), and should be considered in conjunction with the NI43-101 Report for Thor Explorations (Gillman, et.al, 2016), the PFS report GOA and Peter O'Bryan &Associates (POB&A) technical notes (Langille, et.al. 2017, 2017A) completed as part of the review.

The fundamental requirements of the review were to:

- Assess whether the GOA report would constitute a PFS level mining geotechnical assessment sufficient to advance the project to a full (Bankable) Feasibility Study
- If not, provide recommendations for work to bring the project to PFS level
- Identify and recommend work required to bring the project to PFS level for a Bankable Feasibility Study

A preliminary optimised pit shell was provided by Thor Explorations (*20170712_basecase_s38_rf1-04_draft.dtm*)

The designs provided in the GOA report were considered appropriate for PFS level; however, the following gaps exist in the design analysis:

- The pit shells analysed in the GOA report were for a 140 m deep southern pit and shallow central and northern pits, whereas the proposed Segilola southern pit extends to 220 m depth.
- There was a lack of information directly relevant to the west wall slope.
- While the east wall was drilled more comprehensively, few holes extended beyond ~ 10m to 15m beyond the proposed slope position.
- All holes were drilled on an azimuth perpendicular to the lodes, thus any cross-cutting structures will be underestimated.
- The folding mentioned in the GOA (2010) report is not accounted for explicitly
- The hydrogeology conditions have not been fully investigated.
- Intact rock and defect strengths were assumed based upon observation and experience.

  Preliminary results were provided by Thor Explorations for UCS testing; however, direct shear testing has not been completed on critical defects, nor has significant testing been completed in the footwall calc-silicate unit at this stage.

The GOA report provides adequate design justification for the proposed pits, although the west wall design, given limited data, is likely conservative.

Additional drilling in the west wall of the pit shell was analysed to further validate the PFS design, based on data from three (3) holes located in the southern pit and a fourth hole for the west of the northern pit.

Based on our analysis conducted on data extrapolated into the west wall for the southern pit, there is no defendable justification for modifying the initial slope designs and the slope design parameters in GOA (2010) are considered appropriate for a PFS level design.

Summary of Final South Pit Design Parameters (after GOA, 2010)

	East Wall (Footwall)	**West Wall (Hanging wall)**	**North & South End Walls**
Upper batter in Weathered Rock *320-300mRL			
Face angle	55°		
Face height	10		
Berm width	5		
300-180mRL			
Face angle	60°	70°	70°
Face height	20	20	20
Berm width	8^	8^	8^
^160, 200 and 240mRL Berms	12	12	12
Overall Wall angles (pit crest to toe exclusive of ramps)	45°	52°	52°
*This is the inferred pit crest, locally this may be lower or higher and slope designs will need to be adjusted relative to the weathered depth and topography			

The results and base case slope design parameters do not change, and the approach suggested by GOA to use starter or staged pits is endorsed and recommended.

The slope design parameters provided may be used to for preliminary mining evaluation.

While the designs provide a sound foundation it is possible that modifications may be applied at Feasibility Study Stage following analysis of data obtained from proposed investigations, for example, in the upper west wall, where the foliation intensity decreases further to the west, there may well be scope for steepening the slope. Conversely, the lower west wall and lower east wall may require modification to address increased foliation fabric development

All assumptions and caveats presented in the GOA (2010) report were considered applicable to the project.

Recommended Actions for Feasibility Study

In order to bring the project to a full (Bankable) Feasibility Study stage, the following additional work is considered necessary:

- Additional off-section geotechnical holes in the hangingwall (West Wall) and footwall (East Wall) are recommended to define the extent and frequency of E-W steeply dipping structures and rock mass and structure within the pit walls.
- The proposed drilling is summarised as follows:

Proposed Geotechnical Drill Programme, Definitive Feasibility Study

HOLE-ID	Easting m	Northing m	RL, m	Azimuth	Plunge	Hole Length, m
GTFS17-001	701500	831150	370	090	-55	Option
GTFS17-002	701500	831150	370	045	-45	280
GTFS17-003	701500	831150	370	140	-45	280
GTFS17-004	701725	831100	340	090	-55	250
GTFS17-005	701795	831100	340	035	-50	170
GTFS17-006	701795	831100	340	145	-50	170
GTFS17-007	701655	831475	355	140	-55	200
GTFS17-008	701900	831440	340	235	-45	175
GTFS17-009	701765	830750	330	330	-45	175
			Total			**1670 m**

To achieve best quality drilling and core recovery use of triple tube core barrels is considered essential.

- Hydrogeological studies are required to identify the potential and impact of pressurised groundwater occurring along faults/shears and fracture networks. It is understood a consultant has been commissioned to conduct a preliminary desktop study.
- Hydrological studies to identify surface drainage conditions associated with seasonal watercourses and define appropriate surface drainage systems.
- Shear strength testing of critical structures and additional intact rock strength testing of selected representative samples from the footwall calc-silicate rock unit.
- A site visit to review local conditions, topography, water and groundwater, and specifically to inspect drill core, assess logging processes, check log selected core intervals and select samples for rock properties testing.
- A particular goal of the visit is to define the extent and frequency of E-W steeply dipping structures, aiming to confirm/ extend the database and, if present, identify and characterise additional structures not detected in the geotechnically logged holes within the pit shell.
- Attaining more confidence in the geotechnical environment and validating and modifying the design will be the objective of the Feasibility Study geotechnical program.

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Appendix B.	Stereonet Plot of borehole data, by hole and lithology
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1.0 Introduction

Peter O'Bryan & Associates (POB&A) was commissioned to conduct a review and analysis of the geotechnical design for the Thor Explorations Ltd Segilola Open Pit Gold Project.

The work was commissioned by Mr Alfred Gillman, Exploration Manager for Thor Explorations Ltd and was conducted by Mr Chris Langille, of Northwind Enterprises Pty Ltd in association with POB&A. This report provides detail on our analysis, findings and recommendations for *base case* pit slope designs for the proposed Segilola Gold Project in Nigeria, and is considered an acceptable level for a Pre-feasibility Study.

This report should be considered in conjunction with the NI43-101 Report for Thor Explorations (Gillman, et.al, 2016), the PFS report completed by George Orr and Associates (GOA, 2010) and POB&A technical notes (Langille, et.al. 2017, 2017A) completed for Thor Explorations as part of this review.

2.0 Background

2.1 Scope of Work

The scope of this stage of the study was to incorporate data from structural and geotechnical logging of three (3) oriented boreholes located within the pit shell (SGD155, 156 and 157 – Figure 1, 2 and 3), but providing partial data coverage for the west wall (hangingwall). Other holes drilled as part of the 2016-17 campaign, but not oriented, were included in this assessment for RQD and summary geotechnical logging information. The drill holes and data used in this analysis are summarised in Table 1.

The purpose of this review was to establish the geotechnical design to a Pre-feasibility Study (PFS) level when incorporated with the design proposed in the GOA report. A fourth oriented diamond cored hole, SGD170, drilled to the west of the northern pit, was considered in assessing conditions in the west wall of the smaller proposed northern pit. Locations of the 2016-17 programme geotechnical holes are shown in Figure 1. The preliminary optimised pit shell was provided by Thor Explorations (20170712_basecase_s38_rf1-04_draft.dtm).

2.1.1 Summary of Work Conducted To Date

The designs provided in the GOA report were considered appropriate for a PFS level study, however, the following were considered gaps in the design analysis:

- There was a lack of information directly relevant to the west wall slope, as most of the drilling conducted prior to the current drill program focused on the east wall and ore definition
- While the east wall was drilled more comprehensively, there were few holes extended beyond ~10m to 15 m into the proposed design slope
- All holes were drilled on an azimuth perpendicular to the lodes, thus any cross-cutting structures will be underestimated. While cross-cutting structures individually are not considered to be potential failure planes, they do act as release planes and will contribute to wedge, planar, and unravelling failure potential
- The folding mentioned in the GOA report is not accounted for explicitly and will need to be confirmed as part of the mine plan implementation
- The hydrogeology conditions were not fully investigated and will need to be in order to finalise the design to Feasibility Study level
- Intact rock strength results were received on samples selected by Thor Resources, however, defect strengths have been assumed based upon observation and experience and not tested in the laboratory. The assumed defect shear strength values used in the GOA report are considered appropriate for PFS designs.

2.1.2 Review Objectives

The fundamental requirements of this review were to:

- Assess whether the GOA report would constitute a PFS level mining geotechnical assessment sufficient to advance the project to a full (Bankable) Feasibility Study
- If not, provide recommendations for work to bring the project to PFS level and complete the recommended analysis and reporting
- Identify and recommend work required to bring the project from PFS level to a Bankable Feasibility Study

2.2 Geology

The Segilola Gold Project is located in the crystalline basement complex rocks of south-western Nigeria within one of the main "schist belts" known as the Ilesha Schist Belt. Schist belts in Nigeria occur as north-south trending domains of Upper Proterozoic metasedimentary, meta-volcanic and intrusive sequences that are oriented parallel to the boundary between the West African Craton and the Pan African province. The Ilesha Schist Belt and associated rocks are thought to be a continuation of the gold-bearing shear zones of the Borborema province located in north eastern Brazil. The mineralized lodes generally comprise highly silicified fine-grained foliated biotite gneiss typically intruded by both discordant and concordant pegmatitic quartz-feldspar veins (Thor Exploration, 2016).

At Segilola, gold mineralisation is localised within structural "compartments" defined by the intersection of two main controlling features: a westerly-dipping footwall calc-silicate suite of rocks and sub-vertical shear zones. The divergence of these structures towards the south creates a shallow south plunging structural compartment in which the gold lodes are developed (Thor Exploration, 2016).

The biotite gneiss tends to be more massive the further west from the mineralised horizon in the sequence, and becomes gradually more foliated as the mineralised shears and calc-silicate footwall contact is approached.

2.3 Data Sources

The sources of data for this study included:

- Prior report conducted by GOA (2010) and preliminary Technical Notes by POB&A (Langille, et.al., 2017, 2017A)
- Structural and geotechnical logging from four (4) oriented HQ resource diamond cored holes (Table 1), all orientations are assumed to be measured with respect to mine grid north (approx. 15° east of Magnetic North)
- Thirteen (13) HQ resource holes which (Table 1) were not oriented, however summary geotechnical data was available (recoveries, RQD, weathering, index strength, lithology)
- Drill core photos for fifteen (15) resource holes
- Preliminary pit optimisation shell (20170712_basecase_s38_rf1-04_draft.dtm), wireframes of lodes and lithology contact, topography and transition zone surface strings, dtms and dxf files
- Preliminary water table levels provided by Thor Explorations
- Laboratory strength test results (Unconfined Compressive Strength, UCS) on samples selected by Thor geologists (Thurber Engineering, 2017)

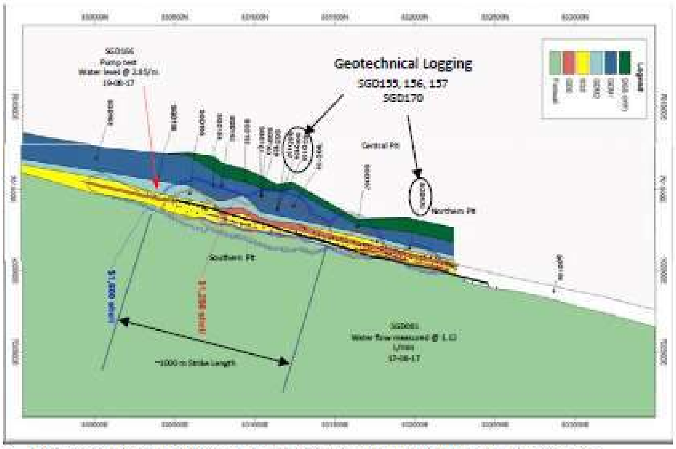

Figure 1. Location of Geotechnical holes for Southern Pit West Wall analysis

Table 1. Summary of Geotechnical Drillhole Data Sources

Hole ID	Easting m	Northing m	RL m	Hole Length, m	Azimuth *	Plunge	Data
Oriented Core							
SGD155	701624.4	831148.5	370.01	218.2	90	-60	Structural Data, RQD, core photos, UCS samples
SGD156	701624.1	831148.7	369.03	268.4	90	-72	Structural Data, RQD, core photos, UCS samples
SGD157	701624.5	831149.1	368.36	296.4	90	-80	Structural Data, RQD, core photos, UCS samples
SGD170	701863	831975	341.7	136.2	90	-70	Structural Data, RQD, core photos
Summary Geotechnical Logging, RQD, Index Strength, Weathering							
SGD153	701684.9	830820.6	329.71	85.7	180	-77	RQD, core photos
SGD154	701718.3	830921.2	344.95	47.2	180	-80	RQD, core photos
SGD158	701604.1	831250.6	362.68	270	90	-66	RQD, core photos, UCS samples
SGD159	701540.4	831049.7	364.33	287.6	90	-60	RQD, core photos, UCS samples
SGD160	701539.8	831050.4	368.05	341.3	90	-76	RQD, core photos
SGD161	701540	831050.2	368.71	302.4	90	-68	RQD, core photos
SGD162	701444.6	830947.7	392.07	332.3	90	-50	RQD, core photos
SGD163	701475.1	830803.6	362.21	290.3	90	-57	RQD, core photos
SGD164	701479.9	830750.7	385.78	265.4	90	-60	RQD, core photos
SGD165	701526.1	830604.1	315.78	242.5	90	-83	RQD, core photos
SGD166	701476.9	830398.6	300.07	242.5	90	-80	RQD, core photos
SGD167	701730	831650	345	197.4	90	-60	RQD, core photos, UCS samples
SGD168	701309	830019	300	362.5	100	-85	RQD, core photos

*Relative to Mine Grid North

3.0 Summary of Findings

3.1 Drill Core Logging & Analysis

The 2017 drill program provides improved coverage of the hangingwall (west) rock mass, particularly in the southern pit, which is proposed to be the deepest sector at ~ 220 m depth. Our initial report (Langille, et.al, 2017) had assumed the depth of the southern pit to be 140m, based upon proposals referred to in the GOA report.

The three (3) holes located in the southern pit, which were oriented and logged for geotechnical and structural data, were within the pit shell. Figures 2 and 3 show the locations of the three holes on Section 831150mN. The following notes summarise observations pertaining to the logged holes:

- While the geotechnically logged holes were located within the pit shell, given the trend of the lithological unit fabric, dipping moderately steep to the west, it was not unreasonable to project these units and associated data into the west wall. This included the GDM1, GDM2 and DGS units (Figure 2). A description of the Segilola Rock Suite, hangingwall, mine and footwall calc-silicate unit sequence was provided and is shown in Appendix Figure C-1.

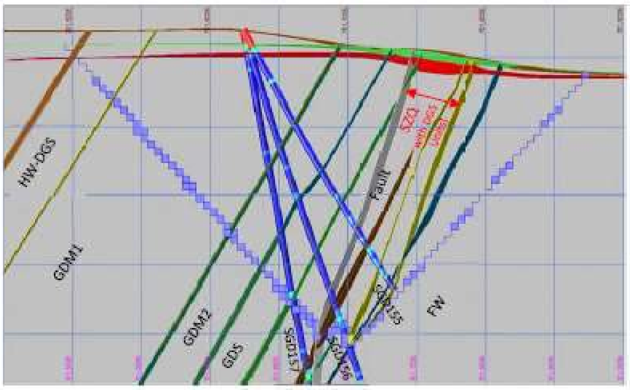

Figure 2. **Section 831150mN** showing detail of the 3 logged geotechnical holes and lithological units projecting into the west wall of the preliminary pit shell

- The holes were drilled on a 090° (relative to mine grid north) azimuth, sub-perpendicular to the deposit, so any cross-cutting structures will be missed or at least underestimated
- Actual structural data (alpha and beta angles from discrete structures) recovered from the logged holes were not complete along the length of each hole due to orientation quality or lost orientation marks (due to spun core), core loss and/ or indistinct orientations. The sections of each hole from which α and β data were obtained are shown in yellow highlight on Figure 3.
- The rock quality is more massive and less foliated the further west in the sequence, away from the ore zone, and thus a better quality rock mass – Fair to Good rock mass quality (Plate 1)
- Closer to the ore zone, increasing levels of alteration/metamorphism have resulted in increased development of gneissic texture and associated foliation, indicating potential for poorer ground conditions in the lower walls of the southern pit (Plate 2)
- As the shear zone hosting the ore lode is approached, the rock mass conditions deteriorate, however, the bulk of this rock mass will be mined out, leaving only the endwalls of the final pit where these conditions may be exposed (Plate 3)
- The footwall calc-silicate unit is more intensely altered and fractured, with zones of broken rock and strong gneissic texture/ foliation (Plate 4)

- The GOA report indicates presence of regional "…east striking and east north-east striking, dextral faults. (reported in Leishman, 2010)", which is not evident in the data other than a few discrete poles on the stereonet plots (Appendix C), although several shallow to the core axis structures exhibiting deformation/ shearing are observed in the core (as examples BH SGD160 @ 58 m, 75 m, 83 m, 101 m; SGD168 @192.5 m, 193.4-196 m, 197.5 m, 198 m213 m, 214 m), indicating that their presence as release planes may be more common (directional bias of the drilling azimuth tends to under-represent cross-cutting structures parallel to the drill section – Plate 5 shows an example)
- A NNW-SSW striking, steep west dipping fault/ shear on the hangingwall of the lodes, intersects the lower west wall. The fault is not readily evident in the core photos or logs.
- Folding reported in the GOA report is not evident in the core photos. The presence and possible impact of such structural features will likely become clear when mining commences.
- Overall, the rock mass is massive, with significant lengths of intact core sticks in the hangingwall rock, with increasing levels of metamorphism/foliation/gneissic texture, the closer to the ore lode and into the footwall – as described in the Segilola Rock Suite slide provided (that is, GDM1 to GDM2 to GDS – Appendix C – Figure C-1).

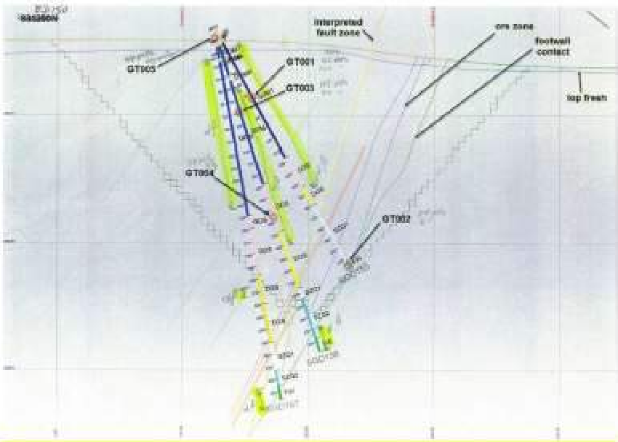

Figure 3. **Section 831150mN** showing geotechnical holes with yellow highlight indicating where good orientation marks allowed for structural orientation data to be collected

3.2 Intact rock Strength

Samples taken for Uniaxial Compressive Strength (UCS) testing are indicated on Figure 3 (and Appendix Figures A-1a to d) with the notation GT001, *et cetera*. Results provided by Thurber Engineering (2017, Appendix A) indicate intact strengths in the range of 88 to 100 MPA for typical fresh Biotite Gneiss (BG) lithology found within the GDM1, GDM2 and GDS units. The test results were lower than the assumed 100-150 MPA values for intact strength listed in the GOA report analysis.

Only two (2) test results were available for the Calc-Silicate footwall metasediment. Photos provided by Thurber (2017) of the failed samples indicated axial failures were typical, demonstrating that failures were through intact rock and not controlled by structure. Therefore, the biotite gneiss units can only be considered moderately strong intact rock.

Table 2. Summary of UCS Testing Results

Lithology	Sample Nos.	Borehole	Average UCS MPa	Comments
GDM1	GT001, 3, 6, 9, 10	SGD155, 156, 158, 159, 167	105.9	Samples failed axially
GDM2	GT007	SGD158	100.5	Samples failed axially
GDS	GT004, 8	SGD156, 158	99.0	Samples failed axially
MS	GT002	SGD155	57.5	Only sample to fail along structure
Extremely Weathered BG	GT005	SGD157	-	Sample arrived in degraded state. Testing not possible

3.3 Weathering Profile

The depth of rock weathering ranged from 11.4 to 15.6m downhole for the holes collared on Section 831150mN, but was as shallow as 0.4 m of residual soil and 6.4 m slightly weathered in SGD170 at 831970mN. The geotechnical hole collar RLs on Section 831150mN were at 360mRL, which was at a greater elevation than the pit slope design indicated in the GOA and Langille, *et.al* reports (where 320mRL was the indicated crest elevation for the *base case* pit design). Any variation between the topography and pit crest, and the depth of weathering will need to be considered in the final design for both bench height design and consistency sector to sector and from a rippability versus drill and blast perspective.

3.4 Hydrogeology, Hydrology and Rainfall

While a detailed hydrogeology study has not been completed, preliminary borehole dipping measurements were provided with initial water table depth measurements for select boreholes and is summarised in Table 3. The measurements are initial readings and it has not been determined if they reflect the general water table, or are a result of water filling the borehole through the intersection with the mineralised zone shears and related structures. A detailed hydrogeology study is necessary to determine the potential impact on pit operations and requirements for wall depressurisation during mining.

Table 3. Water Table Levels Measured in Available Boreholes (Thor Explorations, 2017[1])

Date	HoleID	Easting mE	Northing mN	Elevation mRL	Casing Height ag (m)	Measured Water level(m)	Water Table level (m)	Comment
15/8/2017	SGD155	701624.4	831148.5	370.01	1.45	62.1	60.65	
15/8/2017	SGD157	701624.5	831149.1	368.36	0.6	63	62.4	
15/8/2017	SGD158	701624.5	831149.1	368.36	1.1	51.3	50.2	
15/8/2017	SGD160	701539.8	831050.4	368.05	1.05	62.55	61.5	
15/8/2017	SGD163	701475.1	830803.6	362.21	0.8	65.55	64.75	
15/8/2017	SGD164	701479.9	830750.7	385.78	0.7	51.6	50.9	
15/8/2017	SGD166	701476.9	830398.6	300.07	0.25	2.77	2.52	Pump test to assess recharge rate
Additional holes were identified, however, they were blocked and no longer accessible								

[1] Thor Explorations provided water levels, 22-Aug-17

Re-charge of the water levels needs to be determined and the impact of significant rain events assessed as part of the DFS. A humid tropical climate predominates with a mean average annual rainfall in excess of 1430mm which is concentrated in the rainy season from March to November with a break during August (Gillman, et.al., 2016). This will result in potential accumulation of water in and around the pit, so steps to ensure proper drainage and direction of water away from the pit crest will be required. The interface between the weathered and fresh transition zone may provide a path for water flow, so establishing effective diversion ditches at the base of the weathered profile will be necessary.

4.0 Results of Data Analysis

A summary of key geotechnical data and findings are as follows:

4.1 Rock Mass Quality Assessment

The RQD values were assessed for each of the logged drill holes, as well as key boreholes within the western wall, with the following interpretation:

- The hangingwall units are considered *Good* to *Very Good* quality in terms of RQD, with values ranging from 95-100% (Figure 4 and Appendix Figure C-1a-k)
- There appear to be zones of lower RQD, several of which are assumed to be associated with lithological contacts
- The RQD values in the SZQ and Footwall calc-silicate units range from 60-95%
- There were indications of lower RQDs near the collar of hole SGD159 (Section 831050mN), which is collared closer to the western pit wall. No structural orientation data are available for this hole (Appendix Figure C-1e)
- Plots for RQD, lithology and logged interval are attached in Appendix C.

4.2 Structural Defect Analysis

Analysis of the structural data in DIPS (2017) (Figure 5) indicated the following defect orientations:

- a moderately steep, west dipping set, parallel to the gneissic texture and unit contacts of the main BG lithology
- two (2) moderately steep dipping sets, one dipping to the NE and the second dipping to the SSW. These features can act as sliding planes in either wedge failures or direct toppling failures
- discrete structures plotting parallel to the trend of the borehole, dipping steeply to the NNW and NNE, indicating the presence of cross-cutting structures and validated by shallow to the core axis structures observed in drill core. These features can act as release planes in planar sliding failures, wedge or toppling failures.
- structure set identification and orientation is heavily influenced by the direction of the drill holes.

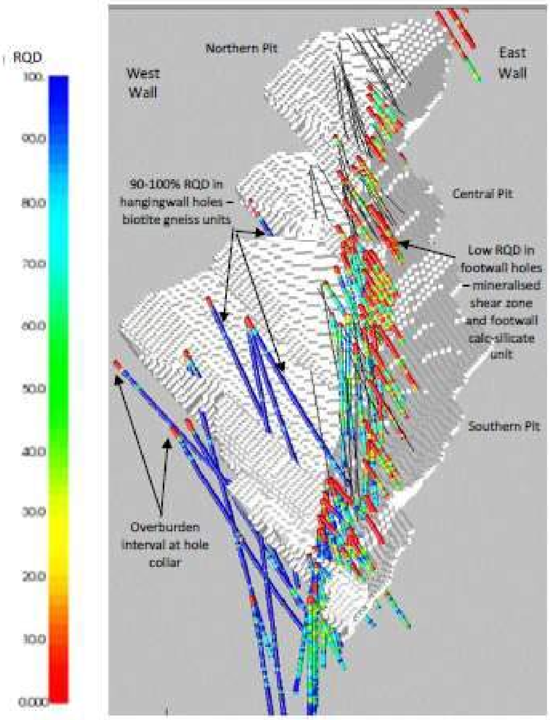

Figure 4. **Optimised Pit Looking North – RQD plotted on boreholes (Plotted using GEM4D – BasRock, 2017)**

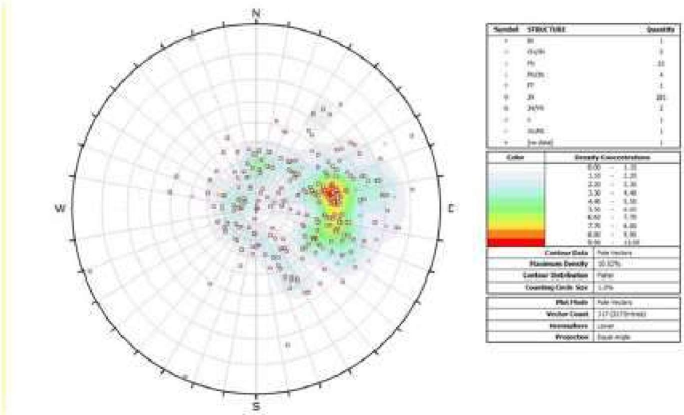

Figure 5. **Stereonet Projection of Defect Data - SGD155, 156 and 157 (DIPS, 2017)**

4.3 Kinematic Analysis

Kinematic and sensitivity analysis was conducted on the poles to all defect features logged. The identified failure mechanisms from this analysis were consistent with those described in the GOA (2017) report. A sensitivity analysis was conducted on variations in the slope dip direction

- The west wall analysis assumed a slope dip direction of 100° and ranges from 080 to 115°
- The slope angle was varied from 50 to 70°, and
- The defect friction angle for planar, sliding and wedge failure was varied from 30° to 50°

The following is a summary of indicative potential failure mechanisms:

4.3.1 Planar Sliding

- The kinematic analysis conducted on 335 poles indicates a low probability of planar sliding failure
- Foliation planes potentially act as release planes for planar sliding in the west wall for slopes/ faces steeper than 70° and in situations where foliation dips steeply to moderately to the ESE due to folding and where/ if the foliation fabric is undercut during mining

4.3.2 Wedge Sliding

- Wedge failures are kinematically possible, and likely to occur on a bench scale in the west wall where slope angles exceed 70°. The occurrence will be controlled by the foliation and possible presence of steeply north or south dipping, E-W structures

4.3.3 Toppling Failure

- The potential of toppling failures occurring is higher in the hangingwall due to the presence of the moderately west dipping foliation and WSW and NE moderately dipping joint set acting as basal sliding planes
- The intensity of the foliation will govern the potential for failure and, as noted, the intensity of foliation decreases the further west in the sequence.

4.3.4 Raveling Failure

- Potential for ravelling failure exists locally, where faulting and shearing is encountered and in broken zones, limited in the hangingwall biotite gneiss and more common in the footwall calc-silicate and mineralised shear zone.

4.4 Rockmass/Circular Failure

The rock mass failure potential assessed by GOA (2010) is considered appropriate for a PFS and the strength parameters and resultant strength factor achieved are reasonable to meet the level of a PFS.

5.0 Recommended Base Case Wall Design Parameters

Figure 6 shows the preliminary economic optimisation shell (*20170712_basecase_s38_rf1-04_draft*) provided by Thor Explorations with design domains defined in the GOA (2010) report.

Slope designs were recommended based upon a staged mining approach, with Stage 1 shallow pits mined at shallower wall angles than final wall designs recommended for the initial mining stage to allow for additional drilling on the west (hanging) wall and pit mapping to validate and modify the design as necessary.

Based upon the initial review conducted on the design and design analysis (Langille et.al., 2017), and this more recent analysis conducted on data extrapolated into the west wall, there is no defendable justification for modifying these initial slope designs and the slope design parameters in GOA (2010) are considered appropriate for a PFS level design.

Table 4 summarises the recommended final wall slope design configuration for batter angles, heights and berm widths appropriate for the assessed conditions.

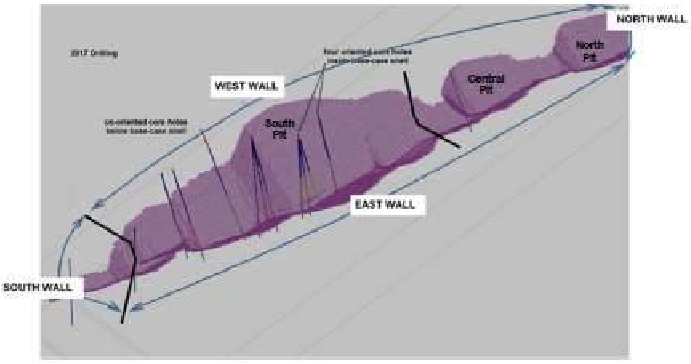

Figure 6. **Proposed pit shell & design domains** (after GOA, 2010)

The results and *base case* slope design parameters do not change, and the approach suggested by GOA to use starter or staged pits is endorsed and recommended. The slope design parameters in Table 4 can be used to put together a preliminary economic pit design. Note the surface RL's vary due to topography, so the upper wall design will need to take that into consideration. While the 10 m batters, with 55° batter face angles were designed to account for the top weathered section, it may be necessary to continue this design into the fresh rock to maintain a consistent slope bench design where topography and thickness of the weathered zone varies.

The designs are considered to provide a sound *base case* and may be modified at the Feasibility Study Stage, with the addition of data from the final pit walls and the proposed data collection programme recommended as part of this study.

Table 4. Summary of Final South Pit Design Parameters (after GOA, 2010)

	East Wall (Footwall)	West Wall (Hanging wall)	North & South End Walls
Upper batter in Weathered Rock **320-300mRL*			
Face angle	55°		
Face height	10		
Berm width	5		
300-180mRL			
Face angle	60°	70°	70°
Face height	20	20	20
Berm width	8^	8^	8^
^160, 200 and 240mRL Berms	12	12	12
Overall Wall angles (pit crest to toe exclusive of ramps)	45°	52°	52°
*This is the inferred pit crest, locally this may be lower or higher and slope designs will need to be adjusted relative to the weathered depth and topography			

6.0 Implications for Mining

The designs provided in the GOA report were considered appropriate for a PFS level study, however, the following were considered gaps in the design analysis:

- There was a lack of information directly relevant to the west wall slope, as most of the drilling conducted prior to the current drill program focused on the east wall and ore definition
- While the east wall was drilled more comprehensively, there were few holes extended beyond ~ 10m to 15m beyond the proposed slope position
- All holes were drilled on an azimuth perpendicular to the lodes, thus any cross-cutting structures will be underestimated. While cross-cutting structures individually are not considered to be potential failure planes, they do act as release planes and will contribute to wedge, planar, and unravelling failure potential
- The folding mentioned in the GOA (2010) report is not accounted for explicitly
- The hydrogeology conditions have not been fully investigated and will need to be in order to finalise the design to Feasibility Study level
- Intact rock and defect strengths were assumed based upon observation and experience, and preliminary results were provided for UCS testing, conducted by Thurber Engineering (2017), but direct shear testing has not been completed on critical defects, nor has significant testing been completed in the footwall calc-silicate unit at this stage.

The fundamental requirements of this review were to:

- Assess whether the GOA report would constitute a PFS level mining geotechnical assessment sufficient to advance the project to a full (Bankable) Feasibility Study
- If not, provide recommendations for work to bring the project to PFS level
- Identify and recommend work required to bring the project to PFS level for a Bankable Feasibility Study

The GOA report provides adequate design justification for the proposed pits, although the west wall design, given lack of data available, is likely conservative.

6.1 Mining Implications

All assumptions and caveats presented in the GOA (2010) report were considered applicable to the project, the most pertinent of which include:

- Identifying local variations in surface topography, rock weathering depths, geological conditions (particularly in the footwall where mixed lithologies are common) and structural geological conditions which may be encountered along strike at the deposit, potentially requiring local amendments to the wall designs during mining.
- The design parameters supplied in the GOA report were *base case* designs which may require local amendment on an "as required" basis, as indicated by additional data collected from drilling, and once mining commences, mapping and ground conditions actually encountered.
- Bench faces mined in fresh rocks are expected to break back to "daylighting" geological structures where these exist near wall positions. Variation in the orientation and intensity of the gneissic texture/foliation planes subject to folding (reported by GOA, 2010 and Gillman *et al*, 2016, but not definitively defined) will result in variations to the batter angle at which daylighting of structures occur. This will impact berm crest stability and the necessary widths of final catch berms.

- It is assumed that good quality drilling and blasting practices will be instituted to minimise the extent of potential break back of batter walls, berm and ramp crests and quality of the final pit walls.
    - The near surface, highly weathered rocks extending from surface to the BOCO should be capable of being freely dug, however, there may be sections where these rocks require light paddock blasted (blasted to loosen) to maintain satisfactory productivity.
    - The irregular rock weathering profile of the deposit and the difficulties in assigning an accurate "average" depth of rock weathering need to be kept in mind and localised hard rock blasting allowed for in portions of the weathered rock profile.
    - Blasting conditions are anticipated to be "difficult" in areas of walls exhibiting extremely variable local rock weathering profiles affected by penetrative rock weathering along faults and within the calc-silicate footwall.
    - Allowance should be made for "wet blasting" conditions from the TOFR downwards.
    - Successful mining to design depends on minimising mining-induced disturbance and care must be taken to ensure that production blasts do not damage or unduly disturb the final wall rocks. This includes:
        - Adjustment and strict control of sub-drill depths in the vicinity of future berms and berm crests (to minimise damage to the berm crests and wall rocks).
        - Allowance should be made for trim blasting (firing to free faces) to be carried out where final walls are being mined should a staged pit approach be adopted and pre-split blasting of final walls mined in the deeper final pit.
        - Batter faces and berm crests are hydroscaled with high pressure water and mechanically scaled to remove loose rock, prior to mining the next bench to reduce future rockfall potential. The interpreted "blocky" and "wedgy" nature of the rocks implies that there is potential for rockfalls to take place from walls should these not be adequately and comprehensively scaled.
        - The services of a specialist, appropriately qualified blasting engineer be obtained during initial mining, and used periodically during the life of the project, to assess appropriate blasting requirements and performance of blast designs.
- Identification of the potential impact of water pressure, noting comments on artesian water encountered in drill holes in the GOA report and measured water flow from boreholes SGD166, which was pump tested, and SGD081, which was measured at approximately 1.12 L/minute flow rate, will need to be made from a detailed hydrogeological study, conducted as part of the Feasibility Study. Allowance may need to be made to carry out wall depressurisation drilling (installation of sub-horizontal drainage holes) within the future southern pit walls mined below the groundwater table in order to maintain wall stability adequate for mining purposes. The design of these holes may only be carried following inspection of hydrogeological conditions encountered during mining.

6.2 *Recommended Actions for Feasibility Study*

In order to bring the project to a full (Bankable) Feasibility Study stage, the following additional work would be required:

- Additional off-section geotechnical holes in the hangingwall (West Wall) and footwall (East Wall) are recommended to define the extent and frequency of E-W steeply dipping structures and rockmass and structure within the pit walls. A summary of required drilling is shown in Table 5. It is assumed that all drilling would be conducted using triple tube (HQ/NQ) drilling techniques as it is vital that as much of the natural fabric as practicably possible is preserved during the drilling process. Drill collar locations are approximate, dependent on accessibility, topography and other local operating factors and can be adjusted to suit local conditions, but should be reviewed by the authors should changes be required.

- The proposed drilling summarized is shown for the west wall (hangingwall) in Figure 7 and east wall (footwall) in Figure 8.

Table 5. Proposed Geotechnical Drill Programme, Definitive Feasibility Study

HOLE-ID	Easting m	Northing m	RL, m	Azimuth	Plunge	Hole Length, m
GTFS17-001	701500	831150	370	090	-55	0
GTFS17-002	701500	831150	370	045	-45	280
GTFS17-003	701500	831150	370	140	-45	280
GTFS17-004	701725	831100	340	090	-55	250
GTFS17-005	701795	831100	340	035	-50	170
GTFS17-006	701795	831100	340	145	-50	170
GTFS17-007	701655	831475	355	140	-55	200
GTFS17-008	701900	831440	340	235	-45	175
GTFS17-009	701765	830750	330	330	-45	175
			Total			**1670 m**

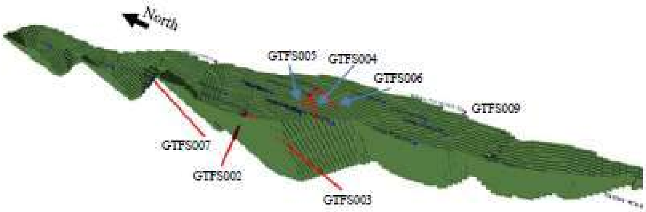

Figure 7. **Preliminary pit shell showing location of proposed geotechnical drilling in west wall – looking NE**

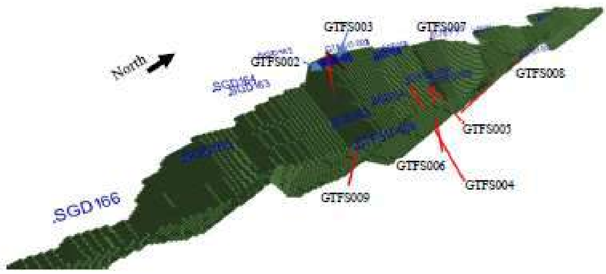

Figure 8. **Preliminary pit shell showing location of proposed geotechnical holes in east wall – looking NW**

- A hydrogeological and hydrology study to identify the potential and impact of pressurised groundwater occurring along faults/shears and fracture networks, and surface drainage conditions associated with seasonal watercourses. It is understood a consultant has been commissioned to conduct a preliminary desktop study. The availably of wall holes for future water level monitoring should be communicated as part of the overall FS.
- Rock property testing of critical structures (structures controlling sliding, wedge failures – direct shear testing of representative structures) and UCS tests of selected representative samples from the footwall calc-silicate rock unit (one sample tested as part of this review). Holes GTFS004, 5, 6, 8 and 9 will be located in the footwall rock unit along the length of the southern pit.
- A site visit to view local conditions, topography, water and specifically drill core, logging process and sample selection for direct shear testing
  - define the extent and frequency of E-W steeply dipping structures, aiming to confirm/ extend the database and, if present, identify and characterise additional structures not detected in the geotechnically logged holes within the pit shell. Ensuring best quality drilling and core recovery is vital – use of triple tube core barrels is considered to be essential.
  - Due to the ~ 1,000m strike length of the southern pit, up to nine (9) additional holes are recommended and include holes into the east wall footwall and two additional holes angled at 120° to the perpendicular holes in the west wall, as well as holes covering the north and south wall of the preliminary southern pit.
  - Additional holes were also considered to better define the designs for the central and northern pits, however, the current level of data is considered appropriate, given the relatively shallow pit depths and to manage the pit design through initial starter pits, using shallower temporary wall angles (Langille, et.al. 2017A; GOA, 2010) and using mapping data collected during Stage 1 to optimize and finalise the final slope designs.

The analysis conducted to date provides the basis for a practical, but possibly conservative design. Given the assumptions for the west wall analysis of projecting the lithology units along their trend into the west wall:

- In the upper west wall, where the foliation intensity decreases further to the west, there may well be scope for steepening the slope
- Conversely, the lower west wall and lower east wall may require modification to address increased foliation fabric development
- Attaining more confidence in the geotechnical environment and validating and modifying the design will be the objective of the Feasibility Study geotechnical program.

All assumptions and caveats presented in the GOA (2010) and summarised in Langille, et.al. (2017) reports are considered applicable to the project, and will not be discussed in further detail in this report.

7.0 Closure

We thank you for the opportunity to contribute to this most interesting project and trust that the information provided meets your immediate requirements.

Analysis and recommendations herein are based upon information provided by Thor Explorations Ltd, and are made on the assumption that appropriate techniques will be used, and performed at a consistently high standard, in all aspects of future mining and wall stability monitoring at Segilola Gold Mine.

Please contact the undersigned if there is any need for clarification and/ or further comment.

PETER O'BRYAN *& Associates*
per:

Peter O'Bryan	**Christopher Langille**, P.Eng.
BE (Mining) MEngSc MAusIMM (CP) MMICA	MEngSc MBA MAusIMM (CP)
Principal	*Associate*

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APPENDIX 3 – HYDROGEOLOGY

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18 September 2017

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Attn: Mr Alfred Gillman, Exploration Manager

Ref: Segilola Gold Project, Nigeria – Preliminary Feasibility Study
Review of Hydrogeology at Proposed Mine Site

Introduction

Thor Explorations Ltd (Thor) is conducting a PFS level review of their Segilola Gold Project (SGP) which is located in Osun State, Nigeria (Figure 1).

This letter report provides a desktop review of hydrogeology at the proposed SGP mining operations. Most information for this review has been provided by Thor, and a minor amount of data has been obtained from internet sources.

The main objectives of this review are:

- Provide an overview of the hydrogeology at the proposed SGP mining operations from available information
- Identify aspects of the hydrogeology that could impact groundwater management at the proposed mining operations
- Identify knowledge gaps in the site hydrogeology where additional information is needed to improve confidence in possible methods for groundwater management prior to commencement of mining
- Provide a work scope for additional investigations to progress the project to a Definitive Feasibility Study (DFS)

This review has been requested in email correspondence dated 31 August 2017 from Mr Alfred Gillman.

Previous Work

A Bankable Feasibility Study (BFS) for a previous owner of the SGP is contained in the March 2012 report by Ratel Group Limited and Tropical Mines Limited[1]. An Environmental Impact Assessment (EIA) was also completed for the project in 2012 by Fugro Nigeria Limited[2]. Both of these documents contain information about the project setting and the proposed mining operations, and are referenced in this letter report.

Project Setting

The SGP is located in south west Nigeria, about 250 km NE of Lagos. The project area is 7°31' north of the equator in tropical southern Nigeria.

Climate

The BFS and EIA present some climate statistics from the settlements of Akure and Oshogbo, respectively. Akure is about 50 km SE of the SGP site, and Oshogbo is about 40 km NW of the SGP site (distances and directions determined from Google Earth). Some climate data for these stations is also available from the weatherbase.com website.

The SGP site is located in a Tropical Savanna climate zone and experiences a pronounced dry season. Average annual rainfall is around 1.35 m at Oshogbo, and 1.57 m at Akure. About 80% of the annual rainfall occurs in the seven month period April to October.

Humidity is characteristically high throughout the year, and average relative humidity is 75% to 77%. Average daily temperature varies by only a few degrees Celsius year round, and the average temperature at both Oshogbo and Akure is 26°C.

Mining Proposal

Open pit mining at the SGP is currently being considered by Thor. The 2012 BFS indicates underground operations may follow after open pit mining is completed.

The conceptual open pit at the SGP is around 1,800 m long by a maximum 385 m wide, and maximum depth of around 160 m.

Topography, Vegetation, and Surface Water Hydrology

The BFS notes that the area was once covered by dense tropical rainforest. Intense farming practices have reduced the original vegetation coverage to secondary forest and bush regrowth, as well as some cash crops. The clearing of the land for agriculture and the presence of secondary regrowth would have had a large impact on the surface water hydrology of the area.

Land surface elevations in the vicinity of the project are reported to range between 300 mASL and 580 mASL (above sea level). Local relief across the prospect is reported to be 30 m, with topography dominated by several NNE trending ridges.

There is an east-west trending watershed divide in the northern part of the prospect, and surface water drainage across the site is predominately to the south. Several small streams flow to the south between the ridges. These streams are described as shallow and transparent, up to 3 m wide and <1 m deep, with sand and gravel bed loads.

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Geology and Hydrogeology

Discussions of regional and district geology are contained in the 2012 BFS and EIA documents, and these are summarised below. A plan of the district-scale geology is presented in Figure 2, and a cross section is presented in Figure 3.

The SGP lies in a sequence known as the Ilesha Schist Belt. This sequence is composed of Proterozoic supracrustal rocks that are infolded into a migmatite-gneiss-quartzite complex, and are part of the south-western Nigerian Shield.

The Ilesha Schist Belt has a north-south strike extent of >200 km, and maximum width is 60 km in the south. The Ifweara Fault or Shear Zone is a regional, dextral strike-slip structure which is present in most of the Ilesha Schist Belt (Figure 2). This structure occurs about 5 km west of the SGP.

At the project scale the SGP rock sequence strikes NNE and dips steeply WNW, and consists of locally refolded biotite gneiss, biotite schist and calc-silicate gneiss, all of which are intruded by pegmatite and dolerite dykes (Figure 4). The SGP mining prospect lies at the boundary of a biotite gneiss (in the hangingwall) and calc-silicate sequence (in the footwall).

Gold mineralization at the SGP occurs in fractured pale to dark grey coloured smoky quartz veins, sheared pegmatite and silica/chlorite/carbonate alteration along a single robust shear zone in mainly biotite ortho-gneiss. The shear is located 2 m to 15 m into the hangingwall from a well-defined contact with calc-silicates and biotite schists.

East-west striking wrench faults occur at the north and south extremities of the deposit and close to the crest of the conceptual open pit. The wrench fault at the northern end of the deposit is regional in extent with over 50 m of dextral displacement. The wrench fault at the southern end of the deposit is less extensive with around 10 m displacement.

Three north-south striking and steeply west dipping normal faults occur within the crest of the conceptual open pit at the SGP.

Sulphide minerals are noted to occur within some of the rock sequence that hosts the gold mineralisation.

The depth of total oxidation of the bedrock sequence at the SGP is generally <10 m. Some of the faults and shears are totally oxidised to depths of >60 m.

Information about groundwater in the vicinity of the SGP is limited. Some hydrogeological data from the SGP have been collected recently by Thor, and these are presented and discussed in the next section. Broad information about regional groundwater resources in Nigeria is available from the Africa Groundwater Atlas which is hosted by the British Geological Survey[3].

The regional hydrogeological zone where the SGP is located is known as "Basement Complex". Aquifers in this zone can occur at the base of weathering or in fractured bedrock. Generally, aquifers in the Basement Complex tend to be low yielding. The quality of groundwater from basement rocks is noted to be generally good.

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The EIA presents hydrochemical analyses from two water supply wells near the SGP, one from Iperindo about 2 km south of the SGP, and the other from Odo Ijesha about 3 km north of the SGP. Presumably these wells are relatively shallow and dug by hand. The total dissolved salts concentrations of these samples are reported to be 279 mg/L and 400 mg/L. The TDS concentrations indicate the shallow groundwater is relatively fresh, and this is consistent with the general hydrogeological setting.

Available Site Hydrogeological Data

Groundwater Level Data

Thor collected a set of groundwater levels from some of the exploration boreholes at the SGP in August 2017. These boreholes are inclined, with dips between 46 and 85 degrees, and range in depth between 60 m and 340 m, ie, they are exposed to the bedrock sequence. All boreholes have surface casing to the top of fresh bedrock, and are open below this depth.

Figure 5 is a location plan of the boreholes in this groundwater level data set, and a contour plan of the calculated groundwater levels is presented in Figure 6. The collars of the boreholes included in this data set are either within or very close to the crest of the conceptual open pit. Figures 5 and 6 have been provided by Thor.

Figure 6 indicates that groundwater levels in the northern half of the SGP are flat lying and generally in the range 332 mASL to 340 mASL. There is more variation in groundwater levels from boreholes in the southern half of the SGP, with groundwater levels in the range 300 mASL to 330mASL.

The vertical depth to water below the land surface measured in the boreholes is <5 m in the northern half of the SGP, and up to 60 m in the southern half of the SGP. Clearly there is a strong contrast between the groundwater levels in the northern and southern sectors of the SGP. The cause(s) of this contrast warrants further investigation as it is likely to influence groundwater management during mining operations.

Pumping Test Data

Thor conducted a pumping test at borehole SGP166 (see Figure 5 for location) in August 2017. This borehole is located at the southern end of the conceptual pit crest. The total length of the borehole is 216.5 m, and is thus exposed to a large section of the fresh rock sequence.

SGD166 was pumped for 3 hours at a constant rate of 0.3 L/sec (26 kL/day). Water levels were measured in SGD166 at 10 minute intervals for the first hour of pumping, and at 20 or 30 minute intervals until the end of pumping. Recovery water levels were measured for 6 hours after pumping stopped.

The total volume of groundwater pumped from the borehole was 3.24 kL, and the end-of-test drawdown in SGD166 was 2.28 m.

There was no measurable drawdown in SGD166 during the first 20 minutes of pumping, and the drawdown after 30 minutes of pumping was 0.07 m. Thus around 0.5 kL of water was pumped from this borehole before any response of water levels in the borehole was observed. This is an unusual response, the cause of which is unknown at this stage.

Figures 7A and 7B present a drawdown versus time graph and a recovery water level graph for the SGD166 pumping test data. The linear trend of drawdown with the logarithm of time after about 50 minutes of pumping is not unusual. The data are, however, very likely to be affected by casing storage and are thus not amenable to analysis. The magnitude of the drawdowns for the given pumping rate suggest a bulk formation transmissivity of order 1 m^2/day, with an equivalent bulk hydraulic conductivity of order 10^{-7} m/sec.

The recovery water level data in Figure 7B show early recovery of drawdowns. This can indicate that pumping caused displacement of fine materials from some of the fractures intersected by the borehole, or rapid aquifer recharge from, eg, nearby surface water.

The short duration pumping test of SGD166 does indicate that the pumping removed a small volume of groundwater from aquifer storage.

Hydrogeological Issues for Mining at SGP

The hydrogeology of the SGP site is not well understood and has not been investigated sufficiently to confidently develop groundwater management strategies for the proposed mining operations. At this stage, however, the available hydrogeological data are sufficient for the purpose of contributing to a Preliminary Feasibility Study for the proposed mining operations.

Some aspects of the site hydrogeology which are apparent from available information are listed below:

- Groundwater levels are close to the land surface over a large portion of the proposed open pit, and thus dewatering and management of groundwater will be required form the beginning of mining.
- Groundwater and surface water across the site are probably closely related. Some groundwater recharge could occur either by means of direct infiltration or seepage from channels. During the dry season some of the stream flow could be sustained from groundwater seepage into the channels. These processes are not unusual in tropical settings.
- Site surface water management will be important during mining at the SGP, and surface water will need to be diverted from the crest of the open pit.
- Descriptions of the larger-scale hydrogeology of the "Basement Complex" in south western Nigeria indicate that, in general, water wells in this setting is not noted to be high yielding. While that would be true as a generalisation, the local setting at the SGP is different because of the presence of NNE and cross cutting structures. The groundwater-yielding characteristics of these structures is not known, and investigations are required to address this deficiency.

- The hydrochemistry of groundwater from the deeper unweathered sequence at the SGP is unknown. It will be important to develop an understanding of this hydrochemistry as it could affect the approach to groundwater management.
- Shallow groundwater from hand dug wells in nearby villages is relatively fresh, and there is clearly reliance on these shallow groundwater resources as a water supply. It will be important for the mining operations to be aware of all nearby groundwater users prior to site clearing and the start of mining.

Recommendations

Site-scale investigations are clearly needed to improve the hydrogeological knowledge base and progress the project to Definitive Feasibility Study level. The broad objective of these investigations is to gain sufficient understanding of the site hydrogeology to confidently define the approach or approaches to groundwater management both during mining and when the mine is abandoned.

A staged investigation program is recommended, and key components are:

- Review available drilling reports for any indications of difficult drilling conditions, groundwater production (from RC air hammer boreholes), or drilling fluid circulation problems (from diamond core holes).
- Conduct air-lift pumping tests in several of the existing open boreholes, noting the yield and whether the yield varies with time. These tests may provide an opportunity to collect samples of groundwater from the deeper rock sequence and structures at the SGP for determination of salinity.
- Depending on the results of the air lift tests, review the site geological and structural models and select at least two sites for the construction of trial dewatering wells. These wells would then be subjected to pumping tests, and this may require nearby observation piezometers to be constructed. Samples of groundwater would be collected from these wells for comprehensive hydrochemical analysis.
- Assess all of the results, and develop a mine groundwater management plan. This plan will need to address any groundwater issues identified by the mining geotechnical consultant, such as the need to lower groundwater pressures in the pit walls. The plan will also need to address the management of any seasonal variations in groundwater levels, and this may require some interaction with the surface water management strategy for the open pit.
- As part of the management plan, identify sites for piezometers, either open standpipes or multi-level vibrating wire piezometers.
- Prior to any site clearing and the start of mining, prepare an inventory of local groundwater and surface water users who may be impacted by the mining operations. If possible, collect water samples for comprehensive hydrochemical analysis. Periodic monitoring of these water resources during mining should occur.

All of the site hydrogeological investigations will need to be conducted in accordance with the requirements of Nigerian government agencies and/or the equivalent agencies in Osun state.

Yours faithfully
Peter Clifton & Associates

P M Clifton
1708_L01(0).DOCX

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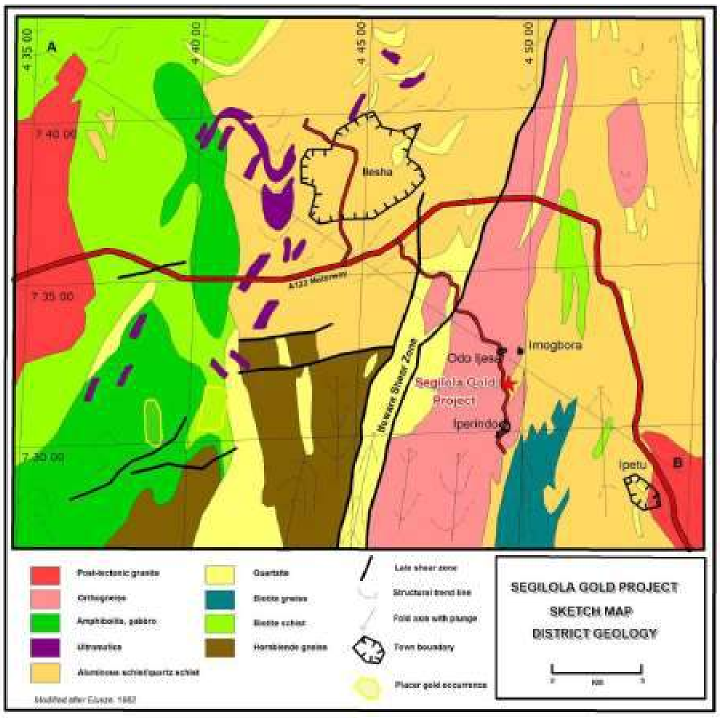
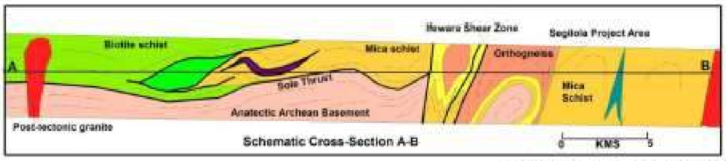
Source: Thor Explorations Ltd

Figure 1: Segilola Gold Project Location Plan



Source: Thor Explorations Ltd (2012 BFS)

Figure 2: Segilola Gold Project District Geology Plan



Source: Thor Explorations Ltd (2012 BFS)

Figure 3: Geology Cross Section (A-B in Figure 2)

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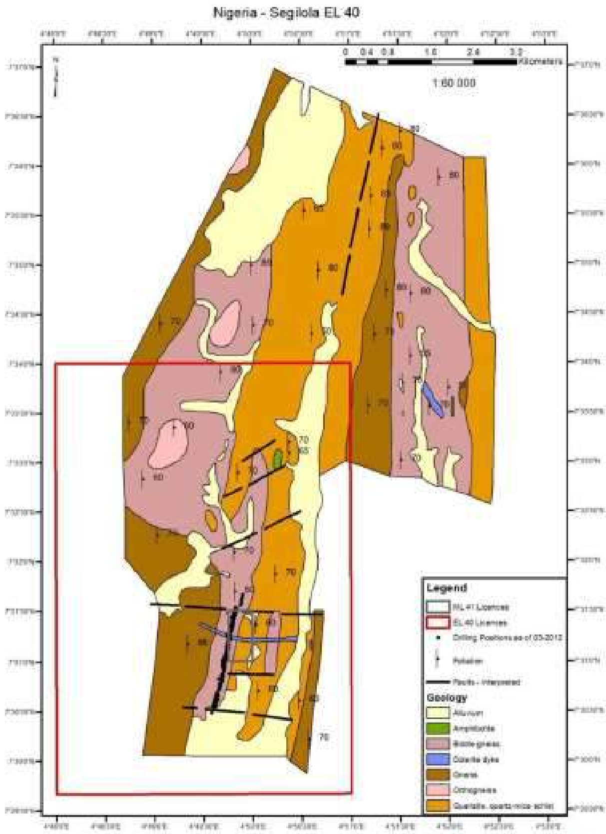
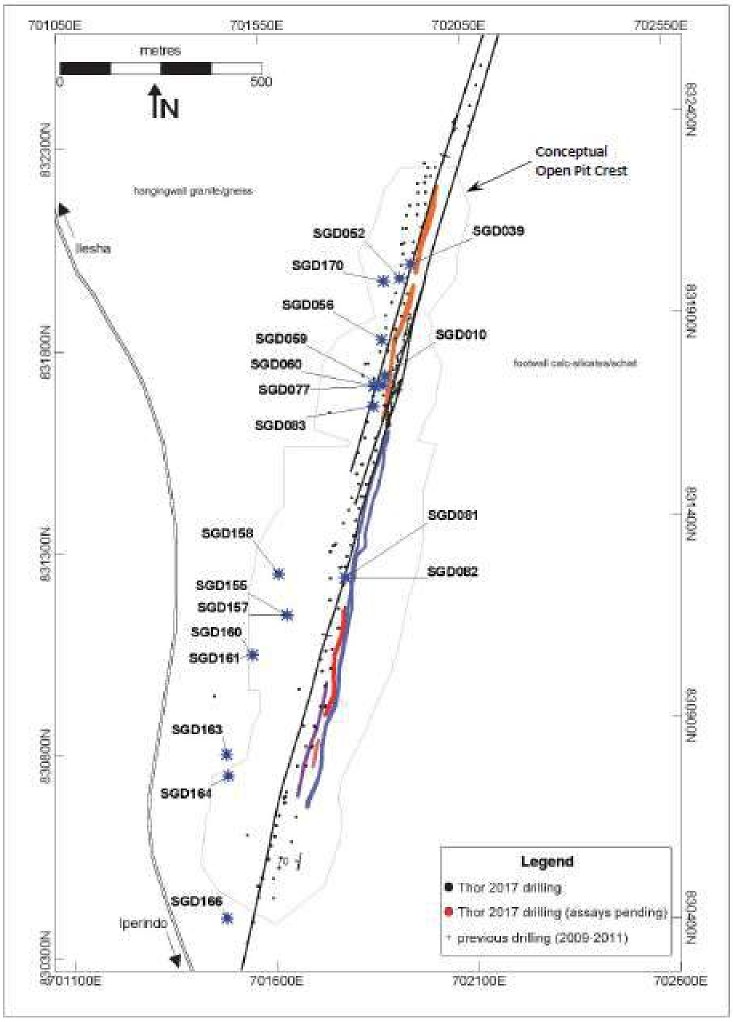


Source: Thor Explorations Ltd (2012 BFS)

Figure 4: Segilola Gold Project Local Geology Plan



Source: Thor Explorations Ltd

Figure 5: Boreholes with Groundwater Levels

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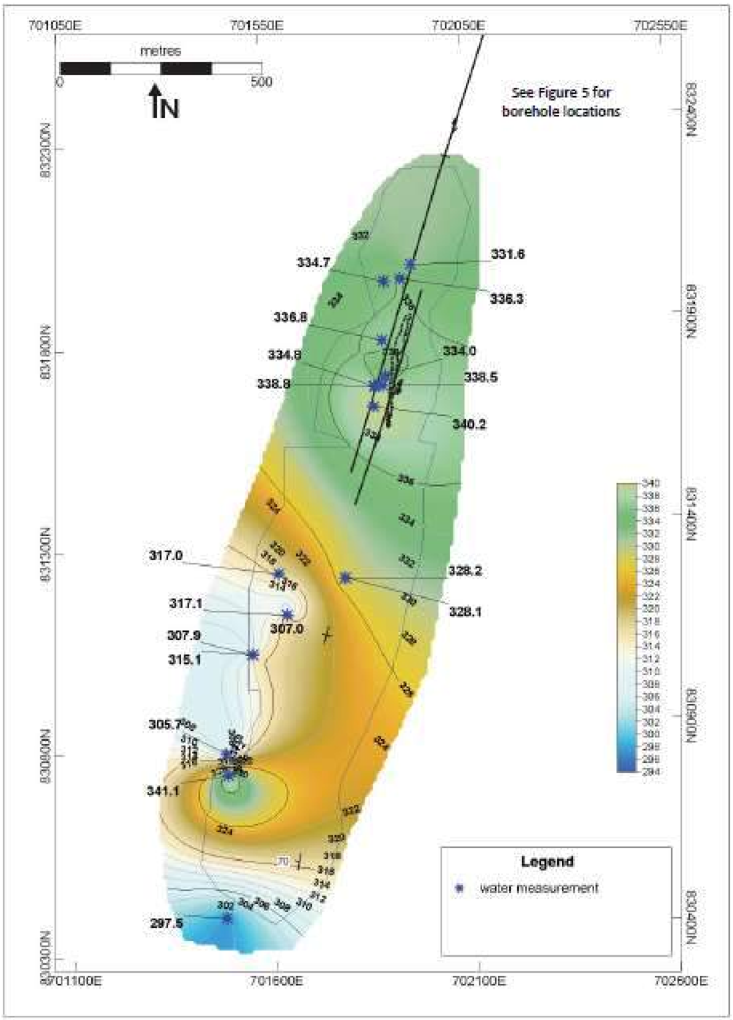

Source: Thor Explorations Ltd

Figure 6: Groundwater Level Contours (mASL)

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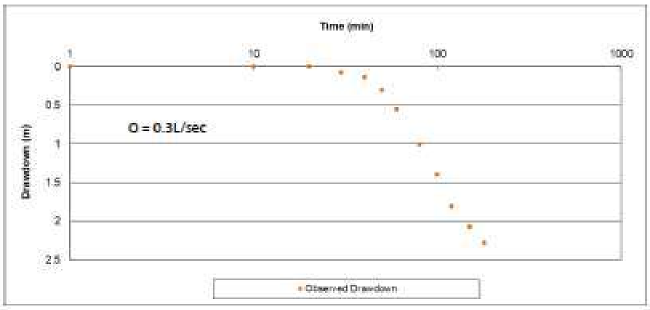

Figure 7A: SGD166 Drawdown vs Time Graph

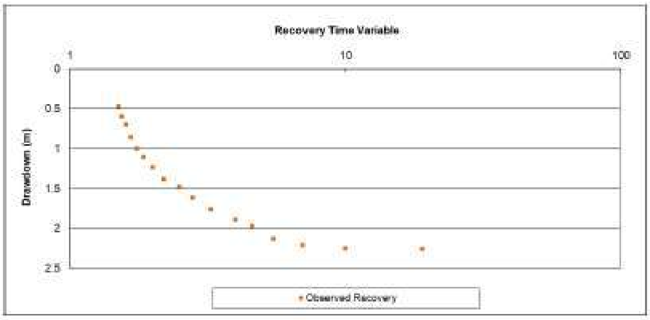

Figure 7B: SGD166 Recovery Water Levels vs Time Graph