Independent Technical Report
Mineral Resource and Ore Reserve Estimate,
Segilola Gold Deposit,
Osun Province Nigeria

Prepared by Mining Associates Pty Ltd

for

Segilola Resources Operating Ltd

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1 SUMMARY

This is an independent technical report on the Segilola Gold Project (or "the Property") of Segilola Resources Operating Ltd in Osun Province, Nigeria. prepared in accordance with Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI43-101). Segilola Resources Operating Ltd is a subsidiary of Thor Explorations Ltd.

The Segilola Gold Project is located in the state of Osun, Nigeria, approximately 120 km northeast of the city of Lagos and 18 km south of the regional centre of Ilesha. The Project is centred on 700,987 mE, 832,281 mN (WGS84 UTM Zone 31N).

The property comprises a Mining Licence (ML41), which covers an area of 1,720 ha (17.2 km^2) and is contained within a larger Exploration Licence (EL19066), which covers 2,700 ha (27.0 km^2). The Mining Licence (ML41) was renewed in September 2016 and is valid for a period of 25 years until 2041.

1.1 PROJECT HISTORY

The area is well known for its gold production from eluvial placers. Modern mining of the alluvial and eluvial deposits began in 1942 and official records state an annual historic production of approximately 23,000 oz gold. The Segilola deposit, formerly known as the Iperindo Reef, was first discovered during the working of the eluvial deposits in 1945.

From 1949 to 1969, the deposit was mined by local operators in a small-scale open pit measuring approximately 5 m wide, 15 m deep, and 300 m along strike. The operators processed the ore with a second-hand stamp mill together with a ball mill and tables acquired from Ghana.

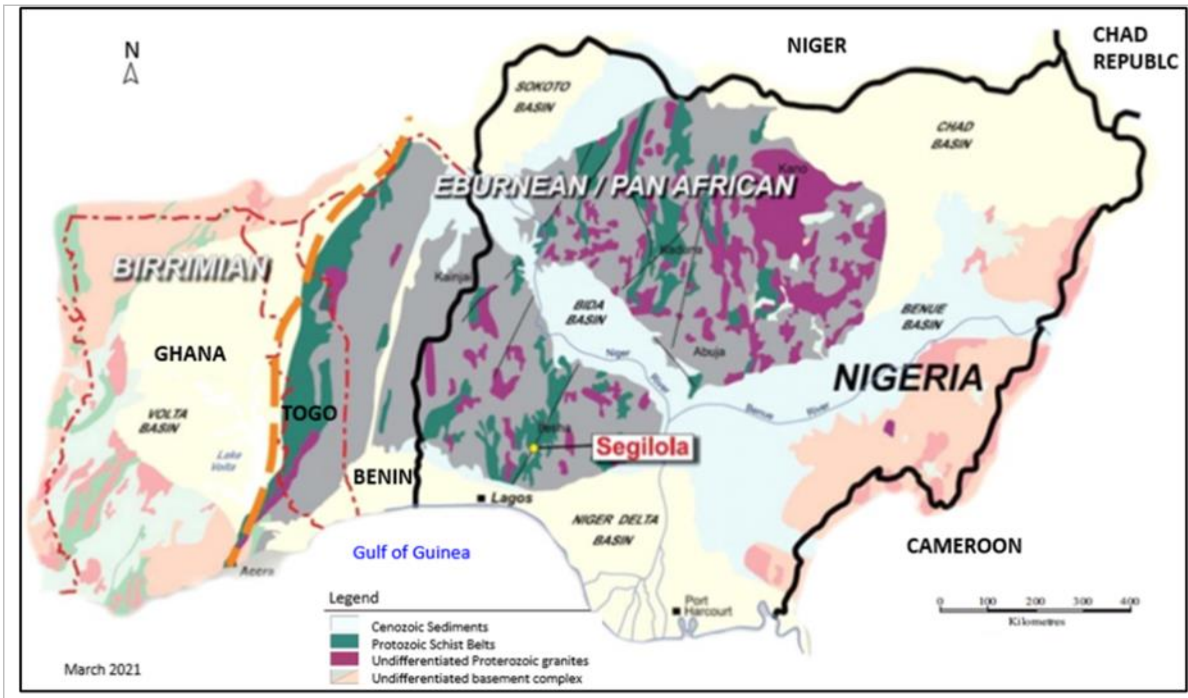
In 1970 the property was acquired by Obokun Minerals Developments Limited (OMDC) and in 1982 Nigerian Mining Corporation (NMC) acquired the Project from OMDC. In 1995 Tropical Mines Ltd (TML) was incorporated as a joint venture company owned 20% by NMC and 80% by Pineridge Nigeria Ltd (PNL). In 1996 Temporary Mining Licence TMiL 19706 was assigned to TML.

In 2007 Segilola Gold Limited (SGL), then a wholly owned subsidiary of CGA Mining Limited (CGA), acquired the right to earn up to 51% interest in the tenements. CGA commenced drilling known mineralized zones. In 2009 CGA declared a maiden Mineral Resource estimate. SGL was transferred by CGA to its affiliate, Ratel Group Limited (RGL), a Toronto Stock Exchange (TSX) listed entity. In 2010 RGL completed a Feasibility Study (FS) for internal purposes. In 2012, a Revised Bankable Feasibility Study was completed but not published. Development of the Project was delayed due to a dispute between TML and RGL regarding earned interest in the Project.

SROL Explorations Ltd (SROL) acquired the Project (100% interest) in August 2016 through the acquisition of Segilola Resources Operating Limited (SROL) and its joint venture partner Segilola Gold Limited (SGL) from Ratel Group Limited (RGL or Ratel), a wholly owned subsidiary of RTG Mining Inc. Thor Explorations (Thor) rights to the property are through its 100% ownership of SROL and 100% of SGL.

1.2 GEOLOGY

The Property is located in the crystalline Basement Complex rocks of southwestern Nigeria within the Ilesha Schist Belt (ISB). Schist belts in Nigeria occur as north-south trending domains of Upper Proterozoic (Eburnean 2,000 Ma) meta-sedimentary, meta-volcanic, and intrusive sequences that are oriented parallel to the boundary between the West African Craton and the Pan African Province. 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 (600 Ma) orogenic suite. Primary gold mineralization in the schist belts commonly occurs in quartz veins within several lithologies.



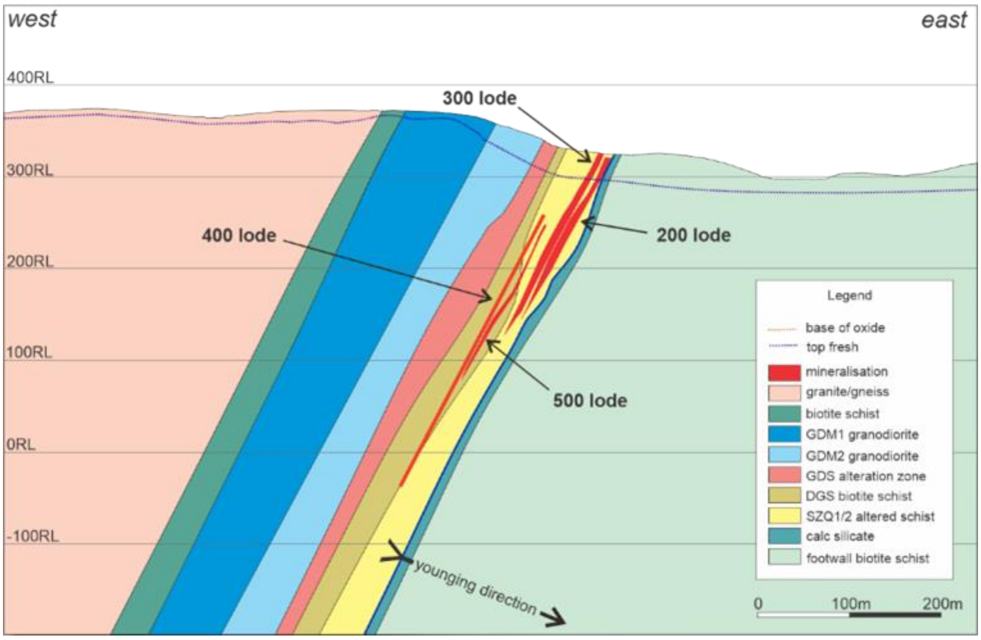
Regional Geology, Segilola Project, Osun State, Nigeria

The ISB 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 Shear Zone (ISZ). The ISB has accounted for a significant proportion of Nigeria’s limited gold production.

The property stratigraphy consists of a series of quartzite schists, a gneissic sequence, and surficial alluvial sediments. The quartzite schists are composed of quartzite, quartz-mica schist, and meta-sediments such as garnet-biotite schist. The gneissic sequence is divided into paragneiss (biotite gneiss), orthogneiss (granite gneiss and pegmatoidal gneiss), and undifferentiated gneiss. The stratigraphy trends north-northeast and generally dips steeply towards the west.

Drilling results demonstrate that gold mineralization occurs in fractured pale to dark grey coloured smoky quartz veining, sheared pegmatite, and silica/chlorite/carbonate alteration. Mineralization is dominantly hosted in steeply dipping vein sets or lodes. The lodes form an elongate mineralized zone striking 010° and dipping 60° to 70° towards the west within a shear zone, primarily in biotite gneiss. The currently drilled mineralized zone is approximately 2,000 m in strike length, between 70 m and 200 m in depth, and between 2 m and 18 m in true thickness.

The lodes lie within an overturned sequence of metamorphosed, strongly foliated quartz sediments (quartzites/quartz biotite schist) at the boundary between the basement biotite gneiss (hanging wall) and calc silicate and mylonitic biotite-garnet schists (footwall). A unit of massive to foliated granodiorite conformably intrudes the sequence between the quartzites and basement gneisses. Pegmatitic veins, which are mostly conformable to schistosity, permeate the quartzite and footwall rocks. Gold mineralization is associated with late stage weakly foliated to undeformed ‘pegmatitic’ veins and is restricted to the quartzite unit. The depth of weathering varies from 1m to 2m in the west to 5m to 10m in the mineralized shear zone.



Cross Section through Segilola deposit showing geology and lodes

The mineralogy of the Segilola deposit is characterised by its general simplicity and consistency. The gold is entirely non-refractory and commonly occurs as visible particles within either pegmatitic quartz-feldspar veins or foliated biotitic selvedges to the veins. There are no significant trace element associations such as silver with gold. However, metallurgical assaying indicates slightly elevated copper (250 ppm to 300 ppm) and mineralogical studies suggest a gold-tellurium association.

1.3 EXPLORATION

Historical exploration activities on the project have included geophysics, geological mapping, soil sampling, trench sampling and drilling. Drilling has been undertaken at the Project by several previous owners. The focus of historical drilling programmes was to test the strike length of known mineralization north and south of the known deposit, mostly with the objective of producing a Mineral Resource estimate.

Between 2008 and 2011, CGA undertook three resource definition drilling programmes which comprised 159 holes (totalling 15,987 m). Data from this CGA drilling is the only historical data used for the current Mineral Resource estimate.

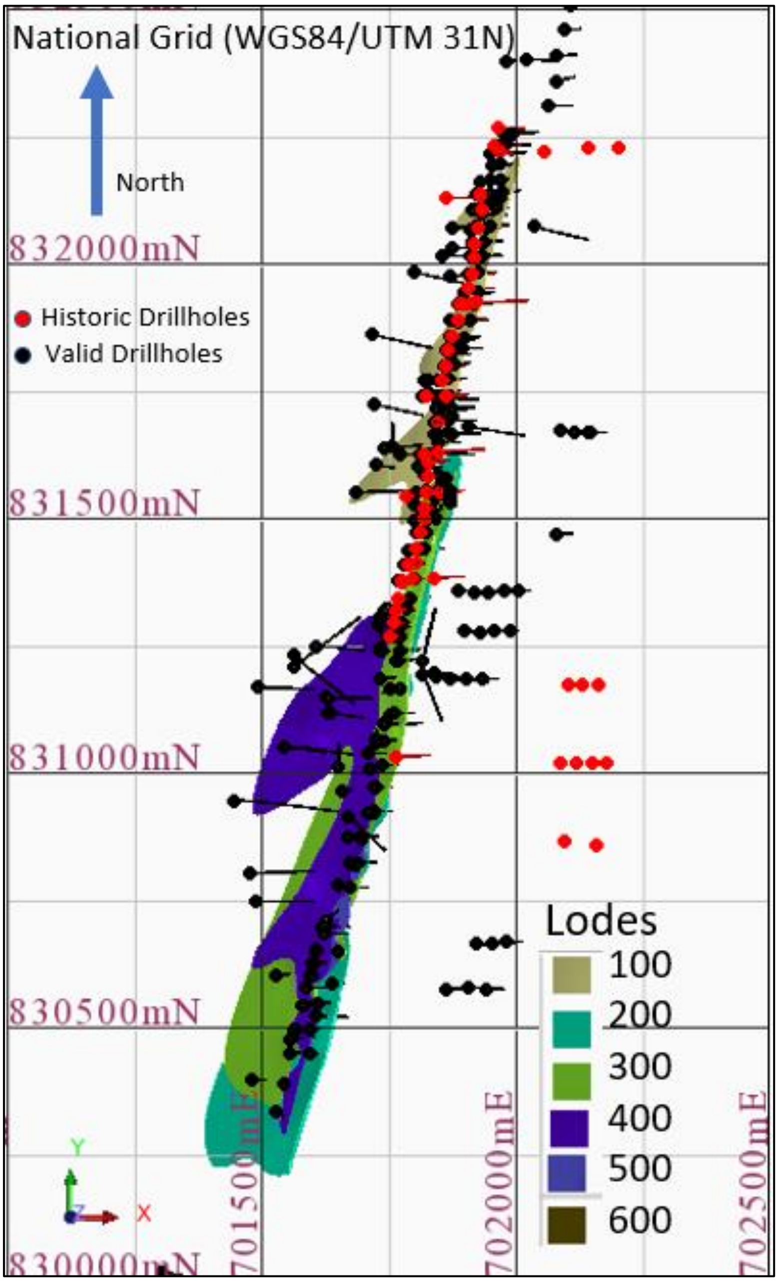
Data from the NMC and Hansa drilling has not been used in the current Mineral Resource estimate due to a lack of quality assurance and quality control (QAQC) data, a lack of verifiable downhole survey data, and the lack of verifiable core intersections due to full-core sampling.

Historical mineral resource estimates were prepared by previous owners prior to SROL's (Thor) acquisition in 2016. Thor is not treating the historical estimates as current Mineral Resources or Mineral Reserves.

1.4 MINERAL RESOURCE ESTIMATE

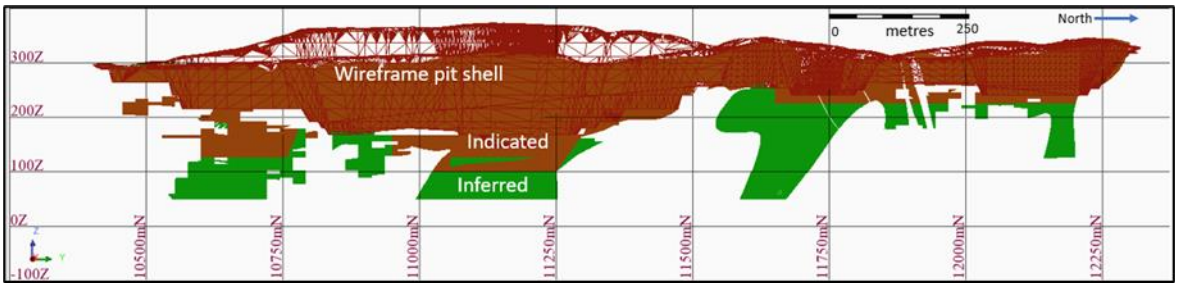
The Mineral Resource Estimate for the Segilola Gold Project, Osun State, Nigeria, has been prepared with an effective data of 4 January 2021 by Mining Associates. Mining Associates employee, Mr I. Taylor, MAusIMM (CP) prepared the Mineral Resource Estimate. Mr Taylor takes Qualified Person responsibility for the Mineral Resource Estimate.

After the Mineral Resource Estimate reported in March 2019 an additional 90 holes for 8,463 m of infill and depth extension drilling has provided a better understanding of the mineralization by providing greater detail which has been incorporated into the models. As a result, intervening waste material was removed from the southern lode, creating two parallel lodes (Lodes 200 and 300) with a consistent and often thin band of waste separating them, and lodes were extended at depth.



Plan View of Deposit and Drilling

The Segilola Gold deposit Mineral Resource comprises an Indicated resource of 4.06 Mt @ 4.66 g/t Au for 608,000 ounces of gold, and an Inferred resource of 0.443 Mt @ 4.8 g/t Au for 68,000 ounces of gold.



Open Pit and Underground Potential Resource Classification (local grid)

All classified resource blocks located between the surface and the designed pit with grades greater than 0.30 g/t Au were included in the reported open pit mineral resources. Mineralization located below the pit shell is considered potentially amenable to underground mining methods when constrained by strings representing continuous mining blocks and reported above 2.5 g/t cut off.

Mineral Resource Estimate March 2021

	Open Pit (> 0.30 g/t Au)			Potential Underground (> 2.5 g/t Au)			Total		
Category	Tonnes (kt)	Grade (g/t Au)	Gold (koz)	Tonnes (kt)	Grade (g/t Au)	Gold (koz)	Tonnes (kt)	Grade (g/t Au)	Gold (koz)
Indicated	3,674	4.51	532	386	6.13	76	4,060	4.66	608
Inferred	32	2.54	3	411	4.95	65	443	4.78	68

Notes:

- Open Pit Mineral Resources are reported at a cut-off grade of 0.30 g/t Au. A designed pit wireframe was used to constrain the resources.
- Underground Mineral Resources are estimated at a cut-off grade of 2.5 g/t Au, beneath the open pit constraint and inside the high-grade wireframe lode models.
- The Mineral Resource is considered to have reasonable prospects for economic extraction by open pit mining methods above a 0.30 g/t Au and within a designed pit wireframe.
- Mineral Resources below the pit shell are considered to have reasonable prospects for economic extraction at a higher cut-off of 2.5 g/t where mineralization is continuous.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- The Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves.
- Totals may not add exactly due to rounding.
- The statement used the terminology, definitions and guidelines given in the CIM Standards on Mineral resources and Mineral Reserves (May 2014) as required by NI 43-101
- Average Mineralized bulk density is 2.68 t/m^3.
- Mr I Taylor, MAuIMM (CP), Principal Geologist of Mining Associates, is responsible for this Mineral Resource statement and is an "Independent Qualified Person" as defined in NI43-101.

1.5 MINERAL RESERVE ESTIMATE

The Mineral Resources were converted to Mineral Reserves by the following process:

- The cut-off grade was determined based on optimal cut-off grade as constrained by the defined pit. This was cross-checked using the economic cut-off grade which is determined by the metal selling price and cost, processing cost and recovery, and general and administration (G&A) costs.
- Appropriate mining dilution and mining recovery factors were determined based on a re-blocking exercise of the resource block-model. This resulted in an edge dilution that amounted to approximately 12% weighted average across the deposit.
- Final and interim pit shells were defined using the Lerchs-Grossmann algorithm in Geovia Whittle software, incorporating project specific contract mining costs.
- Pit designs were completed based on the selected pit shells, incorporating appropriate geotechnical mining constraints.
- A Life of Mine (LOM) schedule was formulated based on the pit designs, incorporating appropriate mining equipment production rates consistent with the basis of the quoted mining costs.
- A project economic evaluation was completed.

The ultimate Grade Control Model will have a Selective Mining Unit (SMU) block size of 0.375m x 3m x 1.5m, suitable for the scale of primary mining machinery at the mine. As at the end of March 2021, the Proved and Probable Mineral Reserve is estimated as 4.0 million tonnes at 4.0 g/t Au.

The Indicated Mineral Resources have been converted to Probable Reserves. Unclassified/Inferred resources that fall inside the $1,650/oz ultimate pit design are excluded from reserve reporting.

The mining production schedule for the design Mineral Reserve pit is described under Item 16 of this report.

The reserve ore tonnes have increased from 3M to 4M tonnes at a consistently high 4 g/t Au of Probable material. A large majority of the increased tonnes have come from successfully converting inferred resources to indicated resources, particularly in the southern extent of the Segilola deposit.

Mineral Reserve Estimate March 2021

Open Pit	Tonnes (Mt)	Grade (g/t Au)	Contained Metal (kOz Au)
Probable	4.0	4.0	517
Total Ore Reserves	4.0	4.0	517

The deposit is amenable to conventional open pit mining methods and gold processing using conventional comminution, gravity concentration, and Carbon in Leach (CIL) recovery.

The Project designed in the DFS is an open pit operation feeding a conventional gold processing process plant. The projected Life of Mine (LOM) is approximately five and a half years, comprising approximately four years of open pit mining with processing continuing for a further 14 months. The LOM ore production is 4.0 million tonnes (Mt) at an average grade of 4.0 g/t Au. The process plant is designed for a throughput of 715,000 tpa and gold recovery is projected to be 97%. The total gold mined over the LOM is 517,000 oz Au. The process plant produces an annual average rate of 83.6koz Au for approximately six years recovering a total of 502koz's Au.

1.6 MINERAL RESOURCE AND MINERAL RESERVE RISK ASSESSMENT

Structural complexity is associated with the cross-cutting dykes. There may be additional unidentified dykes to the south.

The offset between Lodes 100 and 200 is only defined by current drilling. Forest prevents surface mapping of the interpreted off-set.

Gold does occur as visible gold and nuggety gold deposits have highly skewed grade distributions.

The hanging wall lodes (Lodes 400, 500) are less continuous than the main lodes within the Segilola mineralized zone. While these lodes add to the tonnes and grade available, they are not the main drivers for the pit shell. The lodes have been modelled to highlight the potential locations of structures carrying grade, areas that should be targeted with grade control drilling.

A review of the financial model supports the robustness of the reserve.

Minor design aspects may pose operational risk (slope stability and rainy season pit dewatering).

In view of the tight mining width of the various lodes in several areas, dilution in excess of the tight optimisation assumption of 12% is possible. This will have a marginal impact on production (head grade), and thus financials. This poses mainly a financial risk and not a Mineral Reserve risk.

The mining contract structure will require strict adherence to planned volumes. If these are not achieved, unit rates will increase, which places performance risk in the mining section with SROL. This poses a financial risk not Mineral Reserve risk.

1.7 MINING RISK ASSESSMENT

Slope design parameters especially on the eastern wall must be critically monitored, as batter angles are sub-parallel to foliation in, especially footwall contact areas.

The final bench height of 24m on the final pit walls in the southern section of the pit could pose operational challenges. It should therefore be considered to halve these to 12m heights, without flattening the overall angles – 12m heights will enable all areas to have similar mining heights, berm positions and decrease toe loading (risk of failure).

In-pit dewatering holes will create more of a production nuisance than benefit over the relative short mine life. External dewatering holes in correct aquifers will add value.

The 6-month rainy season will have a significant impact in terms of water volumes collecting in the pit.

Current pit topography has overburden on high slopes, with the lowest part of the valley being on mineralised outcrop. Water ponding during the rainy season may hamper ore mining. Overburden pre-stripping will be critical to enable alternate water ponding and collection on waste areas, away from ore mining faces. The creation of sumps and installation of adequate pumping and piping infrastructure in the pit will be critical to ensure achievement of the mining volumes according to budget and plan.

The proximity of housing and other public infrastructure close to the mining activities will have to be approached with due care and supportive of a long-term relationship, as mining will impact on the communities with regards to safety, health, environment, and infrastructure. The blast clearing radius (exclusion zone) of the southern-most part of the pit (later stage of the LOM) will overlap with some public infrastructure. Issues may include the effects of blasting (fly rock, dust, noise, vibration), water quality for downstream users, lowering of groundwater levels potentially impacting water supply wells and boreholes in the surrounding community, and control of access to prevent ingress of people and livestock into areas where heavy equipment operates.

Based on the ore lode dimensions, dilution will remain a risk and achieving 12% or less dilution will require appropriate control and supervision over the ore mining operations.

1.8 RECOMMENDATIONS

The offset between Lodes 100 and 200 is only defined by current drilling. Better definition of this offset (fault) needs to be identified during grade control drilling and pit mapping.

The hanging wall lodes (Lodes 400, 500) should be targeted with grade control drilling.

The area between 10,800 to 11,000 mN has down dip gaps of 100 m between drill holes. These gaps in drilling coincide with the projected base of the pit. The surface above this target is steep country, and MA recommends early clearing (within the pit design) to enable additional drilling to optimally target the bottom of the pit.

Future grade control drilling should be optimised as grade continuity at Segilola is known to be erratic, which is expected in an orogenic gold deposit. The proposed close spaced drill program will be used to define better (higher resolution) variograms providing insights into the required drill spacing to define measured resources and optimise the grade control drill spacing required for accurate prediction of the feed grade on a daily or weekly basis.

Slope design parameters especially on the eastern wall must be critically monitored, as batter angles are sub-parallel to foliation, especially in footwall contact areas.

External dewatering holes rather than the use of in-pit dewatering will be of benefit and less disruptive to the mining operation.

Initial mining and plant feed schedules should be reviewed to ensure realistic alignment.

2 INTRODUCTION

2.1 ISSUER

Segilola Resources Operating Ltd. (SROL), a subsidiary of Thor Explorations Ltd. (Thor), requested Mining Associates (MA) to certify the Mineral Resource and Mineral Reserve at the Segilola Gold Project to a NI 43-101 standard and compile the NI 43-101 Report. This report is prepared in accordance with Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). MA was requested to undertake a site visit, review, and verify the data and processes used, present results and recommendations.

The Segilola Gold Project is located in the state of Osun, Nigeria, approximately 120 km northeast of the city of Lagos and 18 km south of the regional centre of Ilesha. The Project is centred on 700,987 mE, 832,281 mN (WGS84 UTM Zone 31N).

MA has not been requested to provide an Independent Valuation, nor has MA been asked to comment on the Fairness or Reasonableness of any vendor or promoter considerations, and therefore no opinion on these matters has been offered.

2.2 TERMS OF REFERENCE

This Independent Technical Report has been prepared by Mining Associates Pty Ltd ("MA") for Segilola Resources Operating Ltd ("SROL") in compliance with disclosure requirements of Canadian National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101"). MA was commissioned by SROL in January 2021 to prepare this Technical Report.

The Technical Report is prepared in accordance with the requirements of NI 43-101 and in compliance with Form 43-101F1 of the Ontario Securities Commission ("OSC") and the Canadian Securities Administrators ("CSA").

2.3 INFORMATION USED

This report is based on technical data provided by SROL to MA. SROL provided open access to all the records necessary, in the opinion of MA, to enable a proper assessment of the project. SROL has warranted in writing to MA that full disclosure has been made of all material information and that, to the best of the SROL's knowledge and understanding, such information is complete, accurate and true.

Most of the information in Items 4 to Item 11 and Item 13, has been compiled using information from the NI 43-101 Technical Report on the Segilola Gold Project Feasibility Study, Osun State, Nigeria dated March 2019 by Roscoe Postle Associates Inc. (RPA).

Additional relevant material was acquired independently by MA from a variety of sources. Historical documents and data sources used in the preparation of this technical report are listed in Section 27: References. This material was used to expand on the information provided by SROL and, where appropriate, confirm or provide alternative assumptions to those made by SROL.

2.4 CURRENT PERSONAL INSPECTION BY QUALIFIED PERSONS

The Qualified Persons for this Technical Report are Mr I. A. Taylor and Mr A.M. Burger, as defined in the regulations of NI 43-101. The current personal inspection of the property was conducted by Mr Burger between 8 April and 10 April 2021. Due to travel restrictions related to the COVID-19 pandemic in force at the time of the study, no other QP was able to visit the site.

Mr Burger reviewed the geological setting, examined rock specimens and field locations of interest, reviewed geological procedures, databases, and general geological practices. Mr Burger also reviewed all current and future mining areas, related infrastructure and site controlling conditions.

3 RELIANCE ON OTHER EXPERTS

The authors have relied on reports, opinions or statements of other experts who are not Qualified Persons for information concerning legal, environmental, political and taxation issues and factors relevant to this report.

MA has assumed, and relied on the fact, that all the information and existing technical documents listed in the References section of this Technical Report are accurate and complete in all material aspects. While MA has carefully reviewed all the available information presented to us, MA cannot guarantee its accuracy and completeness. MA reserves the right but will not be obligated to revise the Technical Report and conclusions if additional information becomes known to us after the date of this Technical Report.

For this report, MA has relied on ownership information provided by SROL. MA has not researched property title or mineral rights for the Segilola Gold Project and expresses no opinion as to the ownership status of the property.

Select technical data, as noted in the Technical Report, were provided by SROL and MA has relied on the integrity of such data. A draft copy of this Technical Report has been reviewed for factual errors by the client and MA has relied on SROL's knowledge of the Property in this regard. All statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this Technical Report.

4 PROPERTY DESCRIPTION AND LOCATION

The Segilola Gold Project is located in the state of Osun, Nigeria, approximately 120 km northeast of the city of Lagos and 18 km south of the regional centre of Ilesha (Figure 4-1). The site is situated between the town of Iperindo and Odo Ijesha village, in the Atakunmosa East Local Government constituency. The Project is centred on 700,987 mE, 832,281 mN (WGS84 UTM Zone 31N).

SROL Explorations Ltd (SROL) acquired the Project (100% interest) in August 2016 through the acquisition of Segilola Resources Operating Limited (SROL) and its joint venture partner Segilola Gold Limited (SGL) from Ratel Group Limited (RGL or Ratel), a wholly owned subsidiary of RTG Mining Inc. Thor's rights to the property are through its 100% ownership of SROL and 100% of SGL.

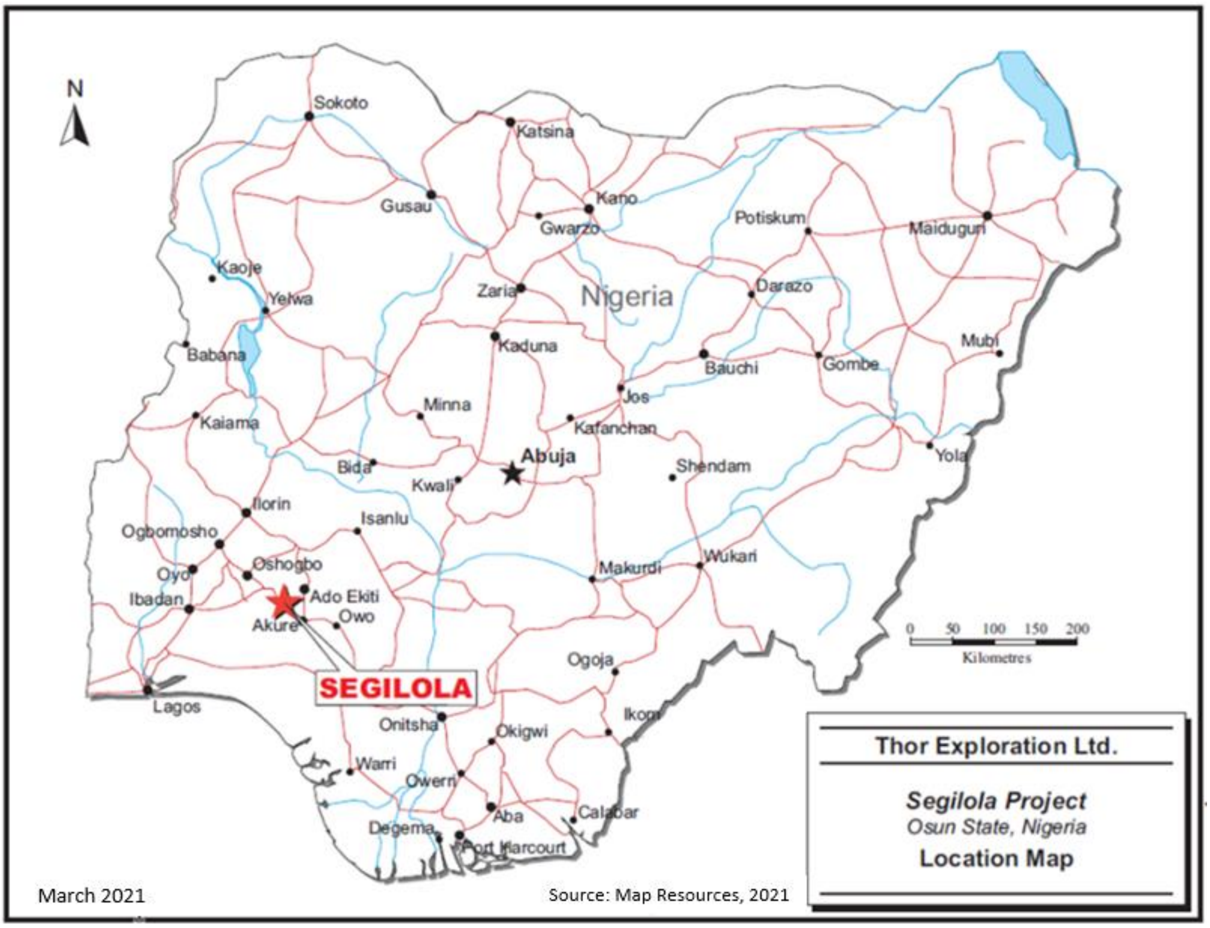


Figure 4-1. Location Map, Segilola Project, Osun State, Nigeria

4.1 PROPERTY TENURE

The property comprises a Mining Licence (ML41), which covers an area of 1,720 ha (17.2 km^2) and is contained within a larger Exploration Licence (EL19066), which covers 2,700 ha (27.0 km^2). The location of the licences is shown in Figure 4-2.

Annual fees are payable in respect of all mineral titles. In addition, 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. As of the date of this report, SROL reports that both licences are in good standing.

The Mining Licence (ML41) was renewed in September 2016 and is valid for a period of 25 years until 2041.

Exploration Licence (EL19066) was originally granted on 25 September 2014 and with the first renewal application granted with effect from 25 September 2017. The licence completely underlies ML41, i.e. the ML is not excised from the EL.

After acquiring EL19066, SROL were granted a further eight exploration licences bringing the total certified land holding to 863 km^2 (Table 4-1 and Figure 4-2). An exploration licence is renewable twice for a period of two years each.

Table 4-1.SROL Tenements

Tenement	Certified Area (km^2)
ML41	16.2
EL19066	27
EL26355	77.6
EL26356	120
EL26357	39.2
EL26358	97
EL28801	124.8
EL28802	17.6
EL29977	173.6
EL29978	170
Total	**863.0**

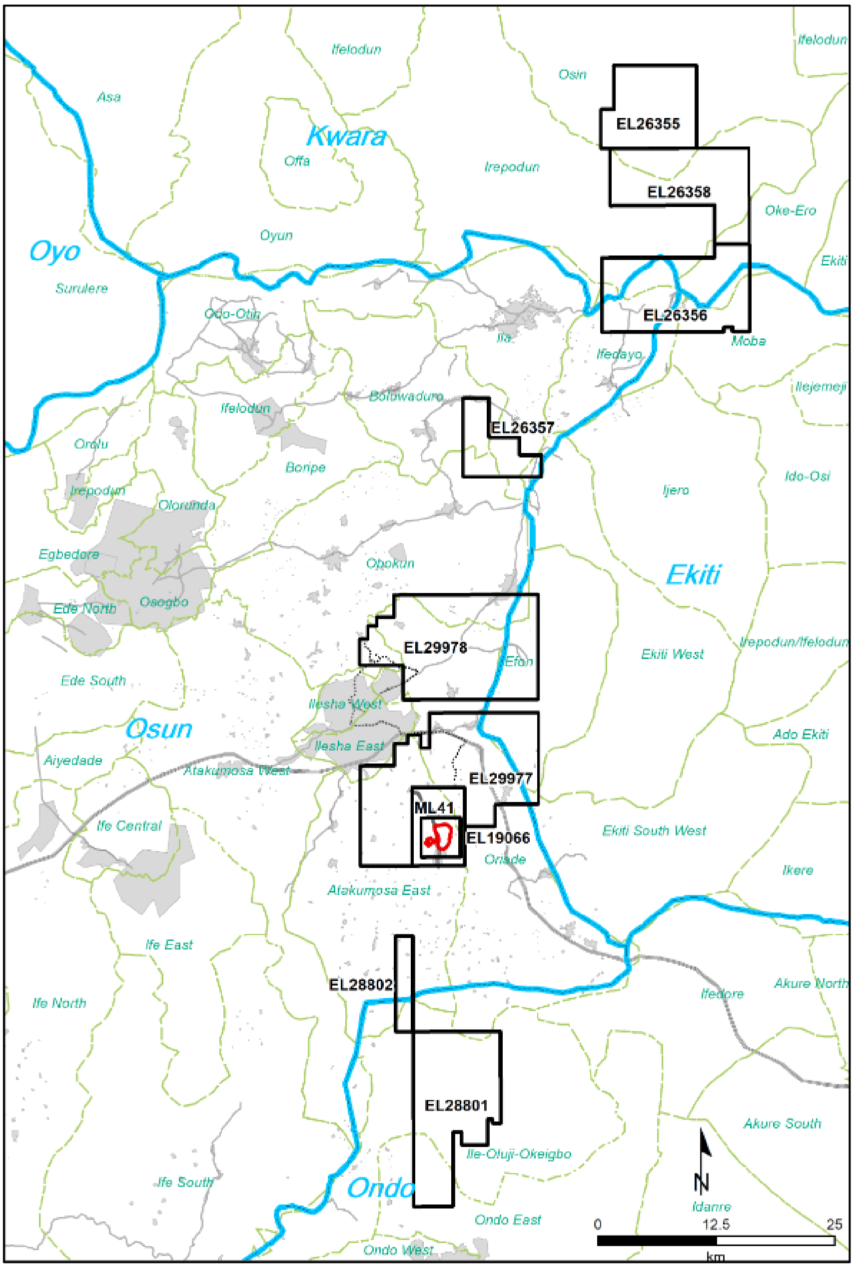


Figure 4-2. Tenure Map, Segilola Project, Osun State, Nigeria

(Source Thor 2021)

4.2 PROPERTY RIGHTS AND OBLIGATIONS

Obligations required to maintain the mining licence include:

- Meeting the prescribed reporting requirements in line with Schedule 5 of the Nigerian Minerals and Mining Regulations 2011, comprising a half yearly report on mining activities.
- Annual service fee of N100,000 per cadastral unit (approximately $275 per cadastral unit) which amounts to N8,100,000 (approximately $22,300) per annum.

Exploration Licence (EL19066) was originally granted on 25 September 2014. The exploration licence is renewable twice for a period of two years each, with the first renewal application granted with effect from 25 September 2017. The licence completely underlies ML41, i.e., the ML is not excised from the EL.

Obligations required to maintain the exploration licence include:

- Meeting the prescribed reporting requirements in line with Schedule 5 of the Nigerian Minerals and Mining Regulations 2011.
- Annual service fee of N2,000 per cadastral unit (approximately $5.5 per cadastral unit) which amounts to N270,000 (approximately $745) per annum.

As of the date of this report, licence EL19066, belonging to SROL, is believed to be in good standing with the above statutory obligations and all fees are fully paid.

Surface rights are locally owned and permission to access the licence area has been obtained from the landowners. This is a requirement when lodging a licence application.

4.3 ENCUMBERANCES, ROYALTIES AND TAXES

Companies engaged in mining activities in Nigeria are subject to a corporate tax of 30% of their taxable profits. They are also subject to an education tax of 2% on taxable profits. A value-added tax (VAT) 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 applicable to a mining operation applies to goods that are exported.

Minerals obtained from mining or exploration are subject to the payment of a royalty 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.

Royalties are payable to the Nigerian government at a rate of 5,400 Naira per recovered ounce of gold. At an exchange rate of 363 Naira to US$, this equates to $14.89 per ounce.

The Property is subject to two additional private royalties as follows:

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

According to the Nigerian Minerals and Mining Act (the Act), the holder of a mineral title enjoys the following tax incentives:

- tax exemption for the first three years of operation, which may be extended for another two 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 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, nationalization, 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 the investor is entitled and the right to a dispute settlement procedure under United Nations Commission on International Trade Law (UNCITRAL) Rules.

The Mining Sector is designated a Pioneer Industry approved by the Federal Executive Council. Pioneer status is a fiscal incentive provided under the Industrial Development (Income Tax Relief) Act (IDITRA), Laws of the Federation of Nigeria.

Eligible companies operating in designated pioneer industries, which apply for and are granted pioneer status, are entitled to income tax holiday for up to five years – three years in the first instance, renewable for two additional periods of one year. In addition to income tax holiday, pioneer companies enjoy other benefits, such as the exemption of dividends paid out of pioneer profits from withholding tax. This incentive scheme has been in place and functional for over 14 years. Thor reports that the Project will be able to benefit from the foregoing tax shield.

4.4 ENVIRONMENTAL LIABILITIES

To the extent known by MA, there are no known environmental liabilities on the Property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESS

The Project is located in the state of Osun, Nigeria, approximately 120 km northeast of the capital city of Lagos, and 18 km south of the regional centre of Ilesha. Lagos has direct flights to the United Kingdom, Europe, South Africa, and the Middle East via regularly scheduled international commercial carriers.

Access from Lagos to the Project takes approximately five hours by road. A paved public road connects the large town of Ilesha to the north and Iperindo to the south and passes through the Project area and provides year-round access to the site. The historical mine site is approximately 600 m from the paved road and can be reached by foot or by four-wheel drive vehicles throughout the year.

Three villages comprising approximately 11,000 people (mainly farmers with family holdings which are linked by minor roads and footpaths) surround the Project.

5.2 PHYSIOGRAPHY

The topography of the local area is undulating with elevations ranging from 300 m to 580 m above 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 of Iperindo. In the immediate vicinity of the Project area, the topography is gently undulating with variations in elevation of approximately 30m.

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. Vegetation in the area comprises crops such as kola nuts, cocoa, banana, and plantain, as well as secondary forest and bush fallow.

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 the Oni River. The watershed cuts across in the northern parts of the tenements. Recoverable groundwater often occurs in the weathered mantle covering the basement.

Weathering is typical of tropical environments 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 metasediments occur at higher levels. In general, saprolite can be reached within less than one metre from the alluvial terraces or other sedimentary cover.

5.3 CLIMATE

The area has a humid tropical climate with an average annual rainfall of approximately 2,500 mm, 80% of which falls in the wet season (May to October). At the Project area there are two distinct seasons (wet and dry) which are influenced by two dominant wind currents.

Mean daily temperatures range between a 32°C in March and 24°C in August. The relative humidity peaks at 81% to 91% during the wettest months (June through August). Operations are possible year-round.

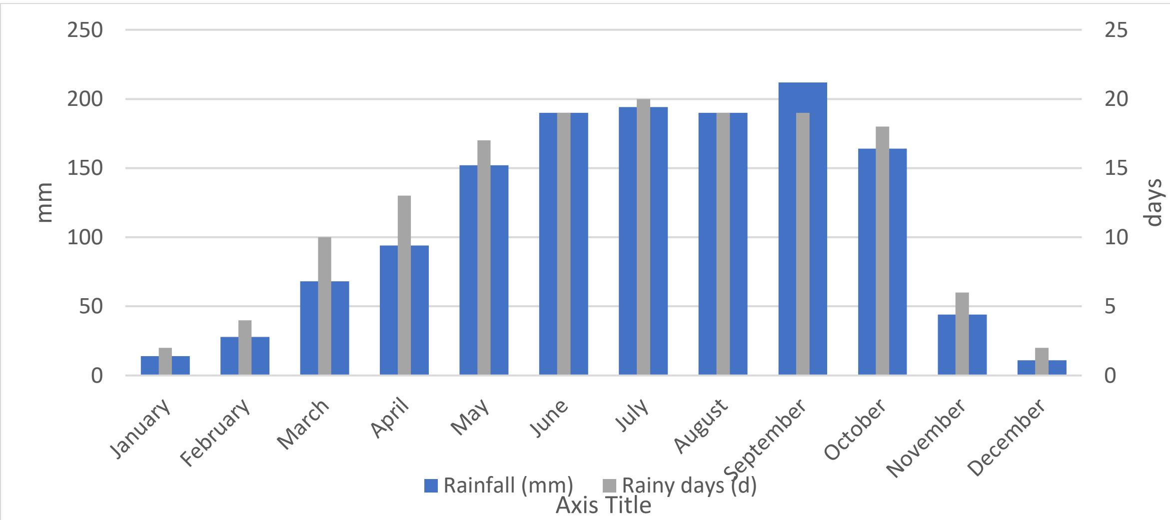


Figure 5-1. Monthly Rainfall Chart

5.4 LOCAL RESOURCES

The Project is situated between the town of Iperindo and the village of Odo Ijesha, in the Atakunmosa East Local Government constituency, with an approximate local population of 11,000. The villages of Imogbara (population 1,230), Odo Ijesha (3,480), and Iperindo (6,145) which surround the mine are agriculture/market-based communities. Housing comprises mostly concrete blocks and corrugated roofing. There are no utility provisions (e.g. electricity, potable water supply, or wastewater collection system and there is a reliance on streams and boreholes for water and gas or wood for energy supply.

Overall, the Nigerian mining industry is underdeveloped. Traditionally, the peoples of the area engage in agriculture with some of them being traders and artisans. Therefore, a large, but relatively unskilled labour pool is readily available.

To date, SROL has maintained good relationships with local stakeholders and there is a common understanding of the Project development process.

5.5 INFRASTRUCTURE

The Project is located close to the bituminised road that connects the town of Ilesha to the north and the village of Iperindo to the south. Construction of the mine camp, plant, water storage dam, tailings storage (TSF) facility and other pre-mining infrastructure is well advanced ahead of the planned commencement of operations in July 2021.

SROL maintains an office, accommodation camp, and core logging and sample preparation facility, which are all housed within a single secured compound on the outskirts of the town of llesha, located approximately 25 km north of the Project area.

At the time of this report, it is believed (based on Thor's public reports and the site visit by the Qualified Person) that the following infrastructure is complete or under construction.

- Camp construction at an advanced state
- Additional office block construction at an advanced stage
- Workshop (clearing and grubbing commenced, concrete pad poured)
- Process facility
    - Foundations, structural concrete, and steel work ongoing.
    - Generator farm concrete foundations complete.
    - Construction of crusher pocket and retaining walls in process, with the mill feed stockpile conveyor structural steelwork in an advanced state of installation.
    - Mill feed stockpile base, feeder draw-down points and conveyor tunnel complete.
    - Mill feed conveyor structure plinths complete, but structural steel still to be installed.
    - Concrete and structural steel for mill train complete, with SAG mill shell installed.
    - Leach tanks are complete, but no piping and wiring completed.
    - Electrowinning and gold plant foundations excavated.
- Water storage dam completed.
- TMF clearing and grubbing completed.
- SROL intends to start commissioning of the process plant by June 2021.

6 HISTORY

6.1 PREVIOUS OWNERSHIP

The area is well known for its gold production from eluvial placers. Modern mining of the alluvial and eluvial deposits began in 1942 and official records state an annual historic production of approximately 23,000 oz gold. The Segilola deposit, formerly known as the Iperindo Reef, was first discovered during the working of the eluvial deposits in 1945.

The ownership and exploration history prior to SROL's acquisition of the Project in 2016 is summarized as follows:

- 1947: Limited underground development was initiated comprising a shaft and an adit by the Odutola brothers.
- 1950s: The prospect was sold to a Mr Gomra, an Abadan-based Lebanese expatriate who began surface mining operations.
- 1953: Investigation of the district by the Geological Survey of Nigeria.
- 1965-1966: The llesha 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 later used by Nigerian Mining Corporation (NMC) and Ijesa GeoMin Mining Development Corporation Limited (IGMDC) in their exploration work.
- 1970: Property acquired by Obokun Minerals Developments Limited (OMDC) which rehabilitated the plant, but operations ceased due to internal company problems.
- 1976: Bureau de Recherches Géologiques et Minières (BRGM) completed mapping and geochemical surveys over the property.
- 1981: Polservice (Polish geologists and engineers) undertook a geological review, petrographic and metallurgical studies.
- 1982: NMC acquired the Project from OMDC and completed an eluvial drilling programme.
- 1983: NMC carried out geological mapping, surveying, and soil geochemistry. Old trenches were cleared, additional trenches excavated, and six holes drilled.
- 1984: NMC carried out additional exploration work and drilled 13 holes.
- 1986: NMC issued a new Exclusive Prospecting Licence (EPL).
- 1987: NMC drilled a further 14 holes.
- 1992: Pineridge Nigeria Ltd (PNL) carried out a detailed pre-investment study and compiled all the data.
- 1994: PNL entered into a joint venture with NMC.
- 1995: Tropical Mines Ltd (TML) was incorporated as a joint venture company (owned 20% by NMC and 80% by PNL). NMC was issued with 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: TMiL 19706 was assigned to TML and approved for 21 years.

- 1997: TML signed a joint venture agreement with Hansa. Hansa operated through its consultancy company Hansa GeoMin Consult and results were reported in the name of the joint venture company IGMDC. IGMDC re-surveyed the licence; rehabilitated, extended, mapped, and sampled the underground crosscut; rehabilitated several old trenches and dug new trenches; mapped and sampled all trenches; completed ground geophysical and geochemical surveys; and carried out drilling. IGMDC also completed a statistical study of the assay results, sampled the tailings, completed petrographic and fluid inclusion studies.
- 1999: TML-Hansa joint venture was terminated.
- 2007: SGL, then a wholly owned subsidiary of CGA Mining Limited (CGA), acquired the right to earn up to 51% undivided interest in the tenements. CGA commenced drilling the known mineralized zones.
- 2009: CGA declared a maiden Mineral Resource estimate prepared by Odessa Resources Pty. SGL was transferred by CGA to its affiliate, RGL, a Toronto Stock Exchange (TSX) listed entity. The same CGA management team remained as overseers of the Project.
- 2010: RGL completed a Feasibility Study (FS) for internal purposes.
- 2011 to 2012: SGL, now a wholly owned subsidiary of RGL, initiated a 4,200 m drilling programme to test the southern and northern strike extensions of the already delineated mineralization. In 2012, a Revised Bankable Feasibility Study was completed but not published. Development of the Project was delayed due to a dispute between TML and RGL regarding earned interest in the Project.
- 2016: Thor acquired a 100% interest in the Project in August 2016 through the acquisition of SROL and its joint venture partner SGL from RGL, a wholly owned subsidiary of RTG Mining Inc.

6.2 PAST PRODUCTION

From 1949 to 1969, the deposit was mined by local operators in a small-scale open pit measuring approximately 5 m wide, 15 m deep, and 300 m along strike (Figure 6-1). The operators processed the ore with a second-hand stamp mill together with a ball mill and tables acquired from Ghana.

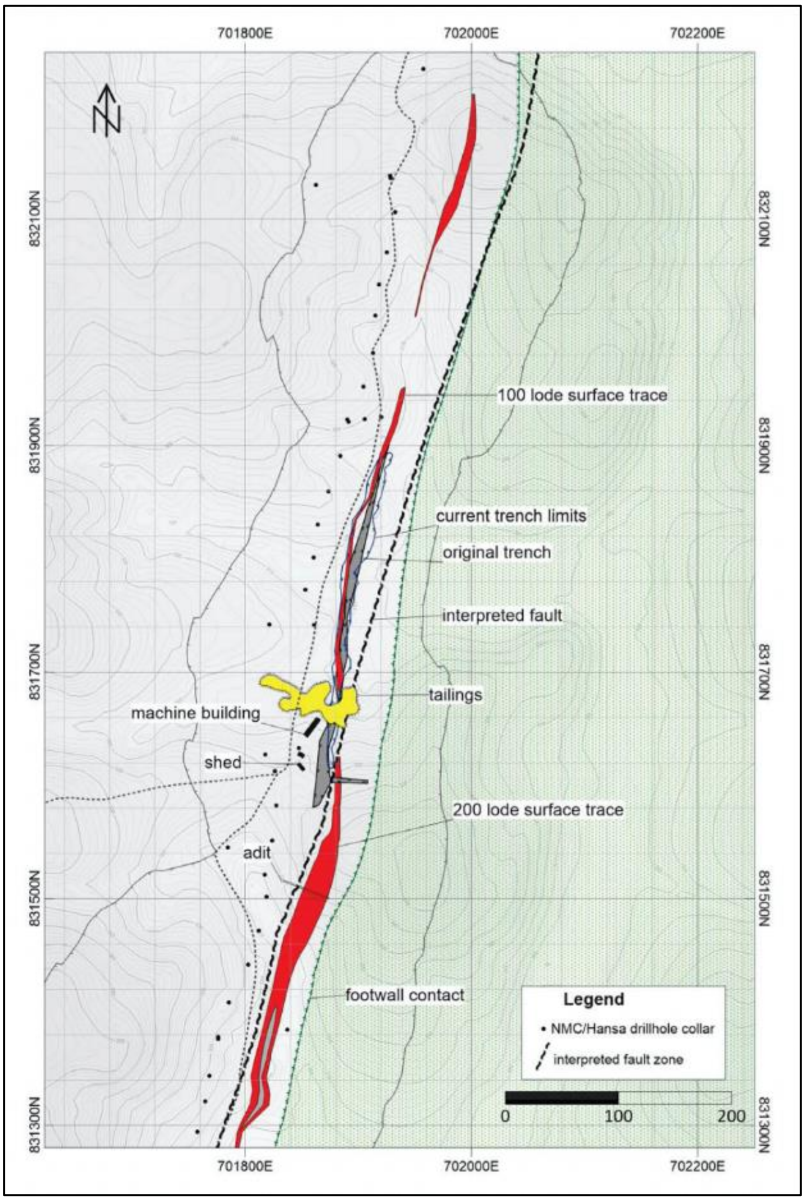


Figure 6-1. Old Artisanal Workings

(Source: Thor 2021)

There are no reliable production records from these historic mining operations.

6.3 EXPLORATION HISTORY

Historical exploration activities on the project have included geophysics, geological mapping, soil sampling, trench sampling and drilling.

6.3.1 Soil Sampling and Trenching

Soil sampling programmes have been carried out by Hansa/IGMDC and CGA.

Hansa/IGMDC collected a total of 1,882 samples between 1997 and 1999. These were taken on lines spaced 25 m to more than 200 m apart, with samples taken every 10 m along the lines, and five contiguous samples composited, so that plotted along-line sample interval was 50 m. The samples were taken from an average depth of 0.4 m, sieved to -80 mesh, and assayed with a detection limit of 0.1 ppb Au.

CGA continued the soil grid north and south along strike of known mineralization. A total of 1,016 soil samples were collected every 25 m along east-west orientated lines, spaced 200 m apart over a strike of approximately 2.8 km. Sample analyses were carried out by SGS Laboratory in Ghana. The soil sampling results from the near vicinity of the deposit and towards the south indicate contamination, most likely produced from dissemination of tailings from the old mining operation.

Both soil surveys identified anomalous gold in soils north and along strike of the known mineralization.

Trenching programmes were carried out by both Hansa/IGMDC and CGA. The trenches were typically between 0.5 m and 1.5 m deep and were focussed primarily on the outcropping vein system.

6.3.2 Geophysics

In 1997 and 1998, IGMDC contracted Terratec, of Heitersham, Germany, to carry out a ground magnetic survey. The survey totalled approximately 400 line-km, on a 100 m by 10 m format. The survey lines were located by a differential global positioning system (DGPS). An EDA Omni Plus proton magnetometer was used, with sensors at two metres and one metre and to permit measurement of magnetic gradient as well as total field. The mineralization occurs near and parallel to the contacts between gneiss (noisy data, low total magnetic intensity (TMI)) and the schist/sediment sequence (quiet data, moderate TMI).

6.3.3 Historical Drilling

Drilling has been undertaken at the Project by several previous owners. The focus of historical drilling programmes was to test the strike length of known mineralization mostly with the objective of producing a Mineral Resource estimate. Step out drilling was also completed on the northern and southern extremities to expand the resource in these directions.

Between 1984 and 1987, NMC completed 33 diamond holes for 2,962 m along the strike length of the Iperindo Reef. The holes were drilled approximately 25 m apart along the strike of the mineralization, with just a single hole on each section. The drilling intersected gold mineralization beneath the old workings.

Between 1997 and 1999, Hansa drilled seven diamond holes totalling 895 m which were designed to check and evaluate a previous study completed on the Project. Drilling was completed by Geo Core Drillers (GCD) and Hansa.

Hansa’s drilling programme comprised three different types of drilling:

- Three twin holes to compare and check the results of previous NMC holes. The assay results and geology logging largely confirmed the results of the previous drilling.
- Two deep holes to demonstrate the vertical extent of the mineralization and intersect the ore body at depth.
- Two exploration holes to step out from previous drilling and clarify the lateral extension of the mineralization.

During this period, Hansa also re-surveyed and re-logged the available core from the earlier NMC holes.

Between 2008 and 2011, CGA undertook three resource definition drilling programmes which comprised 159 holes (15,987 m). Data from this CGA drilling is the only historical data used for the current Mineral Resource estimate.

Data from the NMC and Hansa drilling has not been used in the current Mineral Resource estimate due to a lack of quality assurance and quality control (QAQC) data, a lack of verifiable downhole survey data, and the lack of verifiable core intersections due to full-core sampling.

6.3.4 Historical Resource Estimates

Historical mineral resource estimates prepared by previous owners prior to Thor's acquisition in 2016 are listed in Table 6-1. These estimates are historical in nature and should not be relied upon. A qualified person has not completed sufficient work to classify the historical estimates as a current Mineral Resource or Mineral Reserve and Thor is not treating the historical estimates as current Mineral Resources or Mineral Reserves.

Table 6-1. Historic Mineral Resource Estimates

Company	Date	Indicated Mineral Resources			Inferred Mineral Resources		
		Tonnes (Mt)	Grade (g/t Au)	Contained Gold (ozs)	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (ozs)
Pineridge Ltd	1992				1.06	10.1	347,000
Hansa	1991				1.40	6.0	270,000
Odessa Resources Pty Ltd	2009	3.66	4.40	522,000	0.82	4.1	106,000
Auralia	2017	4.04	4.30	556,000	2.03	4.7	306,000

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Project is located in the crystalline Basement Complex rocks of southwestern Nigeria within the Ilesha Schist Belt (ISB). Schist belts in Nigeria occur as north-south trending domains of Upper Proterozoic (Eburnean 2,000 Ma) 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 7-1). 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 (600 Ma) orogenic suite. Primary gold mineralization in the schist belts commonly occurs in quartz veins within several lithologies.

The ISB 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 Shear Zone (ISZ). This is a dextral strike-slip structure, which may have been active for a lengthy period, from the Proterozoic to the Mesozoic. There is a marked structural contrast between rocks to the east, where the Project is located, and the rocks to the west of the ISZ.

The Pan-African metamorphic event in northwest Africa is generally a high temperature and medium- to low-pressure event. 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 of partial melting within the banded grey gneisses which are interpreted as Archaean (Caby and Boesse, 2001), in the ISB area implies temperatures ≥700°C. For the interpreted metasedimentary sequence, assemblages of quartz – muscovite – biotite - (± staurolite ± garnet ± sillimanite) suggest maximum metamorphic temperatures of 550°C to 620°C and pressures of 4.5 kbar to 5.0 kbar.

The ISB has accounted for a significant proportion of Nigeria's limited gold production. Significant alluvial-eluvial occurrences are known in the amphibolite belt to the west of the ISZ, particularly around Itagunmodi, which is located 15 km to 20 km west of the Project. However, Segilola is the largest known bedrock source of gold in the area. 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.

A plan of the regional geology is shown in Figure 7-1.

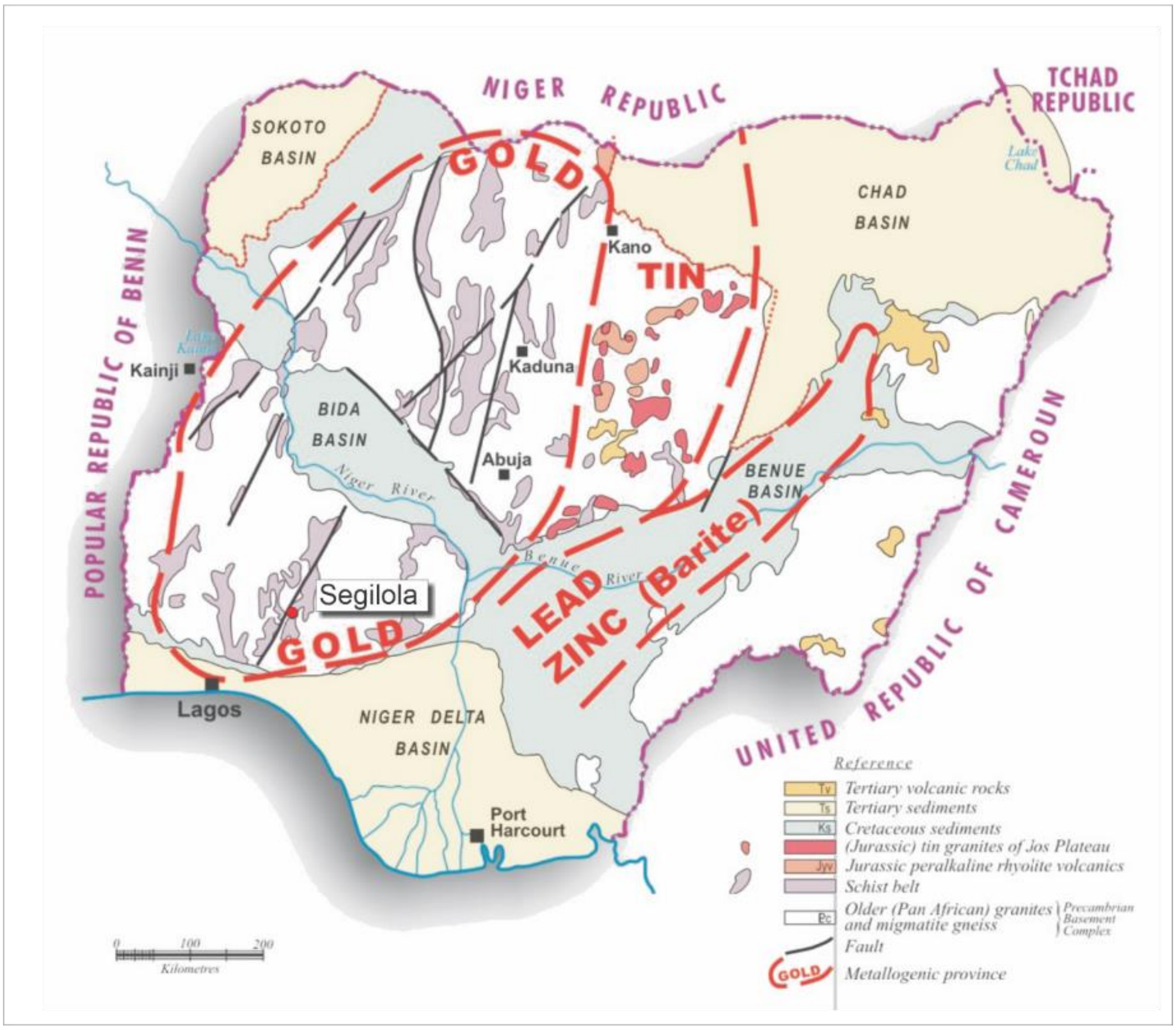


Figure 7-1. Regional Geology showing Metallogenic Provinces, Segilola Project, Nigeria

7.2 LOCAL GEOLOGY

7.2.1 Lithology

According to Caby and Boesse (2001), an Archaean basement (U-Pb zircon ages of 2,600 Ma) outcrops to the west of the area (Figure 7-2). It is typically composed of grey gneiss with lenses of orthogneiss of tonalitic-granodioritic composition, along with some lenses of amphibolite. It is extensively intruded by probable Pan-African granodiorite sheets. Caby and Boesse (2001) also recognise orthogneiss units of late Paleoproterozoic age (U-Pb zircon ages 1,850 Ma) outside the area shown in Figure 7-2, but which may include the orthogneisses around the Project.

The extensive series of aluminous schists, quartz schists, and quartzites present in the Project are interpreted to be Proterozoic sediments. They frequently display preserved sedimentary bedding. The broader sedimentary unit may be interlayered with syn-kinematic orthogneiss after felsic intrusives. Metavolcanics and meta-porphyries of dacitic composition are also recognisable within this broad grouping. A large belt of mafic and ultramafic rocks, known as the Mokuro Massif, occurs to the west of the ISZ. The massif is interpreted to be 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 Mokuro Massif contacts are tectonic, and it is interpreted as a largely flat-lying lens, underlain by schists.

Late-kinematic Pan-African granitic to granodioritic intrusives also occur.

7.2.2 Structure

Caby and Boesse (2001) distinguish two main deformation events; D1 and D2, both of PanAfrican age. D1 generally produced recumbent (flat-lying) high temperature foliations with fold axes typically trending from 120° to 150°. The recumbent attitudes are extensively preserved to the west of the ISZ and are interpreted by Caby and Boesse as a thrust stack, developed above an Archaean basement. Associated stretching lineations trend 040° to 080°, which is interpreted as the overall sense of tectonic movement.

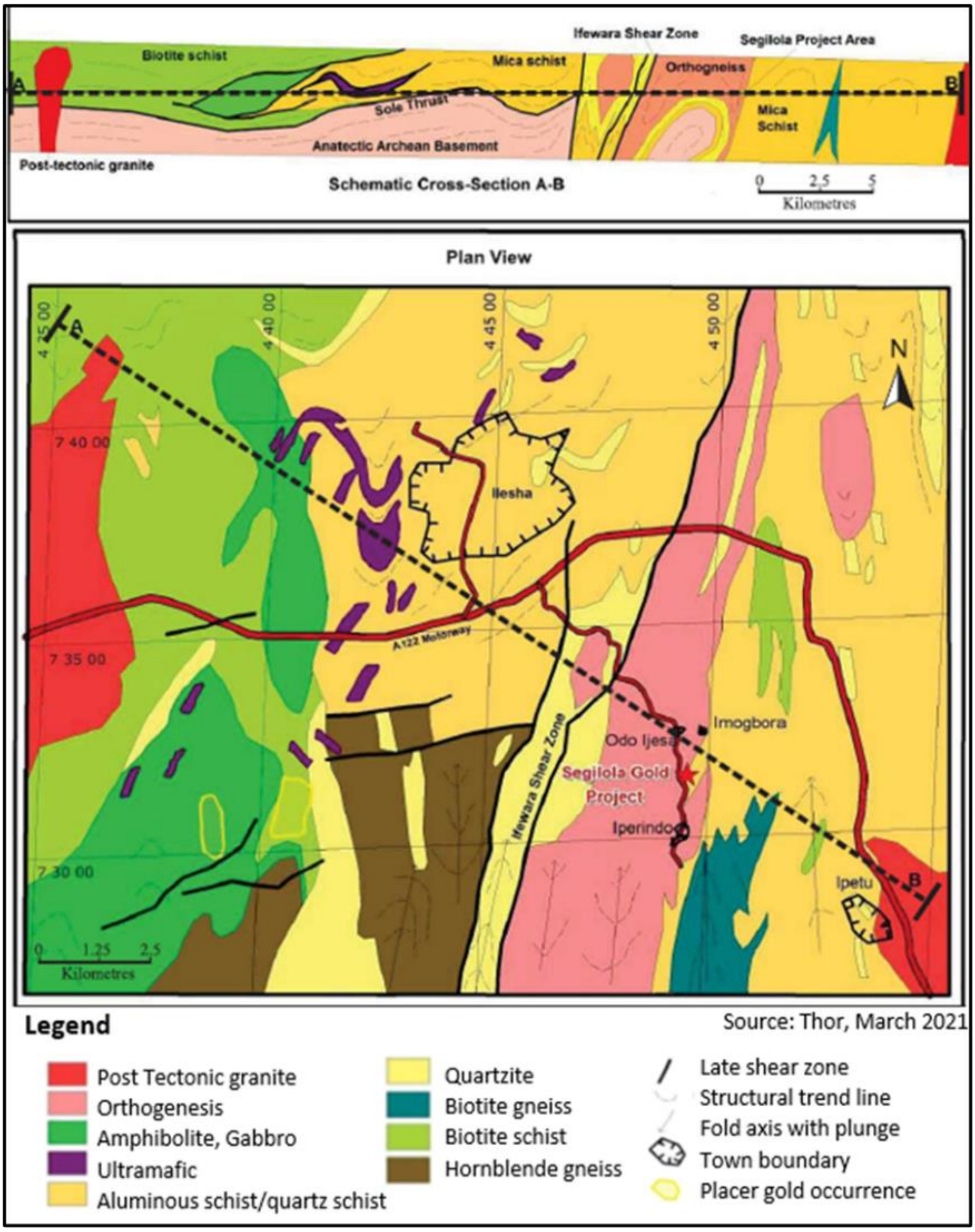


Figure 7-2. District Geology, Segilola Project, Osun State, Nigeria

7.3 PROPERTY GEOLOGY

7.3.1 Lithology

The property stratigraphy consists of a series of quartzite schists, a gneissic sequence, and surficial alluvial sediments (Figure 7-3). The quartzite schists are composed of quartzite, quartz-mica schist, and meta-sediments such as garnet-biotite schist. The gneissic sequence is divided into paragneiss (biotite gneiss), orthogneiss (granite gneiss and pegmatoidal gneiss), and undifferentiated gneiss (those that cannot be differentiated into the above groups). The stratigraphy trends north-northeast and generally dips steeply towards the west.

The orthogneisses underlay topographic highs with rounded tops which are often poor in vegetation. These orthogneisses dominate the western parts of the exploration licence but are also found in the Kajola Ridge at the eastern margin of the licence. The orthogneisses, which are considered to be the basement rocks, are located stratigraphically below the quartzite schists and meta-sediments. The sequence appears to be overturned with the younging direction towards the east which places the older basement gneissic rocks above or in the hanging wall to the younger footwall rocks. The northernmost outcrops of this unit have been found at the lower eastern side of the Kajola Ridge. In the area of Kajola, the orthogneisses are building up the whole eastern flank of the ridge.

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 granitic and a metamorphic overprint has caused a weakly developed foliation.

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 orthogneiss. Garnet bearing schists/gneisses outcrop in the steep valley of the Arafa River. This outcrop is isolated and disappears below the adjacent quartzite hills.

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

According to Oyinloye (2006), in the immediate area of the Project, 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 unmineralized gneiss samples by Oyinloye and Steed (1996) suggested they are S-type granitoids (derived from partial melting of sedimentary source rocks).

In deeper drill holes, 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. Zones of massive carbonate (calcite) occur within the sequence and are presumably marble; although it is possible that they are carbonate veins. It is not yet obvious from the drilling whether the calc-silicates are related to a contact metamorphic environment (i.e. they are skarns). They are locally sulphide-bearing and so have some potential for gold (± copper) mineralization, however, the drilling has not identified gold mineralization in the footwall rocks within the Project area.

7.3.2 Structure

The prevailing strike of metamorphic foliation and banding is to the north-northeast, with dips predominantly steep to the west, but locally steep towards the east. Stereographic projections compiled by Hansa show a preponderance of measured foliations dipping steeply from 270° to 290° or from 090° to 110°. Mapped joints, however, cluster with steep dips towards 005° and 185°. In measurements from the oriented core there is a preponderance of foliations dipping steeply towards 315° or 135° (20 measurements) and fractures and joints dipping steeply towards 320° and 140° (104 measurements). 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, although they do not indicate if this observation is based on a significant number of measurements.

7.3.3 Mineralization

Drilling results demonstrate that gold mineralization occurs in fractured pale to dark grey coloured smoky quartz veining, sheared pegmatite, and silica/chlorite/carbonate alteration. The mineralization is hosted in three steeply dipping vein sets or lodes; the Hanging Wall Lodes (Lodes 100 and 300, and minor lodes 400, 500 and 600) and the Footwall Lode (Lode 200). Together these form an elongate mineralized zone striking 010° and dipping 60° to 70° towards the west within a developed shear zone, primarily in biotite gneiss. The currently drilled mineralized zone is approximately 2,000 m in strike length, between 70 m and 200 m in depth, and between 2 m and 18 m in true thickness.

7.3.4 Sequences

The mineralization is developed within an overturned sequence of metamorphosed, strongly foliated quartz sediments (quartzites/quartz biotite schist) at the boundary between the basement biotite gneiss (hanging wall) and calc silicate and mylonitic biotite-garnet schists (footwall). A unit of massive to foliated granodiorite conformably intrudes the sequence between the quartzites and basement gneisses. Pegmatitic veins, which are mostly conformable to schistosity, permeate the quartzite and footwall rocks. Gold mineralization is associated with late stage weakly foliated to undeformed 'pegmatitic' veins and is restricted to the quartzite unit.

Based on drilling information, the deposit is divided into the 'Hanging Wall Sequence', 'Mine Sequence', and 'Footwall Sequence' which relate to the sequence of pegmatite-intruded gneissic, schistose, and mylonitic rock types that occur to the east of the ISZ (Figure 7-3). The depth of weathering varies from 1m to 2m in the west to 5m to 10m in the mineralized shear zone.

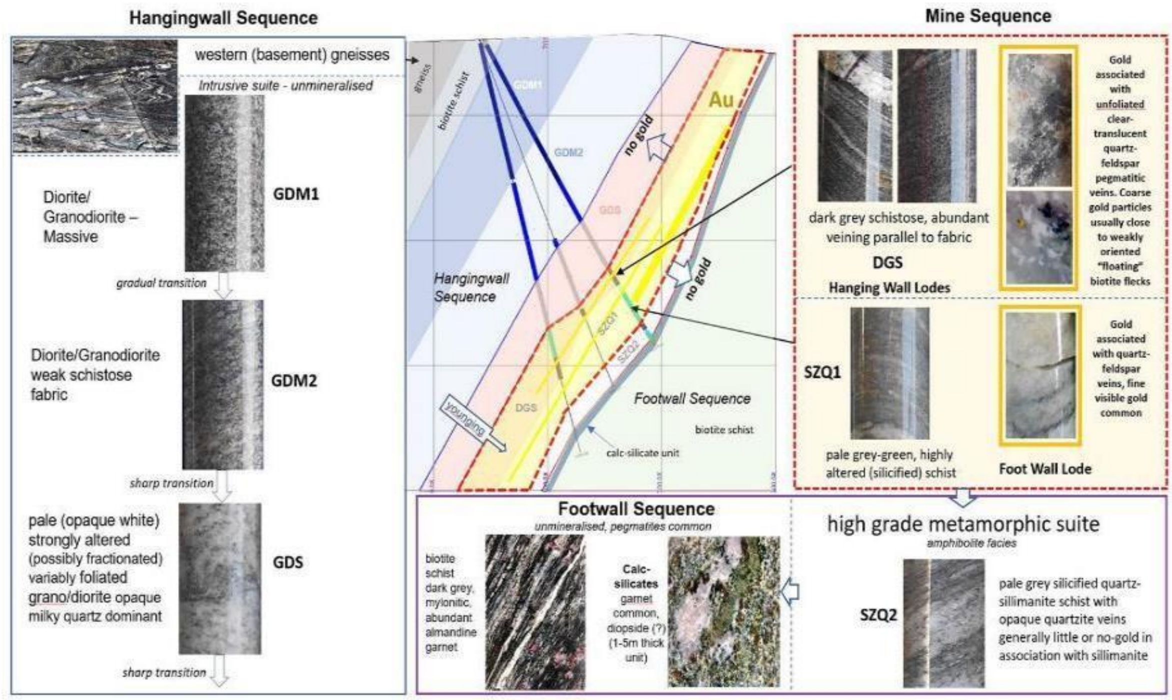


Figure 7-3. Segilola Mine Sequence

(Source: Thor 2021)

The Hanging Wall Sequence consists of a granodiorite unit (GMD1, GDM2, and GDS) that intrudes basement gneisses located to the west and gold-bearing quartzite unit. From west to east, the granodiorite gradually transitions from massive to weakly foliated, and then to strongly foliated as it approaches a sharp transition with the Mine Sequence. Higher gold grades and greater thicknesses are developed adjacent to a 5 m to 20 m thick zone of intense quartz-carbonate flooding located at the eastern margin of the Hanging Wall Sequence. It is possible that the alteration zone could be a differentiated portion of the large granodiorite sill-like body.

The Footwall Sequence consists of a calc-silicate unit and biotite schist. This sequence is separated from the Mine Sequence by a high-grade metamorphic suite consisting of pale grey silicified quartz-sillimanite schist with quartzite veins and generally little or no gold.

The Lode Sequence contains intensely foliated and sheared rocks; DGS and SZQ1. DGS consists of dark grey, quartz-biotite schist with veining parallel to the gneissic fabric and hosts the Hanging Wall Lodes (Lodes 100 and 300). Gold in these lodes is associated with quartz-feldspar-pegmatitic veins and coarse gold particles are usually associated with biotite flecks. SZQ1 consists of pale grey-green, highly altered schist and hosts the Footwall Lode (Lode 200).

Table 7-1 summarises the characteristics of the lodes.

Table 7-1. Lode Nomenclature and Description

Lode	Description	Grade Character	Estimated True Width (m)	Interpretation
100	Hanging Wall Lode	~7 g/t Au	2 - 4	northern continuation of Lode 400
200	Footwall Lode	~3 g/t Au	4 - 5	developed only east of oblique strike-slip fault
300	Footwall Lode	~2.5 g/t Au	5 -8	developed only east of oblique strike-slip fault
400	Hanging Wall Lode	~7 g/t Au	1 - 3	southern equivalent of lode 100
500	Hanging Wall Lode	~2 g/t Au	1 -3	discontinuous, southern lode
600	Hanging Wall Lode	~1 g/t Au	2 - 3	minor discontinuous lode

Lode 100 is relatively discrete with sharp upper and lower contacts. By contrast, Lode 200 is characterised by a wider, more diffuse, and lower-grade mineralization developed around high-grade veins. Lode 300 is located approximately 20 m to 30 m stratigraphically above (west) and parallel to Lode 100 and is best developed in the southern part of the Project. Lode 300 is characterised by some of the highest gold grades with finely disseminated visible gold particles in vein material.

The interpreted geology of the mining licence is shown in plan view and in cross section in Figure 7-4 and Figure 7-5, respectively.

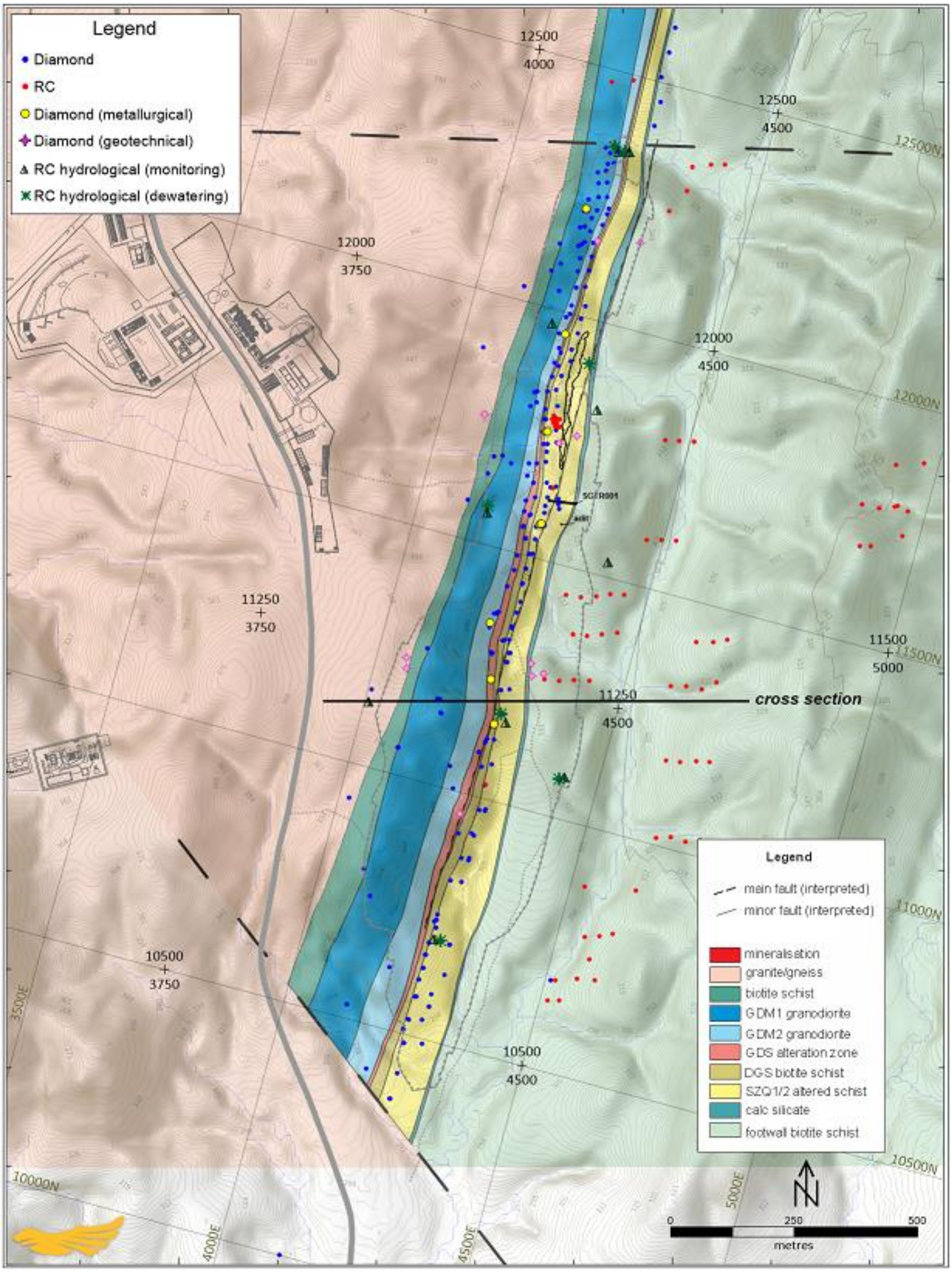


Figure 7-4. Plan View of Interpreted Geology

(Source: Thor 2021)

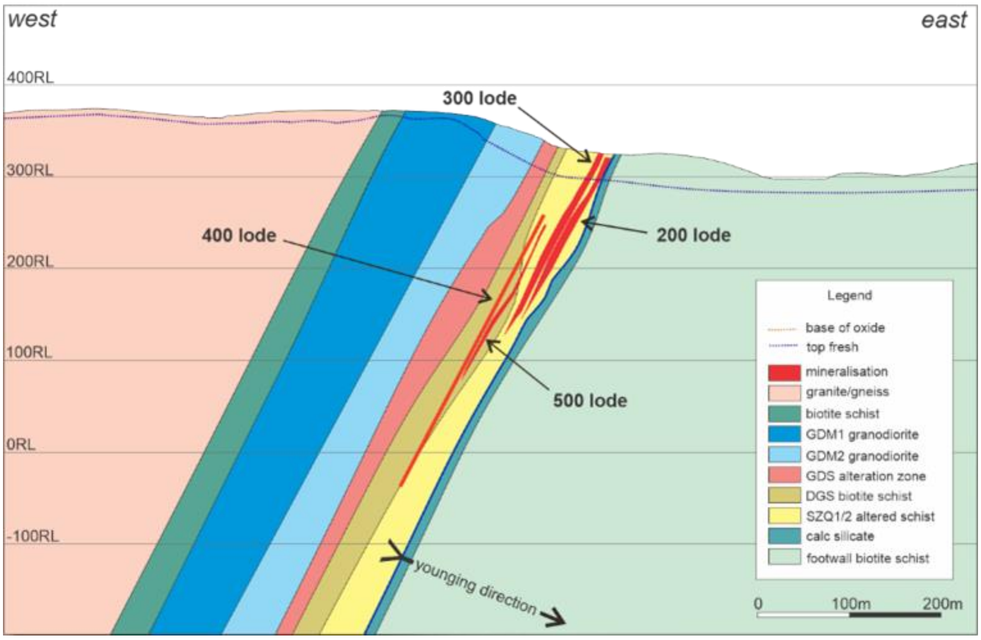


Figure 7-5. Cross Section 831 150 N

(Source: Thor 2021)

There are two styles of faulting: high angle (east-west oriented) oblique faults and strike slip faults. Two dominant oblique faults, which post-date mineralization, are located at the northern and southern extremities of the deposit respectively and have little or no associated displacement (Figure 7-4). A series of north-northeast trending steep strike-slip faults are interpreted within and along the entire length of the mineralized shear zone and are thought to intersect the westerly-dipping footwall mylonite zone in the northern part of the deposit (Figure 7-6).

Strike slip faults occur along the axis of the mineralization which, itself, may be developed either side of the faults and is only lost within the fault itself, over several metres, in the north-central part of the deposit.

To the north, the two features converge with the fault passing into a mylonitic footwall. In this area, the main Footwall Lode 200 is absent as it has been faulted out.

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.

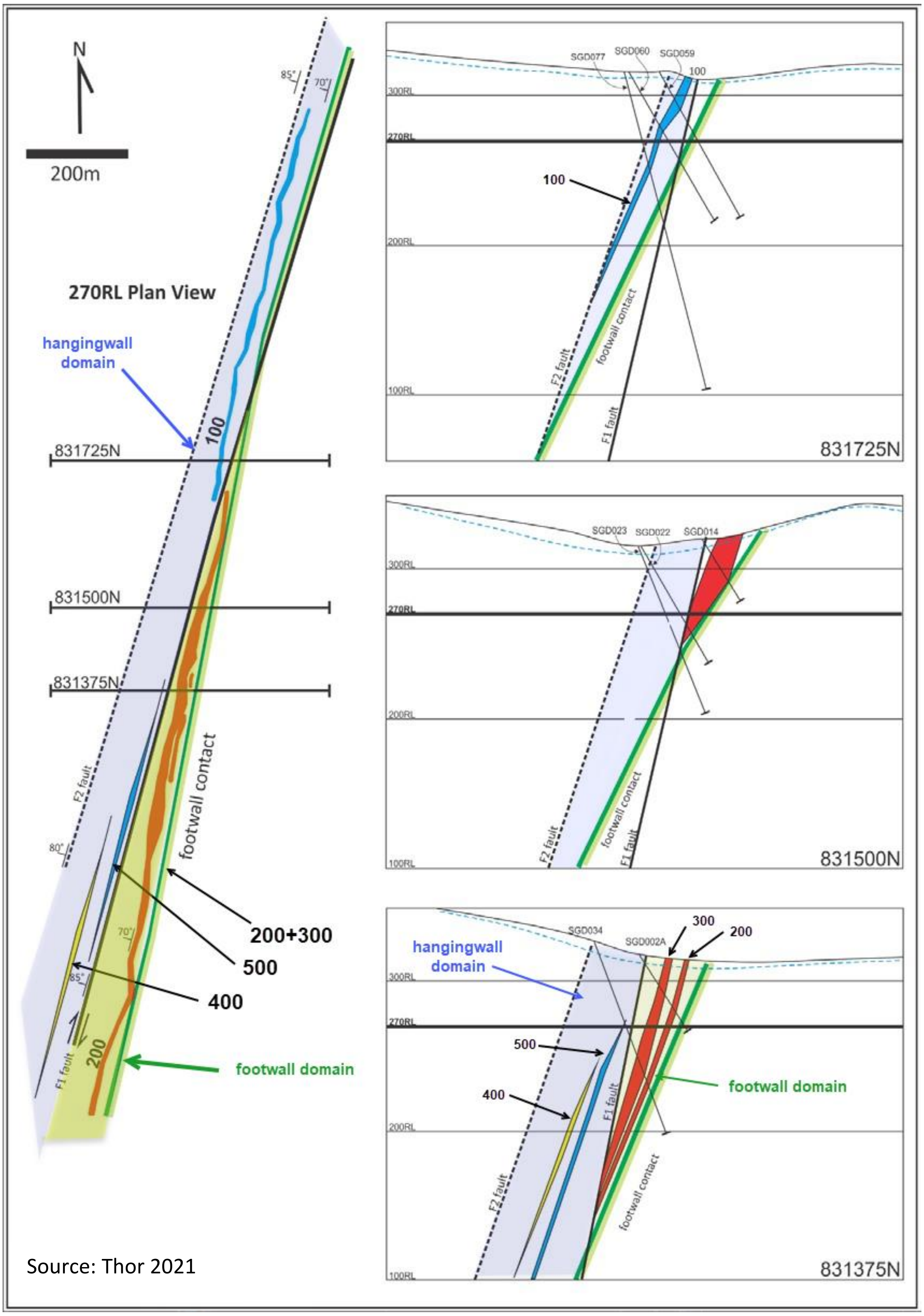


Figure 7-6. Schematic Cross Sections along strike (at 270 mRL)
showing structural interaction of the lodes, footwall, and sub-vertical strike slip faults

7.3.5 Mineralogy

The mineralogy of the Segilola deposit is characterised by its general simplicity and consistency. The gold is entirely non-refractory and commonly occurs as visible particles within either pegmatitic quartz-feldspar veins or foliated biotitic selvedges to the veins. There are no significant trace element associations such as silver with gold. However, metallurgical assaying indicates slightly elevated copper (250 ppm to 300 ppm) and mineralogical studies suggest a gold-tellurium association.

Two styles of gold mineralization are observed:

- Narrow, 1 m to 3 m thick Hanging Wall Lodes within silicified biotite schists (DGS). These lodes locally contain 5 µm to 20 µm grains of visible gold and are developed in the hanging wall to the main (footwall) lode. These lodes appear to have different controls to the footwall lode and have a more vertical continuity over shorter strike lengths.
- Wide, up to 15 m, 'footwall' mineralization within a characteristically grey-green, strongly silicified zone of biotite schists and gneisses (SZQ1).

The mineralized lodes generally comprise highly silicified fine-grained, foliated biotite gneiss typically intruded by both discordant and concordant pegmatitic quartz-feldspar veins.

Shearing, fracturing, and alteration influence the location 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.

Minor sulphides, typically pyrite, are associated with the lodes. 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. A cursory examination suggests most of these do not contain pyrrhotite. These either relate to a late episode of mineralization, or to remobilisation of sulphides.

Native gold is visible in both altered wall rock and in quartz-feldspar veins. It occurs with petzite (a silver gold telluride), within pyrite, and quartz veins. The typical size of native gold blebs is approximately 10 µm. 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 hanging wall quartz-feldspar vein mineralization (SGD156, approximately 40 g/t Au) and one of footwall lode silicified gneiss (SGD155, approximately 20 g/t Au) were prepared for polished section mineralogical study conducted by Townend Mineralogy Laboratory (Perth).

Geochemical analysis of 310 samples indicates an absence of deleterious elements. It also shows that there is no correlation between gold and silver or any base metal elements.

8 DEPOSIT TYPES

Segilola is considered to be an orogenic-style lode gold deposit within a regional scale shear zone. Primary gold mineralization in the schist belts commonly occurs in quartz veins within several lithologies.

Host rocks comprise an overturned sequence of high-grade amphibolite-facies metasediments intruded by a large, possibly differentiated, granodiorite sill-like body. The mineralization is developed within a series of steeply dipping, tabular, very continuous, late-stage quartz-pegmatite veins that do not exhibit any form of significant deformation such as folding or faulting. The geological and mineralogical characteristics of the mineralized veins are consistent throughout both the strike and down dip extents of the known resource.

9 EXPLORATION

Thor acquired the Project in 2016 and initiated exploration which included soil and stream sediment sampling, tailings sampling, drilling, and light detection and ranging (LIDAR) surveys.

9.1 SURFACE SAMPLING

9.1.1 Soil Sampling

From 2016 to 2018, SROL undertook a soil sampling programme, comprising 2,132 samples, including field duplicates. Sampling was carried out at a depth of typically 0.5 m at 50 m spacing along 200 m spaced east-west lines. Multi-element ICP analyses were carried out on a total of 1,338 samples. No trace element associations with gold were detected, although chromium showed a minor correlation.

The most significant results comprised a point anomaly of 0.17 ppm Au located 2.1 km north of the resource and adjacent to the northern projection of the mineralization (Exploration Target Area 2 in Figure 9-1).

9.1.2 Stream Sediment Sampling

During 2018, SROL undertook a reconnaissance stream sediment sampling programme comprising 180 samples. Sample sites were selected to capture the main drainage systems on the property particularly over the more prospective areas underlain by schist. Samples comprised 3 kg to 5 kg of coarsely screened creek bed sediment. Analyses were carried out by MS Analytical laboratory (MS Analytical) in Vancouver, BC, Canada using an 80-mesh screen, minus fraction analysed by aqua regia digest with inductively coupled plasma mass spectrometry (ICP-MS) finish.

The most significant results were 1.10 g/t Au and 1.46 g/t Au in Exploration Target Area 3 which is located 500 m to the west of a known structure.

A total of eight Exploration Target Areas have been delineated from surface sampling and will be assessed in forthcoming exploration programmes (Figure 9-1).

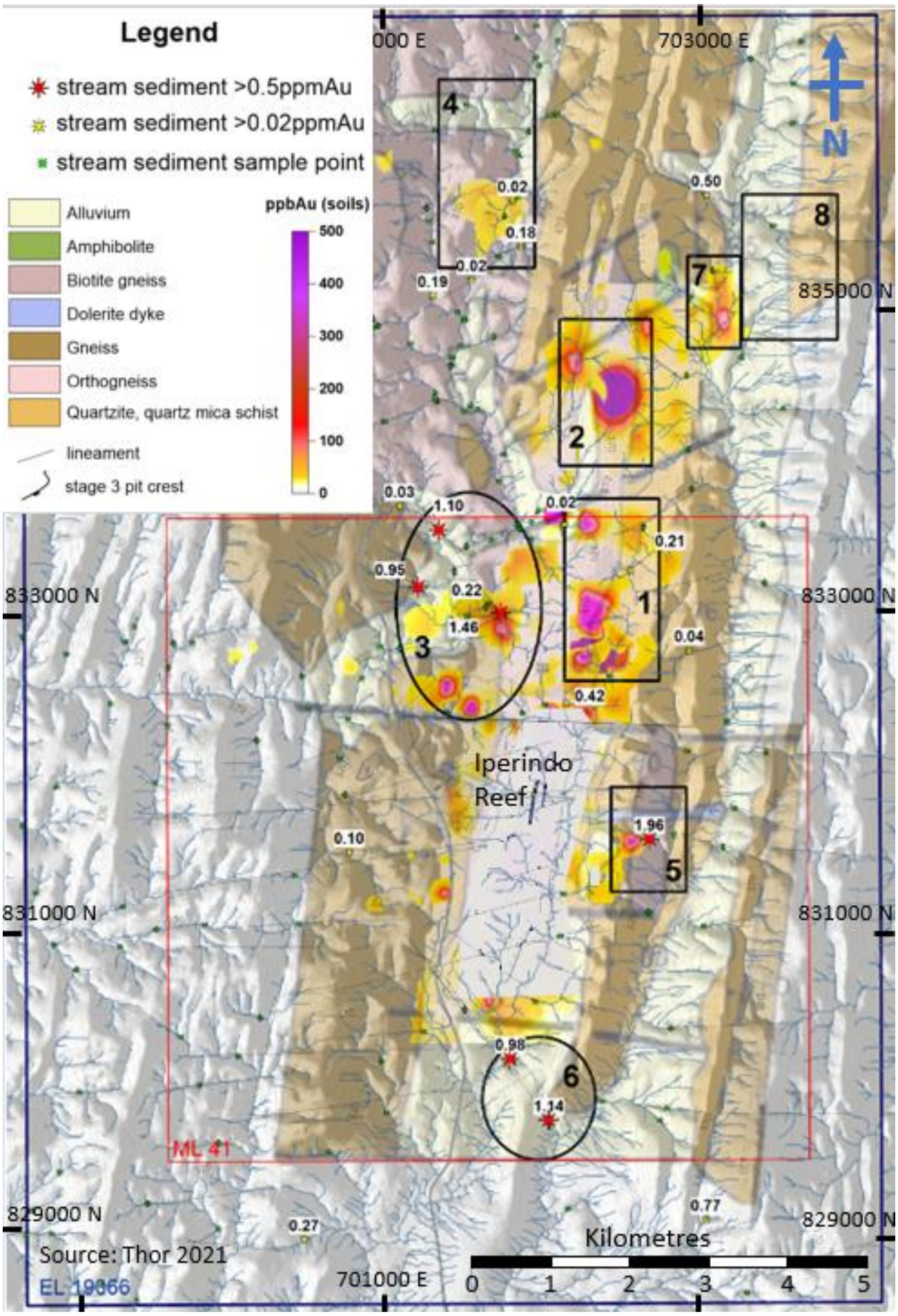


Figure 9-1. Exploration Target Areas

9.1.3 Historical Tailings

In 2018, a total of 40 samples were collected from an area of tailings located north of the concrete foundations of the machine buildings (Figure 6-1). The tailings cover an area of approximately 1,600 m^2 and have an average thickness of 1.5 m. The samples returned values ranging from 0.3 g/t Au and 10.5 g/t Au.

9.2 TOPOGRAPHIC SURVEY

In 2018, SROL contracted Southern Mapping Company (Pty) Ltd (Southern Mapping), South Africa to generate a series of base maps for the Project. Using LIDAR data taken from a fixed-wing aircraft, Southern Mapping produced 45.78 km^2 of digital imagery with a 0.10 m pixel resolution as well as a 45.78 km^2 of topographic coverage with a 5 cm vertical accuracy and 1 m by 1 m pixel size digital terrain model (DTM). Contour maps with 0.5 m and 2.5 m contour intervals were produced in AutoCAD drawing exchange (DXF) formats.

The LIDAR data and DTMs were provided with orthometric heights determined using the EGM2008 model.

9.3 GEOPHYSICS

In 2018, SROL obtained the raw data from the Nigeria Nationwide High-Resolution Airborne Geophysical Survey.

This data was released in 2008 and was used to compile a total magnetic intensity map for the Ilesha (243) 1:100,000 area. Data was collected by Fugro Airborne Surveys using a Scintrex CS2 Caesium Vapour magnetometer. Flight line spacing was generally 500 m, locally infilled to approximately 250 m. Flight lines trended at 135° and sensor mean terrain clearance was 75 m.

Total magnetic intensity in the vicinity of the Project shows several regional-scale faults are evident, including the ISZ to the west of the tenement, and a sub-parallel to parallel structure in the east of the mining licence. Several structures trending at approximately 045° are also apparent, including one that crosses the northern part of the drill-tested area. However, the line spacing is too large to resolve smaller-scale features.

10 DRILLING

Drilling on the Project was conducted by SROL in 2017 and 2018 mainly with the intention of testing the down dip extension of the mineralized zone. An in-pit, extension and sterilisation drilling program totalling 63 holes was completed in 2020. SROL's exploration concept has been to define potential high-grade extensions capable of supporting an underground mining operation to complement the currently proposed open pit operation.

A total of 53 diamond drill holes (8,359 m) were completed by SROL in 2017 and 2018. Of these holes, seven were for metallurgical purposes and seven were for geotechnical purposes (Table 10-2).

Most recently SROL drilled 18 diamond drillholes to target the lower in-pit resource. The program was designed to target and de-risk the lower portions of the in-pit resource by upgrading those portions previously classified as Inferred Resources in the 2019 Feasibility Study.

The company was not able to adequately verify the historical data through chemical assays or other means as required under the CIM Best Practices. As a result historic drilling is not used in the resource estimate (1980's NMC drilling BH01 -BH33 and the 7 Hansa holes from a 1997-98 programme).

As to be expected clearing of vegetation for overburden stripping, establishment of roads and other access ways into the pit has damaged some drillhole collars (Figure 10-1). Several hole collars were visited by the QP, and positions verified with handheld GPS to determine if collar coordinates in the database are comparable with field positions. Collars are closed off by inserting blue poly pipe into the top part of the hole, number marked with spray paint (Figure 10-2). Although not best practice, it fulfills its duty of preventing collapse and objects entering or falling into holes.

Figure 10-1. Pit surface area being stripped of vegetation and prepped for overburden stripping (9th March 2021)

Figure 10-2. Typical drillhole collar (hole SGD 282) – blue polypipe without a concrete base (9th March 2021)

10.1 DRILLING SUMMARY

The Segilola deposit has been drilled over a strike length of over 2 km. The average strike of the lodes is 010° with dips to the west of 60° to 70°. In the denser areas of drilling the holes are located on mostly 25 m to 30 m spaced sections and are generally 25 m or less across strike. In some areas of steep terrain, up to three holes were drilled at different dips from the same drill pad.

Where possible, the holes were inclined at -60° to the east, however, some holes, particularly towards the south, were inclined up to -90° to intersect the lodes. Gold mineralization is developed within a linear lode that dips at 65° to 70° towards the west. The lodes vary in thickness from 2 m to 18 m true width. The dominant sample length is one metre, with sample breaks shortened to accommodate to geological contacts.

A plan view of drill hole collar locations is shown in Figure 10-3 and the drilling statistics are shown in Table 10-1 and Table 10-2.

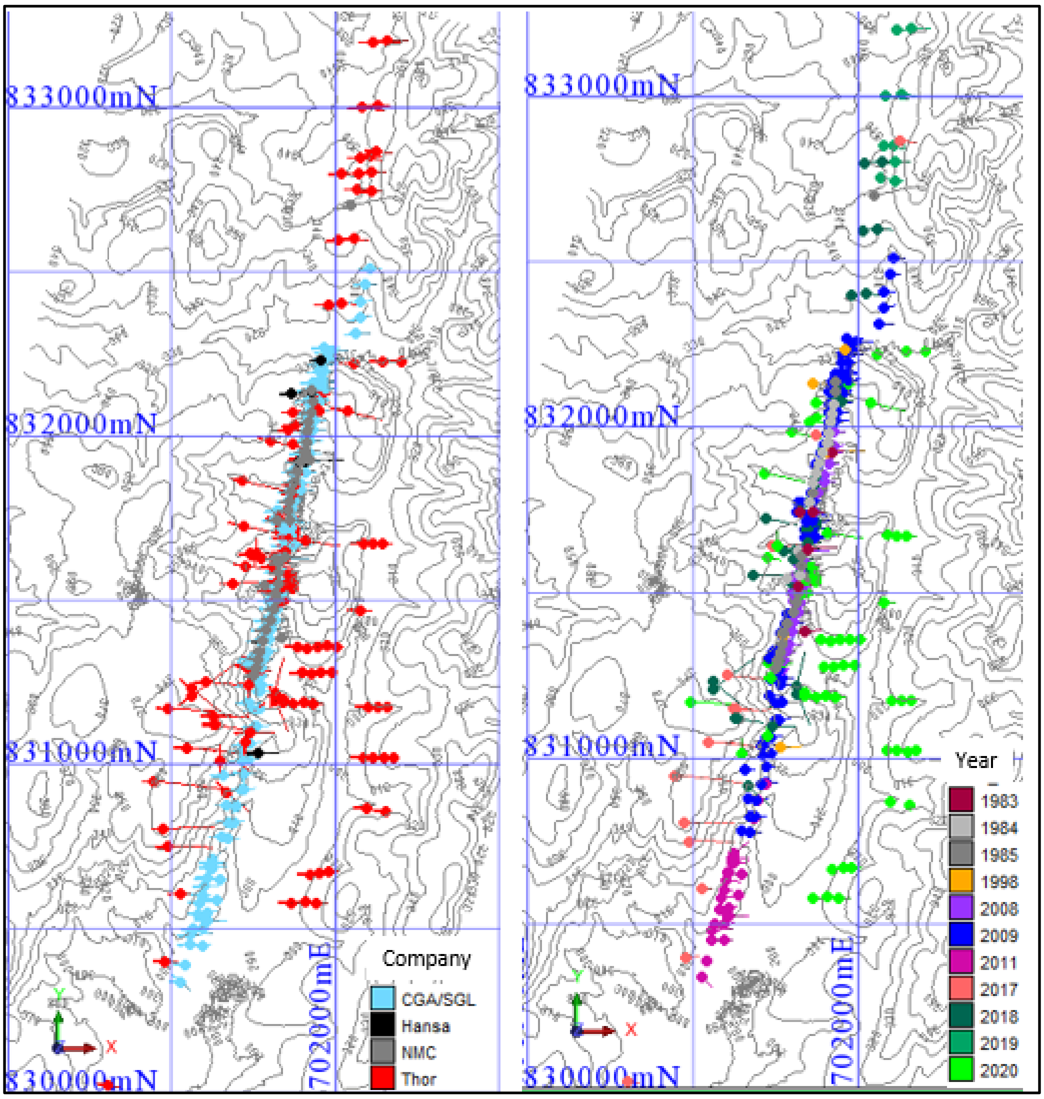


Figure 10-3. Drillhole Collar Plan

by company (left) and year (right)

Table 10-1. Drilling Summary by Year

Year	NMC No. Holes	NMC Metres Drilled	HANSA No. Holes	HANSA Metres Drilled	CGA/SGL No. Holes	CGA/SGL Metres Drilled	SROL No. Holes	SROL Metres Drilled	Total No. Holes	Total Metres Drilled
1984-87	33	2979							33	2979
1997-98			7	878					7	878
2008					34	2578			34	2578
2009					89	9644			89	9644
2011					36	3705			36	3705
2017							16	4156	16	4156
2018							55	5268	55	5268
2019							9	1006	9	1006
2020							63	6391	63	6391
Total	33	2979	7	878	159	15927	143	16821	342	36604

Table 10-2. Drilling Summary by Drill Method

Company	Year	Hole type	Count	Total metres	Start hole_id	End hole_id
NMC	1983	historic	6	579.53	BH01	BH06
NMC	1984	historic	13	1193.88	BH07	BH19
NMC	1985	historic	14	1205.19	BH20	BH33
Hansa	1998	historic	7	878.24	NIG13	TIG31
CGA/SGL	2008	DD	34	2577.92	SGD001	SGD031
CGA/SGL	2009	DD	89	9643.6	SGD032	SGD119
CGA/SGL	2011	DD	36	3705.43	SGD120	SGD154
SROL	2017	DD	16	4156.1	SGD155	SGD170
SROL	2018	GT	9	1377.3	GTFS17-002	GTFS17-013
SROL	2018	RC	18	1038	SGGC01-45	SGRC008
SROL	2018	DD	28	2852.58	SGD171	SGD198
SROL	2019	RC	9	1006	SGRC009	SGRC017
SROL	2020	RC	36	2530	SGRC018	SGRC053
SROL	2020	DD	16	3047.5	SGD199	SGD214
SROL	2020	GC	8	286	GCD01	GCD08
SROL	2020	GT	3	527.2	GTFS17-014	GTFS17-016

MA is of the opinion that SROL's drilling programmes has largely continued with common industry-standard drilling, logging and QAQC protocols and procedures established by CGA.

There is insufficient information on the procedures used for the NMC and Hansa drilling, and it is not documented here and the data from this drilling has not been incorporated into the current Mineral Resource estimate.

CGA drilling was completed by three different contractors using predominately NQ (47.6 mm diameter) sized core and minor HQ (63.3 mm diameter).

All SROL diamond drilling programmes were carried out by Century Mining Co Ltd (CMC) using Atlas Copco CS14 track-mounted rigs. HQ core was obtained in most cases except when drilling difficulties necessitated the use of NQ core.

10.2 SURVEY CORDINATE SYSTEM

The coordinate system used for surveying and data collection is the Universal Transverse Mercure (UTM) projection, Zone 31 North, using the World Geodetic System 1984 Datum (WGS84). Primary data is collected and stored in the UTM projection. To facilitate mining activities the information is converted to a local mine grid using a two-point coordinate conversion (Table 10-3). No adjustment is made to the reduced levels. Drill hole azimuths are rotated -15.4061 degrees from true north.

Table 10-3. Two-Point Grid Conversion

Grid	UTM WGS84 (31N)		Local Mine Grid	
Points	North	East	North	East
Point 1	820800.21	697073.05	0.00	2425.00
Point 2	834480.00	700740.00	14162.92	2425.00

Drill hole sites were initially located using a hand-held GPS (ProMark 2 GPS). Once the sites were located, the qualified surveyors accurately positioned the planned drill location using either a Kolida Digital Total Station or a Trimble R8 DGPS-RTK. (Differential Global Positioning System and Real Time Kinematic positioning to enhance the precision). The Trimble uses the Global Navigation Satellite System (GNSS).

10.3 DOWNHOLE SURVEY

Most of the core from SROL holes were orientated using the Reflex orientation system.

For the CGA drilling, downhole surveys were carried out by Spektra Geotek personnel using a Flexit SmartTool Downhole Survey System. Surveys were generally acquired at 25 m spaced intervals downhole on completion of each hole.

For SROL drilling, CMC drillers used a digital Single Shot Reflex camera with shots taken at between 25 m and 30 m intervals.

Drill core was recovered by the drillers and stored in boxes with markers inserted after each run to indicate the depth and any core loss or gain. At the end of the shifts, the boxes were closed and transported to an enclosed storage area at the Ilesha core shed.

10.4 DRILL HOLE LOGGING PROCEDURES

The following information was recorded from the drill core:

- Geology – Rock type, colour (using a standard colour chart), texture, grain size, weathering (oxide, transition, fresh), alteration, veins, sulphides, mineralogy.
- Structure – Azimuth/dip and dip direction, shear, fracture, joint, infill, colour, thickness, bedding, crenulation, veins, quality of the measurement.
- Sample sheet - Number, weight, mineralogy, and abundance (volume %) of veins and mineralization.
- Geotechnical - Rock strength, weathering, joint sets with type, count, angle, alteration, infill, roughness.

All data was captured directly onto paper and then transferred to Microsoft Excel spreadsheets. All parameters were logged using codes specific to the Project and these were checked daily by the Senior Geologist for completeness and accuracy. Relevant nongeological data such as Hole ID, declination, azimuth, hole depth, core diameter, date, and water ingress, were also recorded.

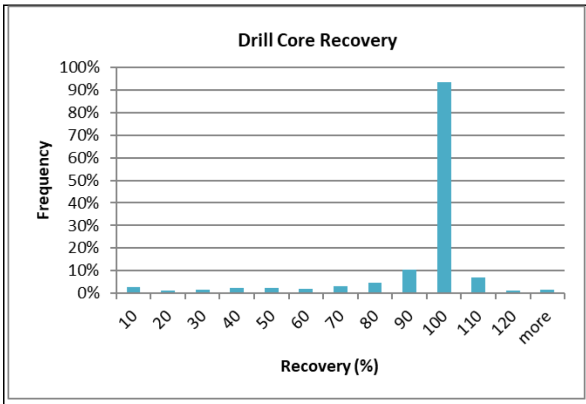
All core was photographed before being marked and cut for assaying (Figure 10-4).

A number of hard copy borehole logs were matched against the core and the database for accuracy during the site visit.

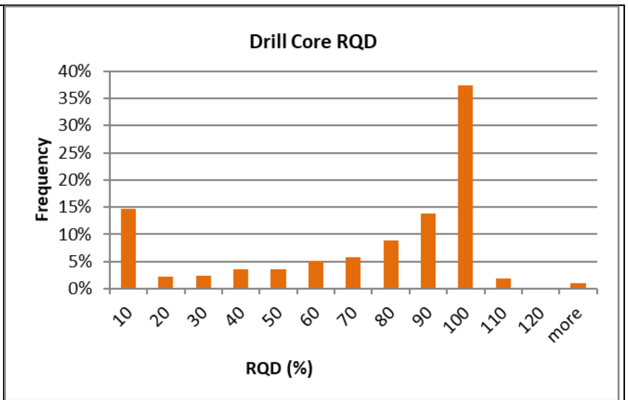
Figure 10-4. Example of Core Photography available on the database

10.5 CORE RECOVERY AND ROCK QUALITY DATA

MA has reviewed the drill core recovery results and found that recovery is good with average recoveries of 94.75% being achieved with the majority of core near 100% recovery (Figure 10-5). The hanging wall shows good recoveries. Areas of low recovery are noted to be restricted to areas within the footwall of the lodes or within the upper 20 m of each drill hole. Recovery in mineralization averages 88.87%. Rock quality data is also recoded as a percentage with the majority of core maintaining greater than 10 cm pieces (Figure 10-5).



Core Recovery Histogram



RQD Histogram

Figure 10-5. Core Recovery and RQD Histograms

No correlation between recovery and grade was observed.

11 SAMPLE PREPARATION, ANALYSES AND SECTURITY

11.1 SAMPLING METHOD AND APPROACH

As with the drilling procedures, the sampling procedures introduced by CGA were also followed by SROL. The only exception to this was the introduction of quarter-core sampling by SROL.

Samples were selected using the following principles:

- Sampling commenced at significant geological boundaries that were considered to represent a distinct change in potential grade. Such boundaries could be structural, lithological, or alteration zone contacts. The sample lengths either side of this boundary were not less than 0.5 m and no more than 2.0 m and returned to 1.0 m intervals as soon as geologically sound.
- Where barren zones were clearly identified, at the discretion of the Senior Geologist, half core was sampled over 5.0 m on both sides of the ore zone at 1.0 m intervals.

The sample intervals were recorded on the drill log. An aluminium tag (or a core marker) showing the sample number and depth from and to, was then wired or riveted into the core tray at the start of the interval.

Both half-core and quarter-core sampling was carried out, CGA dominantly sampled half NQ core (Table 11-1), whereas SROL conducted both quarter-core and half-core sampling on predominantly HQ core. SROL introduced RC chip sampling.

Table 11-1. Drill Core Sampling

Company	Sample Size	HQ	NQ	RC	Total	Percentage of Database
CGA	Half core	34	8,186		8,220	49%
CGA	Quarter Core	38	22		60	0%
SROL	Full Core	138			138	1%
SROL	Half Core	931	464		1,395	8%
SROL	Quarter Core	2,544	21		2,565	15%
SROL	~3kg Chips			4,546	4,546	27%

The quarter-core sampling was carried by SROL during 2016 to reduce sample weights airfreighted to Vancouver for analysis.

The recently completed core yard is clean, open and areas well demarcated (Figure 11-1). There are sufficient, unencumbered core cutters, with stacking space and permanent water supply (Figure 11-2). Core trays are neatly stacked/stored in metal frames, clearly labelled, with easy access (Figure 11-3).

Sampling procedures involved marking the sample boundary on the core then cutting or breaking the core at that boundary. A diamond saw was used to cut the core lengthways along the core axis of the sample interval. One half was sent for analysis, the other half was retained in the core tray. For quarter-core sampling, the half-core split was re-cut along the core axis.

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

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

- Samples for each hole are consolidated at site and the sample numbers are entered into a single submission form (i.e. one submission number).
- Weights were recorded for individual samples.
- Bagged samples were put into manageable loads in large polyweave bags.

Figure 11-1. New Core Shed Structure (9th March 2021)

storage on the left and handling, cutting and sampling on the right

Figure 11-2. Core Cutters (9th March 2021)

Figure 11-3. Core Tray Storage (9th March 2021)

11.2 SAMPLE PREPARATION

For CGA drilling, the sample preparation was completed at the SGS analytical laboratory in Tarkwa, Ghana.

For SROL drilling, sample preparation was completed in two different locations. Before 2017 the samples were prepared at the MS Analytical (MSA) laboratory in Vancouver, Canada (Table 11-2). During 2018, MS Analytical established a sample preparation laboratory in Abuja, Nigeria, and thus samples were prepared there for that drilling campaign. After sample preparation, the pulps were air-freighted to Vancouver for analysis.

All SROL samples were weighed upon receipt (method code PWE-100). The core was then dried, crushed to 70% passing 2 mm, split to a 250 g sub-sample, and pulverized to 85% passing 75 µm (Method code PRP-910).

11.3 ANALYTICAL LABORITIES

Table 11-2 summarises the analytical laboratories used by CGA and SROL.

Table 11-2. Analytical Laboratory Summary

	Operator	LaboratoryLocation	Time Period	Sample TypeAnalysed
CGA	SGS	Tarkwa, Ghana	2008 - 2011	Soil Samples
CGA	SGS	Tarkwa, Ghana	2008 - 2011	Drill Hole Samples
SROL	MS Analytical	Vancouver, BC, Canada	2017 - 2018	Surface Samples
SROL	MS Analytical	Vancouver, BC, Canada	2017 - 2018	Drill Hole Samples

SGS and MS Analytical are both ISO9001:2008 accredited laboratories. The QP has not audited the sample preparation or assaying laboratories in Abuja or Vancouver. Both laboratories are independent of CGA or SROL.

CGA drill core produced grades within a similar range to SROL samples. However, the 2019 QP recommended a selection of high-grade intercepts of CGA holes should be re-assayed for verification purposes.

11.4 ANALYTICAL METHODS

11.4.1.1 Fire Assay

CGA samples were analysed by SGS for gold by fire assay with an atomic absorption spectrometry (AAS) finish (SGS FAA505 method). This method used a 50 g charge and had a 0.01 ppm Au detection limit. Analysis for Ag, Cu, Pb, Zn, As, Mo, and Sb was also completed using an aqua regia digest on a separate 50 g charge with an AAS finish (SGS ARA155 method).

SROL samples were also analysed by fire assay with an AAS finish (MS Analytical FAS-221 Method). An aliquot of 50 g was weighed, mixed with flux (a blend of litharge, soda ash, borax, silica, silver, and various other essential reagents), and then fused to produce a lead button. The gold-containing lead button was cupelled to remove the lead and yield a bead which contains precious metals. The bead was then digested with nitric and hydrochloric acid. On completion of the digestion, the solution is bulked up to volume with dilute hydrochloric acid. The final solution was analysed by AAS.

The analytical methods used on drill core and check assays from the Project are summarised in Table 11-3.

Table 11-3. Analytical Methods

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

11.4.1.2 Metallic Screen Fire Assay

From 2017 to 2019, any assays greater than 10 g/t Au (a total of 48) were re-analysed by metallic screen fire assay (method code MSC-150).

The metallic screen fire assay technique is different from fire assay in that approximately 1 kg of material from the sample is analysed, compared to a 50 g sub-sample analysed by the fire assay technique. The sample size is particularly relevant where the gold is “nuggety” and the absence or inclusion of individual gold particles can significantly influence the fire assay result.

The results of the fire assay and the metallic screen fire assay are compared in Figure 11-4

- Based on 33 samples, metallic screen fire assay generally returns grades, on average, approximately 10% lower than the original fire assay for values less than approximately 22 g/t Au.
- Based on 12 samples, the metallic screen fire assay returns results, on average, approximately 10% higher than the original fire assay for values between approximately 22 g/t Au and 45 g/t Au.
- Based on three samples, the fire assay method generally returns grades, on average, approximately 19% higher than the metallic screen fire assay for values greater than 45 g/t Au.

Overall, there has been a positive impact on high grades (> 22 g/t) using Metallic Screen Fire Assays, although there are insufficient samples to define a Mineral Resource using them. Gold assays between 11 g/t and 22 g/t replicate well using either method.

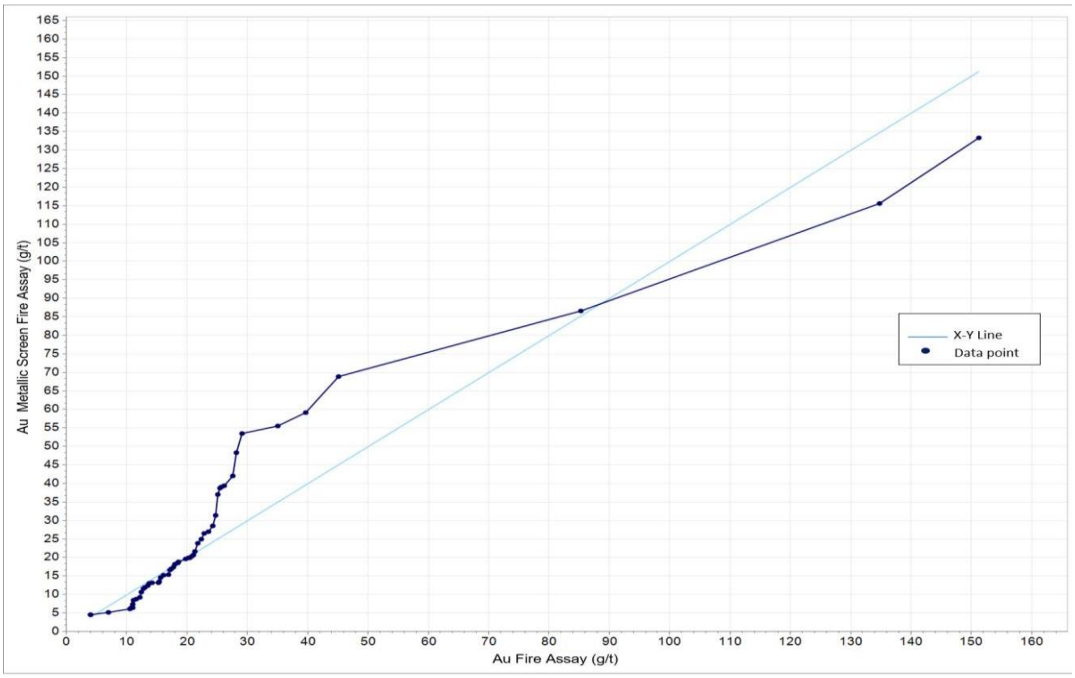


Figure 11-4. Fire Assay and Metallic Screen Fire Assay QQ Plot

Metallic screen fire assay data was not used in the resource estimate because the analyses were only carried on a small percentage of the total samples. Material at cut off is identified well by standard Fire assay. The metallic screen fire assay data indicates the presence of a coarse gold fraction as confirmed by subsequent metallurgical testing.

11.5 SAMPLE SECURITY

Prior to dispatch, the sample 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 were packed onto an independently owned and operated vehicle by senior company technicians under the supervision of senior staff geologists.

Senior CGA personnel transported the samples to DHL couriers in Lagos for delivery to SGS Laboratory, Tarkwa, Ghana for the CGA samples. SROL samples were initially sent to MS Analytical (MSA) Laboratories in Vancouver, BC, Canada via air freight. In November 2017 MS Analytical commissioned a laboratory in Abuja and samples are prepared at the MS Analytical preparation facility. Samples were collected and transported from the Ilesha compound by MS Analytical staff. Prepared sample pulps were flown to MSA Labs in Vancouver for assay.

11.6 QUALITY ASSURANCE AND QUALITY CONTROL (2017 TO 2018)

SROL has instigated a set of QAQC procedures to ensure the reliability of the assay data. This section details the QAQC from SROL's 2017 to 2018 drilling programmes, is documented in the PFS completed by RPA (2019).

The QAQC is divided into 'Field' samples submitted by SROL and the 'Laboratory' samples internally submitted by MS Analytical.

11.6.1 Field QAQC

Table 11-4 summarises the field QAQC data.

Table 11-4. QAQC Summary

Type	Number	Insertion Rate Desired	Insertion Rate Calculated
Field Standards	71	25	24.51
Field Blanks	18	25	96.67
Field Duplicates	68	25	25.56
Total Number of QAQC Samples Assayed	157		
Routine Samples Analysed	1,740		
Total Number of Samples Assayed	**1,897**		11.08

11.6.1.1 STANDARDS

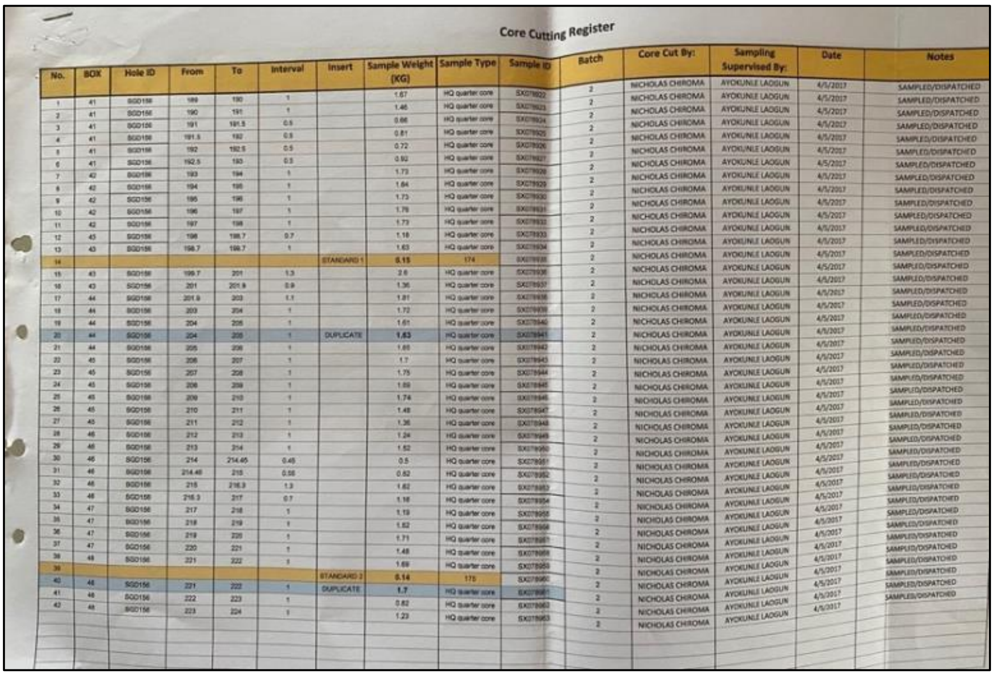
To validate the performance of the laboratory, standard samples (also referred to as Certified Reference Materials, or CRMs) were added to each batch of samples, typically after every 25th sample (Table 11-5). The Core Cutting Register for hole SGD 156 is shown in Figure 11-5 as an example.

SROL used standards supplied by both Geostats Pty Ltd and African Mineral Standards, South Africa (AMS). All the standards were supplied in jars except for AMS 0175, which was supplied in pre-measured 50 g packets. The standards supplied within jars were weighed into a small bag by on-site staff.

Table 11-5 and Figure 11-6 show the results of the standards analysis. In general, the variability is within acceptable limits and the results indicate an acceptable level of accuracy for the analytical laboratory and the assay method.

Table 11-5. List of Standards

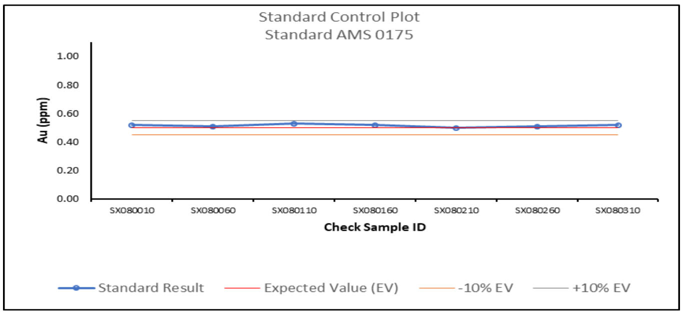
Gold Standard	No of assays	Expected Value (g/t)	Expected Value Range (g/t)	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	Standard Deviation	% Samples in Expected Value Range	% Bias
AMS 0175	7	0.50	0.45 to 0.55	0.50	0.53	0.52	0.01	100	-3.14
G910-8	9	0.63	0.567 to 0.693	0.55	0.65	0.60	0.03	89	4.41
G912-4	26	1.91	1.719 to 2.101	1.73	2.09	1.91	0.09	100	0.02
G913-2	8	2.40	2.16 to 2.64	2.09	2.40	2.33	0.09	88	3.13
OXG-124	12	0.918	0.826 to 1.01	0.85	0.99	0.90	0.04	100	1.87
OXK-119	9	3.604	3.244 to 3.964	3.24	3.93	3.53	0.18	89	2.09

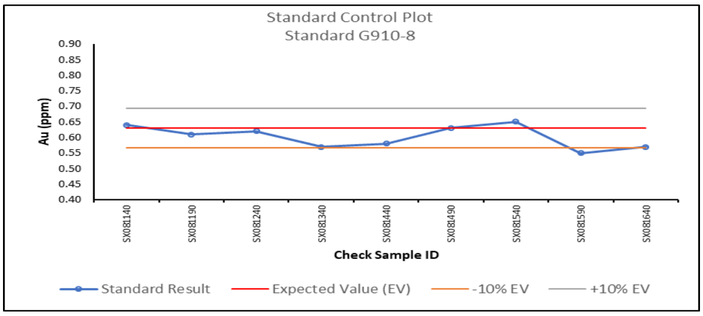


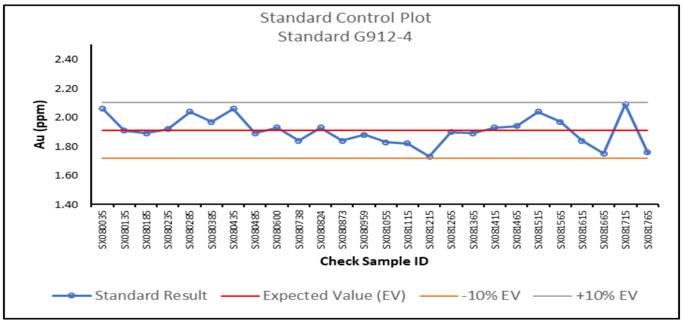
Core Cutting Register

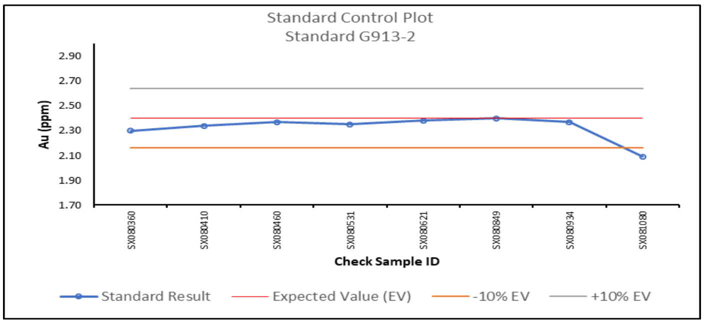
No.	BOX	Hole ID	From	To	Interval	Insert	Sample Weight (KG)	Sample Type	Sample ID	Batch	Core Cut By:	Sampling Supervised By:	Date	Notes
1	41	SGD156	189	190	1		1.67	HQ quarter core	SX078922	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
2	41	SGD156	190	191	1		1.46	HQ quarter core	SX078923	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
3	41	SGD156	191	191.5	0.5		0.66	HQ quarter core	SX078924	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
4	41	SGD156	191.5	192	0.5		0.81	HQ quarter core	SX078925	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
5	41	SGD156	192	192.5	0.5		0.72	HQ quarter core	SX078926	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
6	41	SGD156	192.5	193	0.5		0.92	HQ quarter core	SX078927	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
7	42	SGD156	193	194	1		1.73	HQ quarter core	SX078928	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
8	42	SGD156	194	195	1		1.64	HQ quarter core	SX078929	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
9	42	SGD156	195	196	1		1.73	HQ quarter core	SX078930	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
10	42	SGD156	196	197	1		1.76	HQ quarter core	SX078931	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
11	42	SGD156	197	198	1		1.73	HQ quarter core	SX078932	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
12	43	SGD156	198	198.7	0.7		1.18	HQ quarter core	SX078933	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
13	43	SGD156	198.7	199.7	1		1.63	HQ quarter core	SX078934	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
14						STANDARD 1	0.15	174	SX078935	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
15	43	SGD156	199.7	201	1.3		2.6	HQ quarter core	SX078936	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
16	43	SGD156	201	201.9	0.9		1.36	HQ quarter core	SX078937	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
17	44	SGD156	201.9	203	1.1		1.81	HQ quarter core	SX078938	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
18	44	SGD156	203	204	1		1.72	HQ quarter core	SX078939	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
19	44	SGD156	204	205	1		1.61	HQ quarter core	SX078940	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
20	44	SGD156	204	205	1	DUPLICATE	1.63	HQ quarter core	SX078941	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
21	44	SGD156	205	206	1		1.65	HQ quarter core	SX078942	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
22	45	SGD156	206	207	1		1.7	HQ quarter core	SX078943	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
23	45	SGD156	207	208	1		1.75	HQ quarter core	SX078944	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
24	45	SGD156	208	209	1		1.69	HQ quarter core	SX078945	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
25	45	SGD156	209	210	1		1.74	HQ quarter core	SX078946	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
26	45	SGD156	210	211	1		1.48	HQ quarter core	SX078947	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
27	45	SGD156	211	212	1		1.36	HQ quarter core	SX078948	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
28	46	SGD156	212	213	1		1.24	HQ quarter core	SX078949	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
29	46	SGD156	213	214	1		1.52	HQ quarter core	SX078950	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
30	46	SGD156	214	214.45	0.45		0.5	HQ quarter core	SX078951	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
31	46	SGD156	214.45	215	0.55		0.82	HQ quarter core	SX078952	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
32	46	SGD156	215	216.3	1.3		1.62	HQ quarter core	SX078953	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
33	46	SGD156	216.3	217	0.7		1.18	HQ quarter core	SX078954	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
34	47	SGD156	217	218	1		1.19	HQ quarter core	SX078955	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
35	47	SGD156	218	219	1		1.82	HQ quarter core	SX078956	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
36	47	SGD156	219	220	1		1.71	HQ quarter core	SX078957	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
37	47	SGD156	220	221	1		1.48	HQ quarter core	SX078958	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
38	48	SGD156	221	222	1		1.69	HQ quarter core	SX078959	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
39						STANDARD 2	0.14	175	SX078960	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
40	48	SGD156	221	222	1	DUPLICATE	1.7	HQ quarter core	SX078961	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	SAMPLED/DISPATCHED
41	48	SGD156	222	223	1		0.82	HQ quarter core	SX078962	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN	4/5/2017	
42	48	SGD156	223	224	1		1.23	HQ quarter core	SX078963	2	NICHOLAS CHIROMA	AYOKUNLE LAOGUN		

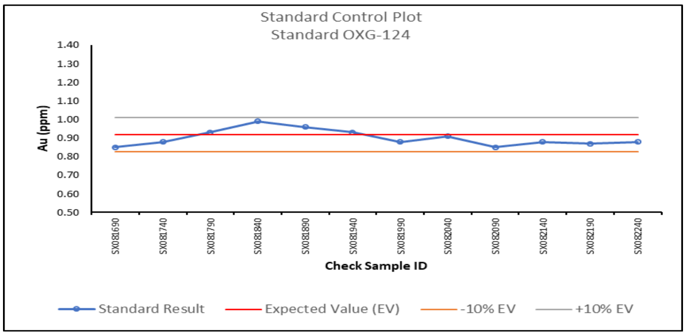
Figure 11-5. Core Cutting Register for hole SGD156 indicating blank- and duplicate samples as sent for analysis











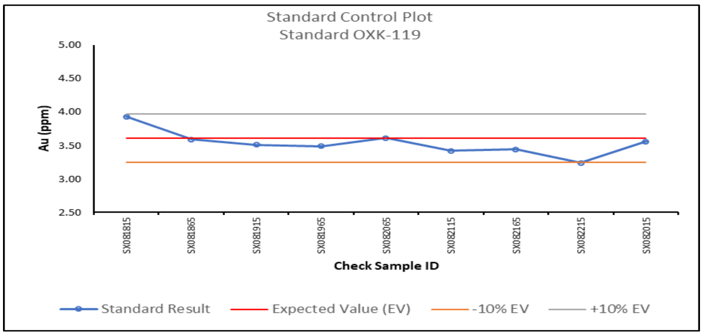


Figure 11-6. Field Standard Control Plots

11.6.1.2 BLANKS

To check for contamination blank samples (in this case gold-free samples) were inserted into batches of samples after every 100th sample.

Certified laboratory blanks supplied by AMS were used. These coarse blanks (Blank No. 0166) were made from homogenised silica quartz and had a gold content of less than 0.001 g/t Au.

Table 11-6 and Figure 11-7 show the results for the blank sample analysis. In the QP's opinion, the blank results are acceptable, with only one failure.

Table 11-6. Field Blanks

Standard	No of Assays	Minimum	Maximum	Mean	Standard Deviation	Pass Rate (%)
Blank 0166	18	0.01	0.07	0.01	0.01	94

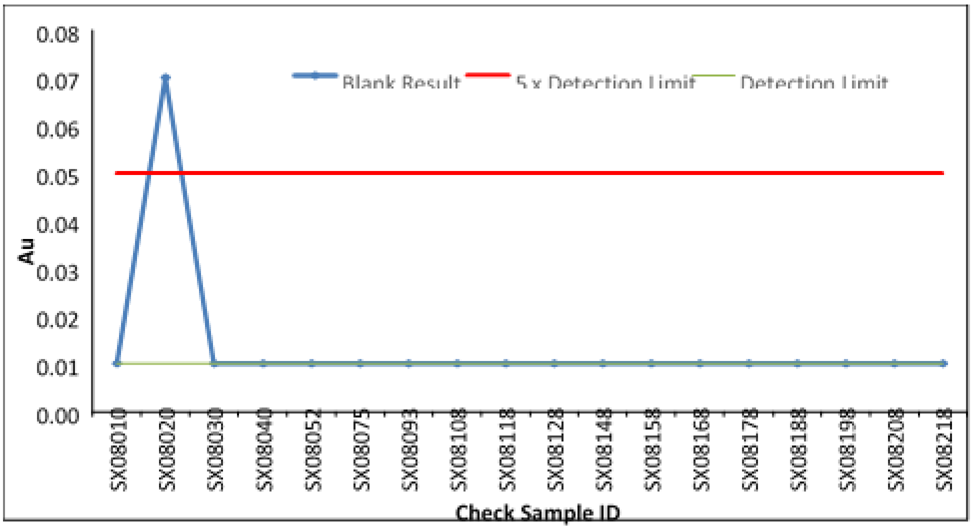


Figure 11-7. Field Blank Control Plot

11.6.1.3 DUPLICATES

Field duplicates are used to determine sampling error and to give an indication of the precision of the data pairs (original versus duplicate). The quality of the data will depend greatly on the quality of the actual duplicate prepared in the field. Representative diamond drill duplicates are difficult to prepare in the field, compared to other drilling methods, as the sample is not coarsely crushed and homogenised. The archived portion of the half core is often used as a field duplicate, however, two halves of a length of core may not be comparable and can produce poorly correlated results.

A total of 68 diamond drill field duplicates were analysed, which represents an insertion rate of approximately 1 in 25 samples. The duplicate samples were produced from both half and quarter core.

Figure 11-8 shows a scatter plot of duplicates versus original values and Figure 11-9 shows the duplicate pair mean versus the half real difference (HRD). The plots show some variability which is consistent with the nuggety nature of the mineralization. In general, the data indicates reasonable precision for the sampling method given the nuggety nature of the mineralization.

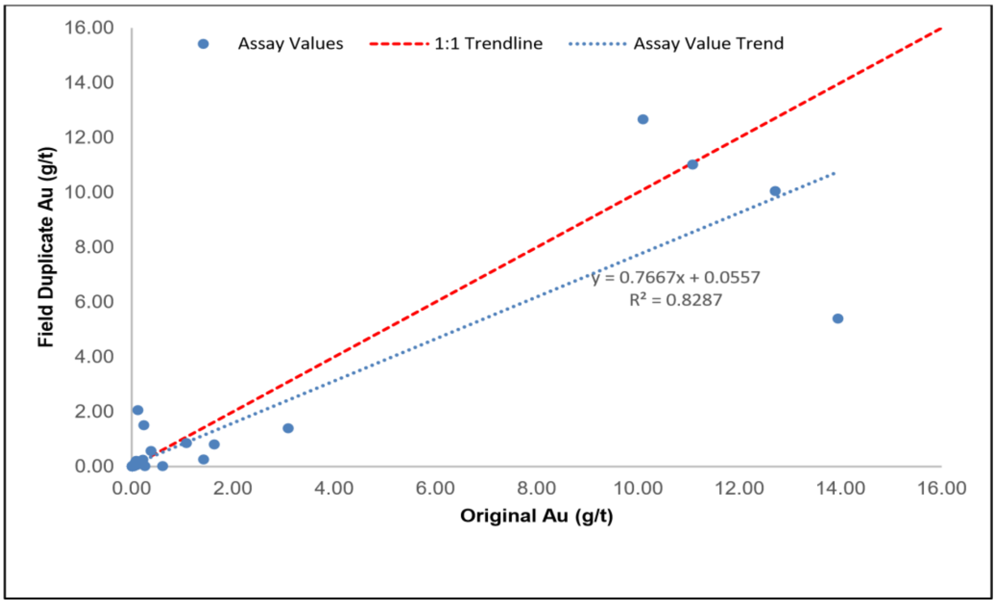


Figure 11-8. Field Duplicate Scatter Plot

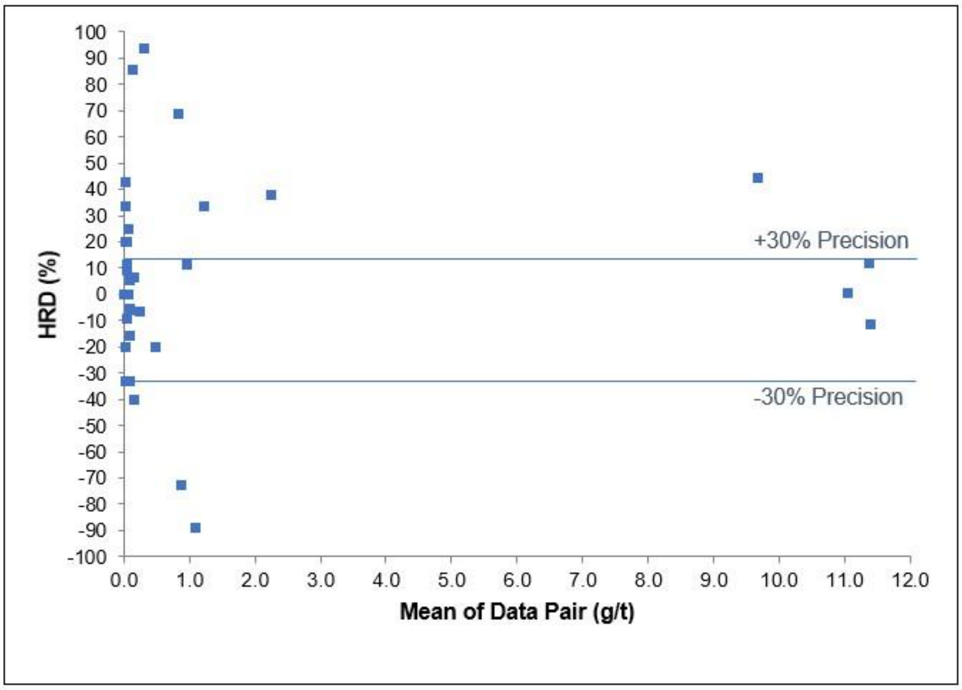


Figure 11-9. Duplicate Mean vs HRD Plot

11.6.2 Laboratory QAQC

11.6.2.1 STANDARDS

MS Analytical inserted a total of 93 standards during the 2017 and 2018 assaying programme. None of the standards failed the control limits, however, it was recommended that the laboratory investigate an observed low bias trend for standard CDN-GS-1U.

11.6.2.2 BLANKS

MS Analytical inserted 151 blanks to assess contamination within the process flow. No failures, i.e. values above the control limit of 0.03 ppm, were present for the 151 observations.

11.6.2.3 DUPLICATES

The laboratory inserted both pulp duplicates and coarse reject duplicates in the sample stream to test the accuracy and repeatability of the sample preparation and analysis.

The pulp (analytical) duplicates were generally inserted every 40th sample. A total of 48 were analysed and the duplicate results were plotted against the original results in Figure 11-10. The visual check and the calculated R^2 value of 0.992 indicate a good correlation between the original and duplicate samples.

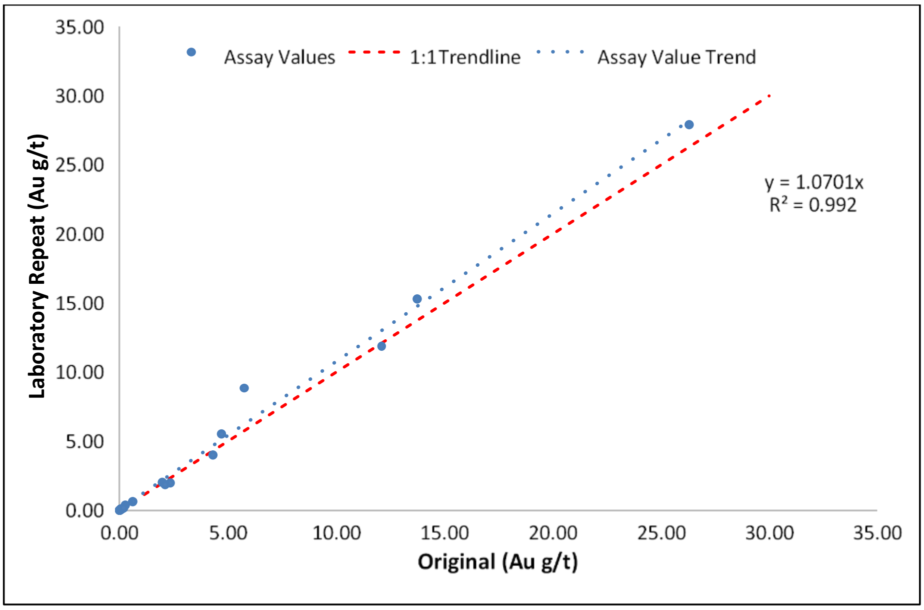


Figure 11-10. Pulp Duplicate vs Original Scatter Plot

Coarse reject (preparation) duplicates are created by splitting the sample after the crushing stage. The results of the preparation duplicates are plotted against the original results in Figure 11-11. The visual check and the calculated R^2 value of 0.9053 indicate an acceptable correlation between the original and duplicate samples.

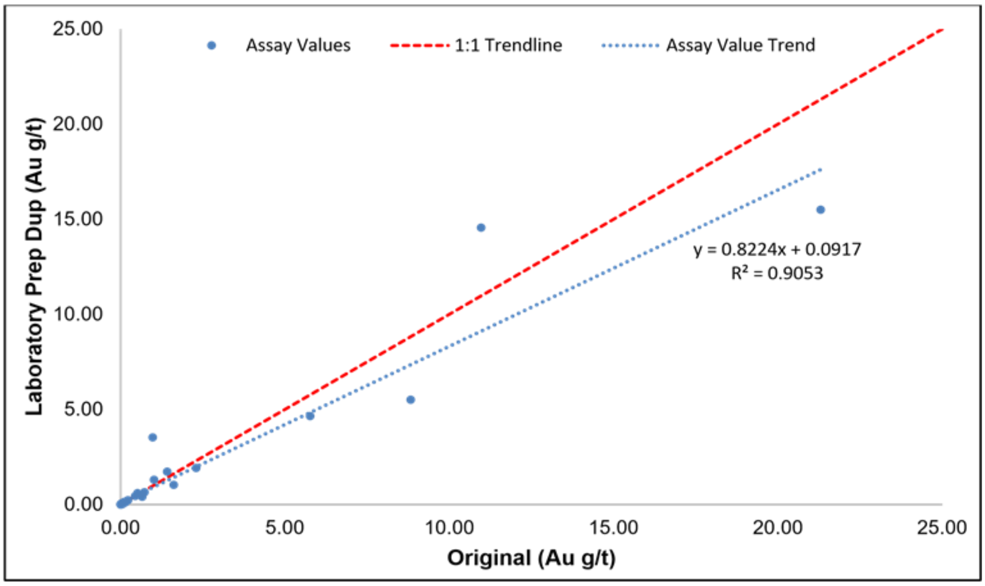


Figure 11-11. Coarse Reject Duplicate vs Original Scatter Plot

11.6.2.4 UMPIRE LABORATORY ANALYSIS

CGA performed inter-laboratory (umpire analysis) checks on SGS Ghana by sending 31 sample pulps for analysis at an independent laboratory: Genalysis Laboratories, (Johannesburg). The checks indicated no systematic bias in the SGS assays (Figure 11-12).

No inter-laboratory checks were completed by SROL. The QP recommends continued interlaboratory checks be performed for all future drilling programmes.

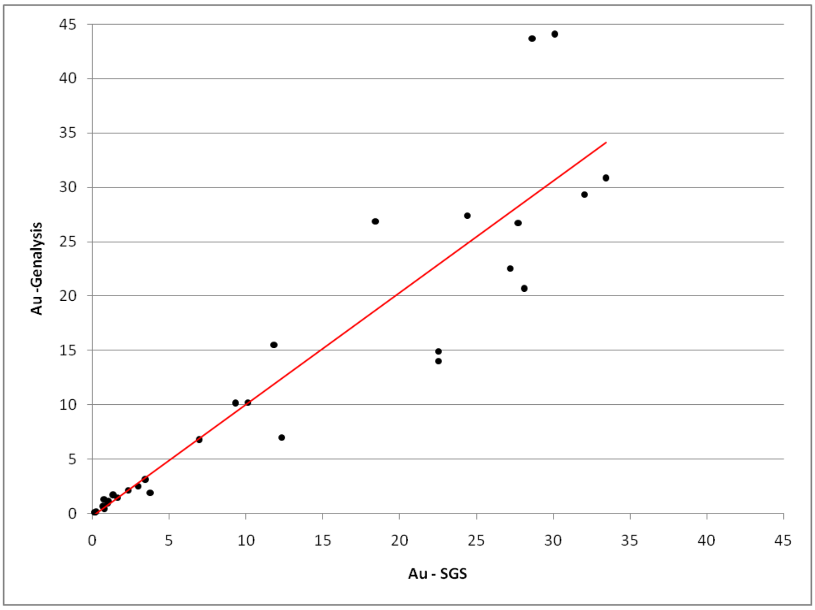


Figure 11-12. Inter-Laboratory Check Assays Scatter Plot

11.7 QUALITY ASSURANCE AND QUALITY CONTROL (2017 TO 2021)

Evidence for compliance to QAQC procedures on site was observed and are considered adequate.

QAQC sample data has been reviewed by Cube Consulting of Perth, Australia ("Cube") and laboratory data was reviewed by MS Analytical.

11.7.1 Review of QAQC Sample Data by Cube

In March 2021 Cube independently assessed all available QAQC sample data collected from May 2017 to January 2021 from the Segilola Gold Project.

11.7.1.1 Certified Reference Material

Cube reviewed the QAQC protocols and control assays for all diamond core sample data analysed at the Segilola Gold Project from May 2017 to July 2018 and from the more recently analysed diamond core samples from November 2019 and January 2021. The 2017-2018 evaluation comprised analytical data for 3,472 original diamond core samples with the inclusion of 124 CRMs, 33 Blank values and 59 duplicate core samples, which were submitted to MS Analytical in Canada between May 2017 and July 2018. The 2019-2021 evaluation comprised analytical data for 1,164 original diamond core samples, with the inclusion of 58 CRM and 11 Blank values and 113 field duplicates, submitted to MS Analytical in Canada between November 2019 and January 2021.

Table 11-7. Summary of Drill hole QAQC Sample Statistics

				Number of Samples and QC Types					Percentages		
Count of Holes	Total Metres Drilled	Hole Type	Year Drilled	Standard	Blank	Duplicate	Repeats	Routine Sample	STD	BLK	DUP
16	4.156.1	DD	2017	66	19	80		1779	3.7%	1.1%	4.5%
28	2852.6	DD	2018	58	14	51		1693	3.4%	0.8%	3.0
44	7008.7	Total		124	33	59		3472			
18	3457.2	DD	2020	60	11	55	58	1164	5.2	0.9%	4.7%
62.0	10465.9	Grand Total		308	77	245	58	8108			

Cube concluded that the MS Analytical (MSA) laboratory during 2017-2018 demonstrated analytical accuracy at an acceptable level within 95% confidence limits for 3 of 5 CRMs and the Blank. 2 CRMs failed the precision test.

Table 11-8. MS Analytical CRM and BLANK Summary Au – Segilola project 2017-2018

Laboratory	CRM	Number of Samp	EVal	StdDev	Accuracy	Precision	% Passing 3SD	% Bias	Period In Use	Comments
MSA	AMIS0174	20	2.13	0.10	PASS	PASS	100	-3.31	2107	
MSA	AMIS0175	28	0.50	0.05	PASS	PASS	100	0.50	2017	1 outlier removed
MSA	G910-8	10	0.63	0.04	PASS	PASS	100	-3.97	2018	
MSA	G912-4	32	1.91	0.09	PASS	PASS	100	0.23	2017-2018	
MSA	G913-2	10	2.4	0.08	PASS	PASS	90	-2.96	2017-2018	1 fail observed
MSA	OXG124	12	0.918	0.017	PASS	FAIL	75	-1.87	2018	3 fails observed
MSA	OXK119	10	3.604	0.105	PASS	FAIL	80	-1.78	2018	2 fails observed
Total		**122**								
MSA	Blank	30	0.01	0.03	PASS	PASS	100		2017-2018	2 outliers removed. Source of BLANK?
Grand Total		**152**								

The MS Analytical laboratory during 2019-2021 demonstrated analytical accuracy at an acceptable level within 95% confidence limits for 6 of 8 CRMs and the Blank. 2 CRMs failed the precision test while 2 of the CRMs contained insufficient samples to effectively review.

Table 11-9. MS Analytical CRM and BLANK Summary Au – Segilola project 2019-2021

Laboratory	CRM	Number of Samp	EVal	StdDev	Accuracy	Precision	% Passing 3SD	% Bias	Period In Use	Comments
MSA	G907-5	6	1.34	0.07	PASS	PASS	100	-0.37	2019	
MSA	OXF162	3	0.832	0.027	PASS	PASS	100	-5.45	2021	Not enough samples to effectively review
MSA	SH82	17	1.333	0.027	PASS	PASS	100	-1.15	2019-2021	2 outliers removed
MSA	SI81	7	1.790	0.030	PASS	FAIL	100	-1.92	2020-2021	Precision fail
MSA	SJ80	6	2.656	0.057	PASS	PASS	100	-0.10	2020	2 fails observed
MSA	SJ95	12	2.789	0.054	FAIL	PASS	67	-5.34	2020-2021	Accuracy failed; 4 failures observed; < -5% pass
MSA	SK109	3	4.102	0.098	PASS	PASS	100	-2.24	2020	Not enough samples to effectively review
MSA	SK94	4	3.899	0.084	PASS	PASS	100	0.15	2020	
Total		**58**								
MSA	Blank	11	0.01	0.03	PASS	PASS	100		2019-2021	Source of BLANK?

11.7.1.2 Duplicates

The results for duplicate samples (original and duplicate) were plotted as Q-Q plots and relative paired difference plots (RPD). The RPD plots evaluate the relative differences in percent between pairs and allow for determining the relative precision of samples through the calculation of the average coefficient of variation (ACV). The calculation of the ACV was set to consider only values >0.1 ppm Au, which is considered to be the threshold for mineralised material. These types of charts also allow for the visualisation of any bias or trends.

There were 194 field duplicate samples presented as NQ or HQ quarter or half core (not always for both the original and the duplicate sample). These variations were charted separately, resulting in small datasets which were reduced further when filtered for a mineralised threshold of Au >0.10 g/t (23 samples) with a varying ACV of between 31%-86%. Most samples were outside of acceptable limits for a field duplicate (ACV = 20-40%). This may be a result of the varying sample sizes. The recommended duplicate material for diamond core is to source the coarse reject material from the crushing of the original half core sample.

There were no coarse reject duplicates provided for review.

There were 58 re-assayed core samples provided for review which were presented as HQ quarter or half core (not always for both the original and the duplicate sample). These variations were charted separately, resulting in small datasets which were reduced further when filtered for the mineralised threshold of Au > 0.10 g/t (41 samples). ACV varied between 43.8%-45.9%, which is outside of the acceptable limits of 20%-40% for a core duplicate and may be a result of the varying sample sizes.

No umpire duplicates were provided for review.

Cube concluded that the duplicates in this review do not provide any meaningful conclusions with respect to the precision associated with the nature of the mineralization, sample collection, sample preparation, sample size and assay methodology.

Table 11-10. MS Analytical Laboratory Duplicate Summary – Segilola Gold Project 2017-2021

Description	Dia Size	QC Hole	Hole Type	Lab	Num Samp	PeriodInUse	Num Samp >0.10ppm	Average Pair Mean Difference	ACV	Asy 10%	Asy 20%	Asy 50%	Comments
HalfCore	HQ;NQ;Unk	Field	DDH	MSA	22	2017-2018	13	-20.6	60.6	7.7	15.4	38.5	Varying core size issue
QTRCore	HQ;NQ;Unk	Field	DDH	MSA	103	2017-2018	12	-4.1	64.9	8.3	25	50	Varying core size issue (5 HC orig included but all have Au<0.10ppm then excluded)
			Total		**125**	**2017-2018**	**25**						
HalfCore	HQ	Field	DDH	MSA	35	2019-2021	14	-10.7	57.9	28.6	35.7	42.9	
QTRVsHalfCore	HQ	Field	DDH	MSA	8	2019-2021	1	120	84.9	0	0	0	Not enough samples to effectively review
HalfVsQTRCore	HQ	Field	DDH	MSA	41	2019-2021	34	14.7	45.9	14.7	29.4	64.7	
QTRCore	HQ	Field	DDH	MSA	32	2019-2021	9	47.4	61.5	22.2	22.2	44.4	
			Total		**116**	**2019-2021**	**58**						
QTRVsHalfCore	HQ	Reassay_Field	DDH	MSA	8	2020	1	120	84.9	0	0	0	Not enough samples to effectively review
HalfVsQTRCore	HQ	Reassay_Field	DDH	MSA	41	2020	34	-9.60	45.9	14.7	29.4	64.7	
QTRVsQTRCore	HQ	Reassay_Field	DDH	MSA	9	2020	6	21.10	43.8	33.3	33.3	50	
			Total		**58**	2020	**41**						

11.7.1.3 Cube Conclusions and Recommendations

Issues reported by Cube included:

- The CRM failures at MS Analytical particularly in 2021, were highlighted as an immediate issue requiring attention.
- The overall CRM insertion rate is low at 6.5%
- The core field duplicates have been carried out on different sample sizes.

- The field duplicate insertion rate is low at 3% for core and 4% for RC samples.
- No coarse reject or umpire duplicate sampling has been undertaken.
- A total of 20 mislabelled CRMs were identified in the dataset.

Cube recommended that:

- Future core sample batches should be carried out on half core intervals to maintain consistency in an effort to eliminate any possible bias due to sample size.
- Failed CRMs be raised with the laboratory on a batch-by-batch basis and that pulp re-assays occur as soon as the batch has been uploaded to the database and a first pass QC check has been completed.
- Ideally the practise of field duplicate re- assay for core sample should be carried out on the same sample size as the original. This would deplete the site record for the chosen interval, thus Cube recommended that the optimal source for a diamond core duplicate sample is to utilise the coarse reject material from the original half core crush to maintain consistency on sample size and material.
- It is recommended that coarse reject samples be retained for all batches. Coarse rejects can be used as duplicate sample material to assess the precision and accuracy of the laboratory assays.
- A retrospective pulp duplicate re-assay undertaking is recommended to gain confidence in the original analysis which contains limited duplicate information.
- It is recommended that an umpire laboratory be used to test duplicates on a regular basis to determine if there is any assay bias at the primary laboratory.
- Coarse blanks being inserted into the sample stream to assess the sample preparation laboratory should be increased from the current rate of 1% to 5%.
- Pulp grind checks should be introduced for 1 in 20 samples per lab job to monitor sample preparation and compliance with the assay contract.

11.7.2 Review of Laboratory QAQC data by MS Analytical

In March 2021 MS Analytical personnel evaluated data generated for laboratory quality monitoring (325 laboratory blanks, 433 Certified Reference Materials (CRM) and 423 laboratory duplicate pairs) during the assaying of 13,819 drill core and RC samples originating from the Segilola Gold Project which were assayed in 158 analytical jobs between May 2017 and December 2020.

The samples had been subjected to sample preparation and the determination of gold (Au) by lead collection fire assay followed by Atomic Absorption Spectroscopy (AAS).

The aim of the data evaluation was to establish the overall method performance in relation to the laboratory acceptance criteria and to report on the accuracy and general precision of the analytical method.

The overall insertion rate for laboratory quality monitors was 5%. Every 42 samples included at least:

- 1 analytical blank.
- 1 analytical duplicate.
- 2 certified reference materials (CRM) that are randomly distributed.

No failures in analytical blanks were present for the plotted 325 observations.

Overall, there were 19 duplicate failures out of the 423 duplicate pairs, a pass rate of 96%. No consistent relative bias was observed between the original and duplicate data.

Accuracy of the 28 different CRMs used was quantitatively assessed and expressed as bias. The % bias was deemed acceptable as a maximum of ±5% should be achieved (-5% to 1.7% was observed). No CRM observations failed the control limits.

Evaluation of CRM precision concluded that the stated method precision is generally achieved (except for CDN-GS-P4G where the nature of the material cannot be discounted as having an impact on the performance of the CRM). MSA concluded that no CRM observations fail the prescribed control limits.

Overall MS Analytical concluded that the parameters used to assess the laboratory quality monitors adhere to specified control limits thereby increasing the confidence in the associated reported final assay results.

11.8 SAMPLE PREPARTION, ANALYSES AND SECURITY COMMENTS

MA considers an overall CRM insertion rate for an established sampling protocol of 5% to be sufficient for continual monitoring of quality. MA agrees that a retrospective pulp duplicate re-assay programme (e.g. 30 samples every 6 months) be implemented to check the original assay results. These samples could also be sent to an independent laboratory to determine if there is any assay bias at the primary laboratory. MA agrees that coarse blanks should be increased from the current 1%; one coarse blank should be inserted after logged high grade mineralisation.

Cube noted CRM's (OXG124 and OXK119) failed the precision tests, MSA Labs found CDN-GC-P4G also failed but stated the nature of the material cannot be discounted as having an impact on the performance of the CRM, this may also be the case for OXG124 and OXK119. The results of the QAQC program, should be reviewed by SROL on an ongoing basis, documenting actions taken because of failure CRM's in a timely manner (vis before uploading to database).

MA is of the opinion that the sample collection, preparation, analysis, and security used by SROL were generally performed in accordance with common industry procedures and practices and are suitable for use in Mineral Resource estimation.

The QAQC procedures and management are consistent with common industry practice and the assay results within the database are suitable for use in Mineral Resource estimation. The QP has not identified any issues which could materially affect the accuracy, reliability, or representativeness of the results.

The QP is of the opinion that the geological and analytical database quality is of sufficient quality to support Mineral Resource estimation.

11.9 BULK DENSITY DATA

The deposit has a shallow weathering profile. The depth of the fresh rock over the mineralized zone is up to 25 m below the surface, oxidation reduces to 1 m to 2 m either side of the lodes. There are 1776 density reading for the project. The data set consists of 1157 de-surveyed records (SG_Density Master.xlsx), only recording the sample coordinates, density readings and hole identification. A scan of the original data was provided "Segiolola DD Specific Gravity.pdf" detailing hole id from-to's and weights.

A further 619 readings are stored in the drill hole database with collar identification and sample intervals and wet and dry weights, and the density reading.

Densities are based upon specific gravity measurements completed by CGA and SROL. Both companies used the Archimedes Principal (immersion techniques) to determine the density of the core. Table

11-11 shows the previous tenement holder has performed the most density readings. SROL has sent 149 bulk density samples off site to MS Analytical (MSA) Labs. The mean density returned is significantly lighter than both the average density seen in the CGA and SROL data.

Table 11-11. Average Bulk Density by Analysing Site

Laboratory	n	mean	CV
CGA (SRL) site lab	1394	2.67	0.05
MSA Lab measurement	149	2.61	0.07
SROL site lab	233	2.77	0.09

Densities were plotted (Figure 11-13) to determine if a depth relationship could be established, no relationship with depth was found. Figure 11-13 does show the samples sent to MSA are higher in the profile. SROL density samples have the highest scatter (Figure 11-13 and Table 11-11.) with a CV 0.09.

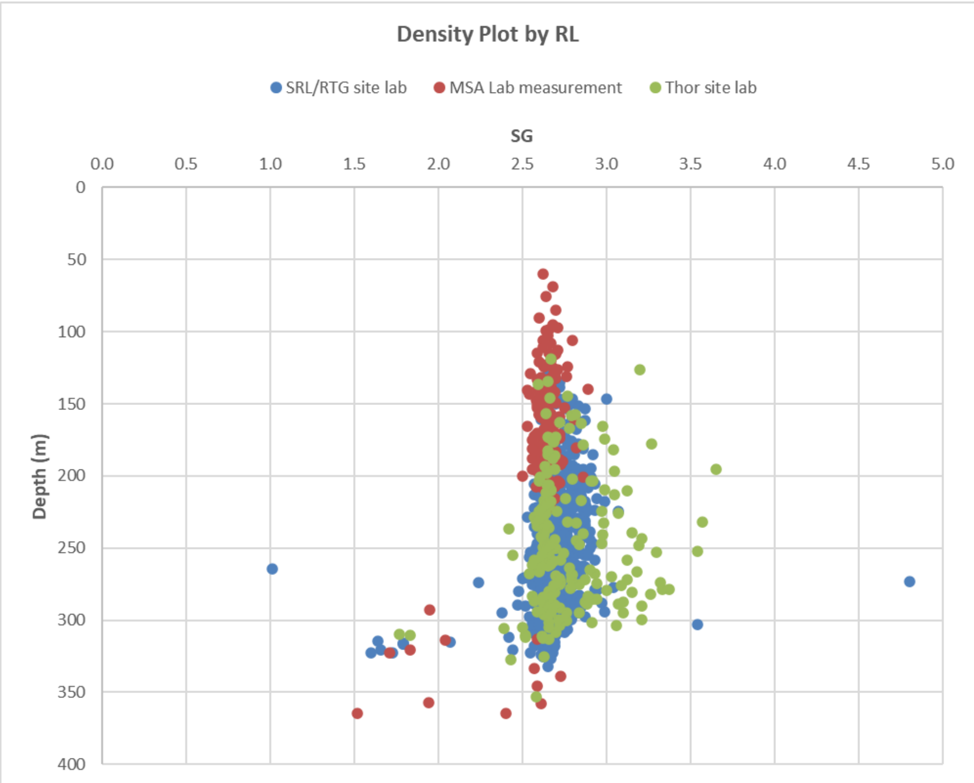


Figure 11-13. Density Measurements by RL and Laboratory

The maximum density reading is 4.8, from a 14 cm interval in hole SGD137 from 34.7 m down hole logged as Fresh Biotite Granite. A density of 4.8 is equivalent to 90% of the sample being pyrite, assuming the remaining 10% has the average rock density at Segilola of 2.68. The lightest sample 1.01, is from 15 cm interval in hole SGD130 from 48.1 m down hole, also logged as Fresh Biotite Granite.

The histogram plot (Figure 11-14) shows most samples fall within the 2.65 bin with a significant number falling in the 2.70 bin.

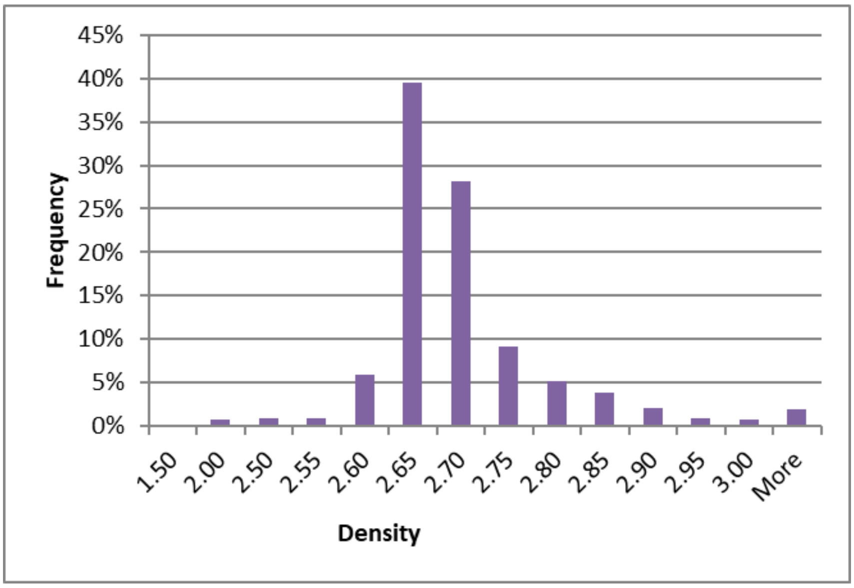


Figure 11-14. Histogram of Density Readings

SROL provided wireframes for top of fresh, base of oxidation. The data was classified into weathered states based on interpreted weathering profiles. The two data sets were statistically summarised (Table 11-12).

Table 11-12. Summary Density Data

	SROL		CGA		Total	
All Material	**No. Samples**	**Average SG**	**No. Samples**	**Average SG**	**No. Samples**	**Average SG**
Oxide	19	2.63	61	2.46	80	2.50
Transition	24	2.67	68	2.62	92	2.63
Fresh	591	2.72	1013	2.67	1604	2.69
	634		**1,142**		**1,776**	

The recent density readings within fresh rock undertaken by SROL are 1.8% heavier than the earlier CGA density readings. Both datasets are used to inform the model.

The density data was further classified into rock types and weathering states (Table 11-13), although some categories are not statistically valid, (n is too small) the average densities (of both data sets) were assigned to the various rock types. The following lithological units are summarised as BS, Biotite Shear; CS, Calc Silicate; DGS, Dark Grey Schistose; DOL, Dolerite; GDM, Granodiorite Massive; GDS, Granodiorite schistose fabric and SZQ is the shear/quartz zone.

Table 11-13. Summary Density Data by Weathering and Rock Type

Lithological Unit	All Material	SROL		CGA		Total	
		n.	BD	n.	BD	n.	BD
SZQ	Oxide	16	2.64	30	2.63	46	2.64
	Transition	20	2.68	40	2.63	60	2.65
	Fresh	446	2.72	630	2.65	1076	2.68
BS	Oxide	-	-	4	1.87	4	1.87
	Transition	1	2.58	2	2.32	3	2.40
	Fresh	23	2.75	129	2.72	152	2.72
CS	Fresh	28	2.79	107	2.72	135	2.74
DGS	Oxide	1	2.63	5	2.66	6	2.66
	Transition	-	-	5	2.69	5	2.69
	Fresh	57	2.69	66	2.66	123	2.68
GDM	Oxide	-	-	19	2.23	19	2.23
	Transition	2	2.57	13	2.59	15	2.59
	Fresh	5	2.69	58	2.66	63	2.66
GDS	Oxide	2	2.54	3	2.65	5	2.61
	Transition	1	2.72	8	2.62	9	2.63
	Fresh	32	2.66	19	2.67	51	2.66
DOL	Fresh	0	0.00	4	2.74	4	2.74
Total		**634**		**1,142**		**1,776**	

* n: number of samples, BD: bulk density

CGA SG data has not translated to the local grid accurately. There is a mismatch between the translation of SG data compared to the translation of the drill holes. Density samples are within 5 m of the projected drill location. Density readings are deemed suitable for determining average density of rock types, but insufficient for direct use in estimating density.

The QP has reviewed SROL's density measuring procedure and considers it appropriate. The results from MS Analytical (an external commercial laboratory) show lighter density readings than either CGA or SROL and may be attributable to the samples generally coming from higher in the profile, though a depth relationship could not be established. The MSA samples have the narrowest spread.

MA recommends original sample intervals for the de-surveyed density data (Segilola DD Specific Gravity.pdf) be added to the drill hole database. MA recommends a density validation programme be implemented, either by cross checking with alternate density methods or by using MSA to provide a check determination.

Assigning average densities to rock types is considered a suitable method for assigning bulk density to the Segilola deposit.

12 DATA VERIFICATION

Drilling data is stored off site. An independent database manager (Cube Consulting) manages uploads and validation. Cube Consulting provides extracts for site and third parties as required. The data base was supplied in the form of a Microsoft Access database export. The data verification was undertaken independently by MA using in-built validation tools in Surpac and by interrogating the database in Microsoft Access.

12.1 INVALIDATED DRILLING

The following holes were not used in the resource estimate (Table 12-1).

Table 12-1. Excluded Holes

Hole id Range	Company	Reason
BH1 to BH33	NMC	Cannot verify information
NIG series (4 holes) and TIG series (3 holes)	Hansa	Cannot verify information
SGD001, SGD002, SGD128	CGA/SGL	Not assayed
SGD175 and SGD176	SROL	Bulked sampled for Metallurgical testing (aka MET03B and MET04B)
SGRC042 to SGRC053	SROL	No down hole survey information at time of resource estimate –the holes are sterilisation drilling for mill site

12.2 ASSAY TABLE

All original assay certificates for the data used in the resource estimate are available. A check was made between the gold values in the Microsoft Access export with the assay and values on the assay certificates. RPA during the March 2019 resource update checked 89% of the assays in the database. A spatially representative selection of drill holes was chosen from all resource zones, covering different years of drilling and assaying. A summary of data requested, and cross checked with the database is given in Table 12-2. A total of 150 drill hole geology logs were provided up to SGD199. Recent logs (post hole SGC200) are in the drillhole database but have not yet been scanned.

Table 12-2. Verification Data Sighted

Requested Holes	Geology	Assay certificates
SGD049	NA	Y
SGD118	NA	Y
SGD122	Y	Y
SGD127	Y	Y
SGD132	Y	Y
SGD148	Y	Y
SGD156	Y	Y
SGD165	NA	Y
SGD184	Y	Y
SGD187	Y	Y
SGD195	NA	Y
SGD206	NA	Y
SGD211	NA	Y

MA found several minor errors within the suppled database (Table 12-3), generally in low tenor material where samples were overlapping. Hole SGD144 had down hole survey errors and were quickly verified and corrected upon request.

Table 12-3. Assay Table Errors

hole_id	Depth from	Depth to	samp_id	Sample Type	Sample Method	Correction
SGRC043	14	42	SX088187	ORIG	CHIPS	Overlapping samples -> Depth from changed 41
SGRC044	24	23	SX088249	ORIG	CHIPS	To depth greater than from depth -> Depth to changed 25
SGRC026	20.8	20.8	SX086664	ORIG	CHIPS	0 m interval, change Depth to 21.8m inline with next sample
SGRC026	38.8	38.8	SX086684	ORIG	CHIPS	0 m interval, change Depth to 39.8m inline with next sample
SGRC026	2.8	2.8	SX086644	ORIG	CHIPS	Depth to 3.8

SGRC019	49	49	SX086224	ORIG	CHIPS	0 m interval, change sample Type to DUP, adjacent samples were consecutive
NIG41	18	44	NIG41_44	ORIG	NR	Overlapping sample interval, Depth to 43
BH28	66.3	71.07	BH28_71.07	ORIG	NR	Sample Type STD, out of step with common RC sampling practices
SGD207	274	274.3	SX087578	DUP	QC_HQ	Sample Type DUP, from to's same as sample SX087577
SGRC037	7.5	8.8	SX087527	ORIG	CHIPS	Overlapping sample Depth to 7.8

The database is managed by Cube Consulting based in Perth. The database is clean, the errors identified, and solutions are listed in Table 12-3.

12.3 INDEPENDENT SAMPLES

No independent samples were collected and submitted by the QP as part of this review.

In MA's opinion, geological data collection and sampling is in line with industry best practice as defined in the Canadian Institute of Mining and Metallurgy and Petroleum (CIM) Exploration Best Practice Guidelines and the CIM Mineral Resource, Mineral Reserve Best Practice Guidelines.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Two metallurgical test work programmes have been undertaken on the Project. The first programme was undertaken for the PFS by AMMTEC Ltd (AMMTEC) in April 2010 and a second, more extensive, programme was completed for the DFS by Independent Metallurgical Operations Pty Ltd (IMO) in July 2018.

A test work programme was completed on a master composite sample and ten variability samples. The test work included comminution testing, gravity recoverable gold (GRG) tests, cyanidation test work, static settling tests, and tailings characterisation.

It was concluded by RPA (2019) that:

- The test work programme is appropriate for development of a process flowsheet.
- The flowsheet selected is appropriate to the test work findings.
- The design parameters agree with the test work findings.
- The Bond work indices and abrasion indices are slightly higher than the average West African hard rock mine but are not excessive or problematic.
- The assayed head grades for the test work were all within economic ranges.
- The proposed flowsheet consists of a conventional hard rock SAG-Ball milling circuit. This circuit is relatively standard for a hard rock gold mine. The grind size nominated of 106 microns is believed to be appropriate for optimal gold recovery versus energy input.
- The proposed flowsheet utilises conventional technology and industry practice.

14 MINERAL RESOURCE ESTIMATE

14.1 INTRODUCTION

The mineral resource estimate for the Segilola Gold Project, Osun State, Nigeria, has been prepared with an effective data of 4 January 2021 by Mining Associates. MA's employee, Mr I. Taylor, MAusIMM (CP) prepared the Mineral Resource Estimate. Mr Taylor takes Qualified Person responsibility for the Mineral Resource Estimate.

The estimation process followed the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" (CIM, 2019). The Mineral Resource Estimate is stated in accordance with CIM Definition Standards (CIM, 2014) and Canadian National Instrument 43-101 (NI 43-101).

After the Mineral Resource Estimate reported in March 2019 an additional 90 holes for 8,463 m of infill and depth extension drilling has provided a better understanding of the mineralization by providing greater detail which has been incorporated into the models. As a result, intervening waste material was removed from the southern lode, Lode 200 was divided into two parallel lodes with a consistent and often thin band of waste separating them, and lodes extended at depth.

Surpac Mining Software was used to estimate gold grades into a 3D block model using ordinary kriging (OK). This estimation approach is considered appropriate based on several factors, including the quantity and spacing of available data, the interpreted controls on mineralization, and the style and geometry of mineralization. A geology and mineralization model was created using implicit modelling in Leapfrog Geo software. This model was used to constrain the resource estimate.

The Segilola Gold deposit Mineral Resource (Table 14-1) comprises an Indicated resource of 4.06 Mt @ 4.66 g/t Au for 608,000 ounces of gold, and an Inferred resource of 0.443 Mt @ 4.8 g/t Au for 68,000 ounces of gold. The resource has been reported as Open Pit (0.3 g/t Au cut-off grade within the designed pit [pit_lom_v16.dt]) and potential Underground (2.5 g/t Au cut-off, resources below the pit with sufficient spatial continuity within a potentially mineable shape) categories.

Table 14-1. Mineral Resource Summary

Category	Open pit (> 0.30 g/t)			Potential underground (> 2.5 g/t)			Total		
	Tonnes (kt)	Grade (g/t Au)	Gold (koz)	Tonnes (kt)	Grade (g/t Au)	Gold (koz)	Tonnes (kt)	Grade (g/t Au)	Gold (koz)
Indicated	3,674	4.51	532	386	6.1	76	4,060	4.66	608
Inferred	32	2.5	3	411	5.0	65	443	4.78	68

To the best of MA's knowledge, there are no environmental, permitting, legal, title, tax, socio-economic, market, political or other relevant factors that would affect the Mineral Resource Estimate presented in this Technical Report.

SROL made available all technical and scientific data and interpretations relevant to the MRE, which have been reviewed and validated by MA. The QP is satisfied that the technical, scientific data and interpretations are sufficiently reliable to estimate and classify the Mineral Resource.

The process followed and decisions taken in the estimation of the Mineral Resource for the Segilola Gold Deposit are summarised in Section 14 including sub-sections below.

14.2 3D LITHOLOGICAL AND MINERALIZATION MODELLING

The Segilola Deposit geology interpretation has been a focus of detailed study in the past year since the previous Mineral Resource Estimate, which had an effective date of 18th March 2019. For detailed descriptions of the geology and the interpretation process, the reader is referred to Item 7 of this report.

The mineralized lodes generally comprise highly silicified fine-grained biotite gneiss typically intruded by both discordant and concordant pegmatitic quartz-feldspar veins. Gold mineralization is controlled by shearing, fracturing and alteration. This relationship has generated multiple zones of gold mineralization hosted within a chlorite-calcite altered shear zone with quartz veining and weak quartz-pyrite veining.

Thor's Group Exploration Manager, Mr A. Gillman, created a geological and mineralization model in Leapfrog Geo software with the resulting wireframes provided to MA for use in the MRE. The geological model was used to define the geological sequence within the block model. The mineralization wireframes are faithful to mineralization with appropriate breaks where drilling dictates. The mineralization wireframes are based on 0.5 g/t Au shells with minimal internal dilution allowed. The minimum modelled lode widths appear to be as narrow as one metre. Six individual mineralization wireframes were supplied, which MA reviewed in conjunction with the lithology model. The QP is of the opinion that the wireframes are acceptable for use in resource estimation.

The mineralization wireframes represent multiple lode structures that trend 010°, dip steeply (80° and 70°) towards the west and extend over a continuous strike length of 2 km (Table 14-2, Figure 14-1 and Figure 14-2).

Lode 100 is 730 m in strike length, has an average thickness of 1 m, and on average extends 100 m down dip (maximum of 280 m). Lode 200 (main footwall lode) is 1,460 m in strike length, varies in true width from 1.65 m to 18 m, and has a down-dip extent of 50 m in the north increasing to 420 m at the southern end. Lode 300 covers 1,300 m in strike, has an average thickness of 1.8 m and extends to a maximum of 300 m down dip at the southern end. Lodes 400 and 500 are in the hanging wall of Lode 300, with Lode 400 having a 1,000 m strike and Lode 500 striking 300 m. Lode 600 is a small lode (100 m strike) in the footwall of the southern end of Lode 100 and sits in the hanging wall of Lode 300.

Table 14-2. Lode Physical Characteristics Summary

Lode	Strike (m)	Width (m)	Down-dip extent (m)
100	730	1-3	100 - 280
200	1,460	1.65 - 18	50 – 240
300	1,300	~2, up to 8	300
400	1,000	1-3	100
500	300	1-3	discontinuous
600	100	2-3	minor and discontinuous

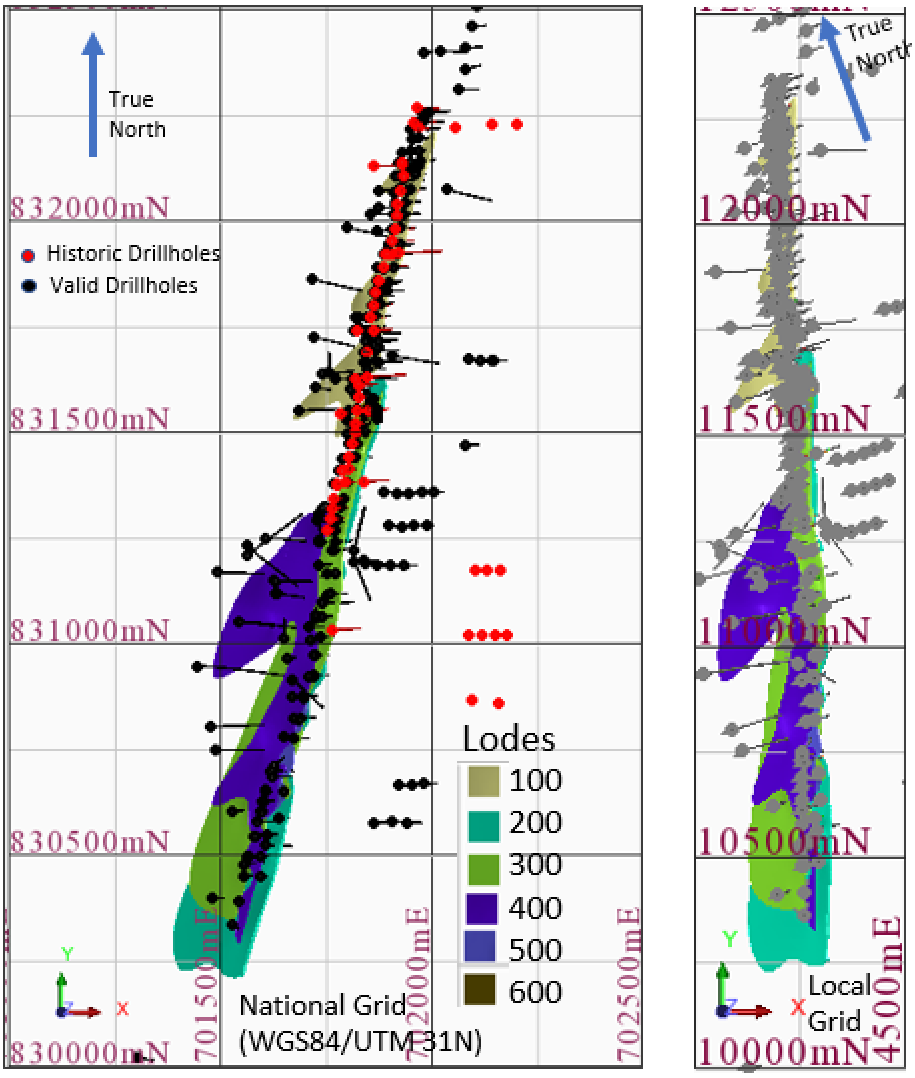


Figure 14-1. Plan View of Deposit and Drilling

(UTM and local Grid)

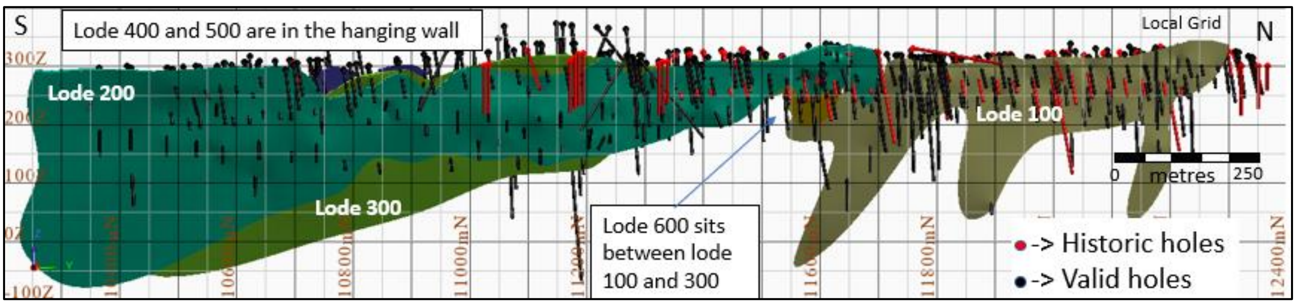


Figure 14-2. Longitudinal Section View

14.3 EXPLORATORY DATA ANALYSIS

MA has undertaken Exploratory Data Analysis (EDA) through statistical analysis of the entire database and on an individual lode basis, including comparison between lodes. Details are provided in the following sections.

Statistical analysis of both Au and Ag (as potentially economic minerals) over the entire data set to determine an appropriate lower boundary cut-off was undertaken, but too few Ag assays exist in the database and the correlation coefficient between Au and Ag is too low to draw any inferences. Summary statistics for all samples are shown in Table 14-3.

Table 14-3. Summary Statistics for all Au and Ag Assay Data

Variable	Au	Ag
Number of samples	15,615	857
Minimum value	0.001	0.11
Maximum value	151.21	85.10
Mean	0.60	1.85
Median	0.02	0.80
Geometric Mean	0.03	0.88
Variance	15.38	20.48
Standard Deviation	3.92	4.53
Coefficient of variation	6.52	2.45
10th Percentile	0.01	0.27
25th Percentile	0.01	0.40
50th Percentile (median)	0.02	0.80
75th Percentile	0.08	1.66
95th Percentile	1.90	7.15
97.5th Percentile	5.29	9.75
Correlation Coefficient	0.0128	

Histograms and log probablity plots were assessed to confirm a natural break existed at 0.5 g/t, confirming SROLs decision to use a 0.5 g/t Au cut-off for the interpretion of lode structures is valid.

14.4 VISIBLE GOLD

Logged intervals of visible gold (VG) are recorded in the database. Downhole locations of logged VG were cross referenced with assay data. In total there are 61 assay intervals where visible gold is present, of which 11 samples had more than one instance of VG recorded. Visible gold tends to cluster and mainly appears within Lodes 100, 200, 300 and 400 (Figure 14-3).

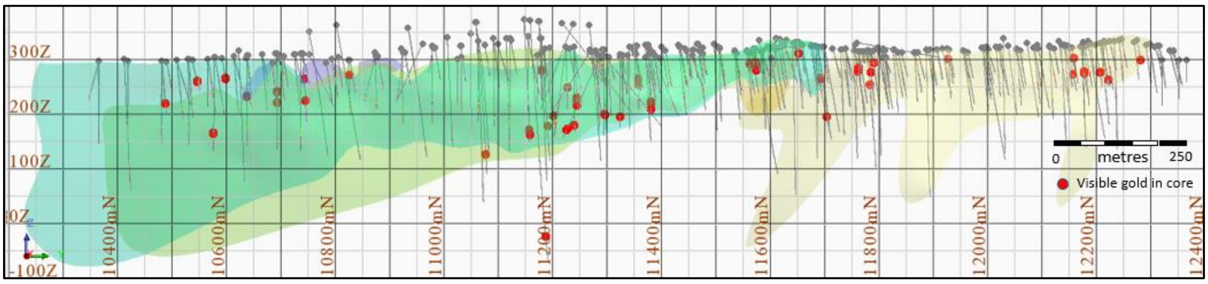


Figure 14-3. Logged Visible Gold in Core

One instance of visible gold occurred in an unsampled portion of drill hole SGD079. The average grade returned for intervals with VG was 24.93 g/t but the largest proportion (~30%) of samples containing VG returned assays of less than 5 g/t. Twenty percent of the samples returned assays between 5 and 10 g/t, 10% between 30 and 35 g/t, and at the top end 15% returned assays greater than 50 g/t (Figure 14-4).

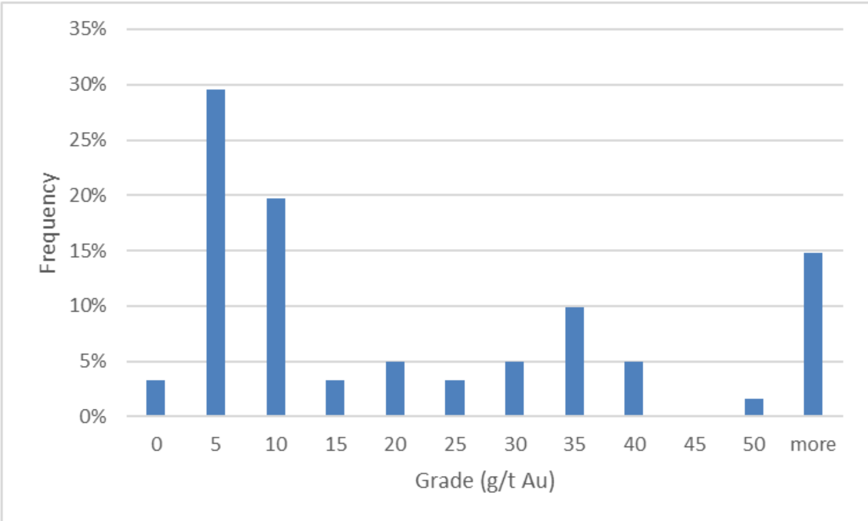


Figure 14-4. Histogram of Assay Grades with Logged Visible Gold

14.5 COPPER AND SULPHUR

The average grades of 1,421 copper and sulphur samples are 71.68 ppm and 0.40% respectively. These levels of copper and sulphur are not considered to be deleterious to the metallurgical recovery process. The highest values (785ppm Cu and 2.81% S) relate to a single hole that intersect Lode 100 below the base of the proposed open pit. The distribution of Cu and S in Lode 100 is shown in

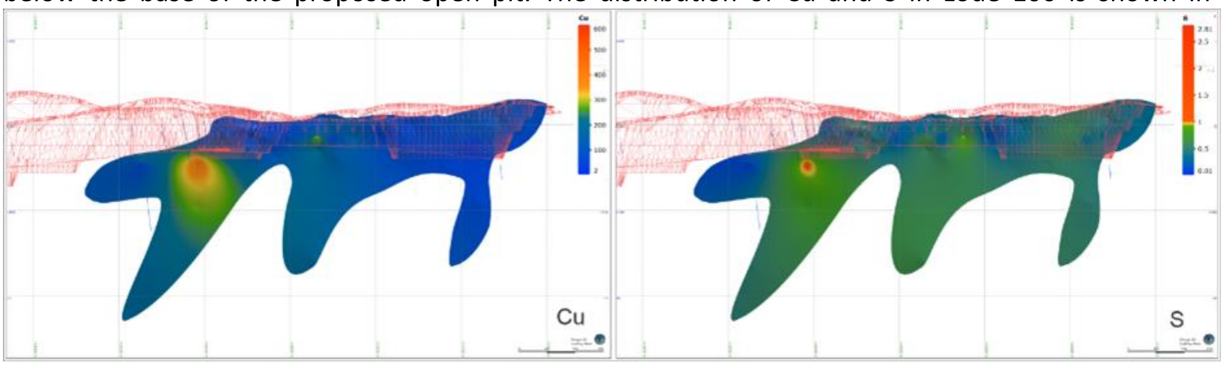


Figure 14-5.

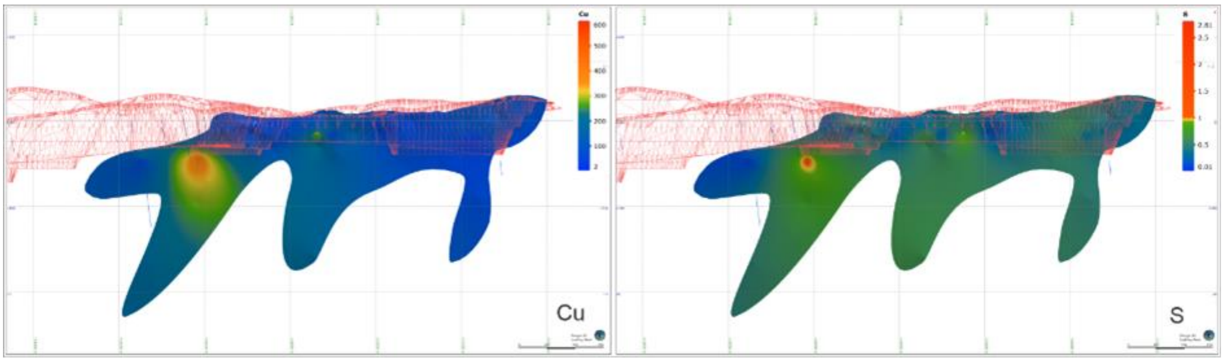


Figure 14-5. Long Section View of Cu and S distribution (Lode 100)

14.6 SAMPLE SUPPORT AND COMPOSITING

14.6.1 Data Flagging

Assays were flagged according to mineralized domain. Each domain was assigned a unique numerical code to allow the application of hard boundary domaining if required during grade estimation. Flagged intercepts were checked to ensure whole samples were included (snapped to drillhole) and that contacts were appropriate (above lower grade boundary). A total of 202 drill holes (1,835.8 m) have intersected the mineralised lodes.

14.6.2 Sample Support

Sample support refers to the length, area, or volume associated with a measurement or observation. In the case of drill samples, it is either the length of core or weight of drill chips that is the main consideration.

The database contains 16,924 acceptable drill samples, the majority of which are half-core (HQ size) samples. There are some quarter-core (NQ size) samples in the database which emanate from a period where SROL were sending samples to Vancouver for analysis and the reduced sample size was required to cut down on freight costs.

Percussion (Reverse Circulation) samples are split using a three-tier riffle splitter. Core samples are listed in the database as either half core (HC), quarter core (QC) or whole core (WC). SROL's drilling is dominantly HQ whilst CGA's drilling was dominantly NQ. The number of each sample type is summarised in Table 14-4. The database contains 1,490 samples of unknown sample type from historic NMC and Hansa drilling which were not used in the resource estimate.

Table 14-4. SROL-Summary of Sample Methods Used

Sample Type	Number	Percentage
RC chips	4546	27%
HQ whole core	138	0.8%
HQ half core	965	5.7%
HQ quarter core	2582	15%
NQ half core	8650	51%
NQ quarter core	43	0.3%

The company have conducted several comparisons of ¼-core vs ½-core and report a slightly higher grade from the ½-core. MA reviewed 41 samples from two holes (Figure 14-6 and Figure 14-7) and notes ½-core reports higher average grade and higher variances (Table 14-5). 64% of ½-core samples report higher grade than their ¼-core equivalent sample.

Table 14-5. Summary Statistics of 1/4 and 1/2 Core Samples

Statistic	1/4 core	1/2 core
Count	41	41
Mean	4.94	5.82
Weighted Mean	4.93	5.66
Minimum	0.02	0.02
Maximum	28.97	35.86
Variance	63.11	94.46
Standard Deviation	7.85	9.60

A QQ Plot (Figure 14-7) shows increased grades report from ½-core samples compared to ¼-core. The effect on the resource estimate is limited as only 13% of assays are ¼-core samples.

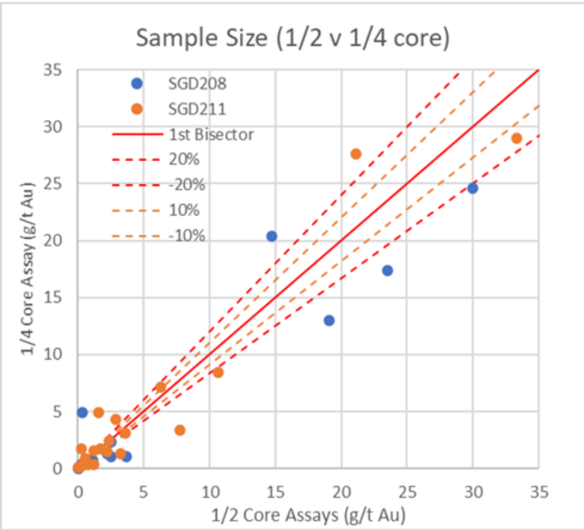


Figure 14-6. Quarter Core vs Half Core (HQ core)

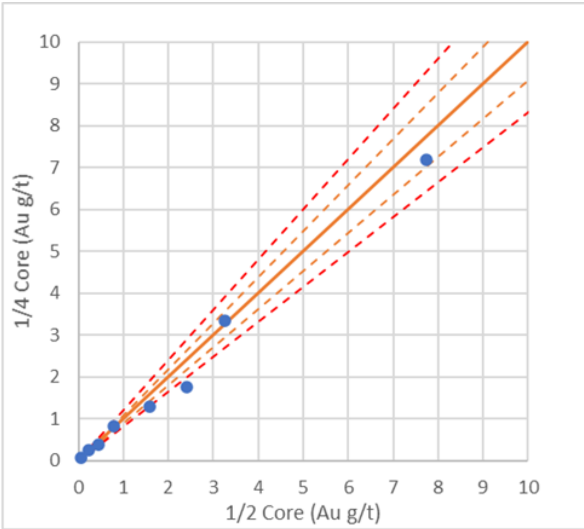


Figure 14-7. QQ Plot of Quarter Core and Half Core

14.6.3 Composite Length

To minimise estimation bias due to differing sample lengths, the drill hole data should be composited to a standard length. Factors to consider when selecting composite length include the original sample length, the vertical dimension of the selective mining unit, and the downhole thickness of the estimation domains.

The sample lengths were statistically assessed prior to selecting an appropriate composite. The majority (91%) of the samples are 1.0 m long (Figure 14-8).

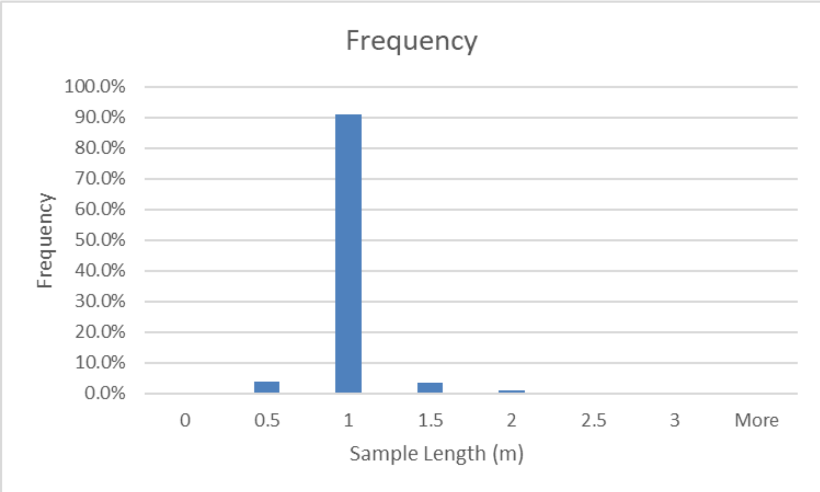


Figure 14-8. Sample Lengths

The samples within the two dominant lodes were composited to varying lengths to determine the effect of composite length on the mean and coefficient of variation (Figure 14-9 and Figure 14-10). Three and four metre composites show an increase in the mean grades, the CV as expected is reducing (increase the sample volume decrease the sample variance). The 1 m composites were selected as most of the database is sampled at 1 m intervals. Compositing honoured the lithology boundaries, and composite lengths less than 0.75 m were discarded.

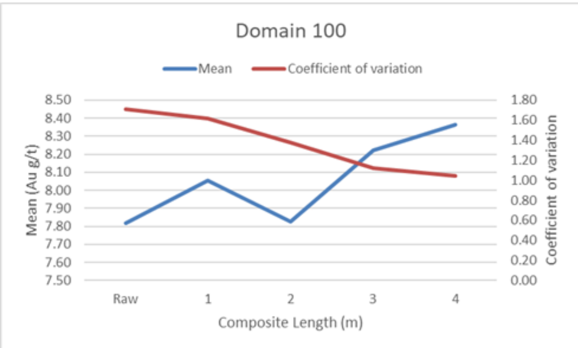


Figure 14-9. Lode 100 Composite Length Test

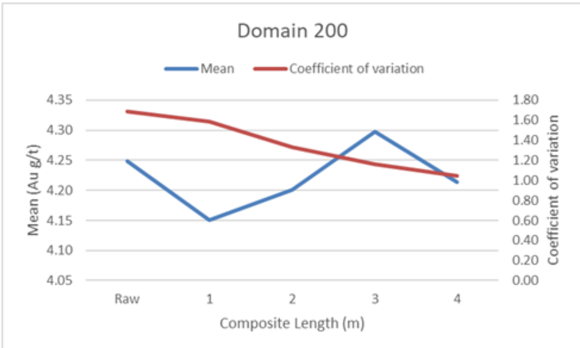


Figure 14-10. Lode 200 Composite Length Test

MA considers one metre composites are appropriate. MA has checked the statistics of the discarded composites to ensure that the estimate would not be biased by removing the short length sample intervals, and no bias has been identified.

14.6.4 Composite Statistics

A domain is a defined volume that delineates the spatial limits of a single grade population. Domains have a single orientation of grade continuity, are geologically homogeneous and have statistical and geostatistical parameters that are applicable throughout the volume (i.e. the principles of stationarity apply). The following charts (Figure 14-11) show generally that the defined wireframes have varying grades and the means shift, particularly in Lodes 200 and 300 where the grade increases to the north. Domains 100, 300 and 400 have extreme outliers. Domains 500 and 600 have limited samples.

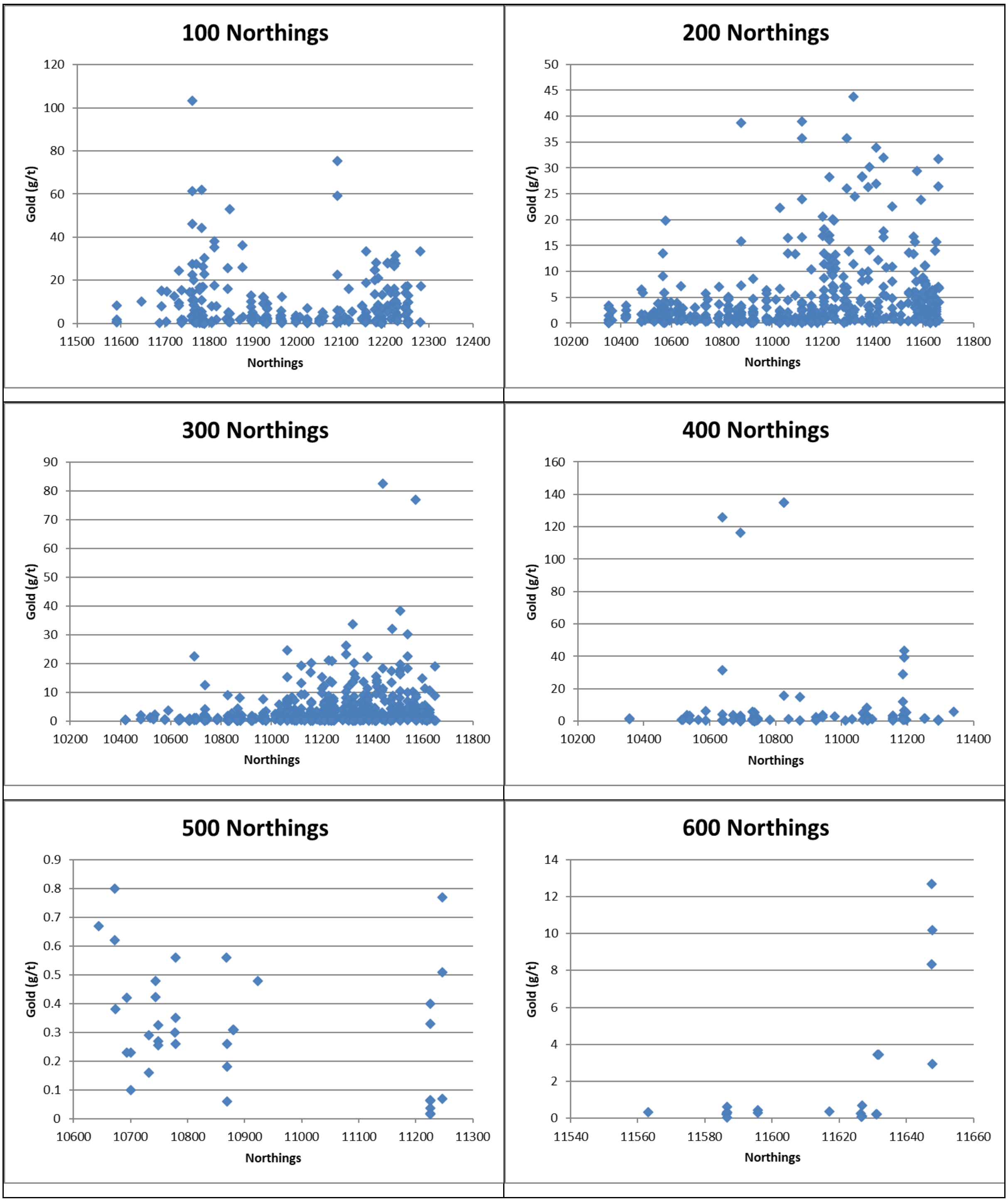


Figure 14-11. Segilola Domain Gold Grades plotted by Northing.

The domains are determined based on geologic knowledge and supported by statistical analysis (EDA). Interpretations were improved by geological constraints that could be confidently correlated between drill holes. The aim is to definite estimation domains that are geologically and statistically self-similar, this is referred to as the definition of stationary zones within the deposit, or stationarity.

An important part of stationarity is the decision on how to group the geological features. After considering the geological history and geometry of the units, and by iteratively combining various groups of lodes and analysing the resulting statistics, MA has decided the estimation domains should be based on individual lodes. The statistics for each lode (uncapped) are summarised in Table 14-6.

Table 14-6. Statistics by Au Estimation Domain

Statistic	Lode/Domain 100	200	300	400	500	600
Str statistics	gold	gold	gold	gold	gold	gold
Variable	Gold	Gold	Gold	Gold	Gold	Gold
Number of samples	290	555	797	104	49	21
Minimum value	0.03	0.02	0.02	0.01	0.02	0.03
Maximum value	103.30	43.71	82.63	135.00	32.20	12.67
Mean	**8.17**	**4.18**	**3.01**	**9.41**	**1.73**	**2.17**
Median	3.00	1.58	1.06	1.19	0.38	0.35
Standard Deviation	12.71	6.61	5.98	27.66	5.31	3.58
Coefficient of variation	**1.56**	**1.58**	**1.99**	**2.94**	**3.07**	**1.65**
10th Percentile	0.47	0.34	0.24	0.32	0.06	0.10
25th Percentile	0.98	0.69	0.49	0.57	0.24	0.22
50th Percentile (median)	3.00	1.58	1.06	1.19	0.38	0.35
75th Percentile	10.03	4.36	3.03	3.30	0.82	3.20
95th Percentile	30.20	17.43	13.25	79.74	13.66	11.42
97.5th Percentile	45.36	26.70	18.19	130.50	26.27	11.42
99th Percentile	61.59	33.00	23.82	135.00	32.20	12.67

14.6.5 Outlier Values

The drill hole database has some high-grade assays that may be due to data entry errors or are just unusually rich and require careful treatment as they have a significant effect on estimated metal content.

Drill hole SGD119 (drilled in 2009) records two isolated samples, each with a length of 3 m. Both assays show high grade mineralization (Table 14-7). The next sample interval, 26 m further down hole, also starts in mineralization. The sampling practice has improved since 2011, the longest samples now permitted are 2 m intervals in barren zones and 1 m intervals in mineralized zones constrained to geological contacts.

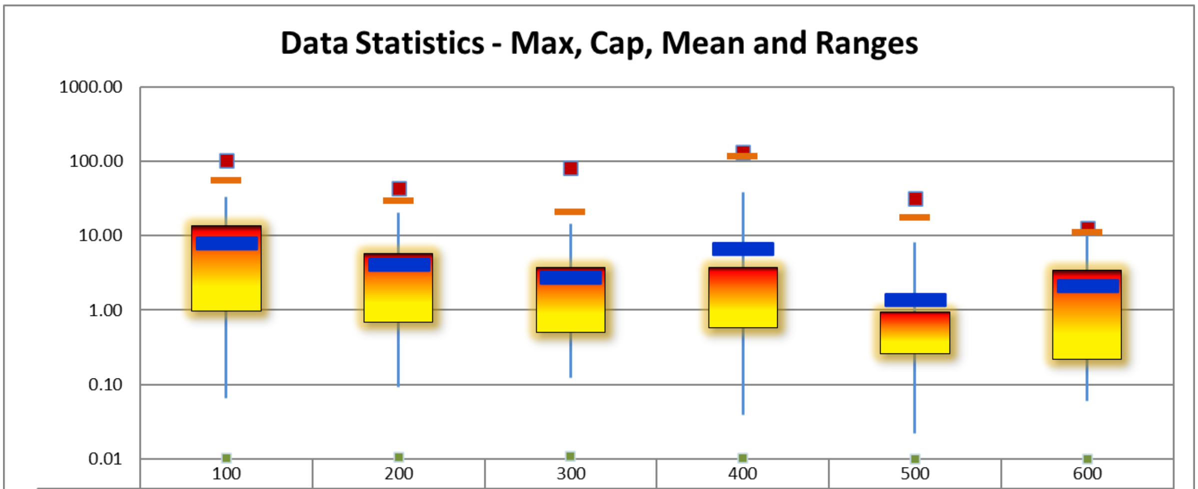
Table 14-7. Outliers Lode 400 Anomalous Sampling

Hole id	Depth from	Depth to	Sample id	Sample Type	Au_Best_ppm	Lab job no	Laboratory ID	Lab Method ID
SGD119	28	31	SX076768	ORIG	135	T0010856	SGS	FAE505
SGD119	31	32	SX076769	ORIG	15.6	T0010856	SGS	FAE505
SGD119	57.85	59	SX076789	ORIG	8.9	T0010856	SGS	FAE505

The highest assay in the database (after hole SGC119) is found in hole SGD174 drilled in 2011 which is 133.27 g/t for a 0.5 m sample from 32.4 m down hole. The next highest assay result is 126,000 ppb Au (126g/t reported in SGD145) followed by 116,200 ppb Au (116.2 g/t reported in SGD146). These are unusual as they are high grade samples but reported in ppb rather than ppm. Fire assays measured and reported in ppb usually have an upper detection limit of 10,000 ppb. There is no evidence in the database that an over-grade assay technique was used for these samples. In MA's opinion these are likely to be data entry errors or conversion errors.

This rudimentary review indicates the assay data is likely in sufficient condition to be used in a mineral resource estimate provided the spurious results are omitted. The database was reviewed by RPA in 2019. MA reviewed 13 holes and accepts RPA's assessment that the database is fit for purpose.

The defined domains have average grades (capped) of between 1.38 g/t (500 – HW) and 7.91 g/t (100 – North). The largest domain (volume and drill intercepts) is 300 – central with an average of 2.78 g/t Au. Capped statistics are shown in Figure 14-12.



	100	200	300	400	500	600
25%	0.98	0.69	0.50	0.57	0.26	0.22
97.5%	33.42	20.59	14.23	38.24	8.06	10.67
2.5%	0.06	0.09	0.12	0.04	0.02	0.06
75%	13.43	5.77	3.70	3.76	0.95	3.44
cap mean	7.91	4.09	2.78	6.80	1.38	2.11
Max	103.3	43.7	82.6	135.0	32.2	12.7
Cap	57.2	30.0	21.2	120.6	17.7	11.4
Count	290	555	797	104	49	21

Figure 14-12. Data Statistics - Box and Whisker Plot

14.7 BULK DENSITY ESTIMATION

The default bulk densities assigned to the block model are based on the weathering profile (Table 14-8). Three DTMS were provided, namely top_FRESH.dtm, Base_Ox.dtm and top_competent.dtm. The weathering categories and associated densities are background densities applied to the block model, later updated with weathered rock densities (Table 14-10).

Table 14-8. Default Densities Assigned to Block Model.

Material	Model Code	No. Samples	Average SG
Completely Oxidised	CX	1	1.99
Moderately Oxidised	MX	80	2.50
Partially Oxidised	PX	92	2.63
Fresh	FR	1604	2.69

Rock types were assigned to the block model based on wireframe interpretations (Figure 14-13). Rock codes assigned to the attribute "rock" are listed in Table 14-9.

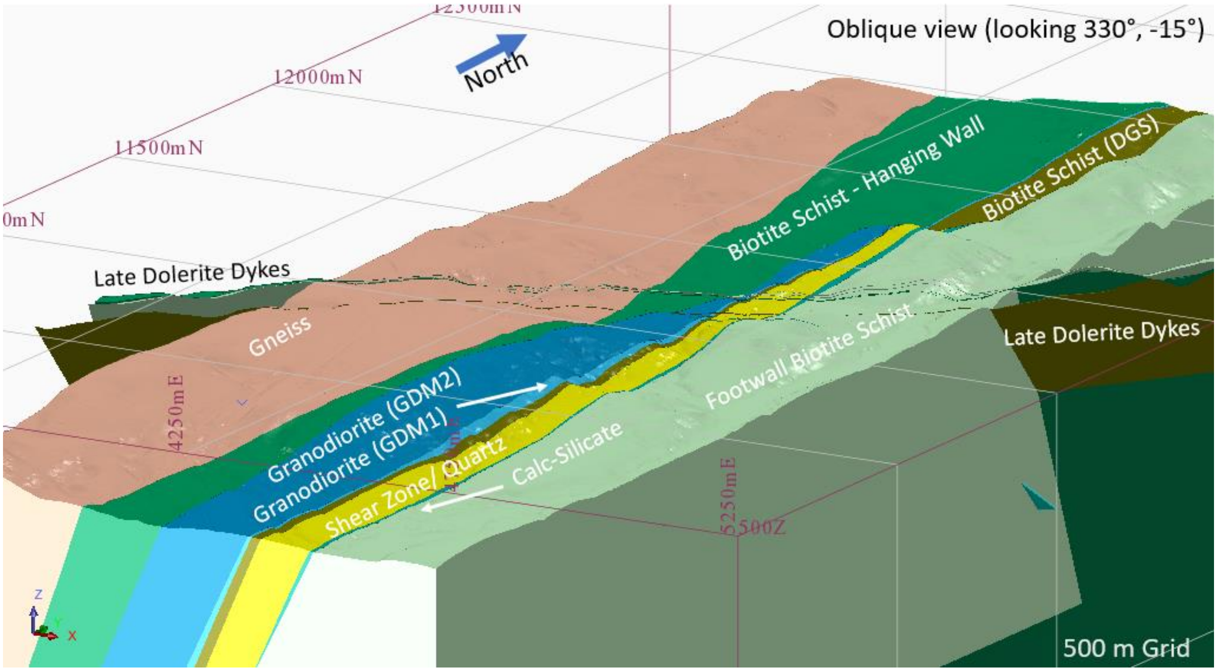


Figure 14-13. Interpreted Lithological Model

Table 14-9. Block Model Attribute "Rock" Codes

Lithological Unit	Block Model Rock Code
Biotite Shear	2
Calc Silicate	3
Dark Grey Schistose	4
Dolerite	9
Granodiorite Massive	5
Granodiorite schistose fabric	6
Gneiss	7
SZQ is the shear/quartz	8

The density is assigned based on rock type and weathering (Table 14-10), for categories that were not statistically valid, (n is too small) the assigned background weathering density remained (Table 14-8).

Table 14-10. Bulk Densities Assigned to Lithology Domains.

Rock Type	Weathering	SG
SZQ	Oxide	2.64
SZQ	Transition	2.65
SZQ	Fresh	2.68
BS	Oxide	1.87
BS	Transition	2.40
BS	Fresh	2.72
CE	Fresh	2.74
DGS	Oxide	2.66
DGS	Transition	2.69
DGS	Fresh	2.68
GDM	Oxide	2.23
GDM	Transition	2.59
GDM	Fresh	2.66
GDS	Oxide	2.61
GDS	Transition	2.63
GDS	Fresh	2.66
DOL	Fresh	2.74

14.8 MOISTURE

No measurements were recorded. All bulk densities used in the resource estimate are dry bulk densities.

14.9 TOPOGRAPHY AND EXCAVATION MODELS

MA was supplied with a high-resolution topography file based on a 2018 Lidar survey. Points within the file represent gridded spot heights at 2.0 x 2.5 m intervals. An historic pit (approximately 300 m long, up to 20 m wide and 7 m deep) is evident in the LIDAR data. The pit exploited the southern extent of the Lode 100. A survey pick up of the historic pit was used to deplete the model.

Blocks above topography (including the historic pit) are excluded from the resource.

14.10 SPATIAL CORRELATION STUDIES

The most important bivariate statistic (spatial correlation between grade and distance) used in geostatistics is the semivariogram. The experimental semivariogram is estimated as half the average of squared differences between data separated exactly by a distance vector ‘h’. Semivariograms models used in grade estimation should incorporate the main spatial characteristics of the underlying grade distribution at the scale at which mining is likely to occur.

Variogram analysis was undertaken in Surpac. Natural 3D experimental variograms were able to be created for the main lodes (100, 200 and 300). Where variogram maps proved difficult to interpret the line of lode (strike) and dip was set as direction one and two respectively, with the third direction generally selected as shallowly plunging to the south, mimicking the general trend of the shoots.

Variogram selection also considered the use of an adjacent domain’s variogram or borrowed from Lode 300, as it is the domain southern lode.

3D experimental variogram modelling used a nugget (C_0) and two spherical models (C_1, C_2), but one spherical model was sufficient at times. The modelled variogram geometry is consistent with the interpreted mineralization wireframes, incorporating a plunge component were identified and modelled accordingly (Table 14-11).

Table 14-11. Variogram Parameters

Copper	Rotation			Variogram					Anisotropy	
Lode	bearing	plunge	dip	Co	C1	A1	C2	C2	Major/ Semi-Major	Major/ minor
100	2.12	9.5	74.92	0.18	0.52	35	0.3	72	1.59	2.3
200	21.25	7.43	57.57	0.35	0.37	40	0.28	70	1.92	2.2
300 - 600	8.9	1.58	65.42	0.4	0.6	40			1.49	2.0

14.11 INFORMING SAMPLE SELECTION AND SEARCH DISTANCES

A kriging neighbourhood analysis was performed to optimise the number of informing samples and search distances. Fourteen blocks were selected (4 on Lode 100 and 5 each on Lode 200 and Lode 300). Blocks from both well and poorly informed areas were selected. Detailed estimation runs were prepared to confirm the proper implementation of the estimation parameters and procedures. Each block was estimated with an increasing number of permitted samples. Kriging statistics (estimated grade, krige efficiency, conditional bias slope, average distance to samples) were then plotted against the number of informing samples to optimise the outcomes. This was done primarily to avoid local conditional biases (too few samples) and over-smoothing (too many samples) of the estimated grade. The selected estimation parameters and procedures were then be applied to the entire domain.

Block estimation uses a three-pass strategy with the number of required samples decreased, and search distance increased for each pass. Reducing the number of required samples limits over-smoothing as search distances are increased.

The maximum number of informing samples varied based on the size of the lode, (quantity of data). Lodes 100, 200 and 300 use a maximum of 16 samples, Lodes 400 and 500 use a maximum of 12 samples and Lode 600 uses a maximum of 10 samples. For subsequent estimation passes the maximum number of samples was reduced to 75% (12, 9 or 7) during pass 2 and further reduced to 50% (8, 6 or 5) for pass 3.

The minimum number of samples were set at eight for all domains. This was reduced to a minimum of six during the second pass and further reduced to two in the final run. By progressively reducing the minimum number of required samples additional blocks could be estimated in each pass.

Search neighbourhoods were defined as an ellipse with the long axis set to 55 m and anisotropic ratios of 1.5:1 for the major/semi major axis (37.67 m) and 2:1 for the major/minor axis (27.5 m). Search distances were doubled for the 2nd pass and tripled for the 3rd pass.

Most blocks (about 80%) have a sample within 40 m of their centroid (Figure 14-14) for Lodes 100 and 300, Lode 200 has most blocks (about 80%) with a sample within 60 m.

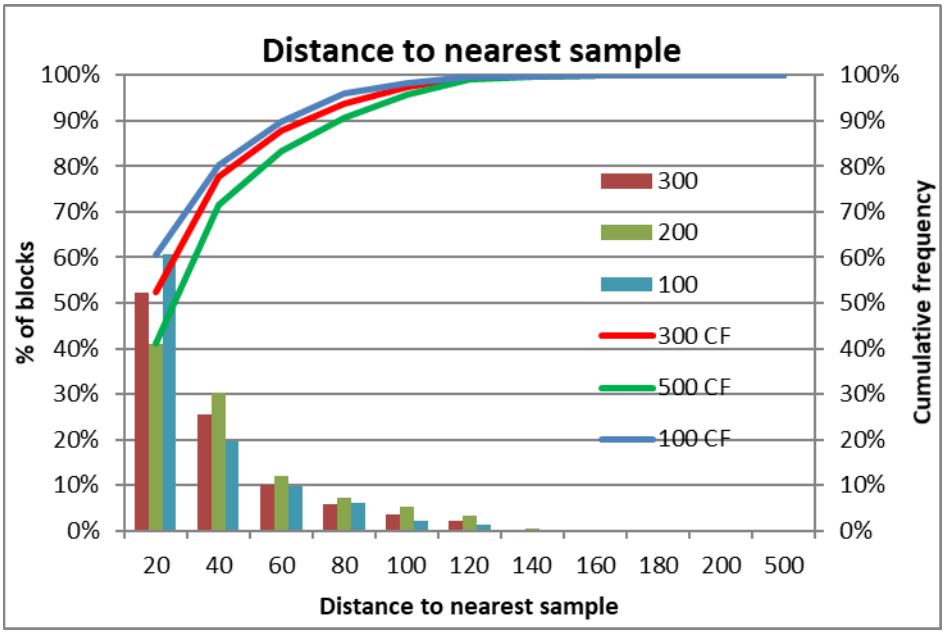


Figure 14-14. Three selected Lodes showing distance to nearest sample.

14.12 MINERAL RESOURCE BLOCK MODEL

The proposed bench heights, geometry of the mineralising controls, drill hole and sample distribution, and the anticipated grade control methodologies being adopted by SROL guided the block size selection.

MA was asked to create a preliminary model in Surpac using the parameters defined in LF by the company. The preliminary model uses a block size of 6 x 24 x 6 m (XYZ) reflecting the intended 6 m working bench (Table 14-12). The model is positioned in line with the 300 m RL being the first complete bench.

Table 14-12. Block Model Parameters

Type	Y	X	Z
Minimum Coordinates	10,000	3,500	-102
Maximum Coordinates	12,424	4,520	450
User Block Size	24	6	6
Min. Block Size	1.5	0.375	0.375

Table 14-13. Block Model Attributes

Attribute Name	Type	Decimals	Background	Description
au_id	Float	2	0	gold ordinary krige estimate capped
au_nn	Float	2	0	gold ordinary krige estimate capped
au_ok	Float	2	0	gold ordinary krige estimate capped
density	Float	2	2.7	Density
deposit	Character	-	WS	Deposit Region
lode	Character	-	WS	Mineralization Domain
lode_id	Integer	-	-99	lode number
rescat	Integer	-	6	Resource classification (1 measured 2 indicated 3 inferred 4 unclassified 5 mined out 6 rock
rock	Integer	-	1	Air = 0 Rock = 1 Andesite = 10
wth	Character	-	FR	FR = FRESH ROCK, PO = PARTIALLY OXIDISED ROCK, OX = OXIDISED ROCK
z_ads	Float	2	0	average distance to samples
z_brg	Float	2	0	bearing of search ellipse
z_cbs	Float	2	0	Conditional bias slope
z_dh	Integer	-	0	number of informing drillholes
z_dhid	Character	-	0	hole_id
z_dip	Float	2	0	dip of search ellipse
z_dns	Float	2	0	distance to nearest sample
z_ke	Float	2	0	krige efficiency
z_kv	Float	2	0	krige variance
z_ns	Integer	-	0	number of informing samples
z_ps	Integer	-	0	1 First Pass; 2 Second Pass Estimate

14.13 GRADE ESTIMATION

In the opinion of MA, the Mineral Resource statement reported herein is a reasonable representation of the Segilola deposit based on current sampling data.

Gold is the primary element of concern. Gold domains are considered to have hard boundaries. To reflect the local orientation of the lodes, dynamic searches were utilised and local undulations in the lodes were determined from the mid-point of mineralized drill hole intercepts. The intercepts were wire-framed and sliced in 10 m sections, which were then smoothed with points every 10 m providing a 10 m grid reflecting the orientation of the lodes. The grid was wire-framed, and the dip and strike of each triangle defined a unique local search orientation for each block.

Grade estimation of gold was undertaken in Geovia's Surpac™ software package (v7.3). Ordinary Kriging ("OK") was used to estimate the gold grades (Figure 14-15). No other elements were estimated.

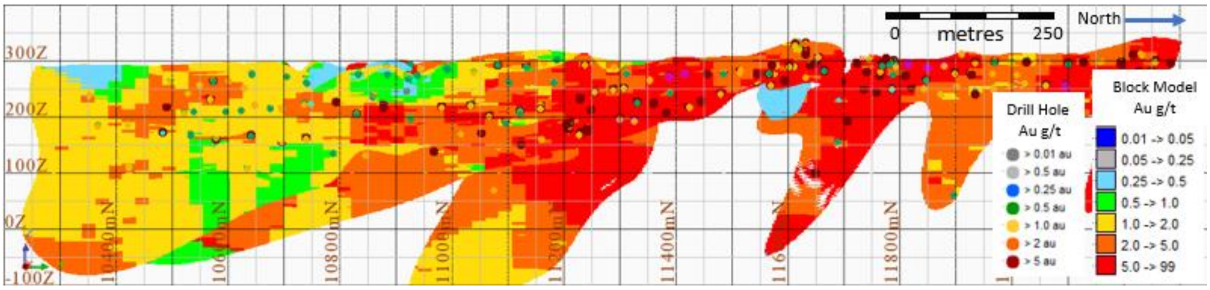


Figure 14-15. Long Section View of Estimated Grades

The krige estimate used a 2 x 5 x 2 discretisation (XYZ), giving discretisation nodes spaced at 3 x 4.8 x 3 m. The distance between nodes approximates 3 times the sample composite length.

An open pit resource is not expected to be developed below 125 m RL. The surface area above the mineralization ranges from approximately 290 m RL to 375 m RL.

14.14 RESOURCE BLOCK MODEL VALIDATION

14.14.1 Validation of Global and Local Estimates and Model Selectivity

The block model was validated by visual and statistical means. The statistical validation included a comparison of drill hole and block grades, and grade-tonnage analysis. An on-screen visual comparison using extracted composite samples and block models was followed by statistical validation with swath plots to compare block estimates with informing sample statistics along parallel sections through the deposits.

14.14.2 Alternate Estimation Methods

To ensure the krige estimate was not reporting a global bias, alternative estimation methods (nearest neighbour and ID^2) were utilised (Figure 14-16). The correlations returned by the alternate estimates were as expected. The nearest neighbour estimate returned less tonnes and higher grade (less contained metal) as block grade is not assigned by an averaging technique (the single closest sample rather than several weighted samples are used to inform a block). The ID^2 estimate is closer to kriging as it does use averaging weighted by distance, but lacks the ability to assign anisotropy, de-cluster the input data, or account for the nugget effect. Using the kriging algorithm provides a reliable estimate due to the ability of kriging to de-cluster data and weight the samples based on a variogram (which incorporates the nugget effect and anisotropy). The biggest impact is seen at the higher cut offs where few high-grade samples are present compared to the neighbouring assay data. This has been partially controlled by applying a grade cap on the outlier values.

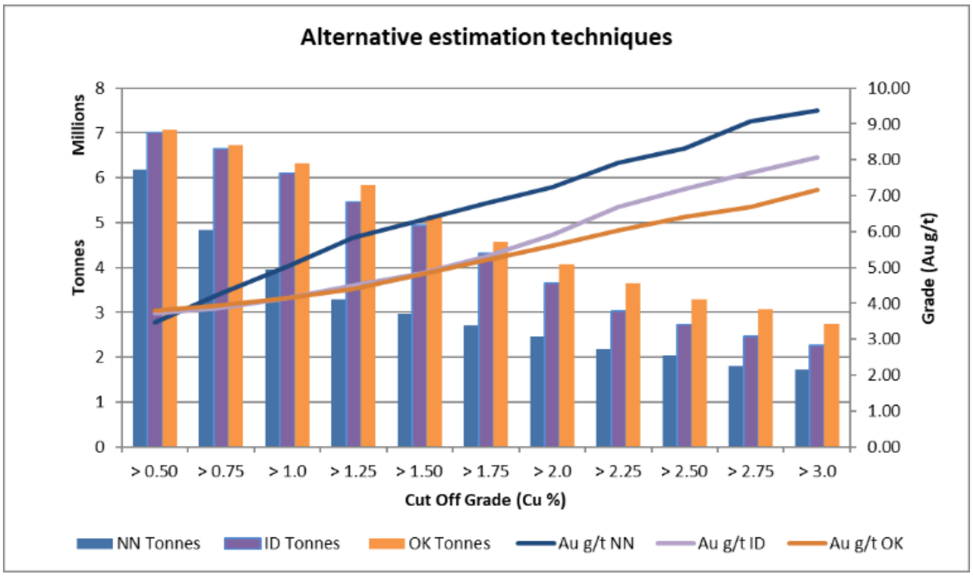


Figure 14-16. Alternative Estimation Results at Nominated Cut-Offs (capped grades)

The tonnages and grades curves (Figure 14-16) provided describe the sensitivity of the block model to differing estimation techniques and should not be interpreted as Mineral Resources.

14.14.3 Global Bias check

A comparison of global mean values within the grade domains shows a reasonably close relationship between composites and block model values (Table 14-14). Lodes 100, 200 and 300 show a good correlation between informing samples and the estimate. Domain 600 shows as a low estimate compared to the informing composite average; the domain has one reasonable grade intercept from hole GCD01, and 6 intercepts of around the 0.5g/t cut off. Domain 400 includes hole SGD119. Domain 500 includes SGD129 which has an intercept of 4m @ 13.40 g/t Au which is supported by 1 m @ 2.25 g/t (SGD145), and 8m @ 1.97g/t (SGD128), with the remainder of the intercepts (11 holes) below 1 g/t. The NN estimate can be considered a de-clustered mean of the input data and confirms the overcall of the composite grades.

Table 14-14. Global Gold Validation by Domain

Domain		Informing Samples		Model (capped estimates)			% Difference OK v Comp grade	% of total volume
lode	volume	Uncapped	capped	OK	ID2	NN		
100	198,808	8.14	7.88	7.06	7.18	6.51	-10%	10%
200	752,648	4.20	4.10	3.65	3.36	3.09	-11%	37%
300	848,831	3.04	2.81	2.81	2.53	2.88	0%	42%
400	179,407	9.19	8.75	8.73	11.56	5.94	0%	9%
500	31,474	2.14	1.69	1.56	1.78	0.79	-7%	2%
600	13,358	2.08	2.03	0.87	0.81	0.67	-57%	1%

14.14.4 Grade Shift through Grade Capping

The grade shift analysis is a check that the grade capping applied has not had undue impact on the global mean grade, and that the capping has achieved its goal of only locally reducing the influence of outliers. MA has prepared a summary table by estimation domain which compare the estimates using uncapped grades and capped grades for Au (Table 14-15)

Table 14-15. Capping Metal Loss Summary by Estimation Domain

Domain	Quantity Blocks	OK Unapped	OK Capped	Grade Shift
100	310	6.53	6.42	-1.68%
200	1,196	3.12	3.04	-2.56%
300	1,182	2.83	2.68	-5.30%
400	436	6.78	6.55	-3.39%
500	36	2.03	1.56	-23.15%
600	15	0.87	0.86	-1.15%

The results show an insignificant impact on the global mean grade. The 500 domain has largest impact, but it affects a small quantity of blocks, while the most significant impact is on domain 300 where a 5.30% shift in mean grade is attributed to outliers. The overall impact is a 3.84% shift in the mean grade. The QP considers that a shift due to grade capping of less than 5% is acceptable.

14.14.5 Estimation Pass Check

The estimation pass check is undertaken to ensure that most blocks have been estimated in the first or second pass, and that only a small proportion of blocks have gone beyond the limits of stationarity within the domain (pass 3). This has been checked visually and by reporting grade-tonnage and total metal by estimation pass from the block model. This calibration was undertaken with consideration for the KNA presented in Section 14.3. Table 14-16 shows percentage of blocks filled (tonnes) in various passes, compared to the percentage of contained metal for each domain estimated within Pass One.

Table 14-16. Gold Estimation Pass Summary

Domain	Pass 1	Pass 2	Pass 3	% contained metal in Pass 1
100	58%	21%	21%	69.2%
200	39%	34%	27%	58.8%
300	58%	25%	17%	66.2%
400	8%	39%	52%	8.0%
500	51%	48%	0%	64.6%
600	42%	56%	2%	72.4%

Most blocks are estimated within the first pass, Domain 400 is the exception, likewise most of the estimated metal resides within the first pass estimates. Domain 400 is well drilled near surface (50 m) and is modelled to depth based on few drill intercepts, requiring a large search distance to provide sufficient composites to fill blocks.

The QP considers that the estimation pass check is consistent with expectations and that the results are acceptable.

14.14.6 Local Bias Check

Swath plots compare the drill hole composite grades to the estimated block model grades over intervals in the Northing (along strike) and Elevation (down dip) directions. Swath plots were generated on vertical E-W 25 m wide swaths to assess local bias along strike by comparing the OK estimate with informing composite means of gold. Results show no significant bias between OK estimates and informing samples and the smoothing effects of kriging are apparent. Notable variances occur at 10580 mN and in the higher-grade lode 100 north of 11650 mN where low tonnes and high grades occur.

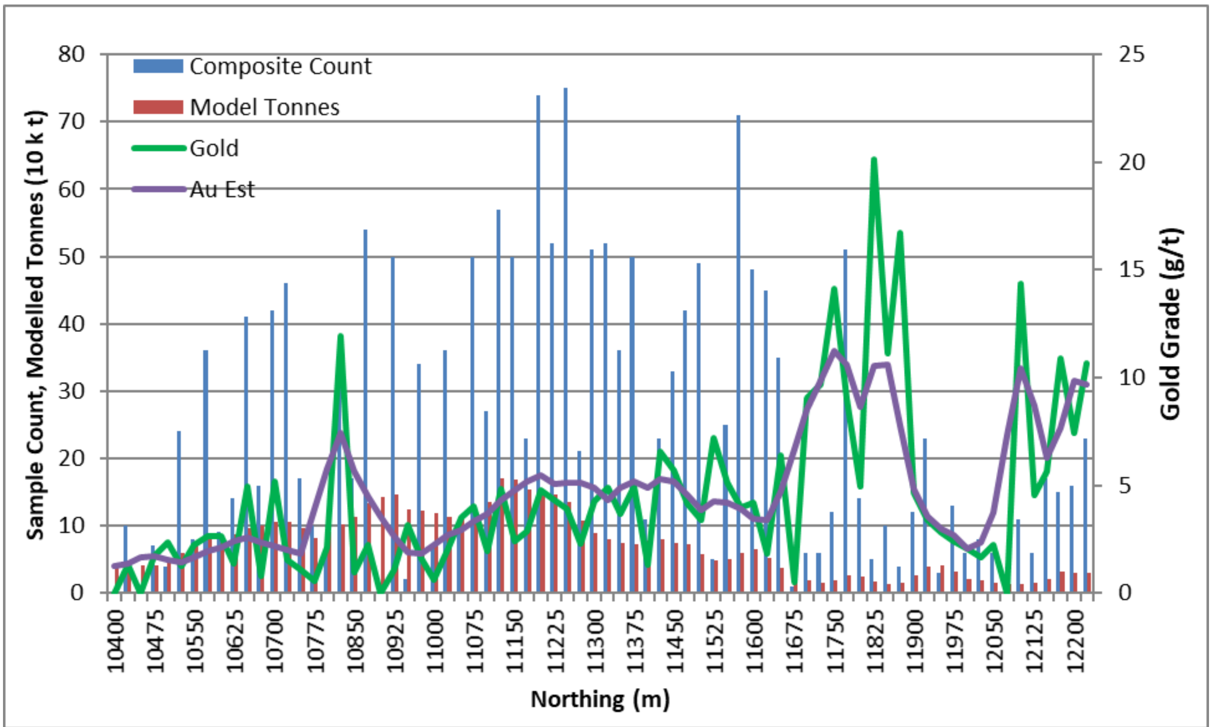


Figure 14-17. Swath Plot – Segilola Deposit by Northing (25m)

A second swath plot was generated on horizontal swaths 12 m wide in the z direction to assess local bias with depth (Figure 14-18). The modelled grades reflect the drill data well. At 270 m RL, intercept grades are high at the top of the shoot developing at the northern end of Lodes 200 and 300. Deeper in the shoot the estimate better reflects the informing sample grades. The grade is erratic at depth where exploration drilling becomes sparse, and the lodes taper off and only the steeper shoots persist.

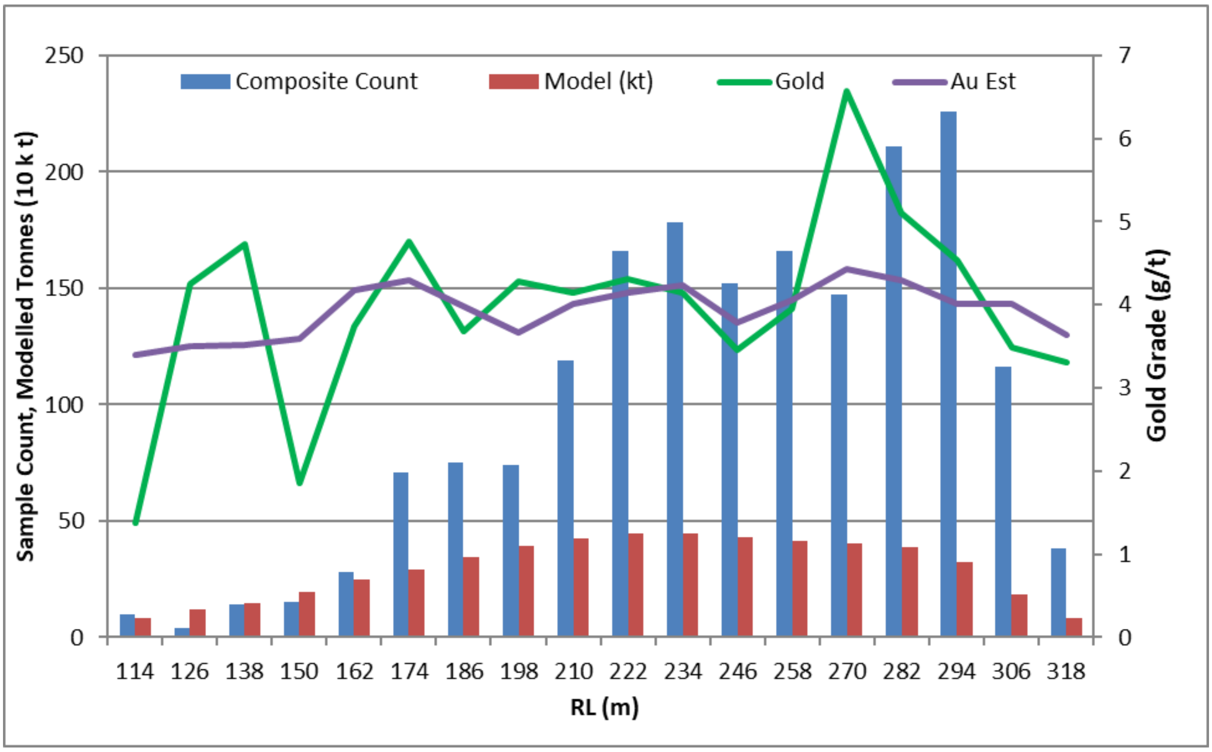


Figure 14-18. Swath Plot – Segilola Deposit by RL (12m)

14.14.7 Reconciliation Studies

Active modern mining of the alluvial and eluvial deposits began in the region around 1942. The Iperindo reef (Lode 100) was first discovered during the working of the eluvial deposits in 1945.

A narrow shallow long pit (about 5 m wide, 15 m deep and 300 m long) was developed during the 1950s. The pit lies on the southern end of Lode 100 (Iperindo reef). Ore was treated by a stamp battery, with manual panning of the crushed material. No records of production are available.

14.14.8 Comparison to Previous Resource

The most recent resource was reported by RPA in 2019. RPA reported an open pit resource above 0.64 g/t Au within an optimised pit shell north of plane 11550mN and within a designed pit shell south of the plane (Figure 14-19,Table 14-18). MA reported the current model using the same parameters with the comparison presented in Table 14-17.

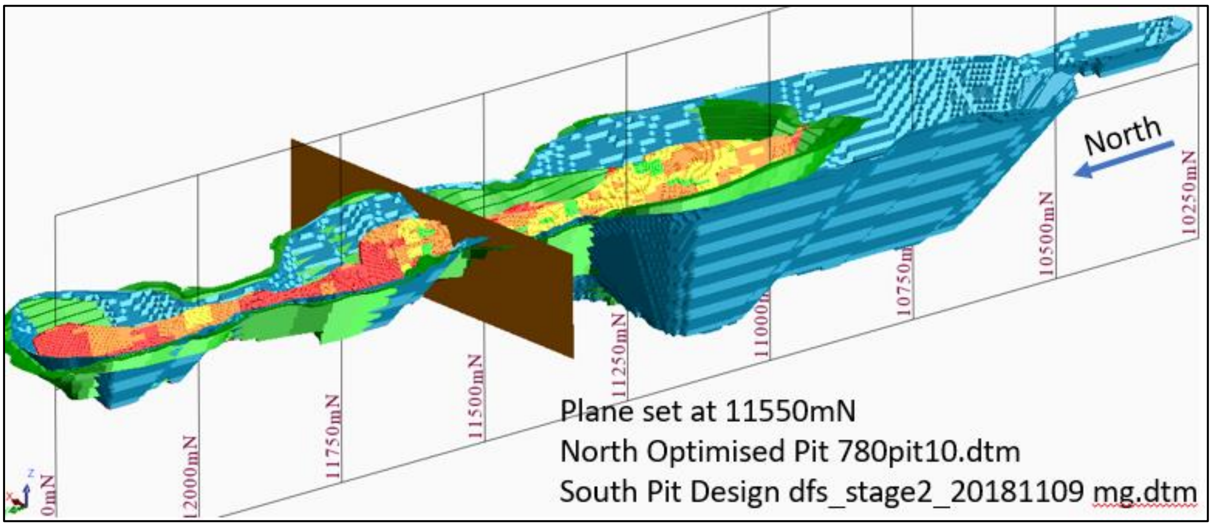


Figure 14-19. Resource Model reported within 2019 Open Pit Constraints.

Table 14-17. Comparison of RPA 2019 Open Pit Resource Estimate and MA Current Model

Resource	March 2019 Open Pit			2021 Resource Model			Var
Category	kt	Au g/t	Au koz	kt	Au g/t	Au koz	ounces
Indicated	3,032	4.52	441	2,531	4.75	386.2	-12%
Inferred	331	6.8	79	2	8.32	0.4	-99%

Total resource reported in 2019 as at 1 December 2018 is shown in Table 14-18 for comparison purposes only.

Table 14-18. Mineral Resource Estimate December 2018

December 2018	Open pit (> 0.64 g/t)			Potential underground (> 2.5 g/t)		
Category	Tonnes (kt)	Grade (g/t Au)	Gold (koz)	Tonnes (kt)	Grade (g/t Au)	Gold (koz)
Indicated	3,030	4.52	441	9	9.39	28
Inferred	33	6.8	73	35	7.9	90

- Open pit Mineral Resources were estimated at a cut-off grade of 0.64 g/t Au and constrained within a pit optimization shell using a Au price of $1,500/oz Au.
- Underground Mineral Resources are estimated by RPA at a cut-off grade of 2.58 g/t Au and constrained within stope shapes using a Au price of $1,500/oz Au.
- Open pit bulk density was interpolated using Inverse Distance Weighting squared.
- Underground bulk density is 2.70 t/m^3.
- High gold assays were capped to 40 g/t Au for open pit resources and 50 g/t Au for underground resources.
- The December 2018 Mineral Resource (Table 14-18) is superseded by the current Mineral Resource Estimate.

The current mineral resource reported as at the 31st of March 2021 is provided in Table 14-19.

Table 14-19. Mineral Resource Estimate March 2021

March 2021	Open pit (> 0.30 g/t)			Potential underground (> 2.5 g/t)		
category	Tonnes (kt)	Grade (g/t Au)	Gold (koz)	Tonnes (kt)	Grade (g/t Au)	Gold (koz)
Indicated	3,674	4.51	532	386	6.13	76
Inferred	32	2.54	3	411	4.95	65

- Open Pit Mineral Resources are reported at a cut-off grade of 0.30 g/t Au. A designed pit wireframe was used to constrain the resources,
- Underground Mineral Resources are estimated at a cut-off grade of 2.5 g/t Au, beneath the open pit constraint and inside the high-grade wireframe lode models,
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability,
- The Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves,
- Totals may not add exactly due to rounding,
- The statement used the terminology, definitions and guidelines given in the CIM Standards on Mineral resources and Mineral Reserves (May 2014) as required by NI 43-101,
- Average Mineralized bulk density is 2.68 t/m^3.

14.15 MINERAL RESOURCE CLASSIFICATION

Block model tonnage and grade estimates have been classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves 2019.

Resource classification is based on data quality, drill density, number of informing samples, kriging efficiency, conditional bias slope, average distance to informing samples and deposit consistency (geological continuity). The confidence in the quality of the data and mining history justified the classification of Indicated and Inferred resources. Data quality does not preclude assigning resources to the Measured category, but geological confidence and grade continuity are not sufficiently defined. Geological continuity has been assumed at 50 m drill-section spacing and is confirmed where drill spacing is tightened.

This Mineral Resource estimate is prepared by digital methods, and the model does have isolated and discontinuous blocks present that have grades above the stated cut-off grade. For the areas considered for underground mining methods these blocks have been excluded from the Mineral Resource statement due to their spatial continuity and size being insufficient to achieve a potentially mineable shape. Blocks of this nature in the open pit area remain in the resource at the lower described cut-off of 0.30 g/t as the blocks form continuous zones at the lower cut-off.

14.15.1 Classification

Indicated resources were classified based on estimates within pass 1 or 2, have the nearest sample within 40 m and an average distance to all samples of less than 60 m, a krige variance of less than 0.6 and a conditional bias slope greater than 0.5.

Inferred resources are estimated within pass two or three, less than 120 m to the nearest sample, and an average distance of 180 m, a krige variance of less than 0.3 and a conditional bias slope less than 0.5.

A block-by-block classification can often result in bullseye or spotted dog shapes that cannot be mined practically, or the outcomes may generate difficulties when preparing Mineral Reserve Estimates. Best practice includes a manual smoothing step to simplify and ensure that the Mineral Resource categories are contiguous. This is achieved by digitising strings in the vertical section, which are then “stamped” through the respective lodes.

MA has completed the following to code the resource classification into the block model and smooth the classification result:

- Average distance to informing samples has been recorded, along with distance to nearest sample, kriging variance and conditional bias slope for each block, considering the four estimation statistics provides a smoother delimitation of boundaries than the distance to the nearest sample,
- Areas of consistent estimation statistics within each lode have been grouped by strings digitized in long section view along the classification boundaries,
- individual lodes were stamped with Mineral Resource categories defined by strings created in long section view.

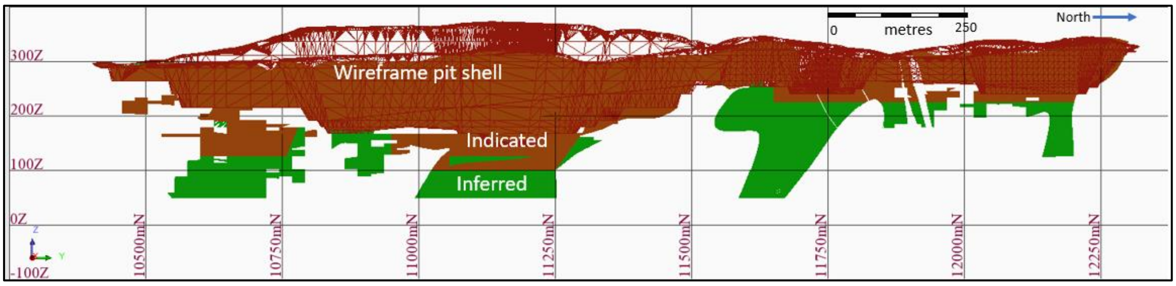


Figure 14-20. Open Pit and Underground Potential Resource Classification

14.15.2 Economic Parameters

The deposit has demonstrable economic value at a 0.30 g/t Au cut off based on the following assumed costs (Table 14-20).

Table 14-20. Assumed Costs for basis of Reasonable Prospects.

	Unit	Oxide	Transition	Fresh
Waste Mining Cost	$/t (waste)	2.27	2.57	3.02
Ore Mining Cost	$/t (ore)	4.07	4.61	5.42
G&A	$/t (ore)	8.7		
Direct Cost	$/t (ore)	12.76	14.46	17.01
Production rate	ktpa (ore)	715		
Processing recovery	%	98.5	97.5	97
Mining dilution	%	*Calculated*		
Mining recovery	%	95		
Royalties	$/t (ore)	2.80		
Freight refining charge	USD $oz Au	6.71		
Gold Price	USD $oz Au	1,650		

Mining factors excluded in this analysis include but are not limited to; capital costs (non-mining, access, and footprint establishment), geotechnical factors, unplanned dilution, and the time value of money. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. However, the defined resource is a contiguous and by virtue of its grade and geometry, should be considered as a mineral resource. As such, the QP considers that the reported Mineral Resource has reasonable prospects for eventual economic extraction by open pit mining methods and Mineral

Resources below the pit shape are considered at a higher cut off to reflect the high costs of underground mining methods.

The Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves. The Inferred resource is of lower confidence than the Indicated resource category. It is reasonably expected that most of the Inferred resource could be upgraded to Indicated mineral resource with continued exploration.

14.16 MINERAL RESOURCE STATEMENTS

All classified resource blocks located between the surface and the designed pit with grades greater than 0.30 g/t Au were included in the reported open pit mineral resources. Mineralization located below the pit shell is considered potentially amenable to underground mining methods when constrained by strings representing continuous mining blocks and reported above 2.5 g/t cut off.

Table 14-21. Mineral Resource Estimate March 2021

	Open Pit (> 0.30 g/t Au)			Potential Underground (> 2.5 g/t Au)			Total		
Category	Tonnes (kt)	Grade (g/t Au)	Gold (koz)	Tonnes (kt)	Grade (g/t Au)	Gold (koz)	Tonnes (kt)	Grade (g/t Au)	Gold (koz)
Indicated	3,674	4.51	532	386	6.13	76	4,060	4.66	608
Inferred	32	2.54	3	411	4.95	65	443	4.78	68

Notes:

1. Mr I Taylor, MAusIMM (CP), Principal Geologist of Mining Associates, is responsible for this Mineral Resource statement and is an "Independent Qualified Person" as defined in NI43-101,
2. This statement uses the terminology, definitions and guidelines given in the CIM Standards on Mineral Resources and Mineral Reserves (May 2014) as required by NI43-101,
3. The mineral resource is considered to have reasonable prospects for economic extraction by open pit mining methods above a 0.30 g/t Au and within a designed pit wireframe. Mineral Resources below the pit shell are considered to have reasonable prospects for economic extraction at a higher cut off of 2.5 g/t where mineralization is continuous,
4. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability,
5. Totals may not add exactly due to rounding,
6. The Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves,
7. Average Mineralized bulk density is 2.68 t/m^3.

14.17 MINERAL RESOURCE RISK ASSESSMENT

Geometry complexity associated with the cross-cutting dykes. There may be additional unidentified dykes to the south.

The offset between Lodes 100 and 200 is only defined by current drilling. Better definition of this offset (fault) will be identified during grade control drilling and pit mapping.

The hanging wall lodes (Lodes 400, 500) are less continuous than the main lodes within the Segilola mineralized zone. While these lodes add to the tonnes and grade available, they are not the main drivers for the pit shell. The lodes have been modelled to highlight the potential locations of structures carrying grade, areas that should be targeted with grade control drilling.

15 MINERAL RESERVE ESTIAMTE

15.1 INTRODUCTION

Detailed technical information provided under this item relates specifically to the Mineral Reserve estimate completed to date and based on the Mineral Resource models and estimates as reported in Item 14.

The open pit Mineral Reserve estimate was prepared by SROL. The Mineral Reserve constitutes the Indicated 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 CIM Definition Standards and excludes Inferred Mineral Resources.

Mining Associates is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserve estimate.

15.2 METHODOLOGY AND RESERVE ESTIMATE PROCESS

The Mineral Resources were converted to Mineral Reserves by the following process:

- The cut-off grade was determined based on optimal cut-off grade as constrained by the defined pit. This was cross-checked using the economic cut-off grade which is determined by the metal selling price and cost, processing cost and recovery, and general and administration (G&A) costs.
- Appropriate mining dilution and mining recovery factors were determined based on a re-blocking exercise of the resource block-model. This resulted in an edge dilution that amounted to approximately 12% weighted average across the deposit.
- Final and interim pit shells were defined using the Lerchs-Grossmann algorithm in Geovia Whittle software, incorporating project specific contract mining costs.
- Pit designs were completed based on the selected pit shells, incorporating appropriate geotechnical mining constraints.
- A Life of Mine (LOM) schedule was formulated based on the pit designs, incorporating appropriate mining equipment production rates consistent with the basis of the quoted mining costs.
- A project economic evaluation was completed.

15.3 MINING PLANNING MODEL

A mine planning model was produced from the Surpac Mineral Resource model described in Item 14. For the purposes of the optimisation software, a mine planning model for the pit was regularised to (X Y Z dimensions):

- 0.375 m x 3.0 m x 1.5 m (x, y, z)

The ultimate Grade Control Model will have a Selective Mining Unit (SMU) block size of 0.375 m x 3.0 m x 1.5 m, suitable for the scale of primary mining machinery at the mine.

15.4 PIT OPTIMISATION

Conventional Whittle software in Surpac (Geovia) was used to determine a design pit shells for the deposit. Optimisations were completed based on selecting a pit shell for the deposit conformable with the practical cutbacks required to achieve the planned production. The optimisation accounted for recoveries of gold metal in dore, as determined for variable Au process recovery relationships.

Final pit shells were generated using Whittle based on the parameters listed in Table 15-1 below.

Table 15-1. Whittle Parameters

	Unit	Oxide	Transition	Fresh
Mining costs – waste:				
Direct Cost	$/t (waste)	2.27	2.57	3.02
Haulage Cost Per Vertical Metre	$/t per m vert	n/a	n/a	n/a
Total Mining Cost	$/t (waste)	2.27	2.57	3.02
Mining costs – ore:				
Technical Services (incl. Grade Control)	$/t (ore)	2.40	2.40	2.40
Direct Cost	$/t (ore)	1.67	2.21	3.02
Haulage Cost Per Vertical Metre	$/t per m vert	n/a	n/a	n/a
Total Mining Cost	$/t (ore)	4.07	4.61	5.42
Overhead costs:				
G&A	$/t (ore)	8.7		
Total Costs	$/t (ore)	8.7		
Processing costs:				
Direct Cost	$/t (ore)	12.76	14.46	17.01
Total Processing Costs	$/t (ore)	12.76	14.46	17.01
Operating factors:				
Production rate	ktpa (ore)	715		
Processing recovery	%	98.5	97.5	97.0
Mining dilution	%	*Calculated*		
Mining recovery	%	95		
Economic factors:				
Royalty (Vendor)	$/t (ore)	1.24		
Royalty (GoN)	$/t (ore)	1.56		
Freight refining charge	USD $oz Au	6.71		
Gold Price	USD $oz Au	1,650		
Geotechnical parameters:				
Total slope angle	Degrees	*Refer to Geotech Design Criterion (15.5)*		

15.5 PIT SLOPE PARAMETERS

Average parameter values for the east and west wall of each pit are summarised in Table 15-2. Table 15-2 summarises the inter-ramp, overall wall angles and bench configuration for the modified slope configuration, proposed by SROL, using 12m benches exclusively in the northern section, two 12m benches in the southern section within the fresh rock, and 3m flitches throughout.

Table 15-2. Revised Pit Design Criteria (12m Bench Height)

Pit	Section mN*	IRA				OWA with respect to Surface				
		West		East		West		East		Pit Floor
		Crest RL	IRA	Crest RL	IRA	Crest RL	OWA	Crest RL	OWA	RL, m
Southern	11165.5	370	52.2	320	42.0	370	51.0	320	38.0	144.0
Central	11745.5	324	49.0	332	48.0	324	40.0	332	39.0	240.0
Northern	12045	337	44.5	322	49.0	337	49.0	322	43.0	240.0
* Northing sections are taken from mine grid northings, perpendicular to the pit wall										

15.6 METAL PRICES

Metal price used is inconsistent with some consensus pricing information tabulated in Item 22.1. The optimisation metal price input was derived after discussion with the Thor Group Management and their medium-term gold price outlook is as follows:

- Gold = $1,650/oz

The pit shell selected was based an equivalent gold price of $1,650 oz Au. This price was used during the pit optimization. Refer to Item 22 for more information regarding gold price.

15.7 METAL RECOVERIES

The input processing recovery projections for Whittle are shown in Table 15-3. Recovery estimations are based on the 2010 Ammtec testwork. The optimisation differentiated between oxide, transitional and fresh recoveries, whereas the economic model assumed a consistent recovery of 97% for oxide, transitional and fresh.

Table 15-3. Metallurgical Recoveries (Ammtec 2010)

Operating factors:	Unit	Oxide	Transition	Fresh
Production rate	ktpa (ore)		715	
Processing recovery	%	98.5	97.5	97.0

15.8 MINING COSTS

Variable mining costs comprising drill and blast, load and haul unit costs, on a bench-by-bench basis, were calculated for each pit, based on its position and the location of its exit ramp, relative to dumps and the ROM tip and supplied contractor rates.

Drill and blast unit costs, based on material type, were calculated according to the desired powder factor for each material type. Drill and blast cost is assigned according to material type and not elevation within each pit. Item 16 has more detail on the calculation of D&B cost.

Item 21 provides more information on the ore and waste mining costs in the pit optimisation process. The overall average mining costs are as follows:

- Average contract ore and waste mining cost = $ 6.8/BCM (including drill, blast, load and haul, ancillary)
- Average overall mining cost = $ 7.73/BCM (including mining overheads in addition to contract costs)

Overall mining costs includes grade control drilling and assaying, as well as ROM ore rehandling.

15.9 PLANT COSTS

Since the Project will be mill constrained, the process operating costs for each area were input as the sum of the fixed and variable costs.

These included:

- Consumables ($/t plant feed)
- Maintenance ($/t plant feed)
- Power ($/t plant feed)
- Laboratory ($/t plant feed)
- Infrastructure ($/t plant feed)
- Personnel ($/a fixed)

Fixed G&A cost over the project was expressed as $/t plant feed for the optimisation.

Table 15-4. Process Plant Costs

	Unit	Oxide	Transition	Fresh
Overhead costs:				
G&A	$/t ore	8.7		
Total Costs	$/t ore	8.7		
Processing costs:				
Direct Cost	$/t ore	12.76	14.46	17.01
Total Processing Costs	$/t ore	12.76	14.46	17.01
Operating factors:				
Production rate	ktpa (ore)	715		

15.10 METAL COSTS

In addition to royalties calculated on spot gold price for metal sold basis, input metal costs for the Segilola Mine gold production comprise:

- Gold refining charge $6.71/oz Au

Item 22 provides an explanation of the derivation of the metal costs used for pit optimisation input.

15.11 MINING DILUTION

The overall mining dilution was estimated to be 12% (Indicated material). The proportion of the model amenable to open pit mining has edge dilution incorporated by re-blocking up one sub-block unit.

The approach of re-blocking compares well with the previous Orelogy method of applying a skin of 0.5 m on the lode interpretation, which determined that the average dilution across within the 2019 DFS pit limits was 14%.

Detailed discussion on the rationale behind the 12% dilution estimation and its impact, is provided under Item 16.

15.12 MINING RECOVERY

A 97% mining recovery factor was applied to account for the quantity of ore that is lost due to spillage and/or re-handling and to account for any unforeseen additional ore losses (ore hauled to the waste dump, etc.).

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

15.13 CUT-OFF GRADE ECONOMIC PARAMETERS

The applied formula for economic cut-off grade used 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 formulation of the Mineral Reserve was calculated as 0.37 g/t Au. Previous estimates used were 0.77 g/t Au and 0.70 g/t Au as per the original DFS in 2019 and Orelogy work completed in 2020.

The 0.37g/t cut-off grade was applied to all material types and all three pits.

15.14 OPTIMISATION RESULTS

Table 15-5 lists the inventories from a selected sequence of optimisation shells. The optimal pit shell is shown as the revenue factor 0,7 shell (pit shell no. 6). This shell was selected as it is operationally the preferred pit shell, on the basis that a lower volume of waste is required to be mined. More detail is provided under Item 16.

Table 15-5. Base Case Optimisation Results

Final Pit ($1,650/oz Au)	RF	Open pit cashflow (M US$)	Waste (M) tons	Ore (M) tons	Grade g/t Au	Strip Ratio W:O	Mine Life (years)	Au (koz) Output
TOTAL	0.7	$ 471.2	61.2	4.5	3.7	13.,6	6.3	478.4

15.15 OPTIMISATION SENSITIVITY

A series of Whittle shells were generated based on different revenue factors against a base case price of $1,650 oz Au as summarised in Table 15-6.

Table 15-6. Optimisation Sensitivity Analysis based on Price

Price $oz Au	Rock Mt's	Ore Mt's	Strip W:O	Max Bench No.	Min Bench No.	Grade g/t Au	Ounces koz's Au	Revenue $M USD	Net Profit $M USD
165	0.5	0.0	11.6	150	123	15.1	20	29.6	27.0
330	3.8	0.4	9.5	150	119	7.1	83	124.0	103.4
495	9.1	0.8	10.7	154	113	6.1	152	226.5	179.4
825	37.6	2.7	13.0	160	91	4.3	369	549.0	363.7
990	59.1	4.0	13.7	160	84	3.8	498	740.3	452.7
1,155	65.3	4.5	13.6	160	81	3.7	530	789.4	471.2
1,320	70.4	4.9	13.5	160	78	3.5	553	822.8	479.4
1,485	72.5	5.1	13.2	160	77	3.4	563	838.8	483.0
1,650	77.2	5.5	13.2	160	75	3.3	580	864.0	484.7
1,815	80.1	5.7	13.1	160	74	3.2	590	879.2	485.1

15.16 DETAILED PIT DESIGNS

The pit has a total length of 1,880 m in a north-south direction, 450 m wide and 270 m from the highest to lowest point. The deepest part of the pit is in the south.

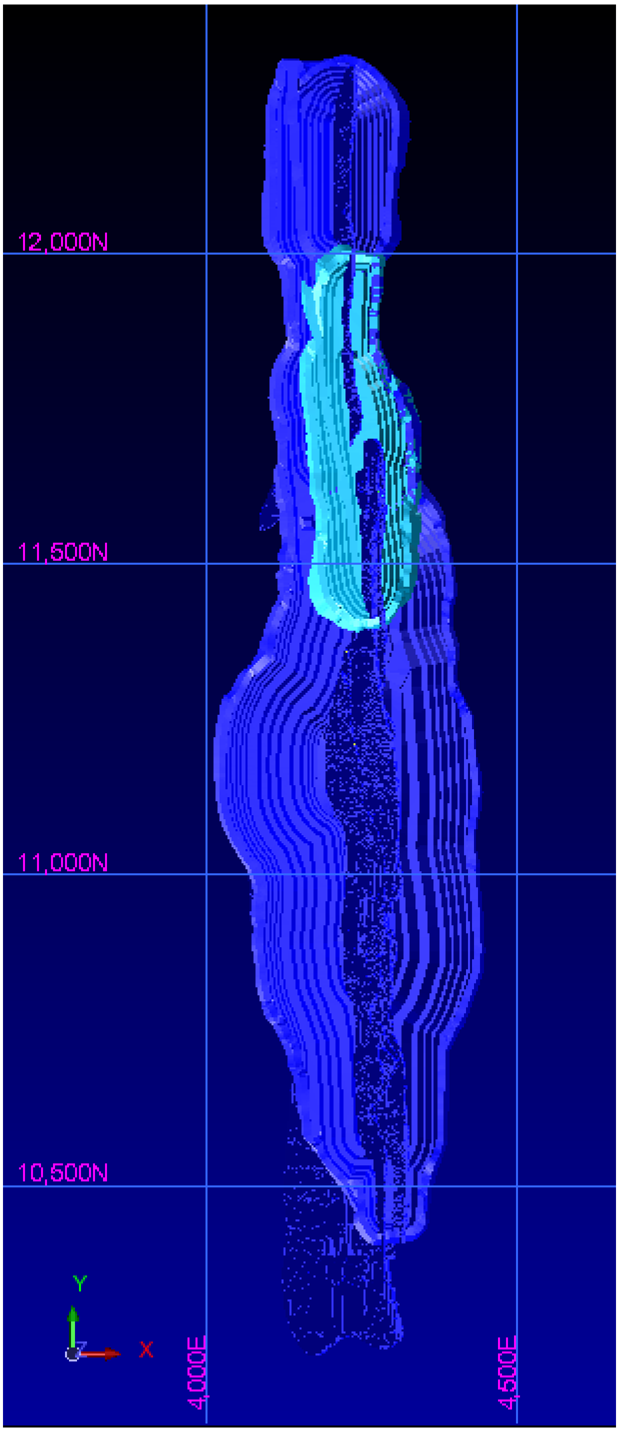


Figure 15-1. Plan View - Detailed LOM Pit and Starter Pit (Local Grid)

SROL believes that it is possible to continue to upgrade the resource in this section of the deposit both at depth and along strike. Therefore, potential for an underground operation exists in the future.

Resource update under Item 14 has more detail on the underground resource volume (Inferred Resource at present).

15.17 MINERAL RESERVE STATEMENT

As at the end of March 2021, the Probable Mineral Reserve is estimated as 4.0 million tonnes at 4.0 g/t Au as depicted in Table 15-8. A comparison between previous reserve estimates and the present, is listed in Table 15-7.

The cut-off grade for the parameters used in the formulation of the Mineral Reserve was calculated as 0.37 g/t Au. Previous estimates used cut-off of 0.77 g/t Au and 0.70 g/t Au as per the 2019 DFS and Orelogy work completed in 2020.

The 0.37g/t cut-off grade was applied to all material types and all three pits.

The Indicated Mineral Resources have been converted to Probable Reserves. Unclassified/Inferred resources that fall inside the $1,650/oz ultimate pit design are excluded from reserve reporting. A breakdown by classification is provided in Table 15-7.

The mining production schedule for the design Mineral Reserve pit is described under Item 16.

The reserve ore tonnes have increased from 3M to 4M tonnes at a consistently high 4 g/t Au of Probable material. A large majority of the increased tonnes have come from successfully conversion of inferred material into indicated material, particularly in the southern extents of the Segilola deposit.

Table 15-7. Reserve Estimate Comparison

	Tonnes (Mt)			Grade (g/t Au)			Contained Metal (koz's Au)		
	DFS 2019	Orelogy 2020	SROL March 2021	DFS 2019	Orelogy 2020	SROL March 2021	DFS 2019	Orelogy 2020	SROL March 2021
Probable	3.0	3.3	4.0	4.2	4.1	4.0	405	422	517
Total	3.0	3.3	4.0	4.2	4.1	4.0	405	422	517

Table 15-8. Reserve Estimate March 2021

	Tonnes (Mt)	Grade (g/t Au)	Contained Metal (kOz Au)
Probable	4.0	4.0	517
Total Ore Reserves	4.0	4.0	517

15.18 MINERAL RESERVE RISK ASSESSMENT

- No significant risks have been detected.
- A review of the financial model supports the robustness of the reserve.
- Minor design aspects are noted in Item 16 that might pose operational risk (slope stability and rainy season pit dewatering).
- In view of the tight mining width of the various lodes in several areas, dilution in excess of the tight optimisation assumption of 12% is possible. This will have a marginal impact on production (head grade), and thus financials. This poses mainly a financial risk and less of a Mineral Reserve risk.
- Mining contract structure does require strict adherence to planned volumes. If these are not achieved, unit rates will increase, which places performance risk in the mining section with SROL. This poses financial risk and not a Mineral Reserve risk.

16 MINING METHODS

The deposit is amenable to conventional open pit mining methods and gold processing using conventional comminution, gravity concentration, and Carbon in Leach (CIL) recovery.

The Project designed in the DFS is an open pit operation feeding a conventional gold processing process plant. The projected Life of Mine (LOM) is approximately five and a half years, comprising approximately four years of open pit mining with processing continuing for a further 14 months. The LOM ore production is 4.0 million tonnes (Mt) at an average grade of 4.0 g/t Au. The process plant is designed for a throughput of 715,000 tpa and gold recovery is projected to be 97%. The total gold mined over the LOM is 517,000 oz Au. The process plant produces an annual average rate of 83.6koz Au for approximately six years recovering a total of 502koz's Au.

16.1 MINE SITE LAYOUT AND INFRASTRUCTURE

16.1.1 Site Layout

The site consists of multiple waste dumps on the north-west, east and ROM pad. In addition to this, several main arterial haul roads will be utilised to access the ROM pad and waste dumps from the main pit as depicted in Figure 16-1.

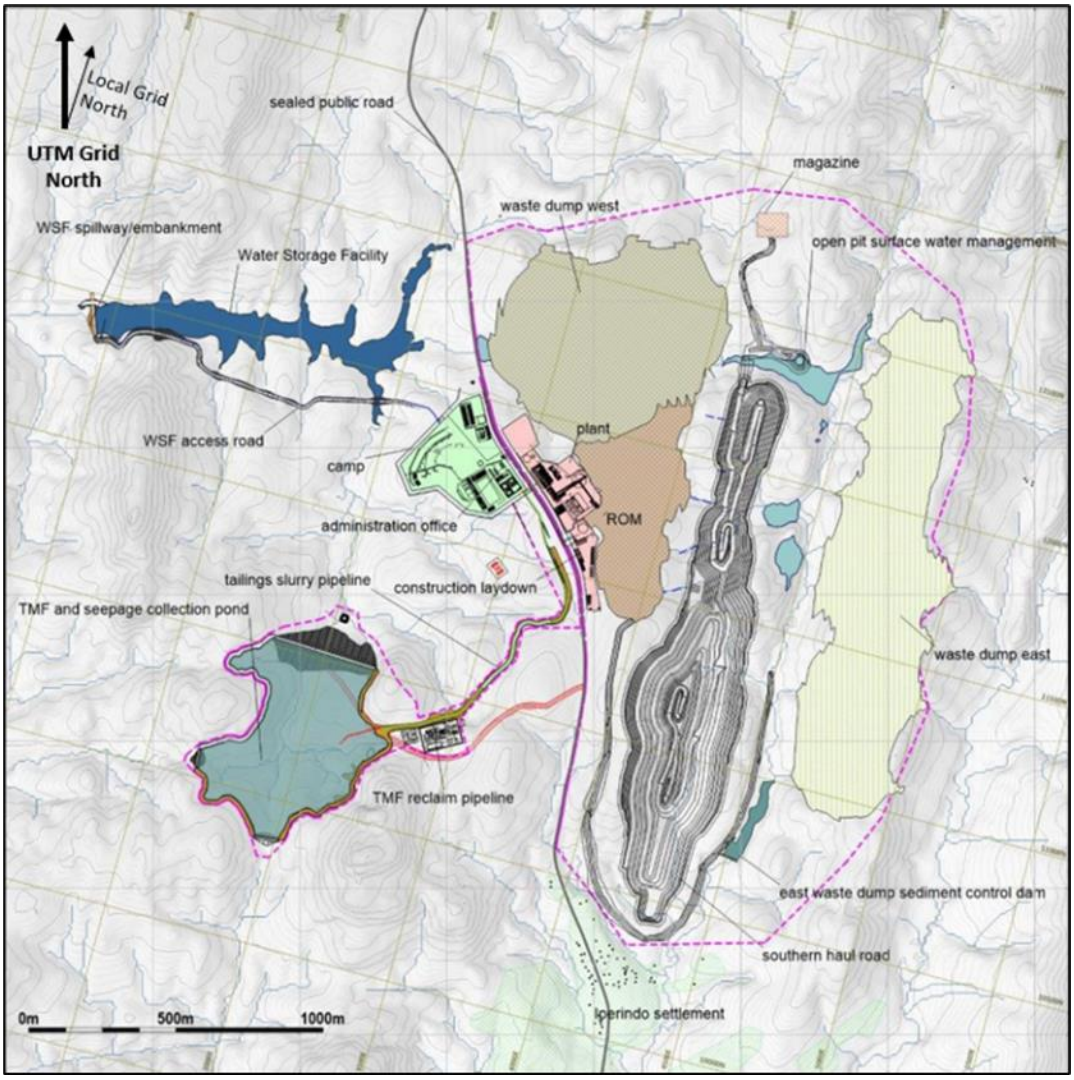


Figure 16-1. Site Layout

16.1.2 Waste Dumps

The dumps were designed using an angle of repose for the waste rock of 37˚. The batter angle was 37˚ as per the expected angle of repose for waste rock. The berm widths used were 10 metres. Thereby providing an Overall Slope Angle of (OSA) of 23˚. The total waste rock volume provided (Table 16-1) can accommodate up to 40M BCMs of waste rock.

The eastern waste dump was set back somewhat from the watercourse to keep it above the 100 year flood line. Additional benefit of setting it back further to the east was to reduce the pressure on the eastern high wall of the pit. The excess volumes in the dump designs provides optionality to expand the pit at depth and along strike.

Table 16-1. Waste Dump Design - Estimated Volume

From	To	North-West (BCMs)	In-pit (BCMs)	Eastern (BCMs)	ROM Pad (BCMs)
228	240	-	-	-	-
240	252	-	0.1	-	-
252	264	-	0.2	-	-
264	276	-	0.3	-	-
276	288	-	0.3	-	-
288	300	-	0.3	-	-
300	312	-	0.4	0.1	-
312	324	0.4	0.4	0.9	0.0
324	336	1.6	0.3	2.1	0.3
336	348	2.9	-	3.3	1.0
348	360	3.2	-	3.8	1.4
360	372	2.9	-	3.4	1.2
372	384	2.4	-	2.5	0.2
384	396	1.7	-	-	-
396	408	1.2	-	-	-
408	420	0.7	-	-	-
420	432	0.3	-	-	-
TOTAL		17.2	2.3	16.1	4.1

16.1.3 ROM Pad

Initial waste rock dumping activities will focus on early completion of the Run-of-Mine (ROM) pad to its full size.

As of the 17th of March 2021, the clearing and grubbing for the footprint of the ROM pad area is nearing its final stages (Figure 16-2). After which, the haul road and pad construction will commence. The initial ROM pad design is estimated to require approximately 300,000 BCM of fill material. This material will be sourced from overburden from the pit.

Figure 16-2. ROM Pad Clearing and Grubbing Activity (17th of March 2021)

The ROM pad is more than 600 metres in length from north to south and 90 to 180 metres wide in places. Conservatively, this should be ample area to stockpile 300,000 tonnes of ore on the ROM stockpile. The overall volume of waste from the pit required to complete the ROM pad is 4 M BCMs.

Furthermore, the ROM Pad can easily accommodate a skyway. A skyway will create a safe divide between the Loader and the Dump trucks thereby enabling efficient dumping of ore on the ROM Pad except where direct tipping is the main activity. SROL are considering different designs for a skyway.

A portion of the ROM ore is tipped directly into the primary crushers. Approximately 80% of ROM ore is tipped onto ROM ore stockpiles to be rehandled by frontend loader into the ROM crusher. Rehandling is done to ensure a controlled grade is fed as is realistically practicable.

Grade control results are used to direct ROM ore tipping into the crusher and onto the various ROM stockpiles. To ensure adequate process control, ore is fed according to a blending schedule, which combines direct tipping and rehandling from the various ore stockpiles.

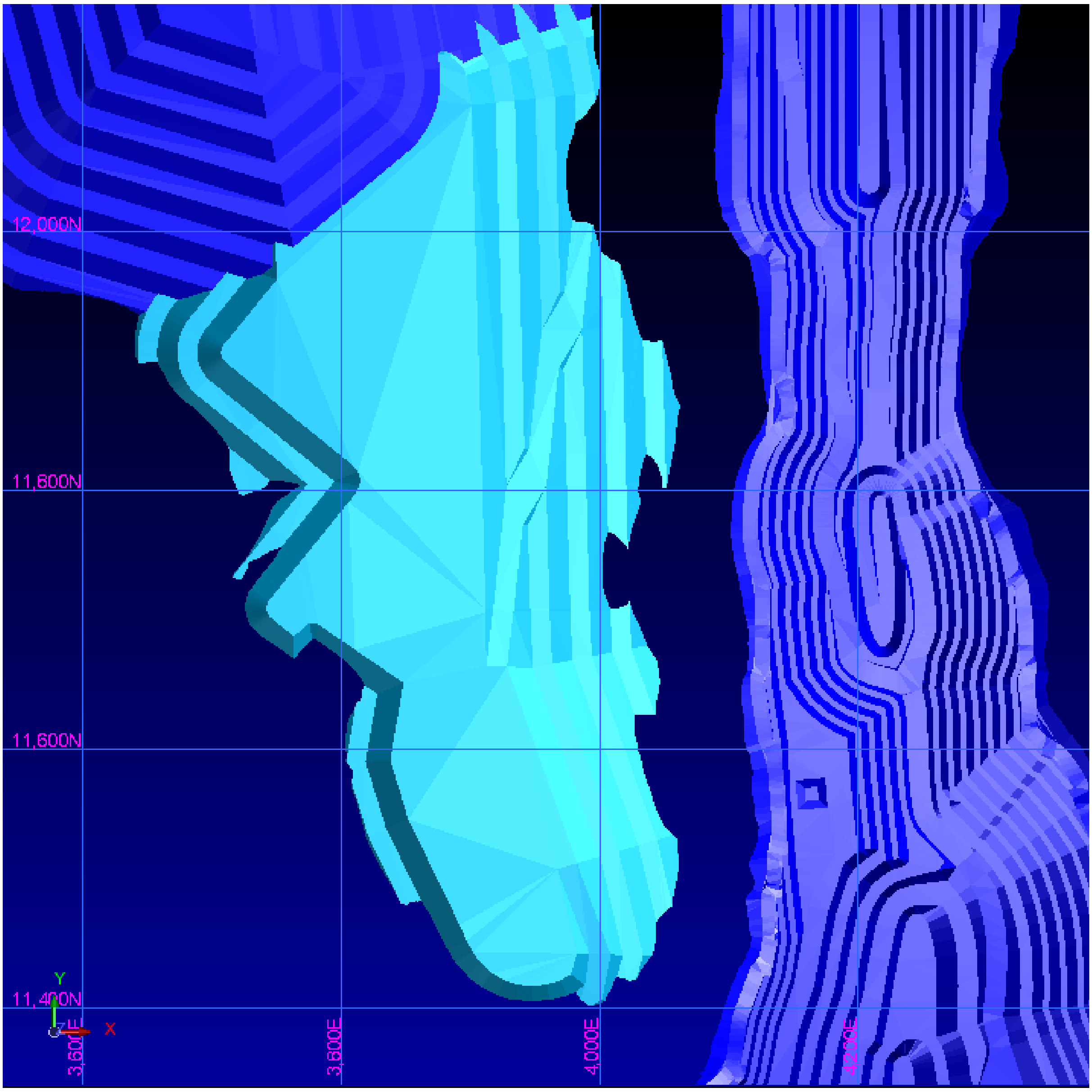


Figure 16-3. ROM Pad Plan View (Local Grid)

MININGASSOCIATES

16.2 MINING METHODS AND OPERATIONS – WORKFLOW

The project is a conventional open cut operation utilising a fleet of drill and blast, load and haul equipment.

SROL has engaged SINIC as the primary contractor to manage the mining contract which entails drill and blast, load and haul summarised accordingly:

- Owner; SROL
- Principal Contractor; SINIC
- Sub-Contractors; Sino-Hydro (Load and Haul) – Auxin (Drill and Blast)

16.2.1 Grade Control

Conventional open pit grade control practices will be used at Segilola Mine, incorporating RC drilling and sampling on a suitably designed drilling pattern and to cover three successive 6.0 m benches (6 x 3.0 m mining flitches). Sampling will employ the use of a rig mounted splitter, with samples collected every metre.

Grade control drilling will be conducted ahead of mining to ensure adequate information to lead short term mine planning and scheduling to limit dilution and ore loss. Reverse Circulation (RC) holes with a 123 mm diameter drill bit, will be drilled on a 5 m (across strike) by 8 m (along strike) spacing, angled at 60° toward 090° (mine grid) and 1.0 m sample interval.

Quality control procedures will be followed, based on international Certified Reference Material (standards), field duplicates, and blank samples at a ratio of 5%. Routine grade control samples will be collected at one metre intervals using a single pass splitter, providing approximately 2 kg of rock chips. A 50 gram charge will be prepared from this split for fire assay (FA) analysis for gold determination at the mine site laboratory. Every second sample of pulp reject submitted to the laboratory is analysed for carbon content.

To assist grade control and better geological understanding, pit floor mapping will be conducted, which in turn will optimise ore block boundary interpretation.

16.2.2 Drilling and Blasting

Blasthole drilling will be conducted with track-mounted mobile drill rigs PowerROC D55 and the XuanHua 370 for the production and presplit drilling respectively.

Current blast design is based on a 127 mm diameter hole. Blastholes are drilled for a 6 m bench height, with a 1 m sub-drill. Drilling for 6 m bench height is done with 115 mm diameter holes when required, with a 0.5 m sub-drill. However, it is possible to use a tighter space drill pattern using 89 mm diameter blast holes where the need for a high degree of selectivity is beneficial when attempting to minimize ore loss and dilution.

There are several rock types and three weathering types in the pit that differ in the required powder factor to effectively break the rock so as to optimize blasting efficiency and fragmentation. Table 16-2 below indicates the relative density of the weathering types, with an indicative burden spacing grid to be drilled in the relevant areas.

Table 16-2. Drill and Blast parameters

		Oxide		Transitional		Fresh		Fresh
	Unit	Waste	Ore	Waste	Ore	Waste	Ore	Ore_Select
Burden	m	4.70	3.9	4.7	3.9	3.4	2.7	2.4
Spacing	m	5.40	4.5	5.4	4.5	3.9	3.1	2.4
Bench height	m	6.00	6.0	6.0	6.0	6.0	6.0	6.0
Subdrill	m	0.80	0.8	0.8	0.8	0.8	0.8	0.8
Total drill depth	m	6.80	6.8	6.8	6.8	6.8	6.8	6.8
Volume blasted	m3 hole	152.28	105.3	152.3	105.3	79.6	50.2	34.6
Stemming height	m	2.50	2.5	2.5	2.5	2.5	2.5	2.5
Hole Diameter	mm	127	127	127	127	127	127	127
Hole Area	m2	0.013	0.013	0.013	0.013	0.013	0.013	0.013
Stemming per hole	t/hole	0.06	0.06	0.06	0.06	0.06	0.06	0.06
Explosive density	t/m3	1.15	1.15	1.15	1.15	1.15	1.15	1.15
Emulsion consumption per hole	kg/hole	62.64	62.64	62.64	62.64	62.64	62.64	62.64
Powder Factor	kg/m3	0.41	0.59	0.41	0.59	0.79	1.25	1.81

Design quality and consistency will be managed using a design approval document detailing the design parameters and expected blast outcomes. The document will be reviewed by the drill and blast, mine planning, geology and geotechnical sections and will be signed-off before being implemented. An in-field quality control process will ensure that the design is executed to a high standard and that the metrics associated with each blast, i.e. hole depth, charge mass and stemming length, are recorded for each hole so that a comparison with the design can be made and blast designs then optimised.

SROL intends to utilise various technologies to further minimise ore loss and dilution such as but not limited to Blast Movement Technologies (BMT) which is now part of Hexagon. The ability to accurately track blast movement is a huge benefit for mines striving to be smarter and more sustainable. Blasting is a highly variable process and movement of the ore during blasting can cost mines millions of dollars in lost revenue per year from ore loss. BMT's solution provides customers with accurate blast information that is used to recover all of a mine's resources, allowing the valuable ore to be sent to the mill, avoiding dilution and misclassification (Hexagon).

16.2.2.1 Explosives

The explosives are emulsion manufactured onsite. The emulsion plant is located near the TMF located to the west of the mine. The explosives consumables include boosters (400 g), nonel det (10 m), surface delays (8 m to 500 ms), emulsion, presplit packaged emulsion, presplit det chord and stemming. The stemming will be supplied from drill cutting and crushed & screened material down to 10 mm.

16.2.2.2 Magazine

The magazine facility is located to the north of the Segilola deposit and is designed to store the detonators and high explosives in a regulatory compliant structure, with required security measures.

16.2.2.3 Bulk Explosives Storage

Raw materials for bulk explosive manufacturing will be stored in a shed at the emulsion plant. The bulk explosives (ammonium nitrate emulsion) are stored in silos at the emulsion plant.

16.2.3 Loading and Hauling

Ore and waste are being excavated by backhoe excavators in discrete flitches, each nominally of 3 m height. Flitch heights can be varied (increased) in areas of known waste to effectively utilise equipment improving productivity and minimising costs.

The mine utilises 85 t and 45 t excavators, with 5.6 m^3 and 2 m^3 capacity buckets, respectively. The 85 t excavators fill the 60 t rigid body trucks with 6 passes, whereas the 45 t excavators fill smaller 32t trucks with 9 passes.

The large Front-end loaders can be utilised when speed and mobility is required, as an example, directly after a blast while excavators are walked back to the working faces. FELs are also employed as and when required to sustain ore feed.

An effective utilization for excavators is planned at 52% (70% availability and 75% utilization). Track dozers ensure loading areas are kept level and free of boulders and large rocks. The haul fleet is comprised of 60 t rigid haul trucks with effective capacity of 55 t per load. An effective utilization for haul trucks is planned at 52% (70% availability and 75% utilization).

16.2.4 Primary Mining Equipment

16.2.4.1 Drill & Blast Fleet

According to the material movement targets, a requirement for 5 units of PowerROC D45 drill rigs. The fleet will be further complimented by the various emulsion delivery equipment.

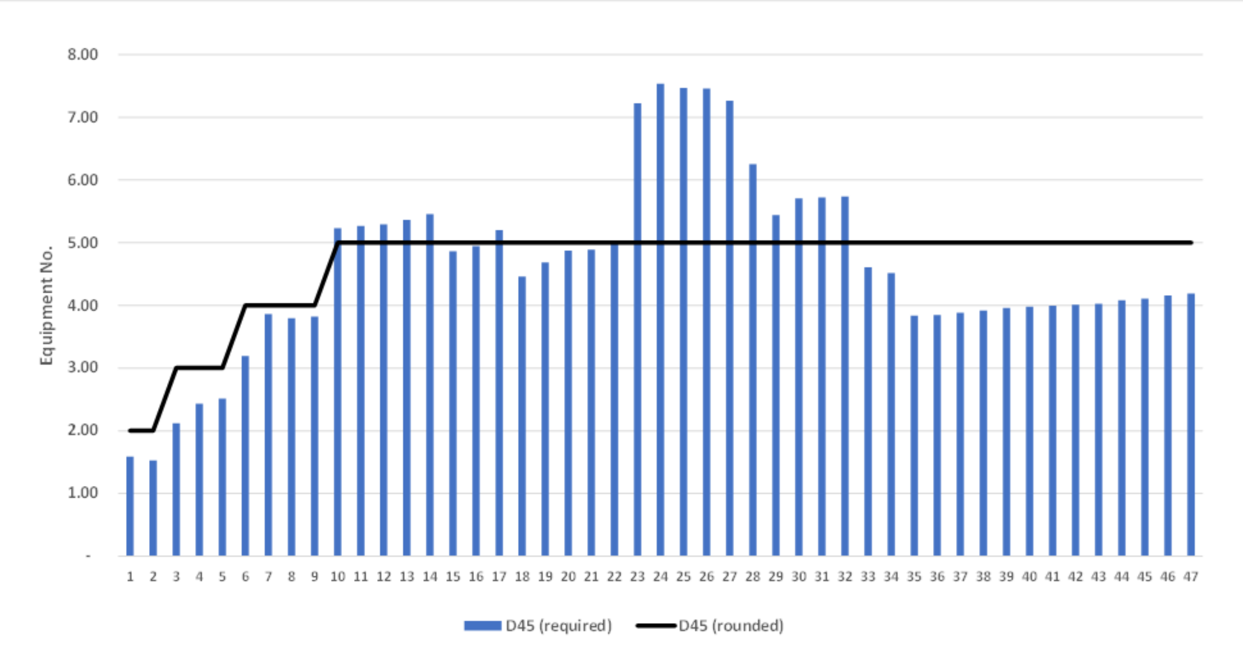


Figure 16-4. Poweroc D55 Requirement

16.2.4.2 Waste Production Fleet

The waste mining fleet will consist of 8 units of the EC950 Volvo 5.6 m^3 excavator. The larger excavators are allocated to the waste predominately and will load Sino-Hydro's fleet of 60 t rigid body trucks (PX90AT) manufactured by Peng Xiang.

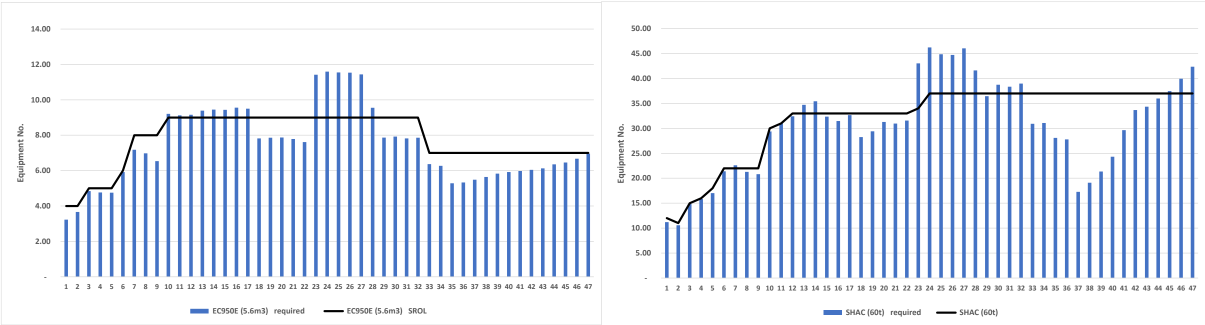


Figure 16-5. EC950 Requirement

16.2.4.3 Ore Production Fleet

The ore mining fleet will consist of 8 units of the EX480 Volvo 2 m^3 excavator. The smaller excavators are allocated to the ore predominately and will be used to load Sino-Hydro's fleet of 32 t rigid body trucks (PX08AT) manufactured by Peng Xiang.

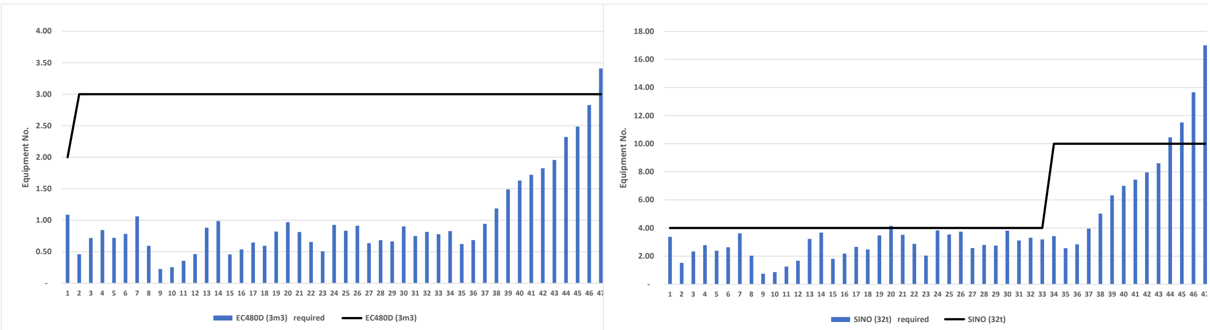


Figure 16-6. EX480 Requirement

16.2.5 Ancillary and Support Equipment

The ancillary and support fleets are provided by SINIC and their associated sub-contractors. In particular, the load and haul fleet is provided by SINIC. Items consist of cranes, dozers, graders, loaders, fuel trucks, water trucks, and light vehicles. A list of the maximum required units per item are summarised below.

Table 16-3. Ancillary and Support Fleet

ANCILLARY & SUPPORT FLEET:	MAX NO.
BCRH-15	2
Aux-D&B	1
Emul-Plant	1
Loader (4m3 - 3.4m height) XCMGLW800KN	2
242KW， XCMG GR3005	2
320HP XCMG TY320	5
420HP XCMG TY410	2
PX65MT-50t	4
3T crane	1
Volvo -sawing machine/crushing head	2
M-SFW6130	12
30t XCMG XS303S	2
5T crane	1
Loader (4m3 - 3.4m height) XCMGLW800KN	2
Loader (4m3 - 4.4m height) XCMGLW800KN	1
Pickup	8
Command vehicle	2
50 seats Golden Dragon XML6122J13	2
Maintenance car Dongte ND1627/Dongte DTA5161	2
Fuelling vehicle-20t Dongte 2534	2
Head:20m Q:100m^3/h (PH=4-10), IH125-100-250+4P/11KW	2
50t Fire Engine, PX65MT	1
Truck mounted crane,8t	1
50t Crane XCMG KY50KA	1
Dongte DTA5042	1

16.3 MINE PLANNING CONSIDERATIONS

16.3.1 Pit Optimisation

Final pit shells were generated using Whittle based on the parameters listed in Table 16-4 below.

Table 16-4. Whittle Parameters

	Unit	Oxide	Transition	Fresh
Mining costs – waste:				
Direct Cost	$/t (waste)	2.27	2.57	3.02
Haulage Cost Per Vertical Metre	$/t per m vert	n/a	n/a	n/a
Total Mining Cost	$/t (waste)	2.27	2.57	3.02
Mining costs – ore:				
Technical Services (incl. Grade Control)	$/t (ore)	2.40	2.40	2.40
Direct Cost	$/t (ore)	1.67	2.21	3.02
Haulage Cost Per Vertical Metre	$/t per m vert	n/a	n/a	n/a
Total Mining Cost	$/t (ore)	4.07	4.61	5.42
Overhead costs:				
G&A	$/t (ore)	8.7		
Total Costs	$/t (ore)	8.7		
Processing costs:				
Direct Cost	$/t (ore)	12.76	14.46	17.01
Total Processing Costs	$/t (ore)	12.76	14.46	17.01
Operating factors:				
Production rate	ktpa (ore)	715		
Processing recovery	%	98.5	97.5	97.0
Mining dilution	%	*Calculated*		
Mining recovery	%	95		
Economic factors:				
Royalty (Vendor)	$/t (ore)	1.24		
Royalty (GoN)	$/t (ore)	1.56		
Freight refining charge	USD $oz Au	0.84		
Gold Price	USD $oz Au	1,650		
Geotechnical parameters:				
Total slope angle	Degrees	*Refer to Geotech Design Criterion*		

The geotechnical parameters are defined in Table 16-14.

The Whittle exercise did not apply a cut-off grade to the re-blocked model. The block size used in the Whittle optimisation were larger and therefore incorporated a higher level of edge dilution at approximately 18%. This demonstrates that further upside exists. The block-size used for the Whittle optimisation was 1.5 m x 3.0 m x 1.5 m (XYZ). The purpose of using larger blocks was simply to reduce the number of blocks being processed and therefore speed up the time required to run the analysis.

The unit rate applied to royalties were back calculated from the total royalties which is calculated as a percentage of gross revenue. Refer to Section 21 for further information.

Based on this, the ideal pit shell is pit shell No.9. However, operationally the preferred pit shell is No.6 on the basis that a lower volume of waste is required to be mined.

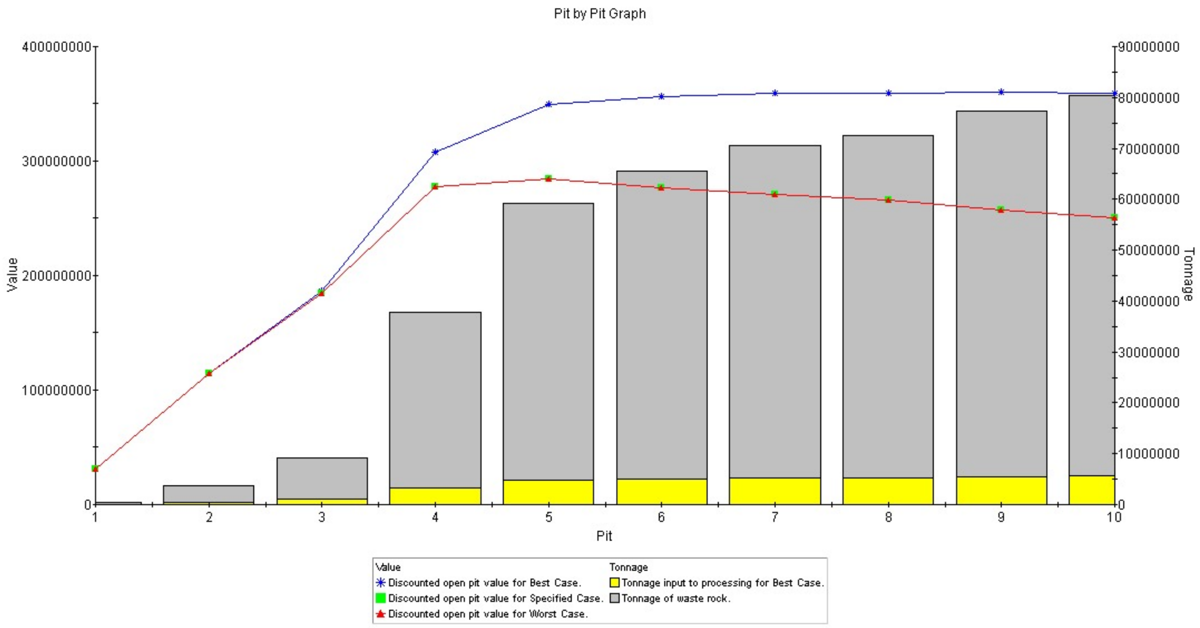


Figure 16-7. Whittle Optimisation NPV Analysis

The detailed pit design was between shell 6 and shell 7 in Table 16-5. On this basis it is reasonable to assume that 530,000 oz's Au is recoverable thereby generating approximately $790 m USD of cashflow during the project's life. The whittle optimisation is based on both indicated and inferred.

Table 16-5. Whittle Optimisation Shell Volumes

Pit	RF	Rock	Ore	Strip	Max Bench	Min Bench	Au_units	Grade	Ounces
		M t's	M t's	W:O	No.	No.	M g	g/t Au	K oz's Au
1	0.1	0.5	0.0	11.6	150	123	0.6	15.1	20
2	0.2	3.8	0.4	9.5	150	119	2.6	7.1	83
3	0.3	9.1	0.8	10.7	154	113	4.7	6.1	152
4	0.5	37.6	2.7	13.0	160	91	11.5	4.3	369
5	0.6	59.1	4.0	13.7	160	84	15.5	3.8	498
6	0.7	65.3	4.5	13.6	160	81	16.5	3.7	530
7	0.8	70.4	4.9	13.5	160	78	17.2	3.5	553
8	0.9	72.5	5.1	13.2	160	77	17.5	3.4	563
9	1.0	77.2	5.5	13.2	160	75	18.0	3.3	580
10	1.1	80.1	5.7	13.1	160	74	18.3	3.2	590

16.3.2 Mining Dilution

The proportion of the model amenable to open pit mining has edge dilution incorporated by re-blocking up one sub-block unit.

The overall mining dilution was estimated to be 12% (Indicated material). The proportion of the model amenable to open pit mining has edge dilution incorporated by re-blocking up one sub-block unit.

The tonnages and grades provided describe the sensitivity of the block model estimates to cut-off grade and dilution and should not be interpreted as Mineral Resources or Mineral Reserves.

Table 16-6. Dilution Summary

Resource	Grade	Resource (0.375x1.5x0.375 sub-block)			Dilution model (0.375 x 3.0 x 1.5 m)			Grade Bin dilution
Category	Bin	M t	g/t	K oz	M t	g/t	K oz	
Indicated	0.3 -> 2.0	1.24	1.22	48.8	1.65	1.12	59.6	133%
	2.0 -> 4.0	1.01	2.84	91.8	1.06	2.85	97.5	106%
	4.0 -> 8.0	0.94	5.62	170.3	0.95	5.60	171.1	101%
	> 8.0	0.48	14.24	221.4	0.46	13.85	205.6	95%
Sub total		3.67	4.50	532.2	4.13	4.02	533.9	112%
Inferred	0.3 -> 2.0	0.21	1.00	6.7	0.02	0.92	0.7	12%
	2.0 -> 4.0	0.00	3.08	0.1	0.00	3.04	0.2	233%
	4.0 -> 8.0	0	4.96	1.9	0.01	4.95	1.9	104%
	> 8.0	0.00	8.59	0.0	0.00	8.58	0.0	20%
Sub total		0.22	1.22	8.6	0.04	2.28	2.8	18%

Edge dilution (mining dilution) is applied by re-blocking the resource models small sub-blocks (0.375 x 1.5 x 0.375 m, XYZ) to re-block up to 0.375 x 3 x 1.5 m (XYZ) and averaging the grade weighted by density of each original subblocks. For blocks entirely within the mineralised boundary, the grade does not change. Blocks on the margin are diluted with waste blocks, and the grade will reduce the corresponding percentage (i.e. 3 mineralised blocks diluted with 1 waste block will dilute 25%).

Figure 16-8 shows the resource model near hole SGD115, the small sub-block on the eastern edge of the central structure is running 7.5 to 8 g/t, when re-blocked (right hand side) the diluted edges range between 2 and 6 g/t. The larger internal blocks maintain the original estimated grade. The mineralization goes from sharp contacts with clear waste between the structures to diffuse contacts dropping in grade reflective of material post blasting.

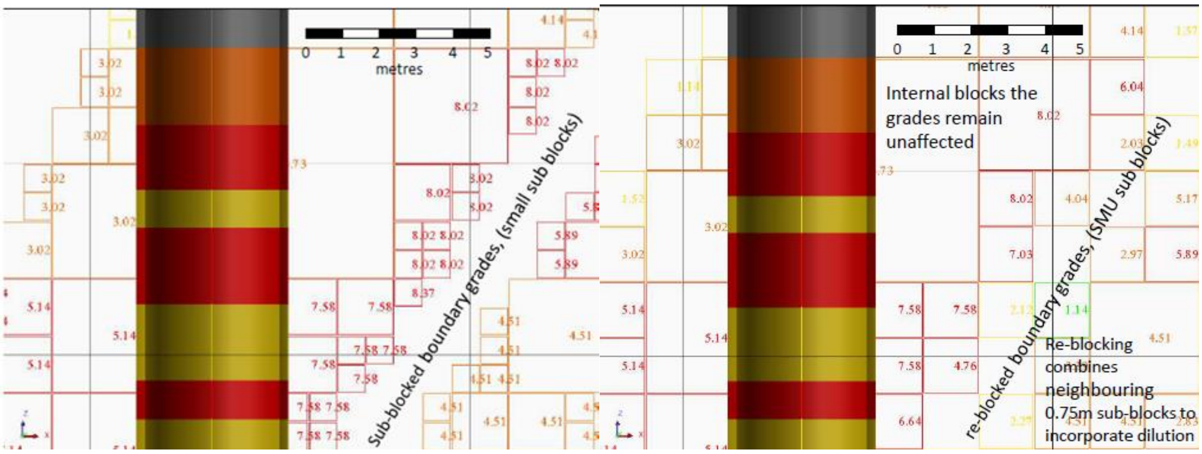


Figure 16-8. Detailed View of Model near SGD115 (11,0765mN)

Left: sub-block model, Right: dilutions model (re-blocked)

16.3.2.1 Skin Dilution (Orelogy)

The approach of re-blocking compares well with the previous Orelogy method of applying a skin of 0.5m on the lode interpretation. As depicted in Figure 16-9 below, re-blocking and skin-out are similar. There are various pros and cons to each approach.

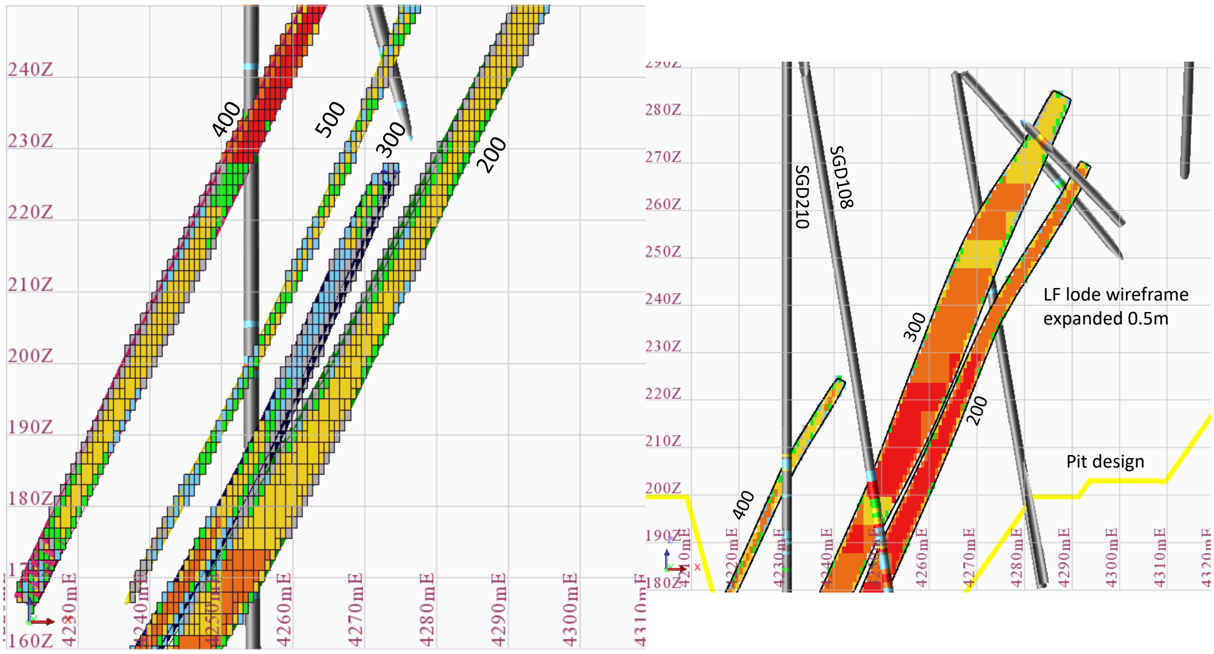


Figure 16-9. Skin Dilution Comparison

By applying the skin-out approach, Orelogy determined that the average dilution across within the original pit limits was 14%.

Table 16-7. Orelogy Dilution Estimate - Skin Method

Description / Source		Cut-off	Total Tonnes	Ore Tonnes	Au Grade	Contained Oz	Dilution	Ore Loss	Oz
		g/t	Mt	Mt	g/t	MOz	%	%	%
D	DFS Dilution factors applied	0.77	52.02	3.03	4.16	12.60	110%	95%	
E	Skin in - Accept Ore loss	0.7	52.02	2.31	4.48	10.37	102%	78%	82%
F	Skin Mix	0.7	52.02	2.74	4.36	11.96	105%	90%	95%
G	Skin out - Accept Dilution	0.7	52.02	3.14	4.00	12.55	114%	95%	100%

16.3.3 Grade Tonnage Curves – Dilution

Grade tonnage curves show the lowering of grade and the increase in tonnes as more dilution is incorporated (Figure 16-10). Sensitivity to dilution re-blocking, grade tonnage curves (above 144 m RL).

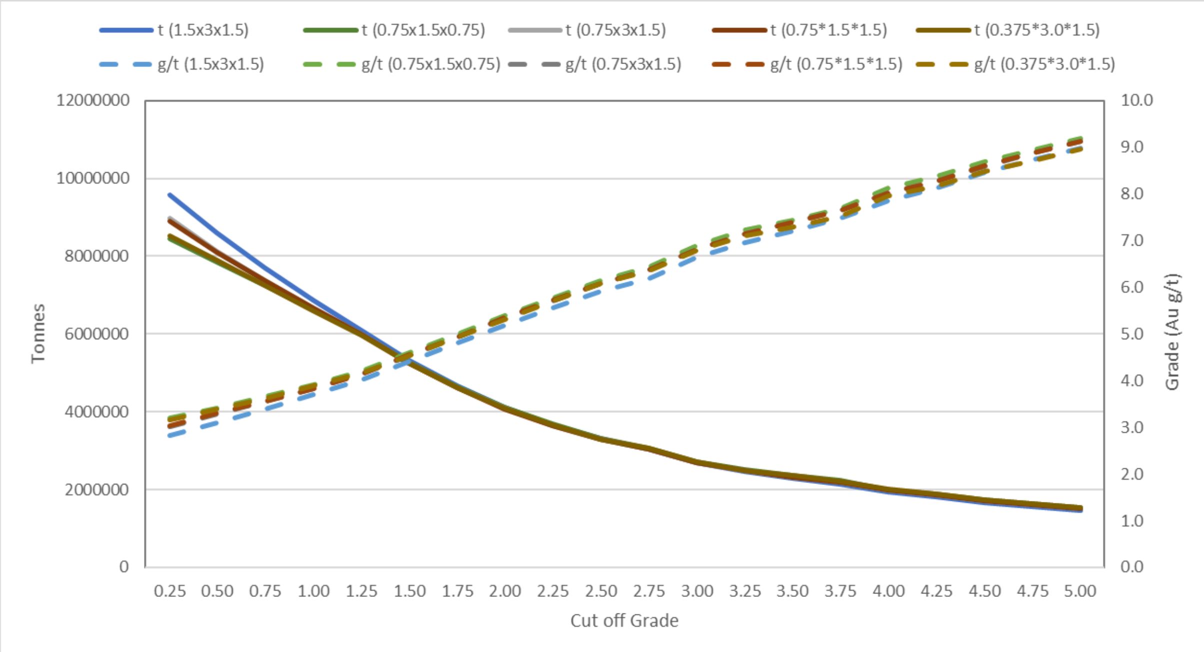


Figure 16-10. Grade Tonnage Curves

16.3.4 Discussion and Analysis - Dilution:

The minimum Block size in the x-axis was 0.375 m. The excavators allocated to the ore are 2 m^3 buckets top loading in broad daylight under the guidance of ore-spotters whilst being enabled by other tools at our disposal. For the sake of conservatism, increasing this further by a factor of 1x would result in a block of 0.75 m which we deemed as excessive. To validate this, SROL compared this approach with Orelogy's approach of applying an outer skin to the lode interpretations of 0.5 m in the x-axis (the "outer skin method"). The global dilution was 12% in contrast to Orelogy's 14%. Whilst slightly lower, SROL consider this to be achievable. Particularly with the right amount of diligence and due care afforded to the GC programme coupled with the D&B and L&H disciplines. Specific activities to assist SROL with our ability to control and minimise dilution include but are not limited to: ore mining to occur on day-shift, use of ore-spotters coupled with the implementation of Blast Movement Technologies (BMT) as required.

16.3.5 Mining Recovery

A 97% mining recovery factor was applied to account for the quantity of ore that is lost due to spillage and/or re-handling and to account for any unforeseen additional ore losses (ore hauled to the waste dump, etc.).

Additionally, some mining loss occurred and has been incorporated as part of the re-blocking exercise taken to estimate the dilution.

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

16.3.6 Cut-off Grade Economic Parameters

When determining cut-off grade, engineers will apply an economic cut-off grade formula to determine this. The formula for cut-off grade is as follows:

COG = (Mining Dilution x Processing Cost) / (Processing Recovery x (Sell Price - Sell Costs))

The cut-off grade for the parameters used in the formulation of the Mineral Reserve was calculated as 0.37 g/t Au. Previous estimates used were 0.77 g/t Au and 0.70 g/t Au as per the original DFS in 2019 and Orelogy work completed in 2020.

Fundamentally, this approach is sound. However, the main shortfall with this method is that the modelling results in the loss of metal and does not account for operational realities such as blending, selective mining and ability to sort the ore in the field according to these parameters.

Henceforth, a different approach becomes necessary once the pit limits and design have been determined and a finite amount of contained metal is thus determined. For this, SROL have adopted the “effective optimum cut-off grade” as guided by the principals set forth in Kenneth Lane’s “The Economic Definition of Ore”. Whereby, the cut-off grade is determined when a maximum economic value is achieved thereby ensuring an optimal utilisation of the resource. The chart below (Figure 16-11) is based on shell 6 from the whittle exercise.

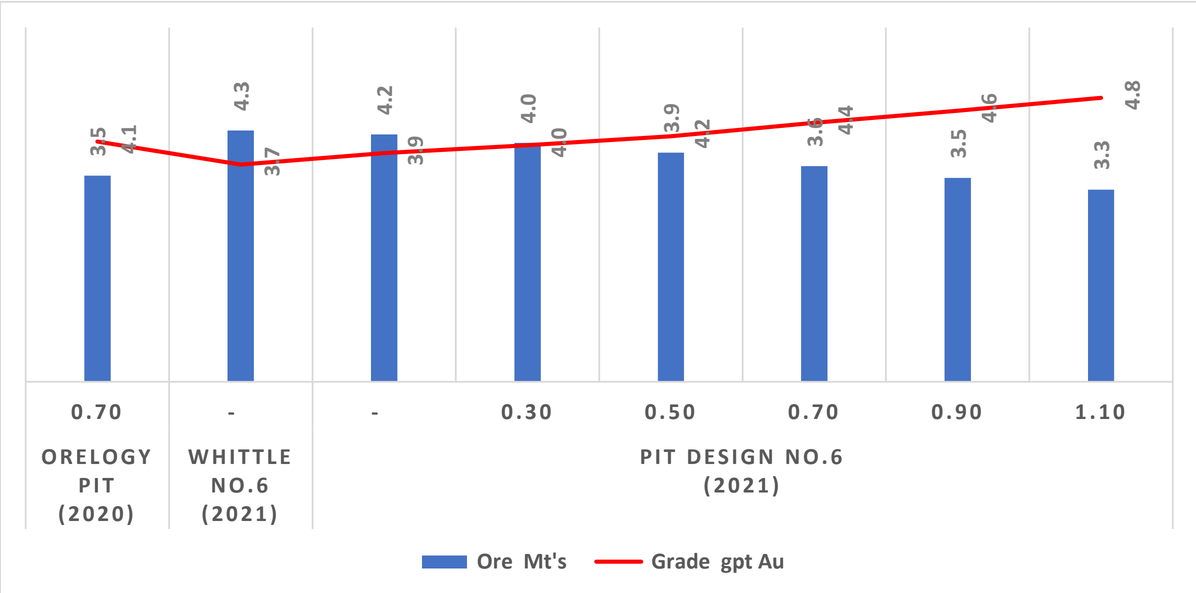


Figure 16-11. Cut-Off Grade Tonnes vs. Grade

SROL determined that 0.3 g/t Au was the preferred cut-off grade. This can also be supported using the Economic Cut-off Grade (ECOG) approach.

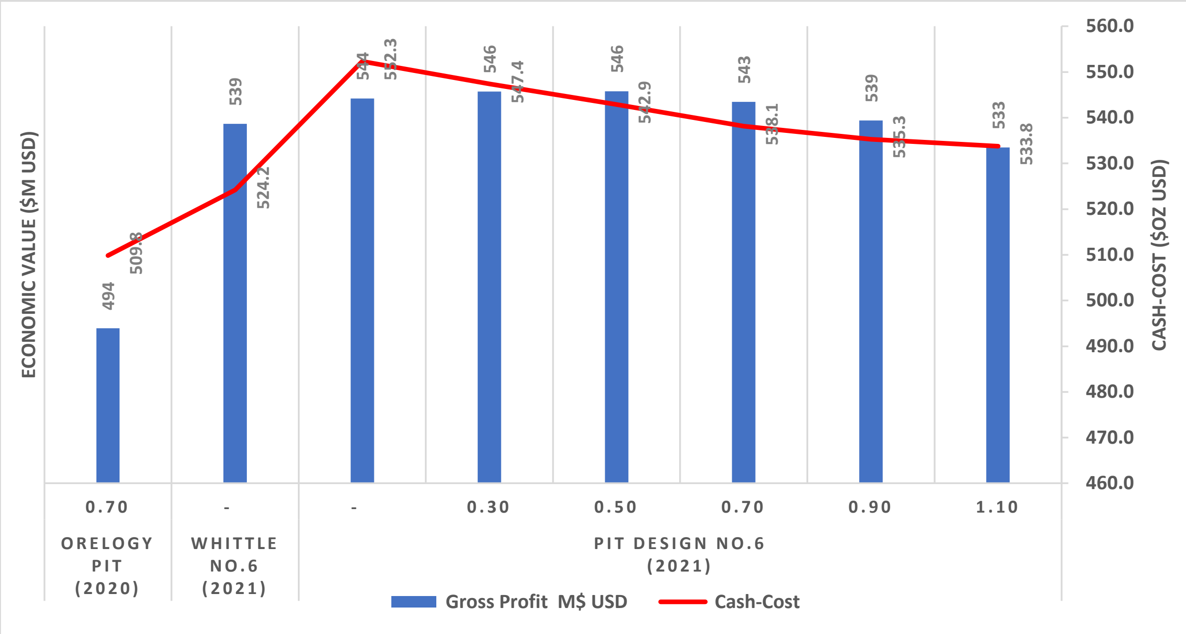


Figure 16-12. Economic Value by Cut-Off Grade

16.4 WHITTLE SHELL AND PIT DESIGN

Based on the updated detailed design (blue), the design conforms well with the selected Whittle Shell No.6 (white).

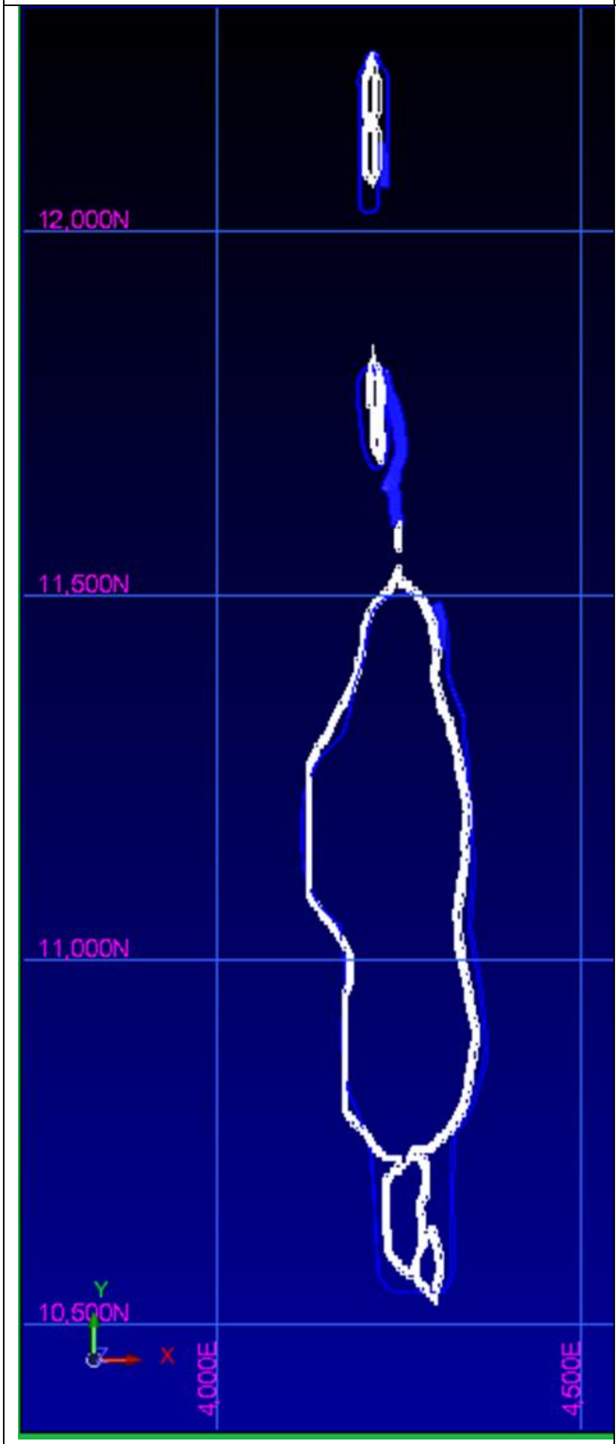
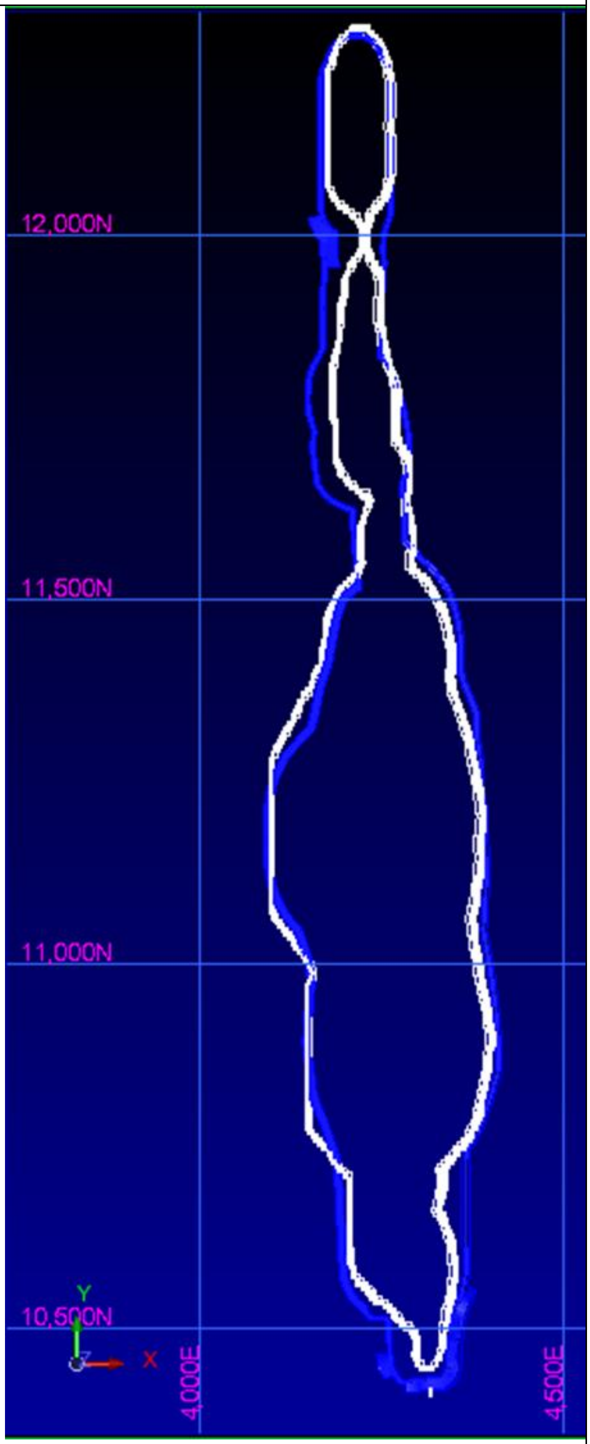
Whittle Shell No.6 and Pit Design (252mRL)	Whittle Shell No.6 and Pit Design (276mRL)
	

Figure 16-13. Whittle Shell No.6 and Pit Designs (Local Grid)

16.4.1 Discussion and Analysis

The Whittle analysis used a base-case of $1,650 oz @ 1x revenue factor (RF). SROL originally selected Pit6 at an RF of 0.7 then gravitated towards a whittle shell that sits somewhere between shell 8 and 9 with an RF of between 0.8 and 0.9, respectively. This equates to a shell selected at a gold price between $1,320 oz and $1,440 oz Au. Although, various consensus forecasts published contemporaneously with SROL's Whittle exercise seemed to suggest that $1,700 to $1,800 was a more reasonable long-term (2025yr) forecast.

16.4.2 Mine Design Parameters

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

Table 16-8. Mine Design

Section	Unit	DFS	Orelogy	SROL
Ramp width	Metres	22	16	16
Ramp gradient		1:9	1:10 (1:8 below 168mRL)	1:10
Minimum mining width	Metres	10	10	10
Geotechnical Parameters	Degrees	Refer to Chapter 16.4.4.5	Refer to Chapter 16.4.4.5	Refer to Chapter 16.4.4.6
Length	Metres	1,650	1,900	1,885
Width	Metres	140 to 430	140 to 430	140 to 430
Depth	Metres	55 to 210	220	230
Area	Ha	43	43	43

Ramp entry/exit points were designed primarily to limit the haulage distance to the waste dump at each stage of the pit. 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.

16.4.3 Pit Design Comparison – Update

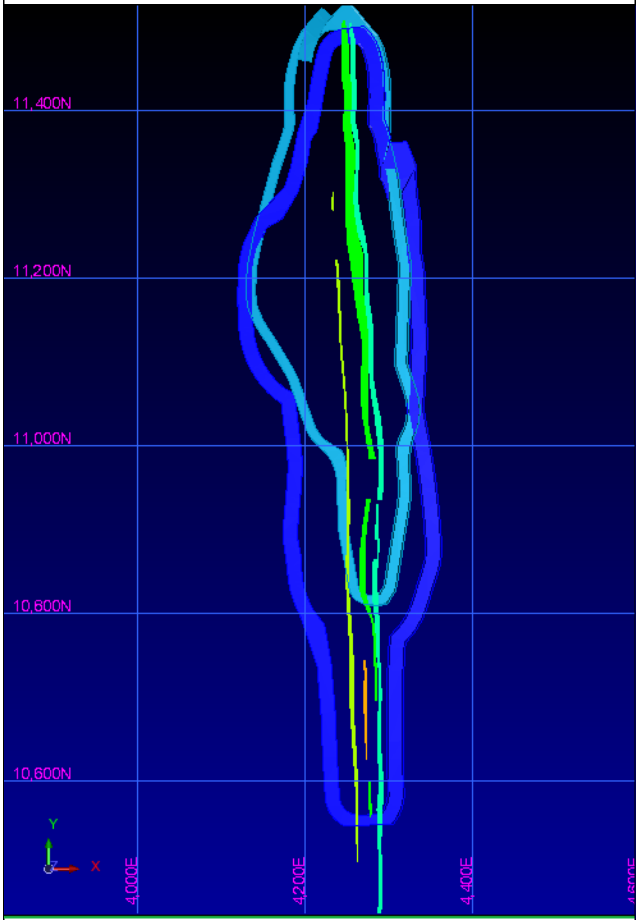
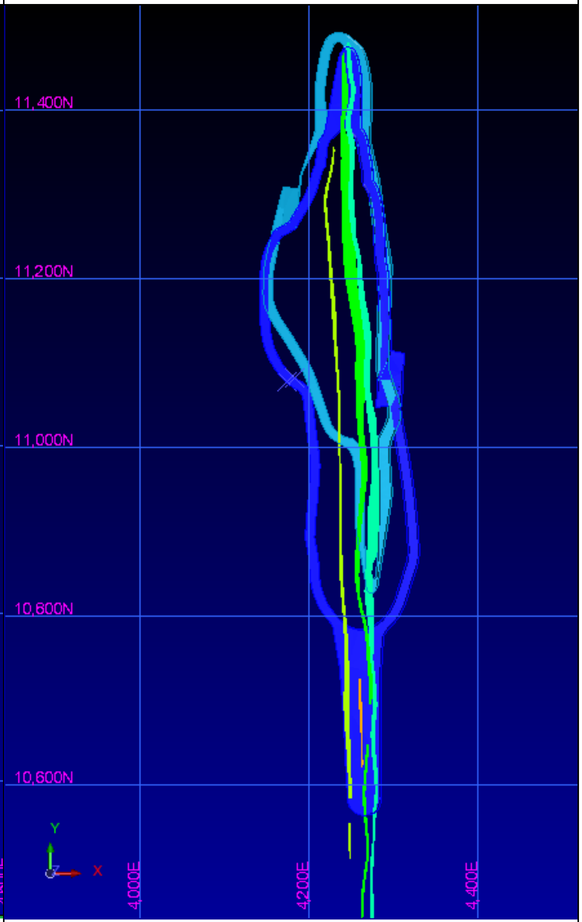
240mRL - 2020 pit vs. 2021 pit	216mRL - 2020 pit vs. 2021 pit
	

Figure 16-14. 2020 Pit vs. 2021 Pit Comparison (local Grid)

Note: The light-blue and dark-blue pits are from 2020 and 2021 (SROL reserve update) respectively.

The primary difference between the two pit designs was the deeper extension to the southern end of the Segilola deposit. This is because of additional indicated material being defined in this area and therefore at a higher level of resource confidence to convert into a reserve.

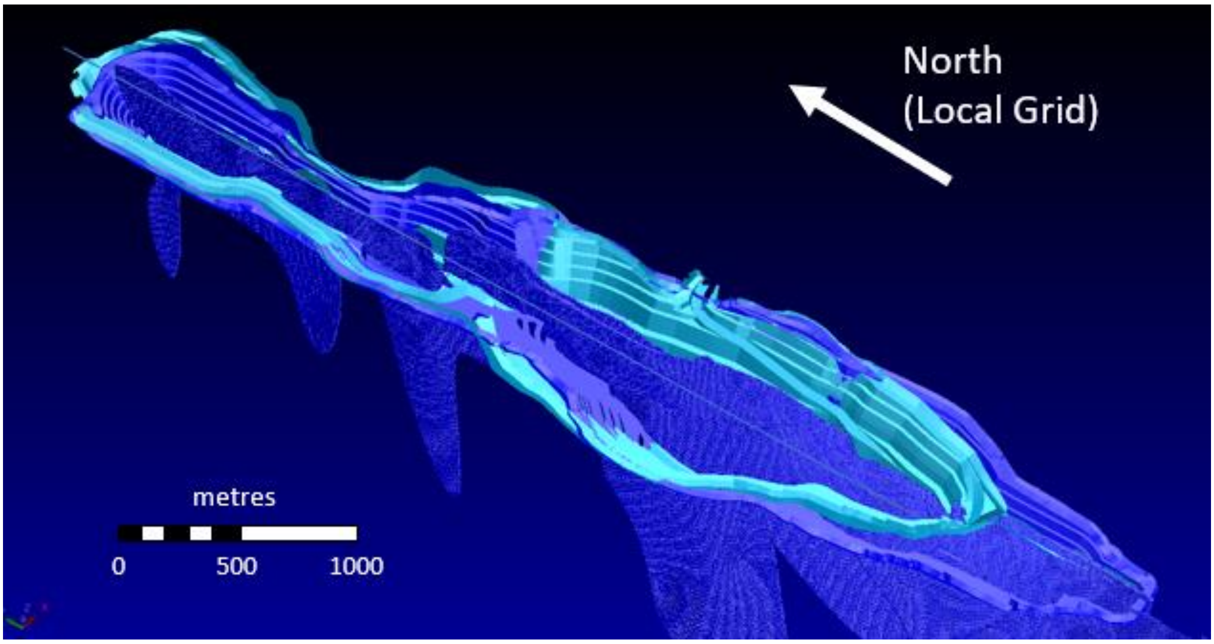


Figure 16-15. Isometric View of Original Pit versus Updated Pit

Note: The light-blue and dark-blue pits are from 2020 and 2021 (SROL reserve update) respectively.

16.4.4 Mine Geotechnical Engineering

16.4.4.1 Rock Strength (UCS & RQD)

The following geotechnical information (UCS and RQD) was used in the pit design process. The information may be important background information for estimating the costs of mining activities (drilling penetration rate, wear rates and the availability of road materials).

Results of the geotechnical rock strength (UCS) assessments are summarised in Table 16-9 and Table 16-11. The test results for the GDM1, GDM2 and GDS units were suspiciously low and for these units the rock strength was assigned as per George Orr's assessment. The strengths of the Biotite Schist and Calc Silica units shows the test work results.

In addition, metallurgical test work (ore zone only) indicates UCS values ranging from 71.7 MPa to 190.6 MPa.

Table 16-9. Rock Strength (UCS) by Lithology

Lithology	Average UCS (MPa)	Comments
GDM1	100 to 150	Values obtained from George Orr analysis
GDM2	100 to 150	Values obtained from George Orr analysis
GDS	100 to 150	Values obtained from George Orr analysis
Biotite Schist (silicate)	54	Strong biotitic foliation, altered
Biotite Schist (unaltered)	165	Only 2 samples, siliceous, violent failure
Calc-Silicate	130	Only 1 sample, FG, greenish, siliceous

Rock-quality designation (RQD) provides information about the quality of the rock mass as indicated in Table 16-10. RQD is a rough measure of the degree of jointing or fracture in a rock mass, measured as a percentage of the drill core in lengths of 10 cm or more.

Table 16-10. Rock Mass and RQD Values

Rock Mass Classification	RQD Range (%)
Very Poor	0-25
Poor	25-50
Fair	50-75
Good	75-90
Excellent	90-100

RQD values are an important input in the pit design slope criteria. From a general mining perspective, zones with high RQD values will need higher powder factors to achieve the desired fragmentation and zones with poor RQD values may have problems with blasthole stability and groundwater inflows.

The RQD assessments for the Segilola Gold Project are graphically displayed in Figure 16-16 and Figure 16-17. The figures demonstrate that the hanging walls (west) have better RQDs than the footwalls (east).

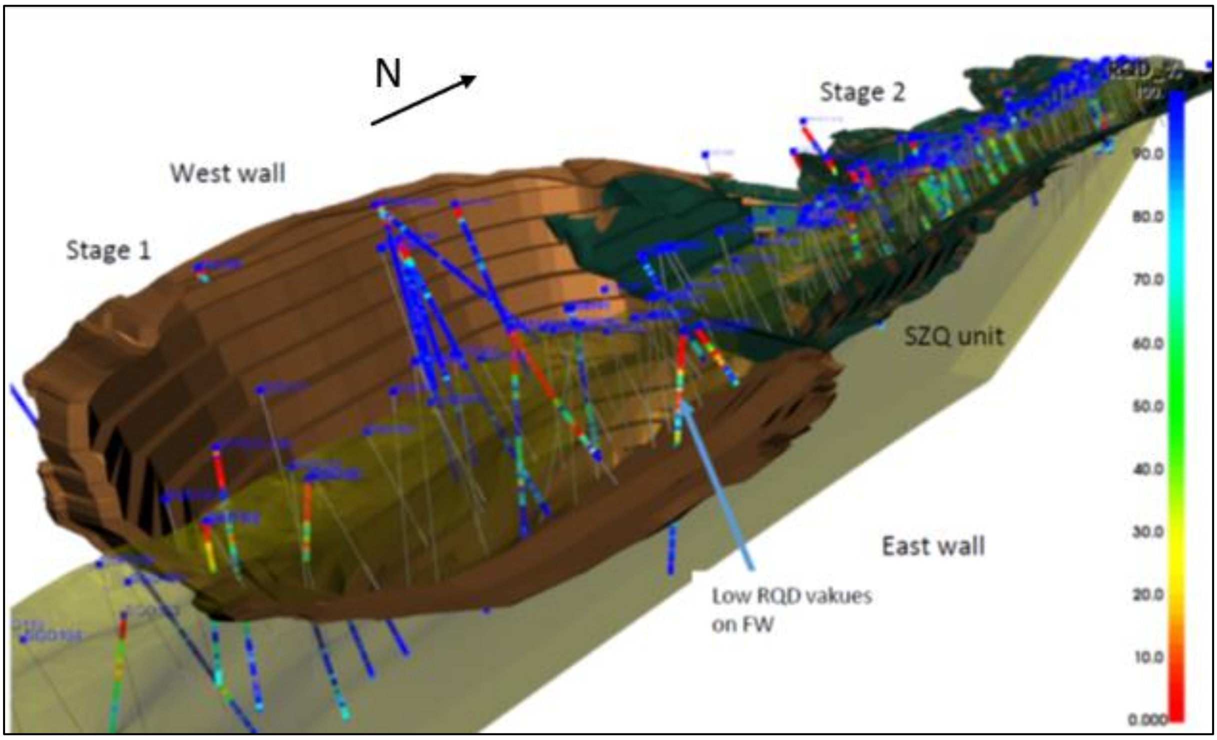


Figure 16-16. Pit Isometric Looking North West - RQD Values

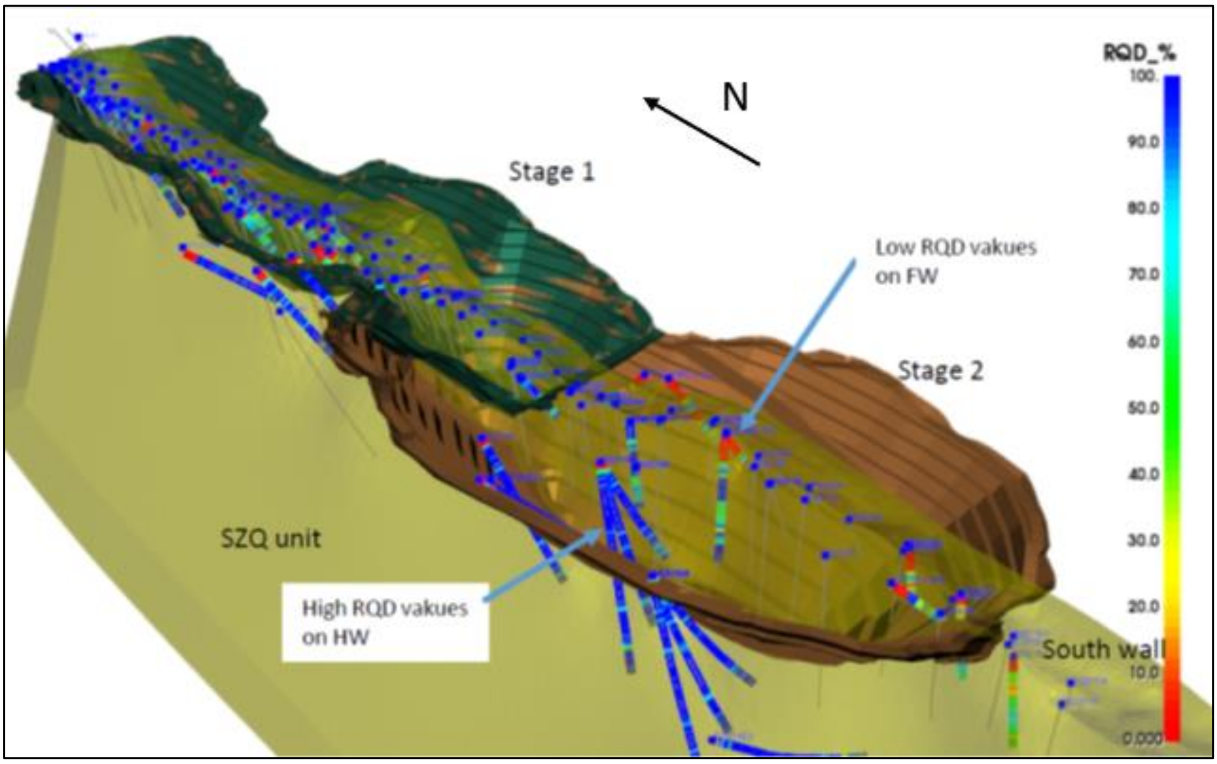


Figure 16-17. Pit Isometric Looking North East - RQD Values

16.4.4.2 Intact Rock Strength (UCS)

All UCS samples from the 2018 drill holes were selected within the east or footwall unit of calc-silicate and biotite schist. In addition to the UCS samples, samples of the structures were selected for further direct shear testing. Results are given in Table 16-11 and Table 16-12.

Table 16-11. 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
BSh	UCS1, 4, 5, 6, 7	GTFS17-005, 006	54	Strongly biotitic foliation, altered
BSh	UCS2, 3	GTFS17-005, 006	165	Siliceous, violent failure – only 2 samples
Calc-Sil	UCS8	GTFS17-011	130	FG, greenish, siliceous, single sample

Note: * These UCS tests are not considered representative as there may have been issues with the testing lab and sample preparation which resulted in lower-than-expected values for UCS at failure.

Table 16-12. Summary of DS Testing Results

Lithology	Sample Nos.	Borehole	Defect Sets	Peak ϕ, c (°, kPa)	Residual ϕ, c (°, kPa)	Comments
GDM1	SDS-1, 2	GTFS17-002, 005	5, 3	33.5, 15.4	31.5, 33.5	2 defect, average values
PEG	SDS-3	GTFS17-005	2, 4	34, 0	31.9, 0	Single, rough, clean defect
BSh	SDS-8, 9, 10	GTFS17-006	2, 4, 3	32.3, 16.6	29.4, 27.0	Siliceous biotite schist
DGS	SDS-11, 12	GTFS17-011	1, 4	31.6, 114.6	26.4, 167.1	Hanging wall lode altered granodiorite gneiss
BSc	SDS-4, 5, 6, 7	GTFS17-005	1, 3	27.1, 9.25	26.4, 20.0	Unaltered biotite schist

16.4.4.3 Seismicity

Segilola is located within a region of Nigeria judged to be at very low risk from future natural seismic events (earthquakes) taking place over the life of the proposed extended mining. This means that there is less than a 2% chance of potentially damaging earthquake at the Project in the next 50 years. Earthquake-induced ground accelerations of this magnitude (if occurring) would not be expected to have a significant influence on future pit wall stability performance (GFDRR, 2018).

16.4.4.4 Geotechnical Parameters

As part of the DFS, published in March-2019, Peter O'Bryan & Associates (POB&A) was commissioned to conduct a review and analysis of wall design parameters for the Project. The mine design and optimisation were based on this geotechnical assessment.

Aside from previous study data, the following was used for this work:

- Structural and geotechnical logging from thirteen oriented HQ diamond cored resource definition drill holes; nine geotechnical holes from 2018 and five resource definition drill holes from 2017.
- Thirteen additional non-oriented HQ resource definition and metallurgy test holes which were summary logged for basic recovery, RQD, weathering and index strength, and photographed prior to sampling, were also assessed to augment the oriented core information.
- A site visit to conduct summary and check-logging, select samples for laboratory testing, and review conditions in the field.
- Laboratory strength test results.
- Historical drill logs.

The main structural fabric of the sequence has foliation/schistosity dipping towards the west between approximately 50° and 75°; a shallow dipping set; and two steep to sub-vertical sets; one dipping north-northwest to south-southeast and the other dipping north-northeast to southwest.

The rock mass is more massive and less foliated further west in the sequence, away from the ore zone. The west wall rock mass is rated as being of Good rock mass quality.

Closer to the ore zone, increasing levels of alteration/metamorphism have resulted in increased development of schistose texture and associated foliation, and increasing silicification, indicating Fair to Poor ground conditions in the lower west and east walls of the southern pit and within the southern and northern end walls of the pit.

The footwall biotite schist unit is more intensely altered and fractured, with intercalated zones of higher siliceous content and weaker schist. This zone is considered to be of Fair to Poor quality in the more siliceous bands, and Poor to Very Poor in the less siliceous bands.

Cross-cutting, approximately east-west striking defects were identified. There is a lower rock mass quality in the eastern biotite schist unit, with more fracturing and a deeper weathered profile.

Overall, the rock mass is massive, with significant lengths of intact core sticks in the hanging wall rock, with increasing levels of metamorphism/foliation/gneissic texture, closer to the lodes and into the footwall, with a section of weaker biotite schist and residual soil indicated at the crest of the western wall, which is not, however, expected to adversely impact the overall wall design.

The soil/residual soil profile ranged from approximately 2 m to 20 m in depth downhole. There is a zone along the central axis of the pit, related to a valley in the local terrain, where depths of soil exceed 25 m. Along the crest of the proposed Stage 2 pit, the weathering depth tends to be less than 10 m and more consistently in the 2 m to 5 m range.

The transition between the soil and fresh rock is sharp in the west wall, with approximately 2 m to 10 m zones of moderately to slightly weathered rock, characterised by iron staining on defect surfaces. This slightly weathered zone with iron staining on defect surfaces is deeper in the east wall, with depths of up to 50 m indicating the presence, current or historically, of water flow through the rock mass and structures. This may have implications for the slope designs.

Voids related to historical underground mining present a potential hazard to operational safety and pit floor and wall stability. Observations indicate or suggest that relatively small voids may be intersected by future open pit mining and a programme of probe drilling will be required to locate and define these openings.

16.4.4.5 DFS Pit Design (March 2019)

In the DFS published in March 2019, the 'Base case' wall design parameters were based on an assessment of likely wall failure modes governed by geological structures (shears, foliation, and joints), and general experience. Empirical, limit equilibrium, and kinematic analysis tools were applied to assess potential instability for the proposed design.

Limit equilibrium analysis indicated both the west and east walls of the southern pit were considered stable, with a factor of safety greater than 1.3. Mohr-Coulomb constitutive models were applied to the massive biotite gneiss and granodiorite, with a linear anisotropic constitutive model adopted for the foliated, mineralised SZQ and footwall biotite schist zones. Kinematic and sensitivity analysis was conducted on the poles to all defect features logged.

Table 16-13. Pit Design Criteria (10m Bench Height, DFS Mar2019)

	East Wall (Footwall)	West Wall (Hanging wall)	North & South End Walls
***Surface**	310-295 mRL	375-365 mRL	317-330 mRL
Upper wall in Weathered Rock/Residual Soil – depth	5-15m	2-9m	2 – 15m
Face angle		35°	
Face height		10m	
Berm width		10m pit crest in rock	
		Upper Wall – Fresh/Transition Zone	
Elevation	***295 (or soil base) to 270mRL***	***365 (or soil base) to 355mRL***	***295 (or soil base) to 270mRL***
Face angle	50°	60°	60°
Face height	10m	10m	10m
Berm width	5m	5m	5m
IRA (exclusive of ramps)	37°	52°	43°
	270mRL to 195 mRL	***355 to 195 mRL****	***270 to 195 mRL***
Face angle	55°	75°	55°
Face height	20m	20m	20m
Berm width	8m	8m*	8m
*295 & 235mRL Berms	-	12	-
IRA (exclusive of ramps)	41°	52°	42°
Overall Wall angles (pit crest to toe exclusive of ramps)	42°	54°	

Note*:* * This is the inferred pit crest, locally will need to be adjusted for weathering and topography.

Assumes: Central pit floor at 220mRL, Northern pit floor at 230mRL, Southern Pit floor at 200mRL – design parameters can be extended at depth but must be reviewed for geotechnical implications of a deeper pit. IRA – Inter-Ramp Angle

16.4.4.6 Revised Pit Design (March 2021)

The original design for the DFS was based upon the assumption of a pit floor RL of 200mRL for the southern pit and was modified to 156 mRL in the October 2019 Design (607_THOR_PITDES_V2_STG2_191004 clipped.dxf). The central and northern pits are relatively shallow at ~240mRL (<100m depth).

Chris Langille (Northwind Enterprises Pty Ltd) was commissioned to review the revised pit design and additional geotechnical drill holes that were drilled as per his recommendation. The review occurred on the 7th of April 2021.

16.4.4.7 Additional Diamond Drill holes - Data Review

For the east wall, original diamond drilling was not extended sufficiently to confirm the design analysis. Three additional oriented diamond cored holes were drilled and logged in 2019, one each on the east wall of the southern, central, and northern pits (GTFS17-014, 015 and 016). The core photos and structural data have been reviewed. The new information outside the pit shell confirmed the assumptions made about rock mass quality, structure, fabric, and continuity that were based upon data from within the pit shell.

In addition, core photos and geological logs of additional resource holes (Figure 16-18) were assessed qualitatively to identify any significant anomalous structures or rock-mass conditions. A further, more robust analysis of the data will be required to assess any future proposals for an east wall cutback and pit deepening.

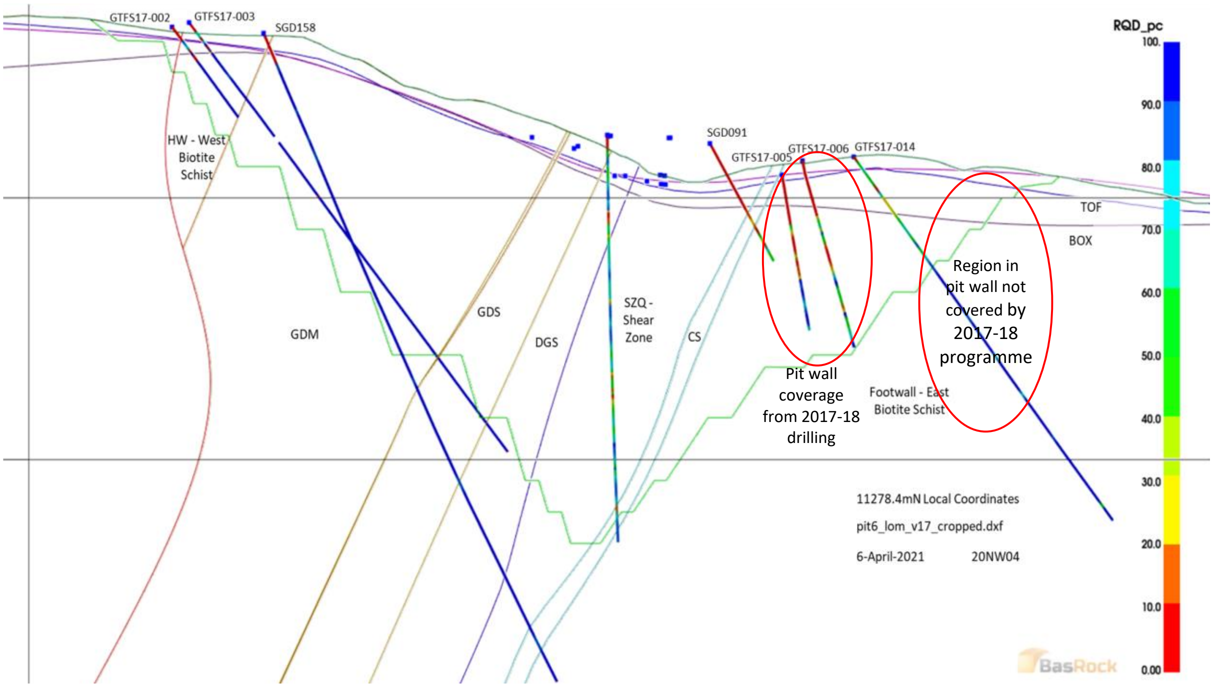
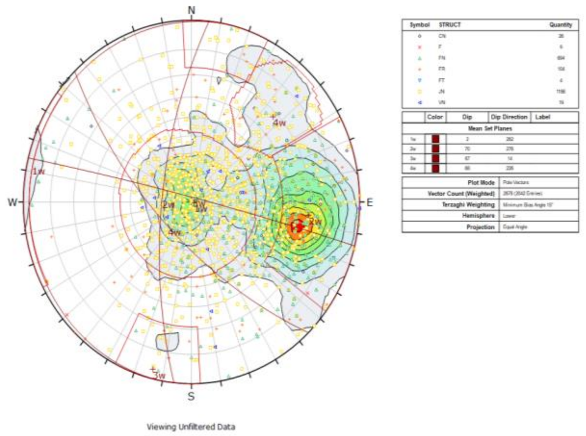


Figure 16-18. Drill Coverage in the East Wall of the Southern Pit

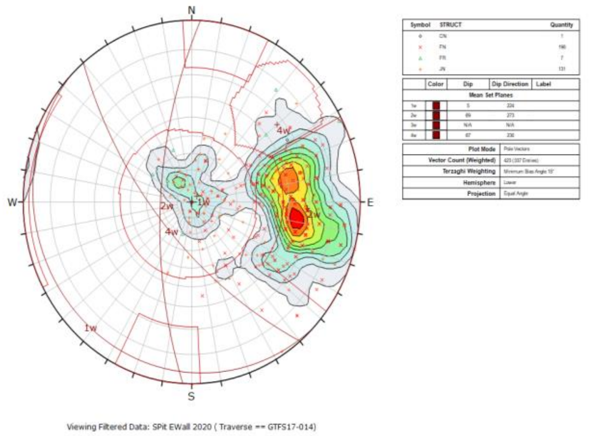
Structural data was compared for each of the geotechnical holes on the east wall of Segilola pit design (pit6_lom_v17_clipped.dtm) in order to validate design assumptions made during the initial design. Stereonet plots of the data are shown in Figure 6-19, A to D. Figure 6-19 A shows a comparison of all structural data for the east wall holes, Figure 6-19 B is the earlier 2018 data, the recent GTFS17-014 is presented separately in Figure 6-19 C and the combined east wall data set is shown in Figure 6-19 D. Similar structural features are evident in each Stereonet plot, including the moderately steep west dipping schistosity/foliation, sub-parallel to the east wall and a steep west dipping set, sub-parallel to the mineralised shear zone hosting the ore. The core photos of hole GTFS17-014 near the proposed east wall of the pit are provided in Figure 16-20.

Segilola Gold Operations - East Wall Structure Data

*Structure orientations are with respect to UTM grid north, approx. 15 deg west of Mine Grid North



A



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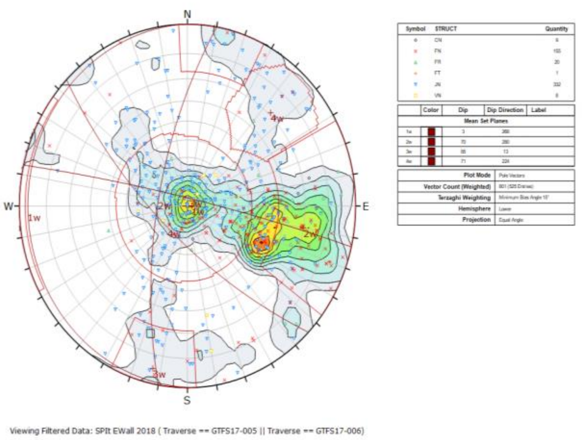

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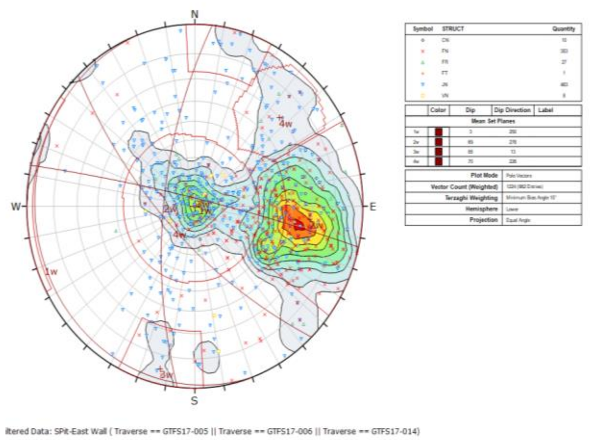


D

Figure 16-19. Stereonet plots -East Wall

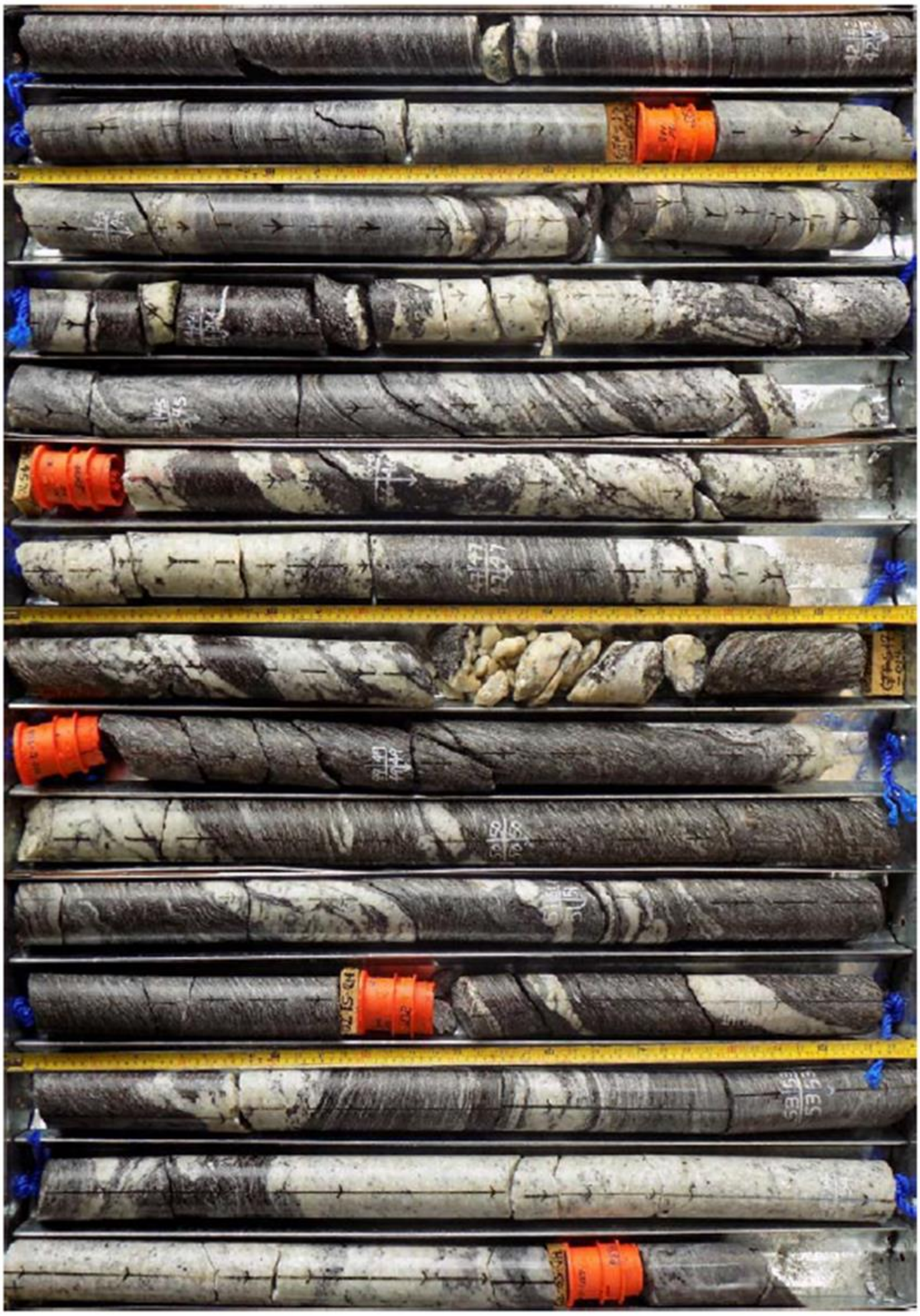


Figure 16-20. GTFS17-014, In proximity of the East Wall, - fair to poor rock mass conditions

16.4.4.8 Revised Pit Design – Geotechnical Assessment

The current proposed design extends the southern pit floor to the 144 mRL (~230 m deep pit from the crest of the southern pit west wall). This results in extension of the pit wall to the east and west. For the west wall (hanging wall), comprising fresh granodiorite gneiss, the current slope designs are considered adequate.

The original DFS design was based upon 10m bench heights, with 10 m batters above 270 mRL on the east wall, and 285mRL on the west wall. The batter heights increased to 20m below the 270/285 mRL, respectively. The design creates three (3) pits: North, Central and South (Figure 16-21) with small saddles between each. A smaller waste dump is designed to the east, approximately 180m from the pit crest. The western waste dump and ROM pad (Figure 16-21 -inset) is located along the crest of the north and west of the central and northern pits.

A section through the southern was taken at mid-point to assess the design versus DFS base case slope design parameters (Langille 2019) with the modified batter/bench configuration proposed by SROL of 24m high batters (2 x 12 m stacked benches) and modified bench widths. The southern pit section at approximately 11210 mN section (local coordinates) is shown in Figure 16-22.

Figure 16-24, Figure 16-25, and Figure 16-26 are sections through the mid-point of the southern, central, and northern pits.

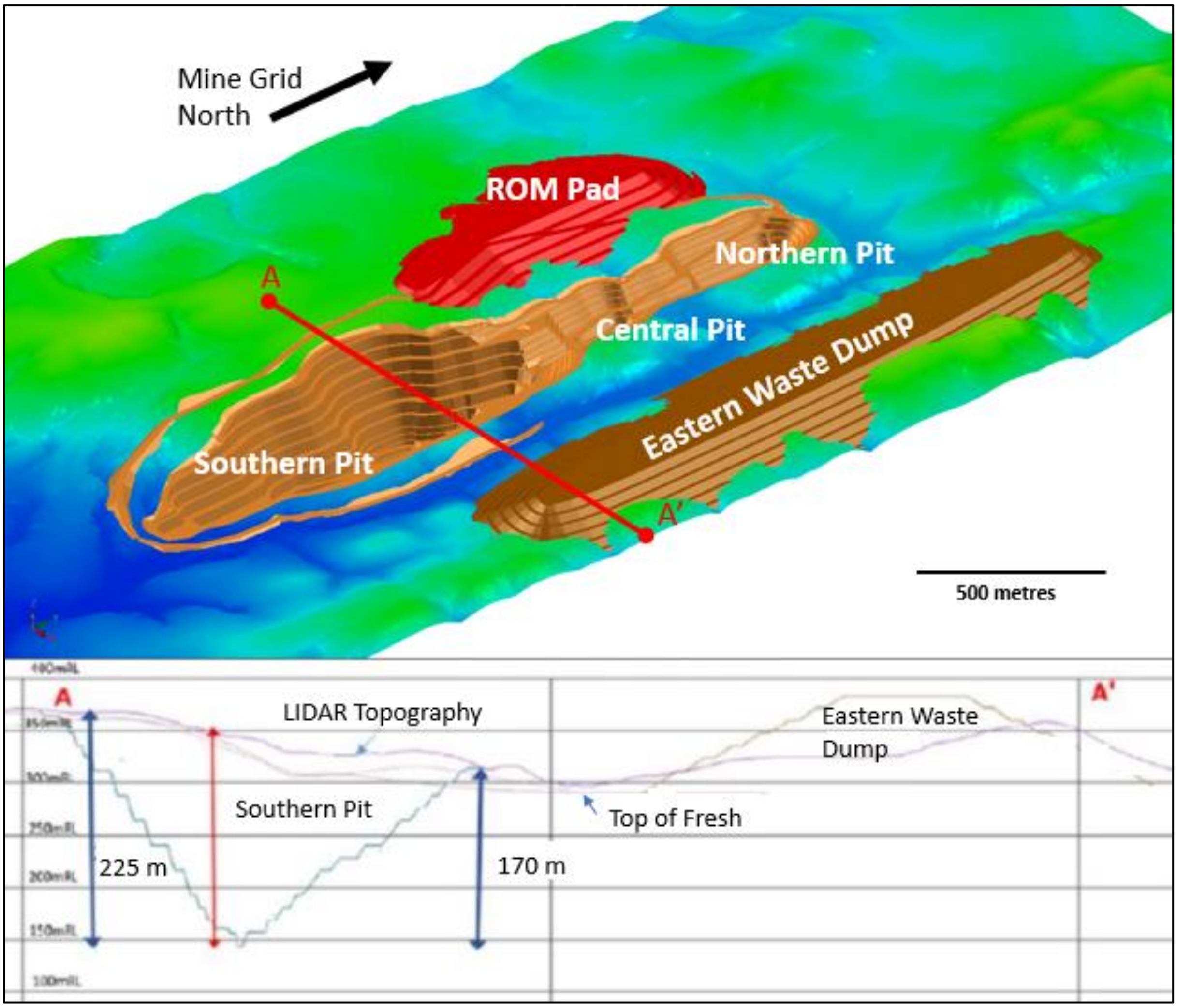


Figure 16-21. Plot of Proposed Pit Shell Design - pit6_lom_v7_clipped.dtm

Batter angles are consistent with the recommendations of 60° in the upper weathered/transition zone on the west wall and 75° in fresh and the 9.3 m berm widths (with a 12.5 m geotechnical berm at the 285 mRL) are adequate for the final 24 m batter heights. The east wall is similarly consistent with the design recommendations (50° in weathered/transition, 55° in fresh, with 9.3 and 11 m berms respectively).

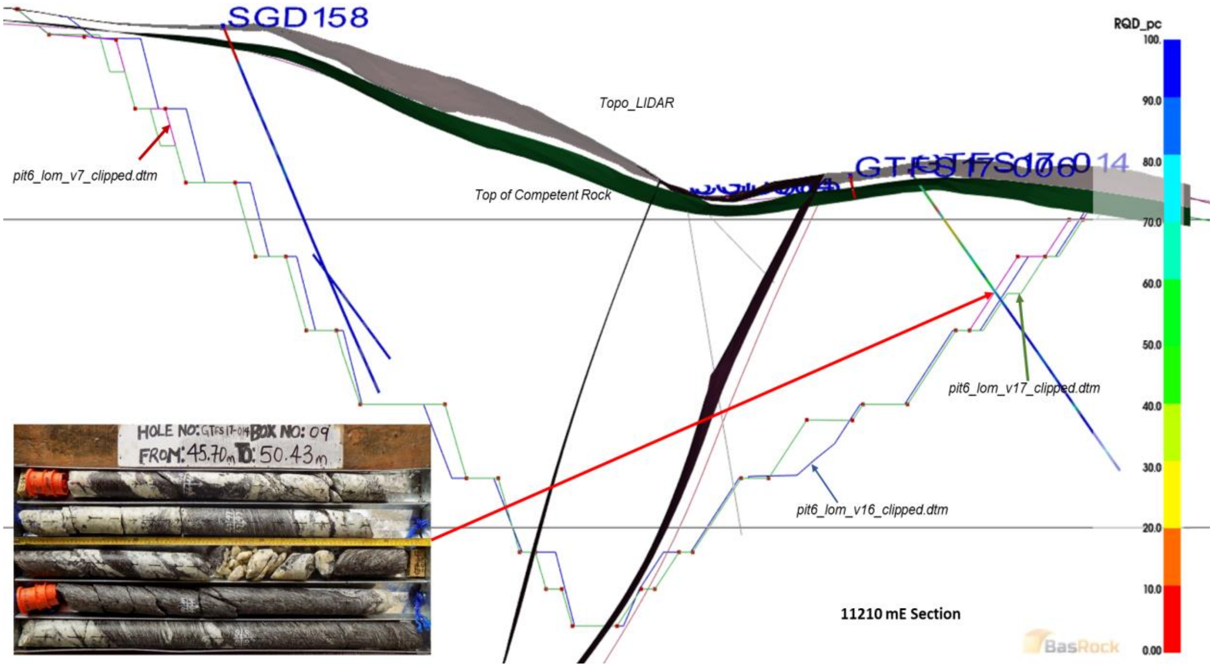
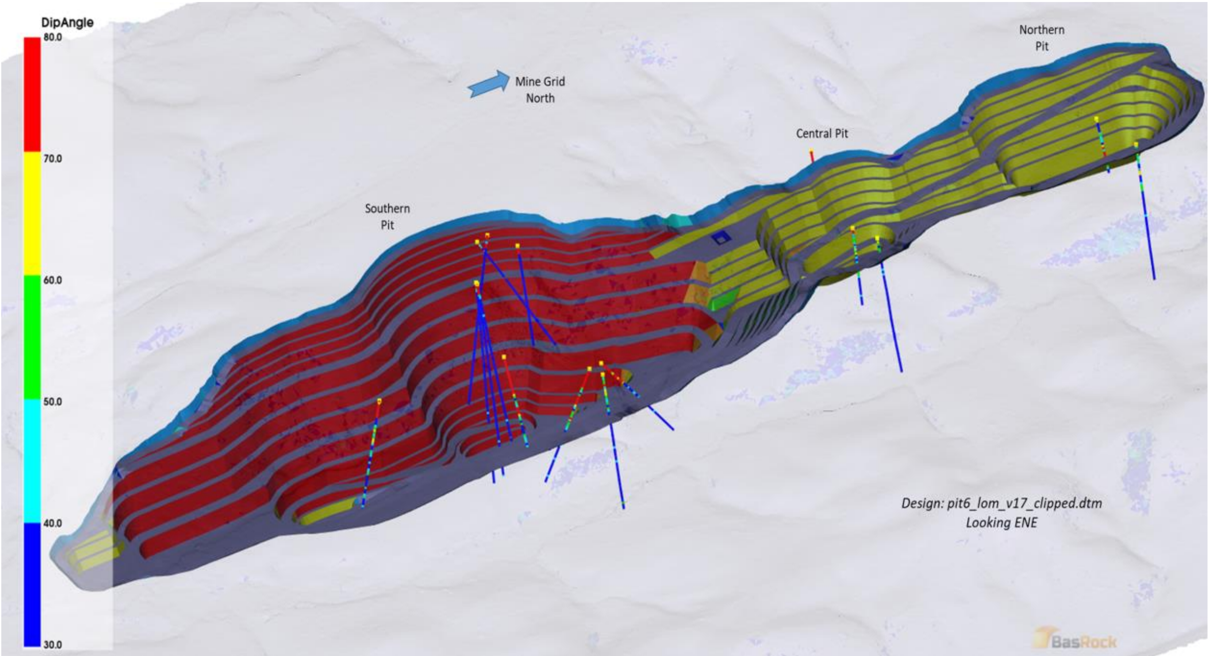


Figure 16-22. Section 11210 mN of Pit 6 LOM V17.dtm design

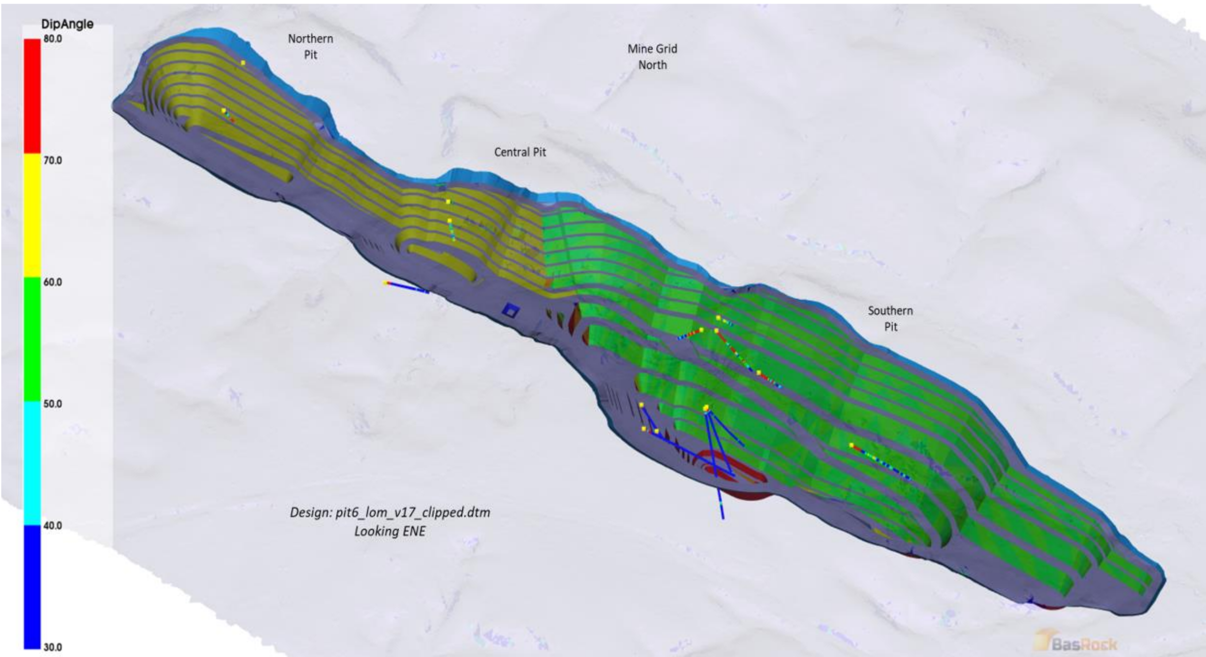
Plots showing colour heat maps of the batter angles in the west and east walls of the pit design are shown in Figure 16-23. Figure 16-23 (a) shows the batter slope colour coded on the west wall (red – 70-75° and yellow 65°) and Figure 16-23 9b) shows a view of the east wall, with shallower batter angles (green 55° and yellow 65°).

The inter-ramp (IRA) and overall wall (OWA) angles are reasonably close to the design 40 degree OWA and 41 degrees IRA in the east wall and 52 degree OWA and 54 degree IRA in the west wall.

Bench widths are consistent with the recommended design, with 2 x 9.6 m wide benches, stacked in 2 batters, separated by a 13.5 m wide bench.



(a)



(b)

Figure 16-23. View of West Wall (a) and East Wall (b) of pit6_lom_v17 design with batter face angles coloured on dip angle.

Average parameter values for the east and west wall of each pit are summarised in Table 16-14. Table 16-14 summarises the inter-ramp, overall wall angles and bench configuration for the modified slope configuration, proposed by SGOL, using double stacked 12 m benches, 3 m flitches, for combined 24 m high batter walls in the fresh rock.

Table 16-14. Revised Pit Design Criteria (12m Bench Height)

Pit	Section mN*	IRA				OWA with respect to Surface				
		West		East		West		East		Pit Floor
		Crest RL	IRA	Crest RL	IRA	Crest RL	OWA	Crest RL	OWA	RL, m
Southern	11165.5	370	52.2	320	42.0	370	51.0	320	38.0	144.0
Central	11745.5	324	49.0	332	48.0	324	40.0	332	39.0	240.0
Northern	12045	337	44.5	322	49.0	337	49.0	322	43.0	240.0
* Northing sections are taken from mine grid northings, perpendicular to the pit wall										

16.4.4.9 Geotechnical review findings

There are no expected issues with small ‘bullnose’ protrusions in the saddle region between the three pits. Geotechnical data collected during early mining in the northern, central, and southern pits will give sufficient confidence in the original analysis results that can be evaluated before progressing deeper, however, this is not a major issue stability-wise.

- The northern and central pits are relatively shallow ~90m depth and will not likely present significant stability issues due to the limited wall height/depth of pit.
- The east wall of the northern pit, however, will need further evaluation during operations. This area of the wall is just south of the Geotechnical drill hole which collapsed into a void (GTFS17-013) and adjacent drillholes were not extended to attempt to define the cause of the problem.
- The waste dump designs were not part of the scope of works for the pit geotechnical design, however; the location of the waste dump and impact on slope stability in the east wall was identified as a potential issue.
    - Given the distance to the pit crest (180m to 200m) and the presence of a natural (seasonal/ephemeral) watercourse between the proposed waste dump and pit crest, Langille (2020) does not see any issues with the waste dump surcharge weight impacting with the east wall pit crest and wall stability. This could change if the dump encroaches closer to the pit crest to that shown & the dump toe is shifted to the west of the watercourse, such that the dump is not allowed to drain effectively.
    - The west/northwest waste dump/ROM pad encroaches on the west wall of the central and northern pits, timing is critical, after those pits are completed and back filling has commenced. Wall surcharge loading will not likely result in instability.
- the overall wall shape and orientation for the southern, central and northern pits meets the design recommendations, with the exceptions noted above.

In general, the revised designs meet the basic slope Overall Wall Angle (OWA) requirements. Modifying the upper portion, above the base of oxidation to single lift benching (12 m high batters) will result in a slightly shallower overall wall angle to meet the recommended criteria.

The OWA is measured from the pit floor toe to the crest of the fresh rock batter and assumes the extremely weathered/residual soil has been pushed back to less than 35°, the natural angle of repose. The Inter-ramp angles (IRA) are measured from crest to crest, above and/or below the included haul ramps.

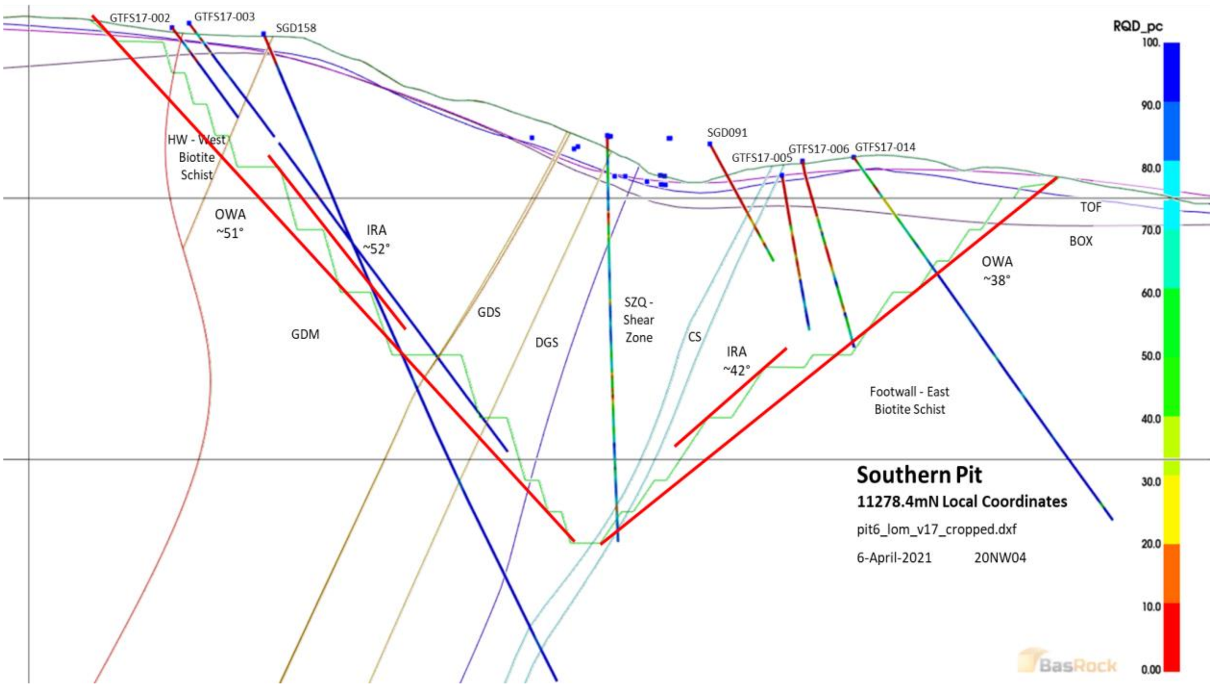


Figure 16-24. Section 11278 mN showing batter/berm configuration of the Southern Pit

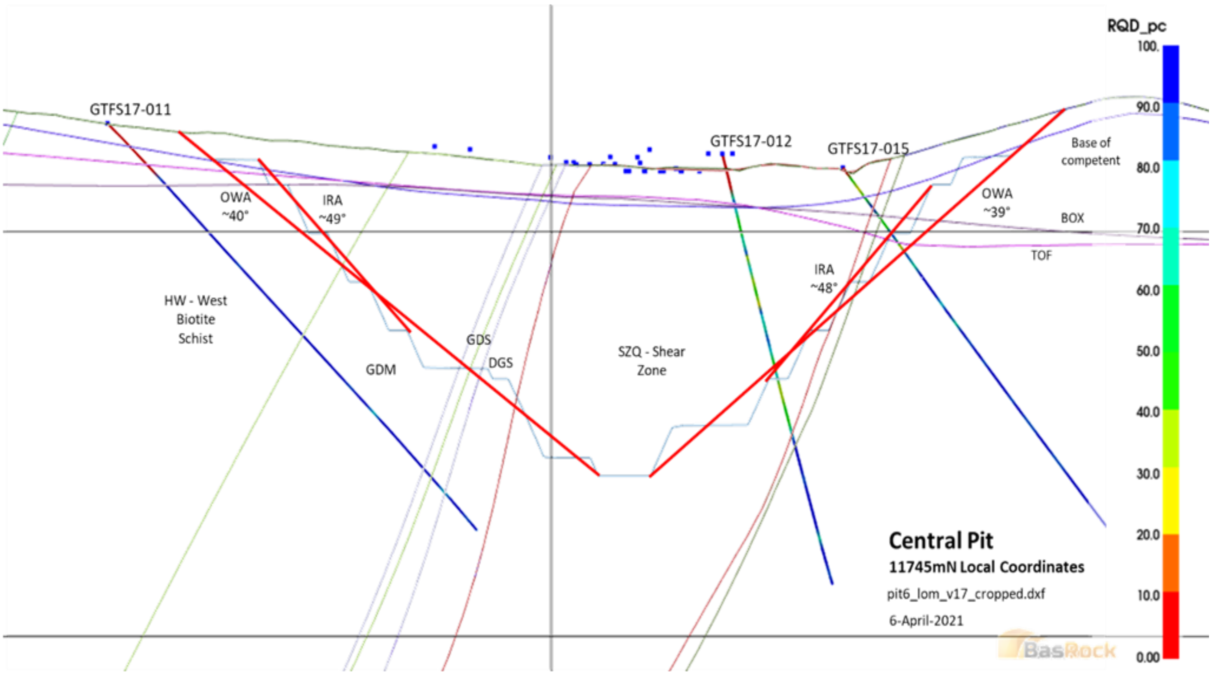


Figure 16-25. Section 11745.5mN showing batter/berm configuration of the Central Pit.

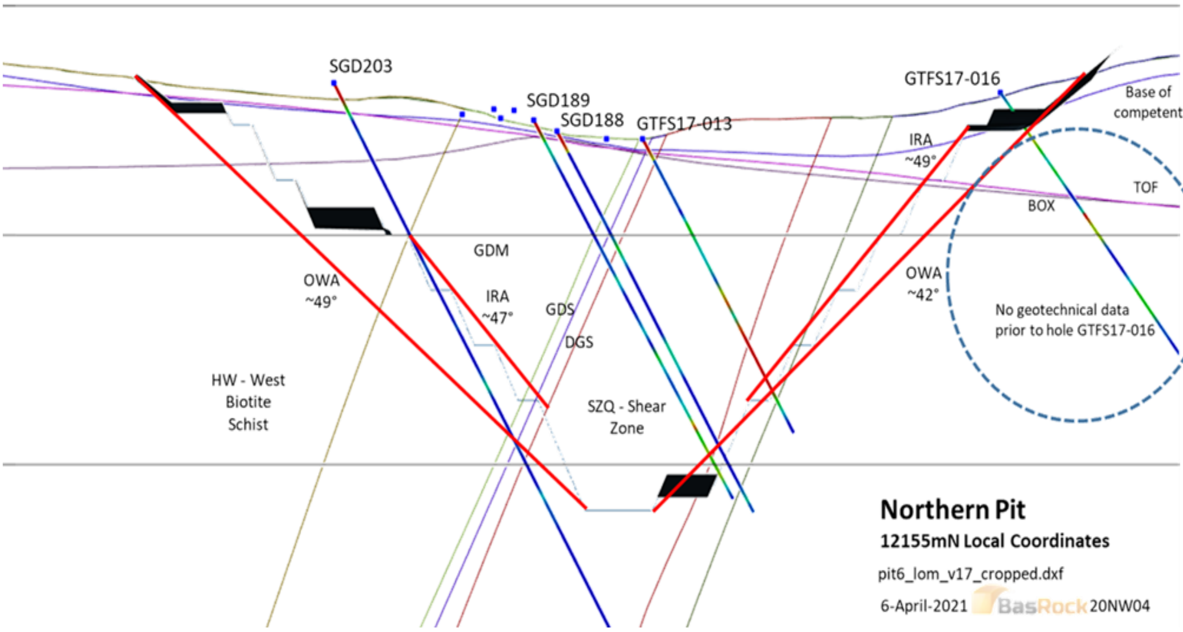


Figure 16-26. Section 12155 mN showing batter/berm configuration of the Northern Pit.

16.5 HYDROGEOLOGY AND MINE DEWATERING

SRK Consulting Ltd (SRK) conducted a field investigation in 2018, with a final analysis completed by Peter Clifton and Associates. The hydrogeological work to date has been limited due to various issues in the field, including hole collapse, so there is some uncertainty around the impact of groundwater in mining and the impact of mine dewatering on the local environment. As such, conservative judgements, such as dewatering for the LOM, in pit dewatering sumps, an interception trench at the base of the weathering zone and horizontal drain holes have been assumed in the mine planning in relation to hydrogeology.

Groundwater depth was measured, and packer tests were performed. Several drill holes collapsed, indicating a high groundwater level. Hydraulic conductivity in the orebody sequence and hanging wall sequence were similar, with a conductivity range of 2.7 x10^{-3} m/day and 5.1 x 10^{-3} m/day. Collapse and abandonment of geotechnical holes indicate potential issues with water.

A relatively small range of groundwater levels were measured given the variation in topography, with groundwater generally occurring within ten metres of ground level. This means dewatering and groundwater inflows into the open pit will need to be managed from the start of mining.

The slightly weathered, fracture rock in the east, north, and south walls of the proposed pits, to depths of up to 50 m, will likely be water-bearing. Water inflows expected into the pit were not confirmed, however, packer testing in the massive granodiorite indicated a low hydraulic conductivity, with the aquifer largely confined to the shear structure hosting the mineralised zone.

Given the mineralised shear hosting the lodes is a likely aquifer, lack of effective dewatering and the relatively shallow water table mean that water management will be a key issue from the start of mining.

Generally, aquifers in the zone of the mine tend to be low yielding with good quality water. The hydrochemistry of deeper groundwater is not known.

Groundwater occurs in the surficial Quaternary, alluvial sedimentary deposits, at the base of the weathered zone immediately above fresh rock and in fractured and faulted sections of the orebody and footwall sequences. The thickness of the friable transitional material ranges from 10 m to 30 m and will pose the most difficulties when dewatering the pit. Large-scale structures in the orebody and footwall are expected to form the main aquifers that require dewatering.

Surface water infiltration would be the main process that recharges the groundwater system at the pit.

Some interaction between shallow groundwater and surface water in drainage channels can be expected during mining. In the ambient setting prior to mining, some of the stream flow during the dry season could be sustained by groundwater seepage, i.e. the streams are gaining groundwater. Mine dewatering may reduce these flows and possibly could cause some streams to cease flowing during the dry season.

Given the uncertainty, partly saturated conditions were assumed for the analysis, and requirements for managing groundwater as part of the mining operations were identified. These included:

- Establishment of effective diversion channels at the base of residual soil slope to manage groundwater flow along this contact aquifer and any potential perched water from structure related aquifers below the fresh interface.
- Use of sumps in the pit floor and identification of a place to pump and store the excess water.
- Allowance for wet blasting conditions and appropriate explosive (emulsions) for wet hole conditions.

- Establishment of a series of horizontal dewatering holes as the pit is developed. This is likely to impact the eastern, southern, and northern walls of the Stage 1 and 2 pits as the relatively impermeable hanging wall sequence is not expected to be difficult to manage. Design recommendations will be provided as part of a mining operations plan.
- Establishment of piezometer holes in the crest of the pit with a series of piezometers at different depths to measure in-wall water pressure and drawdown during mining. All design analyses assumed wet conditions and a water table near surface for the mineralized zone and biotite schist zone and are therefore considered conservative.
- General observation of wall conditions include reference to the presence (or absence) of water in pit walls, and the volume of inflows into, and the effectiveness of, interceptor drains and sumps.
- The humid tropical environment 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.

16.5.1 **Current Work**

The key target for the dewatering is the central shear zone and immediate footwall. Initial drill positions were revised and workshopped based on mine plans, geological information, and hole longevity.

16.5.1.1 Dewatering Programme

Following the unsuccessful attempt in drilling dewatering boreholes in 2018, SRK in 2020 redesigned 7 dewatering boreholes and 10 piezo in and around the Segilola pit (Figure 16-27).

The key target for the dewatering was the shear zone and immediate footwall. Initial drill positions were revised and workshopped based on mine plans, geological information, and hole longevity. Given the strike length of the pit and that the shear zone runs through the centre of the pit, at least one sacrificial borehole was planned. Monitoring boreholes were located largely on the pit perimeter and in some cases twinned with dewatering holes to understand dewatering effects.

SRK provided specifications for the drilling of proposed dewatering and monitoring boreholes, based on the 2018 groundwater review by Peter O'Bryan & Associates, and SROL learning from the previous drilling in 2018.

The specifications, "Specifications for Drilling and Testing of Dewatering Boreholes and Stand-pipe Piezometers for the Segilola Gold Project", were revised in July 2020 after discussions with both drill contractor and the mine.

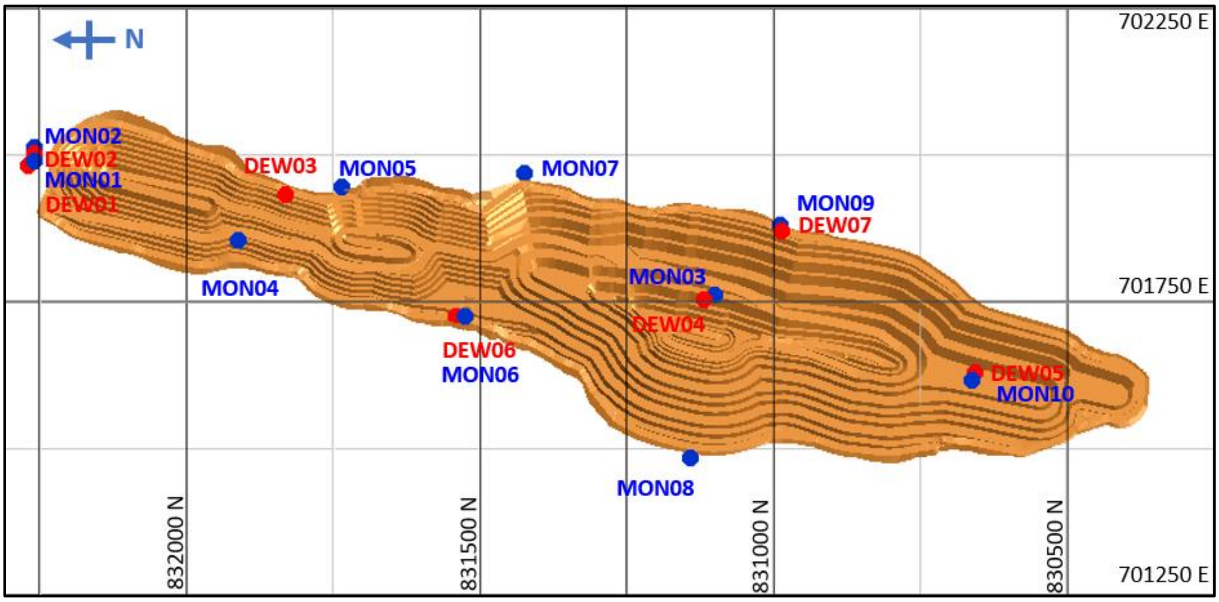


Figure 16-27.Completed Dewatering (DEW) and Piezo (MON) Holes

16.5.1.2 Drilling of Dewatering and Monitoring Holes

A number of large diameter dewatering- and small diameter monitoring boreholes have been drilled, totalling 2,002m. A table of the boreholes are provided in Table 16-15.

Reverse circulation drilling was carried out using an Ashok Leyland DTH drill rig with 1100cfm/330psi compressor system. Hole collars were surveyed using a Differential Global Positioning System (DGPS) and Real Time Kinematic (RTK) positioning to enhance the precision. The DGPS receiver was a Trimble R8 using the Global navigation satellite systems (GNSS).

16.5.1.3 Dewatering Boreholes

Seven (7) dewatering boreholes were drilled with only 6 developed. The seventh borehole DEW 04 drilled dry and was therefore not developed.

Drilling commences using a 14-inch star-shaped bit to penetrate the loose weathered oxides to reach the solid bedrock. 12-inch uPVC casings are installed to the bottom of the weathered oxides to hold-off the loose material.

The borehole is then advanced by drilling with a 10-inch bit size to reach a depth of 101.2 meters.

If the required depth is in excess of 101.2 meters, an 8-inch bit size is used to drill to the optimal depth with advice from SRK consultants.

16.5.1.4 Monitoring Holes

Ten (10) monitoring holes were drilled and constructed. The boreholes were drilled using a 6-inch drill bit to optimal depth except for MON 02(Monitoring hole 2) which was drilled with a 10-inch sized bit.

Table 16-15. Completed Boreholes with Coordinates

hole_id	Purpose	UTM East	UTM North	Elevation	Depth	survey date
DEW01	Dewatering	701982.742	832268.794	312.719	101.2	23/01/2021
DEW02	Dewatering	702003.286	832258.664	315.323	60	23/01/2021
DEW03	Dewatering	701931.764	831829.883	312.269	102	17/12/2020
DEW04	Dewatering	701753.240	831118.001	337.890	140.3	10/02/2021
DEW05	Dewatering	701629.829	830657.956	307.477	142.6	17/12/2020
DEW06	Dewatering	701726.034	831540.680	323.973	101.2	17/12/2020
DEW07	Dewatering	701871.030	830986.116	319.624	138	23/01/2021
MON01	Monitoring	701990.632	832258.895	315.632	50	23/01/2021
MON02	Monitoring	702013.931	832257.585	315.083	101	23/01/2021
MON03	Monitoring	701761.467	831101.086	337.468	140	10/02/2021
MON04	Monitoring	701855.839	831910.575	318.145	120	17/12/2020
MON05	Monitoring	701946.872	831734.853	309.506	142.6	17/12/2020
MON06	Monitoring	701724.795	831526.142	324.124	101.2	10/02/2021
MON07	Monitoring	701968.924	831425.425	322.426	140	17/12/2020
MON08	Monitoring	701484.788	831142.810	372.833	142	17/12/2020
MON09	Monitoring	701881.298	830988.618	319.480	140	17/12/2020
MON10	Monitoring	701616.290	830660.993	309.274	140	17/12/2020

16.5.1.5 Pumping Test

The pumping test program initiated on 9th January 2021 has been successfully completed (Table 16-16). A total of 441 hours of step drawdown and constant discharge pumping tests and subsequent recovery monitoring have been completed on all the dewatering boreholes to determine the behaviour of the aquifers in the area. Neighbouring monitoring holes were also used to collect data during pumping.

The physiochemical parameters of each dewatering borehole were recorded during constant discharge testing. Samples were retrieved from all site boreholes and dispatched to an accredited laboratory for analyses.

Preliminary results indicate compartmentalized aquifers, each with variable hydraulic characteristics and storage capacities. Further analysis is currently being carried out to determine the hydraulic behaviour of the aquifers, determine ideal pumping rates for the boreholes and complete design of the dewatering system.

A two-levelled step test and recovery was conducted on DEW01, and three -levelled step test and recovery were conducted on DEW02. However, four levelled step test and recovery was conducted for DEW05 and 07. This is because the water level did not drop to suction after the two-levelled step test. The step test and recovery were done for a duration of 48 hrs. and 72 hrs. respectively.

Table 16-16. Boreholes with Pumping Test Completed

Borehole	Date Started	Date Completed	Maximum Yield Level
DEW01	05/03/2021	17/03/2021	4.3 l/s
DEW02	17/03/2021	22/03/2021	6.7 l/s
DEW03	13/02/2021	13/02/2021	7.0l/s
DEW05	23/03/2021	29/03/2021	5.1 l/s
DEW06	17/02/21	25/02/2021	13.6l/s
DEW07	27/02/21	05/03/2021	7.9 l/s

16.5.1.6 Water Quality Sampling

Water samples were collected from DEW01, 02, 03 05, 06, 07, some streams and all monitoring boreholes. The samples were sent to the Technology Partners International Nigeria Ltd Laboratory for water quality test. The results for Physico Chemical and Heavy Metals for DEW03, DEW06 and DEW07 obtained have values recorded below their respective WHO thresholds for the regulated parameters.

16.5.2 Storm Water Management

Groundwater inflows are anticipated into the Segilola open pit gold mine, which is in a high rainfall area. Previous work identified the footwall and the shear zone to be the principal sources of water to the pit. A water management plan, that would initially include dewatering boreholes from the pit crest, sump pumping, stormwater diversion and monitoring to determine the effectiveness of the dewatering system, is required.

In this regard, SRK was appointed by SROL, to develop an open pit mine water management plan to deal with the anticipated runoff in and around the open pit area and dirty water run-off from the dump areas.

Source document: 560747_Segilola_Open Pit Storm Water Management_

16.5.3 Diversion Bunds

Diversion Bunds defining paddocks should be constructed around both the waste rock dumps (WRD) footprint to prevent any surface water runoff from the WRDs, from flowing into the nearby stream; and the proposed diversion channel sizes to divert the runoff away from the open pit area are shown in Table 16-17 below:

Table 16-17. Proposed Channel Size for Diversion

Channel	Proposed depth (m) 1:2 side slopes	Bottom width (m)
C1	0.2	0.5
C2	0.2	0.5
C3	0.25	0.5
C4	0.35	0.5

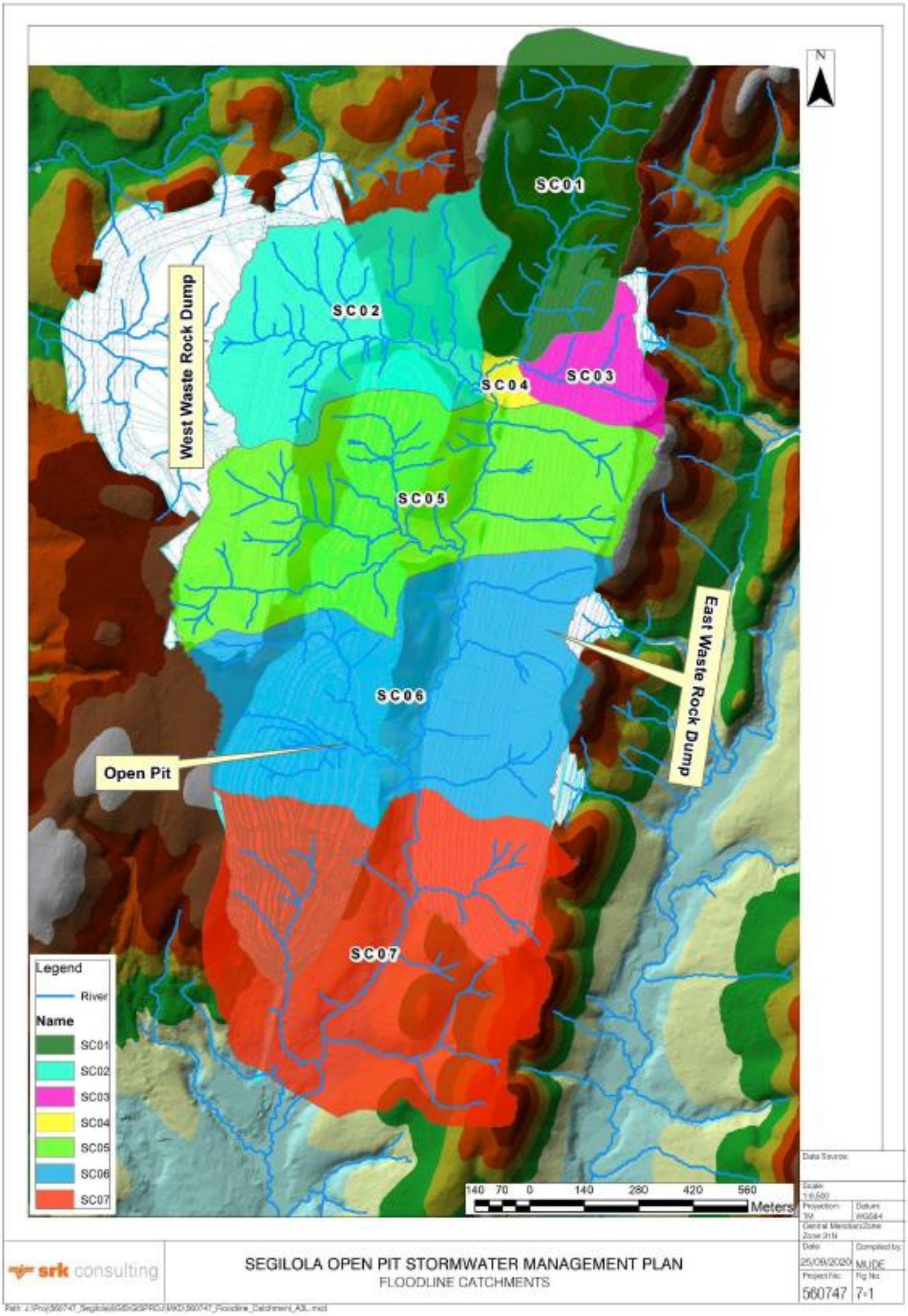


Figure 16-28. Stormwater Management Plan

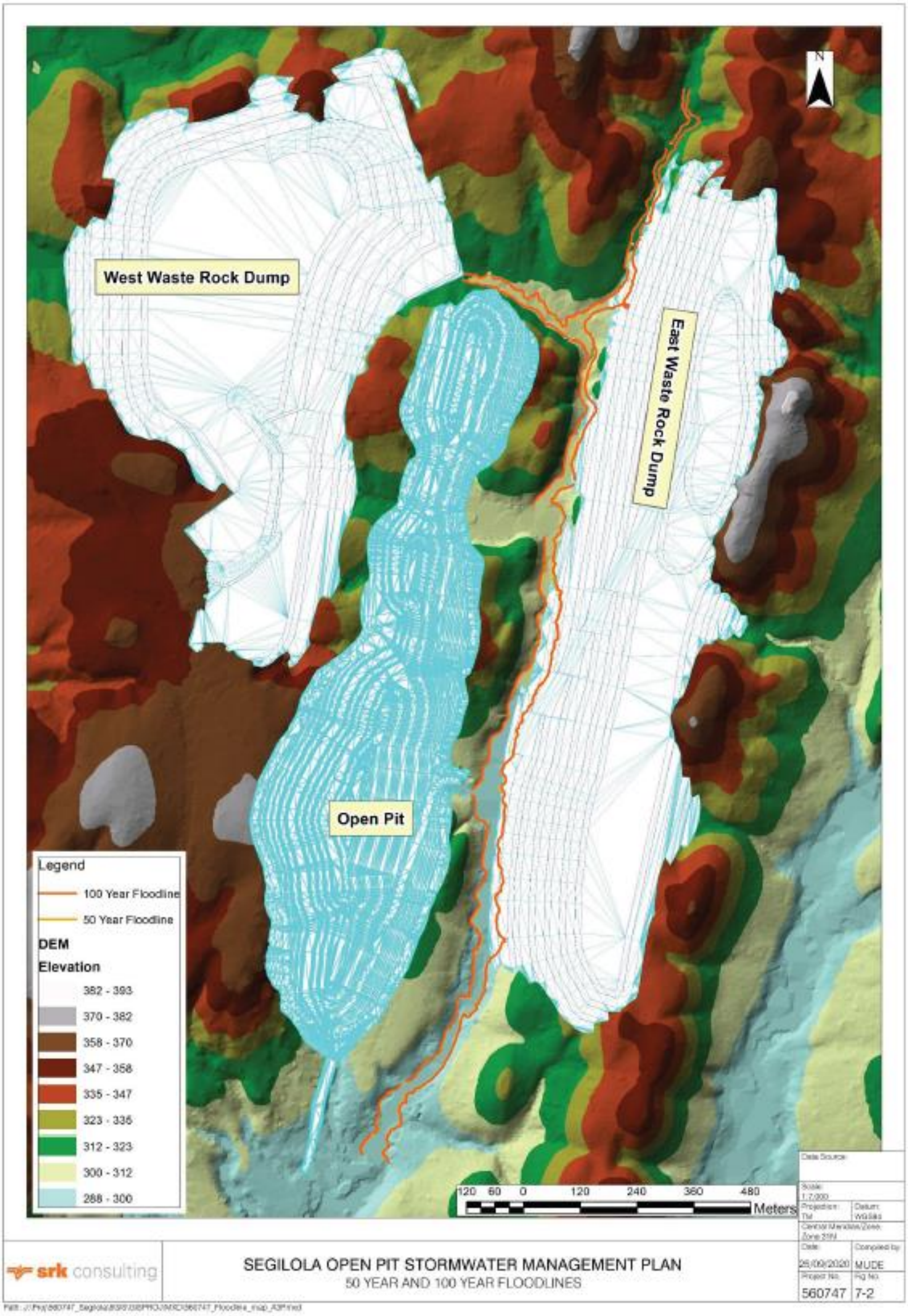


Figure 16-29. 50 Year and 100 Year Flood Lines

16.5.4 Paddock System below the Waste Rock Dumps

To prevent the dirty water generated from the Waste Rock Dump (WRD) entering the stream, Bunds need to be constructed defining staging paddocks. Therefore, paddocks are recommended around the WRD's footprint to prevent any surface water run-off from the WRDs, from flowing into the nearby stream. Spillways will be required from each paddock either interlinking the paddocks and discharging at the lowest paddock or spillway at each paddock and discharging to the environment.

Table 16-18 respectively show the various dimensions of the paddocks required to store the 1:50-year storm event from the WRD runoff, from the embankment side slopes. Figure 16-30 shows a typical paddock cross section.

Table 16-18. Waste Rock Dump Paddock Sizing

Description	Units	East Dump	Waste Dump
SCS Curve number for dump slope		90	90
Height of dump	m	70	60
Lowest dump side slope = x	m/m	0.675	0.675
Slope of paddock = y	m/m	0.001	0.001
Paddock crossfall slope = z	m/m	0.0025	0.0025
Paddock freeboard	m	0.4	0.4
1:50-year rainfall depth	mm	120	120
Mean Annual Precipitation	mm	1356	1356
Mean Annual Evaporation	mm	1166	1166
24-hour stormwater volume from slope	m3/m	9.8	8.4
Maximum height of water above paddock toe	m	1	2
Minimum length of paddock	m	20	20
Maximum water depth below dump toe	m	0.02	0.02
Width of paddock	m	1	1
Vol of stormwater from side slopes	m3	9.8	8.4
Vol of stormwater directly onto paddocks	m3	2.4	2
Total stormwater volume in paddocks	m3	12.2	11
Total stormwater volume below dump toe	m3	1.1	1
Total stormwater volume above dump toe	m3	11.1	10
Depth of water above dump toe	m	0.6	0.5
Required paddock height	m	0.6	0.5
TOTAL PADDOCK HEIGHT WITH FREEBOARD	m	1	0.9
Vol of average rainfall from side slopes	m3	35.2	30
Vol of average rainfall onto paddocks	m3	27.1	27.1
Total average volume of stormwater in paddocks	m3	62.3	57
Annual average lake evaporation from paddocks	m3	19.6	20

Paddocks should be constructed around both WRD's footprint to prevent any surface water runoff from the WRDs from flowing into the nearby stream.

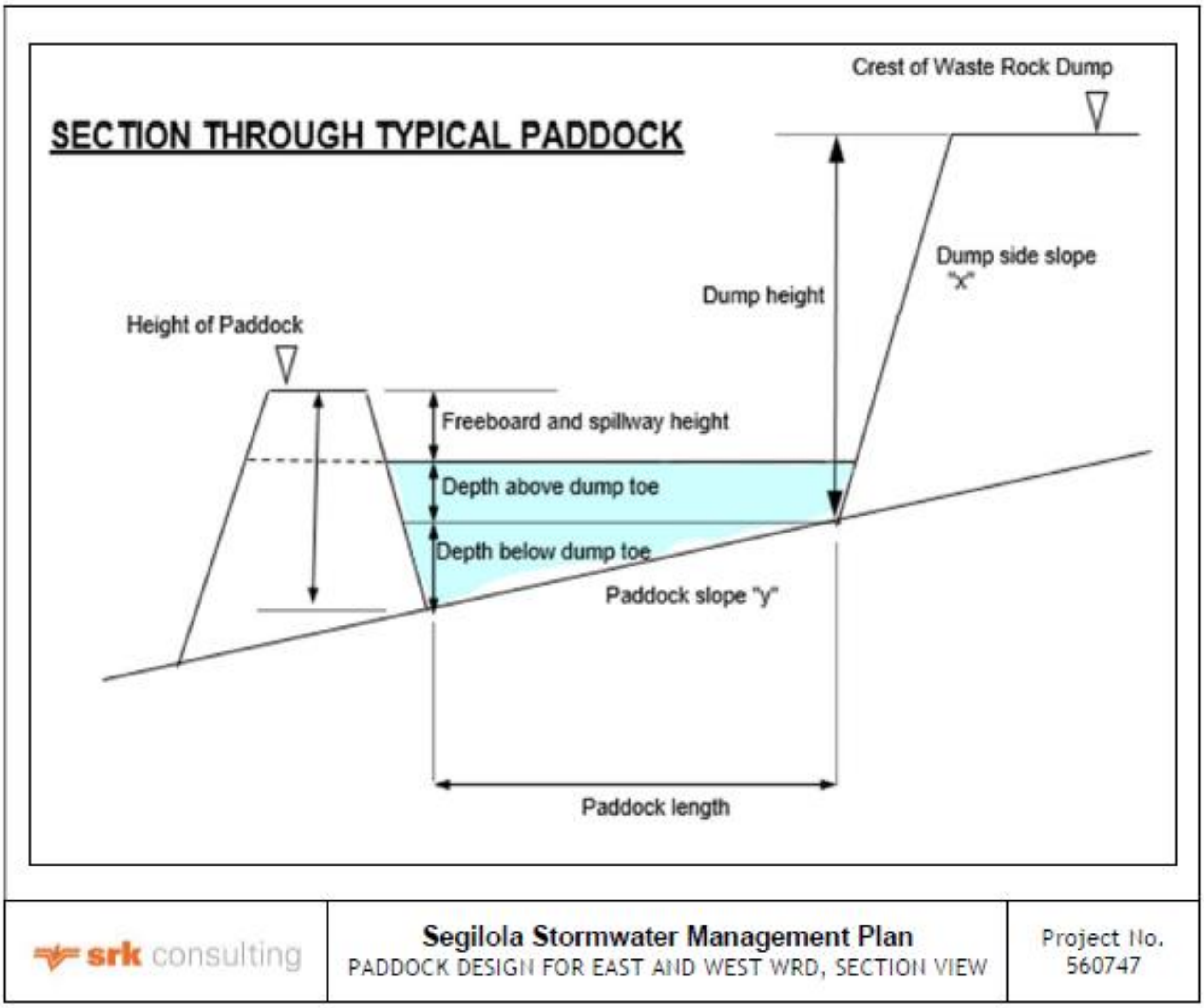


Figure 16-30. Paddock Design for East and West Waste Rock Dump, section view

16.5.5 Open Pit Water Management

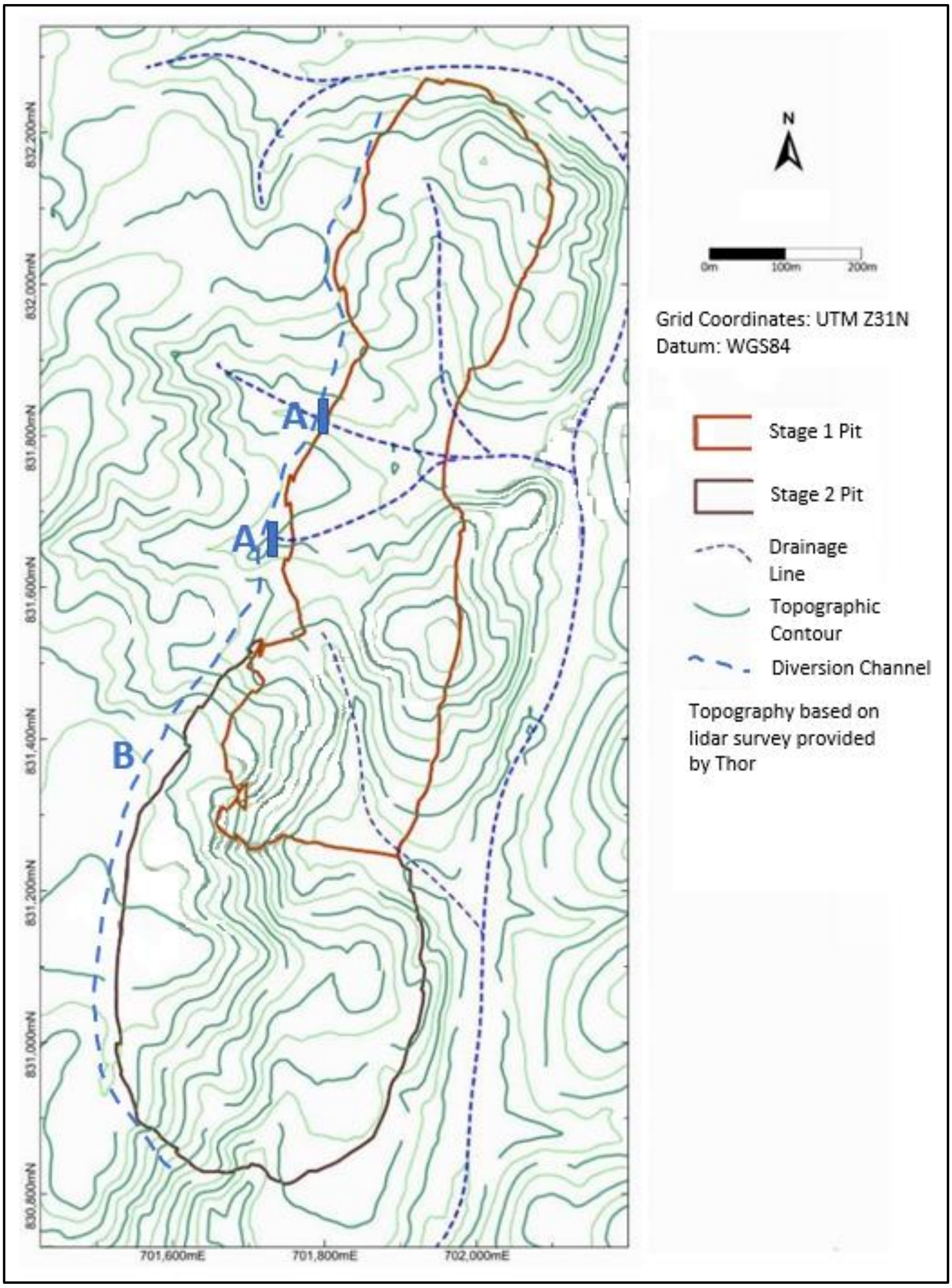


Figure 16-31. Open Pit Water Sources

A comprehensive open pit water management was created as part of SROL's mining readiness plan. This was generated as a separate report (Open Pit Water Management Plan). A bund and diversion channel will be created to divert the source of water at Point A and B, respectively (Figure 16-31).

16.6 ACID ROCK DRAINAGE MANAGEMENT

A report on Acid Rock Drainage (ARD) was prepared by Geochemic Ltd. In November 2020. The findings based on available data suggests that the potential for ARD at the Segilola deposit is low.

Geochemic determined that existing assay data limited outside ore zones (GDM1, GDM2, Biotite Schist). Current assay data set for sulphur grades from existing assay data may be useful to determine PAF/NAF volumes in ore zone area (GDS, DGS, SZQ1/2) where assay test density is higher.

Therefore, a further review will be required as mining commences and more data can be collected for further analysis. In this regard, Geochemic has identified an area that should be of particular focus in the next review as illustrated in Figure 16-32 below.

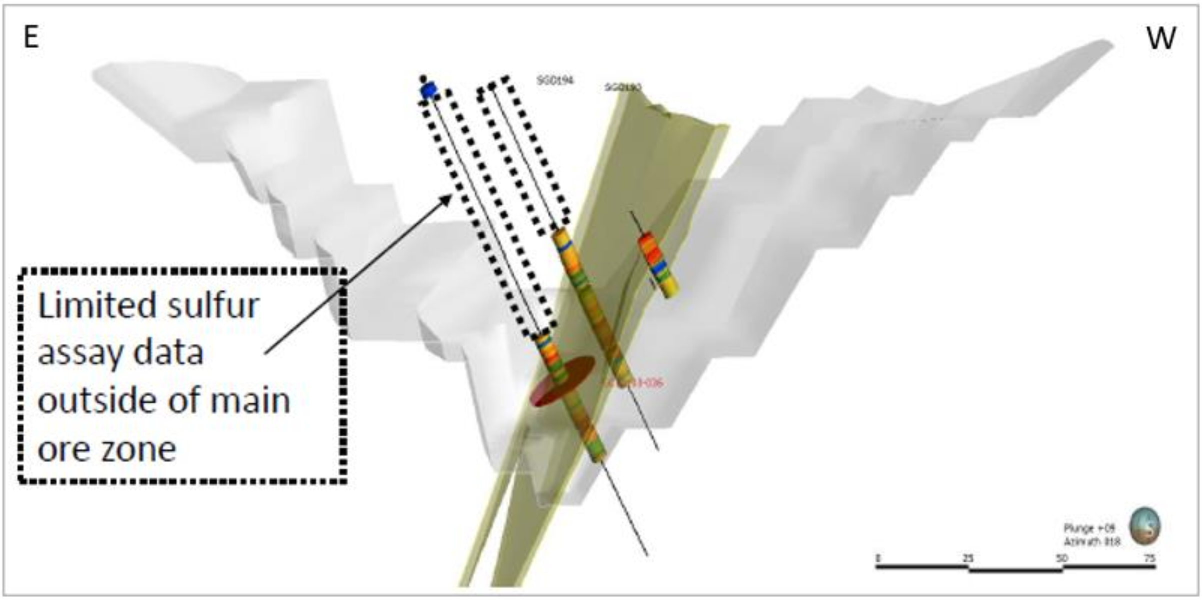


Figure 16-32. ARD Geochemic Proposed Sample Location

Calculation of H-PAF and L-PAF material volumes can be made from the geological model from these waste streams (depending on sulphur data density in existing model).

Currently there is insufficient data on waste to discriminate between NAF, H-PAF and L-PAF waste in block model and mine plan. Modelling of waste zones could be significantly improved with additional drilling (resource/reserve development stage or grade control).

Initially the problematic (i.e. PAF) waste zone could be assumed to likely be restricted to internal waste zones in:

- GDS alteration zone,
- DGS biotite schist
- SZQ1/2 altered schist
- Calc silicates (although these are likely to be buffered by carbonate mineral presence)
- Potential NAF zones could be assumed to include bulk waste zones in the Hanging wall sequence (GDM1+GDM2) and Footwall sequence (Biotite schist)

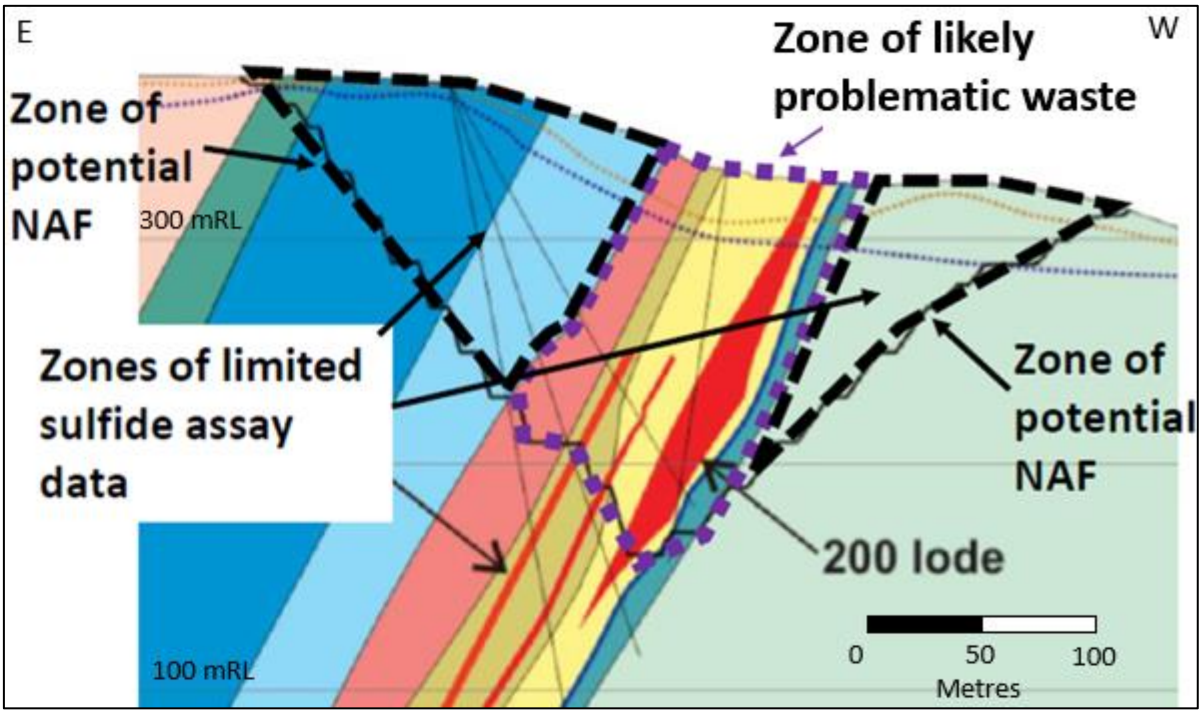


Figure 16-33. Proposed Zones for Further Testwork regarding ARD

16.7 MINING AND PROCESSING SCHEDULES

16.7.1 LOM Schedule

The average target production is 800,000 BCM per month which peaks in the third year of operations at 970,000 BCM per month. This coincides with a cut-back that will be undertaken on a campaign basis using equipment hire.

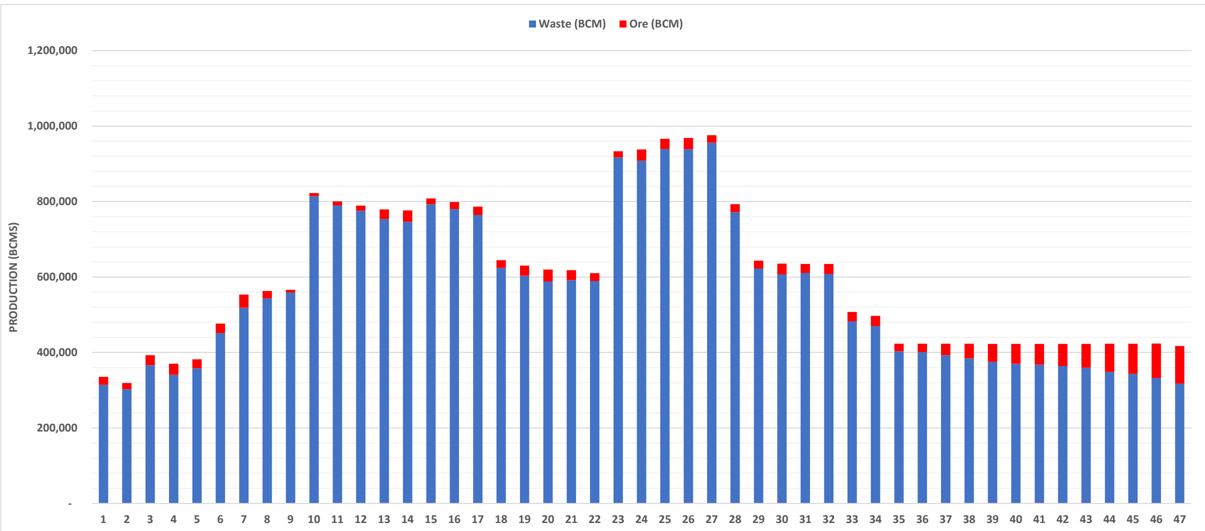


Figure 16-34. Production Schedule

The updated production schedule mines approximately 18% oxide, 10% transitional and 72% fresh material. The timing of each weathering profile is depicted in Figure 16-35.

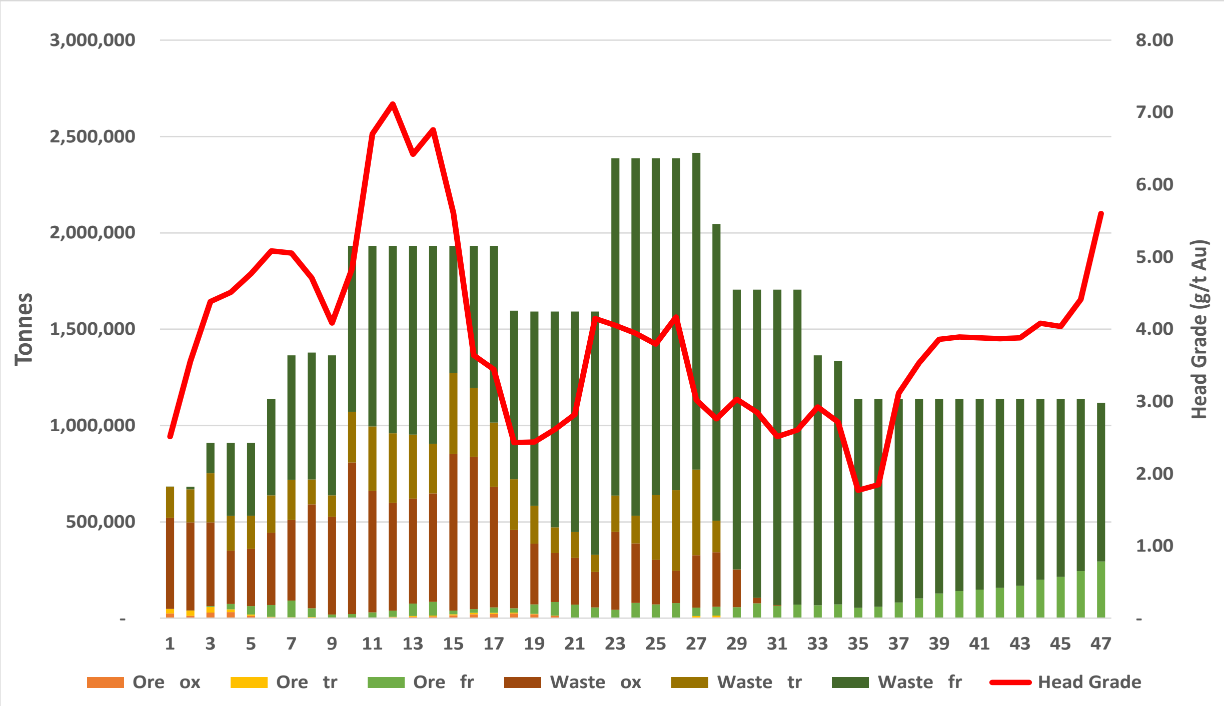


Figure 16-35. Production Schedule by Weathering Profile

16.7.2 Mining Sequence

The Life-of-mine pit was divided into 5 stages between two clear sections. Stage 1a and b are in the northern end, whilst stage 2a, b and c are in the southern end of the pit. These areas were defined based on several qualitative and quantitative factors namely, grade and strip-ratio. Along with various practical considerations such as proximity to potential ramps and access points. The underlying objective of this strategy was to defer waste into discrete periods thereby enabling more ability to manage cashflow demands across the Life-of-Mine (LOM) development.

The initial starter pit is referred to as Stage 1a. This area focusses on the historical artisanal workings. The rationale behind this as SROL's initial starting point was due to its ease of access coupled with the fact that it is clearly defined with relatively well understood high grade zone at a modest strip ratio. Each stage represents between 6 and 9mths of mining activities with the final stage (Stage 2c) representing more than 18 months.

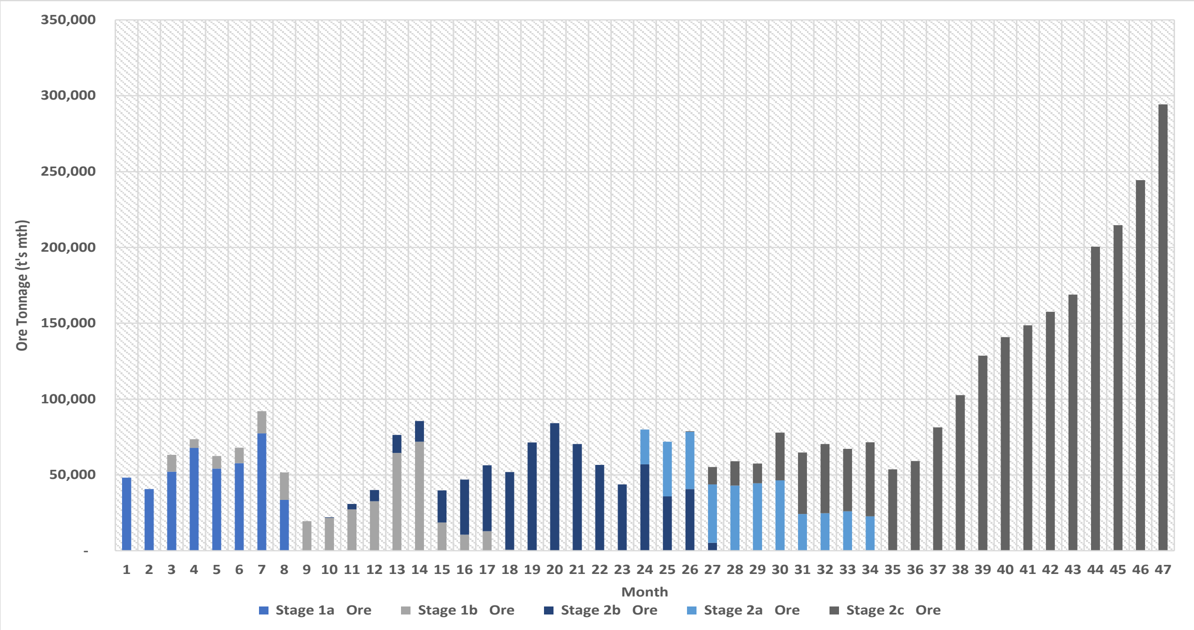


Figure 16-36. Ore Tonnage by Stage

The total volume of material mined for the Life-of-Mine pit is 28M BCM's.

Table 16-19. Pit Material Inventory by Stage

		Waste	Ore	Waste	Ore	SG Waste	Grade	Ounces
	Total	Mt's	Mt's	M BCM's	M BCM's	t/BCM	gpt Au	kAu
Stg1a	4.3	3.8	0.4	1.6	0.2	2.38	4.36	61
Stg1b	9.3	9.0	0.3	3.5	0.1	2.53	6.88	77
Stg2a	9.5	9.2	0.4	3.6	0.1	2.53	3.63	43
Stg2b	12.2	11.5	0.7	4.7	0.2	2.43	3.13	66
Stg2c	35.7	33.5	2.2	13.0	0.8	2.57	3.80	274
TOTAL	71.0	67.0	4.0	26.5	1.5	2.52	4.00	521

****Note:*** inventory table includes undefined ore outside of the Probable material.

The staging is split into five discrete zones thereby enabling flexibility across the mine-life to adapt to changing macro-economic assumptions. Sequencing in this manner enables SROL to focus on the near-term gold production and deferral of waste until later stages of the project's life.

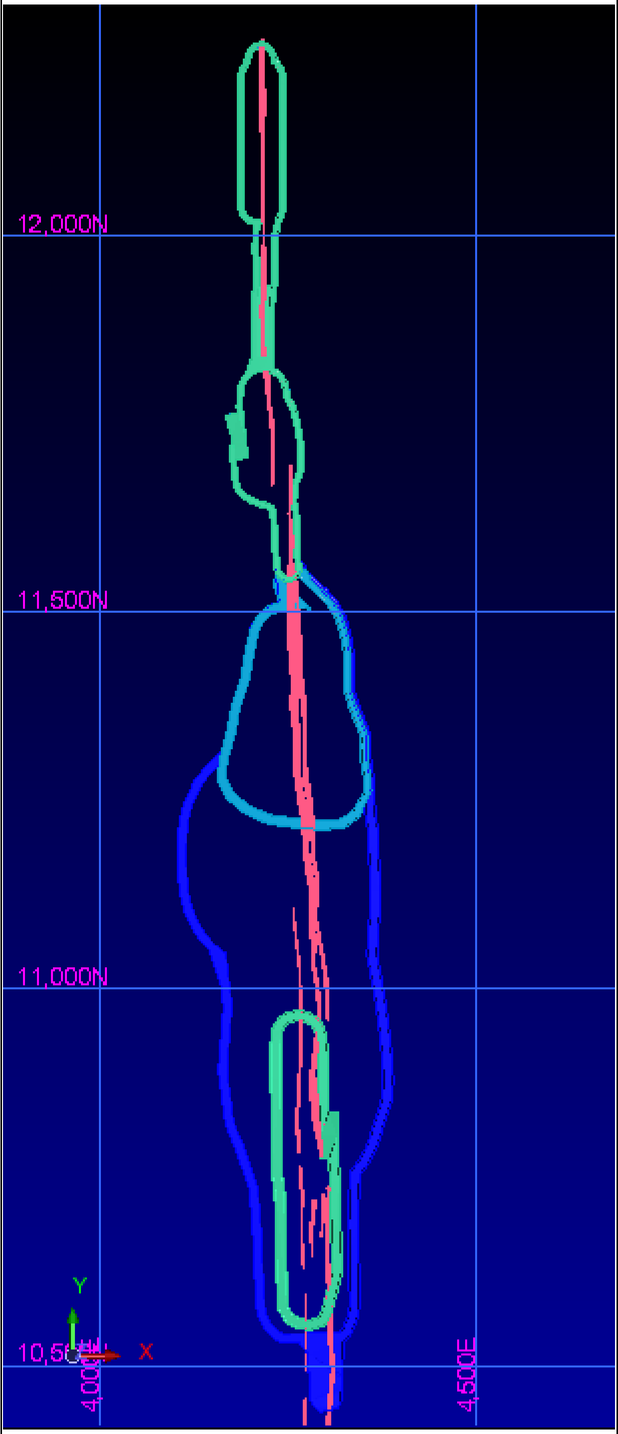
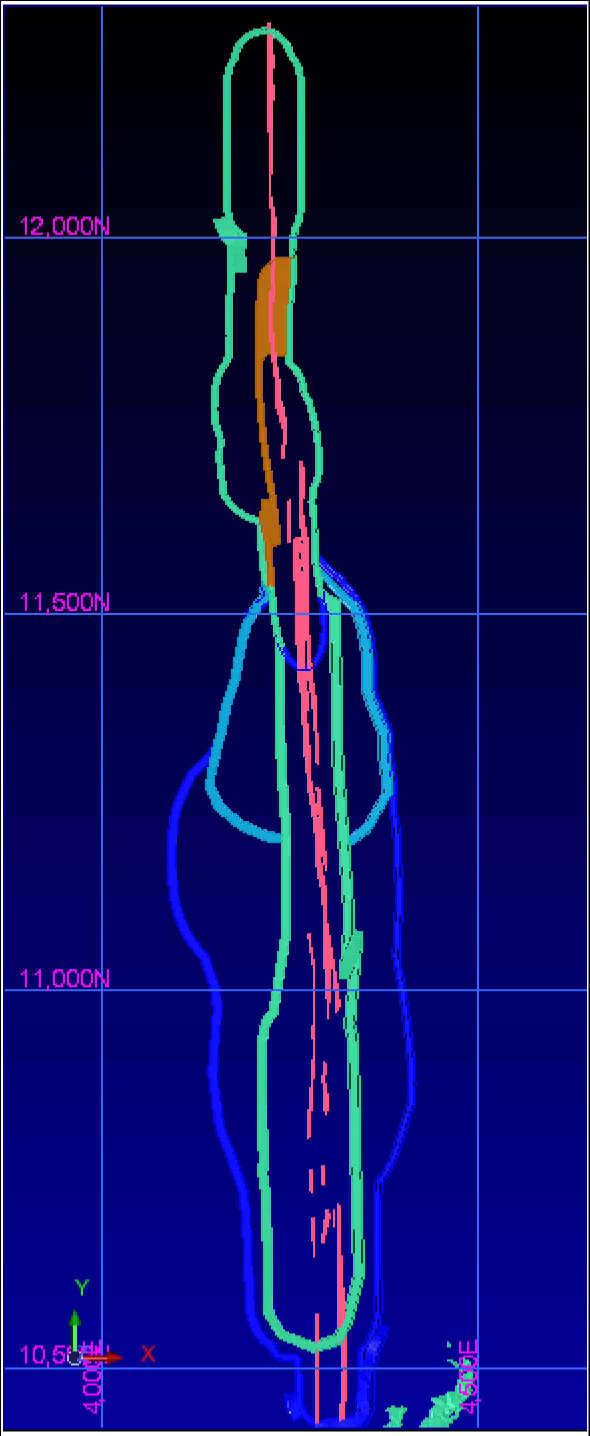
Pit staging (264mRL)	Pit staging (288mRL)
	

Figure 16-37. Pit Staging (Local Grid)

16.7.3 Waste Dumping Schedule

The waste dumps will be prioritised based on proximity from the main area of production. As such, the initial focus will be ensuring the ROM Pad is fully constructed utilising material from stg1a and stg1b in the northern section of the pit.

Thereafter, the waste will be sent to either the North-West Waste Dump or the Eastern Waste Dump depending on whichever is closer at the time. Most of the waste material produced towards the later stages of the mine-life will originate from the southern section (stg2a, b and c).

16.8 MINING RISK

- Slope design parameters:
  - especially on the eastern wall must be critically monitored, as batter angles are sub-parallel to foliation, especially in footwall contact areas.
  - The final bench height of 24 m on the final pit walls in the southern section of the pit could pose operational challenges. It should therefore be considered to halve these to 12 m heights, without flattening the overall angles – 12 m heights will enable all areas to have similar mining heights, berm positions and decrease toe loading (risk of failure).
- Pit dewatering:
  - In-pit dewatering holes will create more of a production nuisance than benefit over the relative short mine life
  - External dewatering holes in correct aquifers will add value.
  - The 6-month rainy season will have a significant impact in terms of water volumes collecting in the pit.
  - Current pit topography has overburden on high slopes, with the lowest part of the valley being on ore outcrop. Water ponding during the rainy season can hamper ore mining.
    - Overburden pre-stripping will be critical to enable alternate water ponding and collection on waste areas, away from ore mining faces.
    - The creation of sumps and installation of adequate pumping and piping infrastructure in the pit will be critical to ensure achievement of the mining volumes according to budget and plan.
- Initial mining- and plant feed schedules should be reviewed to ensure realistic alignment.
- The proximity of housing and other public infrastructure close to the mining activities, especially the southernmost part of south pit (some public infrastructure might fall within the blast radius for certain blasts), will have to be approached with due care and supportive of a long-term relationship, as mining will impact on the communities with regards to safety, health, environment, and infrastructure
  - Blasting (fly rock, dust, noise, vibration).
  - Water quality of downstream users.
  - Lowering of groundwater levels will potentially impact water supply wells and boreholes in the surrounding community.

    - Access control to prevent ingress of people and livestock into areas where heavy equipment operates.
- Based on the ore lode dimensions, dilution will remain a risk and achieving 12% or less dilution will require appropriate control and supervision over the ore mining operations

17 RECOVERY METHODS

This Item is not applicable for this report.

18 PROJECT INFRASTRUCTURE

The Segilola operations will maximize the use of existing infrastructure and the natural landforms, to reduce both their visual impact and reduce costs. Buildings are to be of a simple construction utilizing locally available materials where possible. To achieve this, a number of concepts have been applied:

- Maintaining a natural vegetation corridor between the major Odo Ijesha – Iperindo public road which passes through the Site and the processing plant.
- Damming of a local creek to provide the raw water supply (Water Storage Dam).
- Installing the process plant and camp into a series of terraced areas.
- Buildings to be constructed from local blockwork.

18.1 ROADS

18.1.1 Site Access Roads

Access to the site will be from Odo Ijesha – Iperindo Road via sealed access roads which will service five primary gates:

- Camp and Administration buildings;
- Processing plant / mining compound;
- Mining Contractor Camp;
- TMF / Emulsion storage facility / Laboratory; and
- Water Storage Dam.

An off-loading bay will be provided adjacent to the CNG / diesel storage tanks to allow diesel deliveries to take place without the need to enter the fenced processing plant compound. A turn around bay has also been provided to allow trucks to exit the site.

These roads will be between 3 m and 6 m wide depending upon traffic type and density.

These roads will carry a wide range of vehicles including trucks delivering equipment, fuel and reagents to the site, and light vehicles and buses for personnel movements.

18.1.2 Mine Haul Roads

Mine haul roads will be designed and constructed by the mine operations team to access the pits and the waste dumps and ROM pad, as well as the mining services facilities.

Two major mining roads will be provided, these being the pit ring road and the ROM access road. The ROM access road will connect the pit ramp to the ROM pad and will be primarily used by the mining fleet. This will be unsealed and be of sufficient width to allow two-way traffic flow.

The pit ring road will be constructed along the pit surface perimeter. This will encircle the pit allowing access to the mining compound, and the waste dumps and explosives magazines located on the north side of the pit. Access to the ring road will be achieved from the mining compound, the ROM access road or the pit ramp.

18.1.3 Plant Roads

Plant site roads are internal roads providing access between the administration area and plant site facilities. These roads will be 3m wide depending upon traffic type and hierarchy. The roads will be constructed flush with the bulk earthworks pad to ensure that storm water sheet flow is achieved across the site, avoiding the need for deep surface drains and culvert crossings within this area.

Vehicular access throughout the processing compound will be via sealed roads. These will connect the varying terraced levels and will be sized such that both crane and delivery truck movements can be facilitated.

18.2 BUILDINGS

18.2.1 Plant Workshop and Warehouse

Plant workshop and warehouse will provide a maintenance area, material storage area, activated carbon storage, and an office for employees in workshop. This building with a 2T electric single beam crane will adopt steel structure with cladding.

18.2.2 Plant Reagent Store

Plant reagent store will be of cement block construction with an iron roof. This building will provide for a lime storage room, a NaCN storage room, another reagent storage room.

18.2.3 Laboratory

The laboratory has been designed in consultation with the laboratory provider and is located away from the process plant, for security reasons, off the TMF access road.

18.2.4 Main Camp

The camp is located to the west of processing plant area and is enclosed with a security brick wall. The camp consists of a guard room, recreation room, kitchen, dining room, accommodation buildings, swimming pool and football pitch.

18.2.4.1 Main Block

The Main block will provide kitchen and dining facilities, camp office, camp bar, and male and female ablutions.

18.2.4.2 Recreation Room and Medical Centre

Recreation room will provide for, a reading room, a training room, a fully equipped gym, male and female ablutions, and two male shower rooms & two women shower rooms. A medical centre will have a consultation room, storeroom, emergency room and a sick bay.

18.2.4.3 IT Room

The IT room caters for the server and communications infrastructure.

18.2.4.4 Accommodation Blocks and Guestrooms

There will be a number of blocks that can be configured as two or four rooms. All blocks will be concrete block construction with timber truss and an iron roof.

18.2.4.5 Construction Camp Buildings

The construction camp infrastructure will be used by the mining contractor.

18.2.4.6 Administration Office Area

Administration office area consists of Main Administration office and Technical Office.

The Administration offices will be of cement block construction with timber truss and an iron roof and will provide for 12 offices, a conference room, a coffee area, a server room, a print room, an archives room, an electric distribution room and male and female washing rooms. The Technical Centre has 5 offices and an open plan area for technical planning.

18.2.5 Contractor Mining Camp

The Contractor mining camp is located to the North of the Main Camp. This camp is for the accommodation of mining contract employees.

18.2.6 Contractor Workshop and Warehouse

This area associated with parking area is located on south-east of processing plant area with a fence, and this area has a gate for employees and vehicles to pass in and out. This block consists of contractor workshop and warehouse and associated infrastructure.

Contractor workshop and warehouse will provide a maintenance area suitable for large haul trucks and material storage area.

18.2.7 Contractor Offices

There will be a number of Contractor offices at the workshop, mainly to house maintenance staff. The operations staff will be located at the construction camp.

18.3 POWER SUPPLY

18.3.1 Main Power Supply

Electrical power will be generated on site by the use of power generators. A total of three 2.0MW CNG generating sets and five 1.6MW Diesel generating sets will be installed.

18.3.2 Medium Voltage Switchgear

All medium voltage switchboards, switchgear and motor starters shall be designed, manufactured, assembled and tested to IEC standard.

Equipment will generally be installed indoors, in switch rooms. All 10kV switchgear will be designed for process plant and infrastructure. 10kV Switchgear will generally be air insulated, with ABB vacuum switching devices. The 10kV switchgear will be fully withdrawable type complete with protection, metering and earthing facilities. The outer enclosure shall be rated to IP40 against contact and ingress of foreign bodies, and IP20 internally, between compartments.

In order to the ensure and reliability of the power supply, from the generator set to the 10kV distribution switchgear, the design adopts a double circuit power supply, one circuit on duty and another circuit on standby. Protection will be provided by microprocessor-based protection relays. Control power adopts DC220V/65Ah, and with 3kVA/220V UPS as the standby tripping supply.

18.3.3 Transformers

Distribution transformers will generally be pad mounted, oil immersed with natural cooling. Transformers will have cable boxes for both the high and low voltage terminals. Unless otherwise noted, distribution transformers shall be fitted with an offload tap changer with a range of ±5% of the nominal rating in increments of 2.5%.

Transformers will be designed using the following:

- Alarms and trips.
- Oil temperature alarm and trip.
- Buchholz alarm and trip.
- Pressure relief valve alarm and trip.

18.3.4 Low Voltage Distribution

The maximum transformer rating for the low voltage supply would be 1250KVA. Two sets of 1250 kVA 10/0.4kV transformer will feed a 400 V MCC that supplies power to process plant and infrastructure.

Under normal conditions, the two transformers operate separately and are standby when failure occurs.

One set of 630 kVA, 10/0.4kV transformer will feed the process plant camp, one set of 160 kVA 10/0.4kVtransformer will feed the mining contractor camp and two sets of 125kVA 10/0.4kV transformer will feed the WSD and TMF respectively.

18.3.5 Low Voltage Motor Control Centres (MCCs)

Motor Control Centres (MCCs) will be designed and constructed to IEC standard, with internal separation of functional units of Form 4b. Motor starter panels for packaged equipment may be Form 1 as appropriate.

Each incoming and outgoing circuit shall be provided with the facility to isolate that circuit and to lock it out by attaching lockout hasp and padlock; control circuits are not bound by this requirement. The isolating device shall be comprised of current circuit breaker, isolator, fused switch or disconnector switches.

MCCs shall have the following features:

- Be compartmentalized,
- Be of the non-withdrawable type with moulded case circuit breakers,
- Have magnetic contactors,
- Have an earth bus,
- Enclosures will be of the general-purpose type for indoor service (IP40) or of the weatherproof type for outdoor use (IP54), as required,
- Outgoing power and control wiring will be brought out to the terminals in the wireways; and
- Main buses will have a current-carrying capacity of 2500 A.

18.4 LIGHTING

Provision has been made for site area lighting and structure-mounted lighting to ensure safe working conditions. Lighting will also be installed to ensure that visual security monitoring can be conducted at all times in and around the process plant and associated infrastructure to maintain a safe environment.

18.5 SERVICES

18.5.1 Fuel Storage and Distribution

The process plant fuel storage tanks are designed to serve the daily needs of crushing, milling, leaching, elution heaters, carbon regeneration kiln, gold smelting, and auxiliary facilities. Fuel will be pumped from the fuel storage tanks in the fuel storage area and reticulated to the power station.

Diesel and CNG will be delivered to site by road tanker. Diesel and CNG storage will be housed in a bunded area to prevent leakage to the surrounding environment in the event of spillage. The fuel facility will be operated by a fuel supplier. Sufficient storage for 500,000 litres of diesel is planned.

18.5.2 Sewage Treatment

Effluent from all water fixtures in the plant, Administration office and the camp will drain to respective gravity sewage system. These systems will consist of graded PVC pipe systems connecting to all buildings within these areas.

There are 2 septic tanks, one is employed for processing plant, mining office, mining contractor and the other is for the camp. Site ablutions and other domestic wastewater will feed into a series of localized septic disposal systems. The Contractor shall make provision for suitable and sufficient sewage treatment during construction.

One sewerage treatment plant will be built at the northwest of the plant site, and then the sewage of 2 septic tanks will be collected in the sewage treatment plant, where the treated effluent will be pumped to the reclaim water pond. The treatment plant and the septic tanks' sludge will be transferred away by truck.

18.5.3 Waste Management

The Employer shall adopt an effective waste management practice and shall conduct environmental awareness training in waste handling for personnel. Waste management focal point shall be appointed. Rubbish bins and skips shall be provided at appropriate locations within the site, and illegal dumping of waste shall be prohibited.

18.5.4 Security Fencing

Adequate security fencing (either brick walls, or mesh type fencing approved by the Employer in advance of its installation) will be provided to complement the security risk for each section of the Facility.

The processing plant is designed surrounded with double row fencing. A 4 m High Block wall is used outside the plant and wire mesh fence is placed on the outside of this wall. This will be illuminated at regular intervals to allow perimeter inspection at night. Access to the processing plant compound will be via a single gatehouse located on the Site access road.

The administration office compound will be surrounded by a single brick wall. This compound will be located to the west of the processing plant on the opposite side of the road. Access to the processing plant compound will be via a single gatehouse located on the site access road.

The mining contractor compound will be surrounded by a single brick wall. This compound will be located to the south of the processing plant compound adjacent to the open cut operations. Access to the processing plant compound will be via a single gatehouse located on the site access road.

A 3 m brick wall is used to secure the camp. A fence will run along the road, to provide additional security.

An additional double wire mesh fence will encompass the emulsion storage and explosive magazine.

Entire Site perimeter to be surrounded by a wire mesh fence. All walls and fencing will be topped with razor wire.

18.6 WATER SUPPLY

18.6.1 Raw Water

Raw water for the plant will primarily be provided to site from the plant feed water dam located to the Northwest of the processing plant. And the raw water for the Facility will be pumped to the raw water pond from the dam. Overflow from the raw water pond will reported to the reclaim water pond and provide makeup water to losses contributed to entrained solution in deposited tailings and evaporation to maintain the process water volume.

18.6.2 Process Water

The processing plant reclaim water will be pumped from the TMF. Due to evaporation losses in the TMF, the process water system will be supplemented from time to time. The supplementary water will be obtained from the raw water system. The reclaim water pond will be constructed adjacent to the raw water pond so that the raw water overflows to reclaim water pond. With this arrangement the reclaim water pond can be kept full at all times to make up the balance.

Process water will be stored within a dedicated 2000 m^3 reclaim water pond and will be distributed around the processing plant by a dedicated process water pump.

Duty/standby process water pumps will be provided for the plant water supply. The process water from potable water treatment plant and sewerage treatment plant will be decanted to the reclaim water pond.

18.6.3 Potable Water

18.6.3.1 Daily Potable Water

Potable water for the process plant will be produced by treating raw water in the water treatment plant. The treatment plant will consist of clarification through flocculant addition, sand filtration, carbon filtration, micro filtration, and reverse osmosis. Potable water will be reticulated to distribution points in the plant as required. Additional ultra-violet sterilization units will be installed on outgoing potable water distribution headers.

18.6.3.2 Safety Shower and Eyewash Water

Safety water is supplied by potable water pump to a dedicated water tank for storage. The water tank will offer emergency shower and eye wash by separate safety water pumps.

18.6.3.3 Domestic Water

Water will be withdrawn from the raw water supply to feed the potable water treatment plant. Some of the raw water will be upgraded to potable water standards. The remainder will be treated as non-potable water for daily use, except drinking directly. Domestic uses within the treatment plant include washing faces, showing, toilet flushing, etc. The treatment process is physical filtration which includes sand filtration, carbon filtration and ozone sterilization.

The domestic water (non-potable water) in plant will be pumped to the processing plant, Administration office area, mining contractor area and the camp.

18.6.3.4 Fire Water

A fire water tank will store and supply fire water directly for the whole area which includes processing plant, Administration office area, mining contractor office area and the camp.

Fire water for the process plant will be sourced from sand filtered tank by gland seal water pump.

The fire water pumping system will contain:

An electric fire water delivery pump is designed to supply fire water at the required pressure and flow rate. This will start automatically if the main pressure drops to a pre-set pressure.

A diesel driven fire water pump that will start automatically in the event that power is not available for the electric fire water pump or that the electric pump fails to maintain pressure in the fire water system.

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

Moreover, there are 2 foam pumps (one duty and one standby) for exclusive use at the fuel storage area.

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

18.7 TAILINGS STORAGE FACILITY

Knight Piésold Limited (Knight Piésold) has undertaken the detailed design for the Tailings management facility (TMF) at the Segilola Gold Project in Nigeria. The work was undertaken under the instruction of SROL, to consider the design of the TMF based on a LoM plan between 2021 and 2026, with an estimated total tailings production of 4.04 Mt. An estimated settled dry density of 1.35 t/m is used for the design based on previous settling tests and a tailings solids specific gravity of 2.61 t/m. The Stage 1 North Embankment starter wall is designed to a maximum elevation of 353 mRL with a crest width of 12 m and slopes 1V:3H to provide approximately 12 months of tailings storage. This stage will be split into two phases. Phase 1 will give 6 months initial storage.

The starter wall is a zoned earthworks structure to be constructed with the following materials:

- Fine filter chimney and blanket drain identified as Zone 3,
- Compacted fine fill identified as Zone 1,
- Compacted bulk fill identified as Zone 2,
- HDPE liner placed on the upstream face of the starter wall.

The TMF will undergo a total of five stage raises using the downstream construction methodology where the crest moves farther downstream for each consecutive raise. The final dam geometry has a minimum crest elevation of 353 masl and a crest width of 12 m. The upstream and downstream slopes will be raised from the starter dam at an angle of 1V:3H. The raises will be constructed with the following materials:

- Zone 1 - Compacted fine fill,
- Zone 2 - Compacted bulk fill,
- Zone 3 - Fine filter chimney and blanket drain,
- Zone 4 - Transition material,
- Zone 5 - Bulk rockfill.

The starter wall and four TMF raises and elevations are:

- Stage 1 a starter facility – 335 mRL
- Stage 1b - 337.5 mRL
- Stage 2 - 340.0 mRL
- Stage 3 - 342.5 mRL
- Stage 4 - 345.0 mRL
- Stage 5 - 352.0 mRL l

The TMF Stage 1 starter wall detailed design is part of the North embankment. Two small saddle dams labelled as the West and South embankment will be constructed in subsequent stages to maintain containment up to elevation 352 mRL (Figure 18-1).

- Since this work was completed, various changes have been made to the design to incorporate the revised LOM projections for 4.05 Mt of tailings. The filling schedule based on a new final crest elevation of 353 mRL. This work is still ongoing.

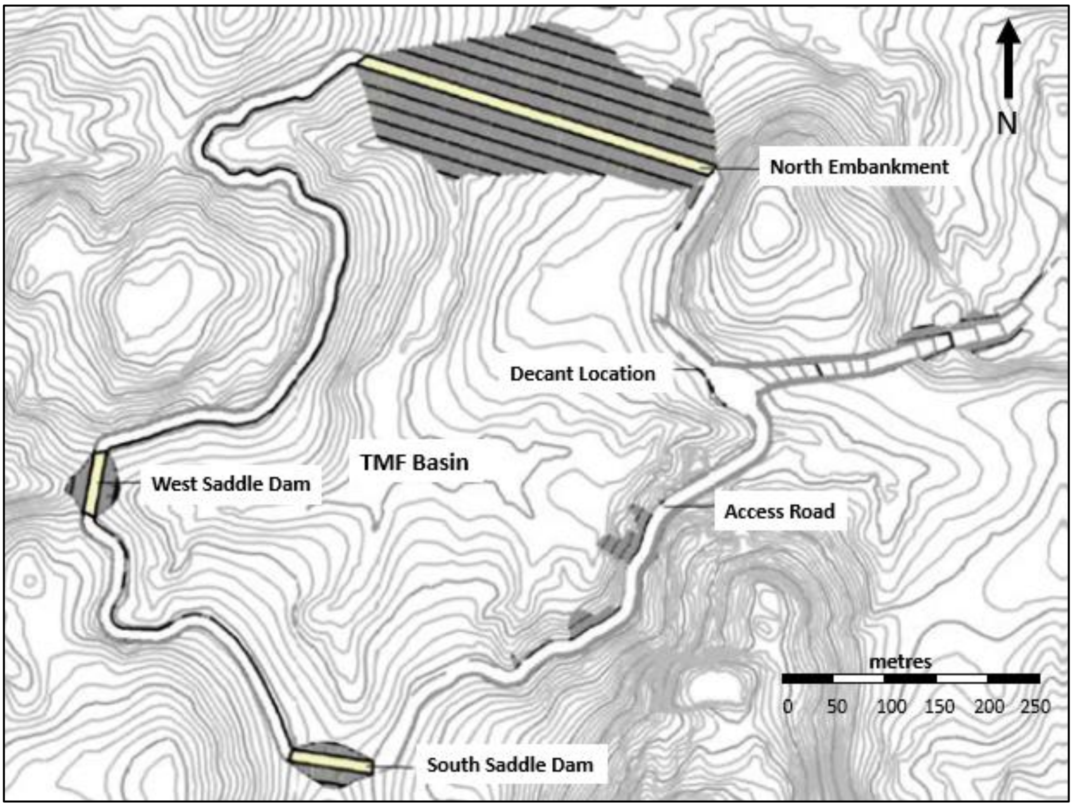


Figure 18-1. TMF Designed Footprint - Plan View

As of the 17th of March 2021, the TMF access road from the process plant and mine is estimated to be completed by the 20th of March 2021 (Figure 18-2).

Figure 18-2. TMF Access Road Construction (DZ0025) (17th of March 2021)

18.8 PROCESSING FACILITY

The processing facility has been modified from the original Definitive Feasibility Study (DFS) from a throughput of 650 ktpa to approximately 715 ktpa. The higher throughput has provided additional flexibility in the mining schedule.

The figure below shows the construction stage currently. The underground conveyors are being installed whilst the leach tanks are already in place.

Figure 18-3. Process Facility (16th March 2021)

(Lambert (SROL geologist) pointing towards leach tanks in the distance)

19 MARKET STUDIES AND CONTRACTS

This Item is not applicable for this report.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

An Environmental Impact Assessment (EIA) was carried out for the Project by independent consultants, in compliance with the EIA Act No. 86 of 1992. The Federal Ministry of Environment (FMENV) issued an Environmental Impact Assessment (EIA) certificate for "...exploration and mining of gold deposits at Iperindo/Odo, Osun State" on 22 March 2013. The EIA is a nationally approved document, and the certificate includes specific and general conditions requirements for Thor/SROL.

The recent baseline surveys are summarised in Table 20-1.

Table 20-1. Environment and Social Baseline Survey Updates

S/N	Surveys Conducted	Date Conducted/To be Conducted	Contractor
1	Environmental Impact Assessment (EIA)	2008/2012	Fugro Nigeria Limited
2	Baseline Ecology Study as part of Biodiversity Management Plan	First (Wet) Season Survey: June 2018 Second (Dry) Season Survey: January 2019	Fundamental Integrated Site Appraisal Services Limited (FISAS, former Fugro Nigeria Limited)
3	Ecology Surveys – Once annually - dry season	Annual Dry Season Survey March 2020 Annual Dry Season Survey February 2021	FISAS (Contract agreed on an annual basis)
3	Water (Surface and Ground Water) Quality undertaken monthly	August 2017 – to date	International Energy Services Limited (IESL)/ SGS/ Biogeochem Associates/ Fundamental Integrated Site Appraisal Services Limited (FISAS)
4	Noise Monitoring undertaken monthly	August 2019 –to date	Internal Fundamental Integrated Site Appraisal Services Limited (FISAS, former Fugro Nigeria Limited)
5	Air/Soil Quality undertaken monthly	Commenced in January 2020	Fundamental Integrated Site Appraisal Services Limited (FISAS, former Fugro Nigeria Limited)
6	Socio-economic conditions in host communities - one-off	Part of Livelihood Restoration Plan (Feb 2020 – December 2021)	Dynasty Global with oversight by DigbyWells Environmental and SROL

Monitoring of baseline environment factors has been ongoing since 2017 commencing with surface water then expanding to include groundwater, air quality and dust, noise, and soil quality. The findings mirror those outlined in the EIA and broadly conform with Nigerian standards.

Cultural heritage assessment was undertaken as part of the EIA. This information was validated as part of the socio-economic and asset surveys undertaken with conjunction with the Livelihood Restoration Plan June 2019 to March 2021. No tangible or intangible cultural heritage has been impacted by the project footprint.

Community development agreements (CDAs) providing project benefits to the local community (as required by the Mining and Mineral Regulations 2011 and EIA Certificate) have been completed and signed for the three communities closest to the mine. SROL has regular meetings with the community leaders so that they are made aware of the project activities and impacts and encouraged them to make inputs in appropriate mitigation measures.

Management plans under the umbrella of an Environment and Social Management System (ESMS) and Environmental Management Plan (EMP) have been prepared, or are in the process of being prepared, to address environment and/or social impacts during the remaining exploration, construction, operation, and closure phases.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

21.1.1 Initial Capital Cost

This process facility represents the bulk of the capital outlay at the Segilola Gold Project. The total capital outlay for the EPC component is $67.95M USD.

Table 21-1. Initial Capex Breakdown

Capex Items	Unit	Amount
EPC	[MUSD]	67.95
Duties & VAT	[MUSD]	4.00
Owners Costs	[MUSD]	10.50
Mining pre-production and pre-strip	[MUSD]	5.83
Initial Capital	**[MUSD]**	**88.28**
First Fill	[MUSD]	1.90
Up-front working capital	[MUSD]	2.24
Total Development Spend	**[MUSD]**	**92.42**

21.1.2 Sustaining Capital Cost

The sustaining capital costs at Segilola pertain to ensuring sufficient capacity in the Tailings Management Facility (TMF) is maintained. This includes an additional lift that will be undertaken in 2024. Other significant items include water management. Specific activities for this may include but are not limited to, sediment control structures, water diversion, and re-routing of minor watercourses away from the pit. The total sustaining capital costs for such items is estimated at $6.75M USD.

21.1.3 Mine Closure and Reclamation Costs

The mine closure costs have been incorporated into the overall economic model which are budgeted at $4.15M USD. The mine closure plan follows that specified in the Definitive Feasibility Study published in March-2019.

21.2 OPERATING COSTS

21.2.1 Mining Costs

The estimated costs for mining were based on the Estimated Incurred Contract (EIC) budget that was established for the original contract. The SINIC contract, as the principal contractor, have entered into a Schedule of Rates style contract based on agreed unit costs by activity. SINIC are the sub-contractors in charge of the Load & Haul operations.

For the purposes of the reserve update, the schedule of rates was applied to the revised Bill of Quantities (BOQ's). The estimated unit rate for the mining inclusive of fuel is $6.8 BCM or $2.70 per tonne of material.

Table 21-2. Estimated Incurred Contract (EIC) Budget

	Unit	Total	0 yr	1 yr	2 yr	3 yr	4 yr	5 yr
Ore	Mt's	4.04	-	0.61	0.76	0.79	1.88	-
Grade	gpt Au	4.00	-	4.7	4.1	2.9	4.2	-
Ounces	koz's Au	521	-	93	100	73	254	-
Ore	BCM's	1.5	-	0.2	0.3	0.3	0.7	-
Waste	BCM's	26.5	-	6.1	8.7	7.8	3.9	-
Total material	BCM's	28.1	-	6.4	8.9	8.1	4.6	-
Unit Cost	$/BCM	6.8	-	5.6	5.6	5.9	6.6	-
Unit Cost	$/t (ore)	47.3	-	64.3	73.9	69.1	18.4	-
Total Direct Cost	$ USD	171.1	5.6	35.8	50.0	48.2	30.5	1.0
Fuel Cost (Provided at Cost by SROL)	$ USD	20.3	-	3.5	6.4	6.2	4.1	-
Total Contract Value (excl. Perf. Bonus)	$ USD	191.4	5.6	39.4	56.4	54.4	34.6	1.0

An additional $2.40 per tonne for grade control is applied. Further costs related to SROL technical support bring the overall estimated cost to $53.48 per ore tonne.

21.2.2 Processing Costs

The processing costs are estimated to be $17.18 per tonne of ore milled.

Table 21-3. Processing Cost Unit Breakdown

	Total	Unit Rate (Ore + Waste tonne)	Unit Rate (Ore tonne)	Cash Cost ($oz Au)
Operating Consumables	27,493	0.39	6.86	54.79
Maintenance	5,392	0.08	1.35	10.75
Power	17,046	0.24	4.25	33.97
Laboratory	6,048	0.09	1.51	12.05
Infrastructure	1,780	0.03	0.44	3.55
SROL Personnel	11,077	0.16	2.76	22.07
Sub-total - Processing	**68,837**	**0.97**	**17.18**	**137.18**

21.2.3 General and Administration Costs

The general and administration (G&A) costs are comprised of office, vehicles, security, personnel, technical, environmental, and social responsibility, and travel.

Table 21-4. G&A Cost Breakdown

		Total	Unit Rate (Ore + Waste tonne)	Unit Rate (Ore tonne)	Cash Cost ($oz Au)
Country Office		5,992	0.08	1.50	11.94
Site Expenses		6,089	0.09	1.52	12.13
Vehicles		2,492	0.04	0.62	4.97
Security		2,862	0.04	0.71	5.70
Personnel		8,311	0.12	2.07	16.56
Technical		1,367	0.02	0.34	2.72
Environmental & Social Responsibility (ESR)		925	0.01	0.23	1.84
Travel		3,245	0.05	0.81	6.47
Sub-total - G&A		**31,282**	**0.44**	**7.81**	**62.34**
Freight and refining charges		3,365	0.05	0.84	6.71

21.2.4 Total Cash Costs

The Project has a life of mine Cash Cost of $663/oz and an All-in Sustaining Cash Cost (AISC) of $685/oz Au. Including capital costs of $180/oz Au, the total All-in Project Cost is $865/oz Au.

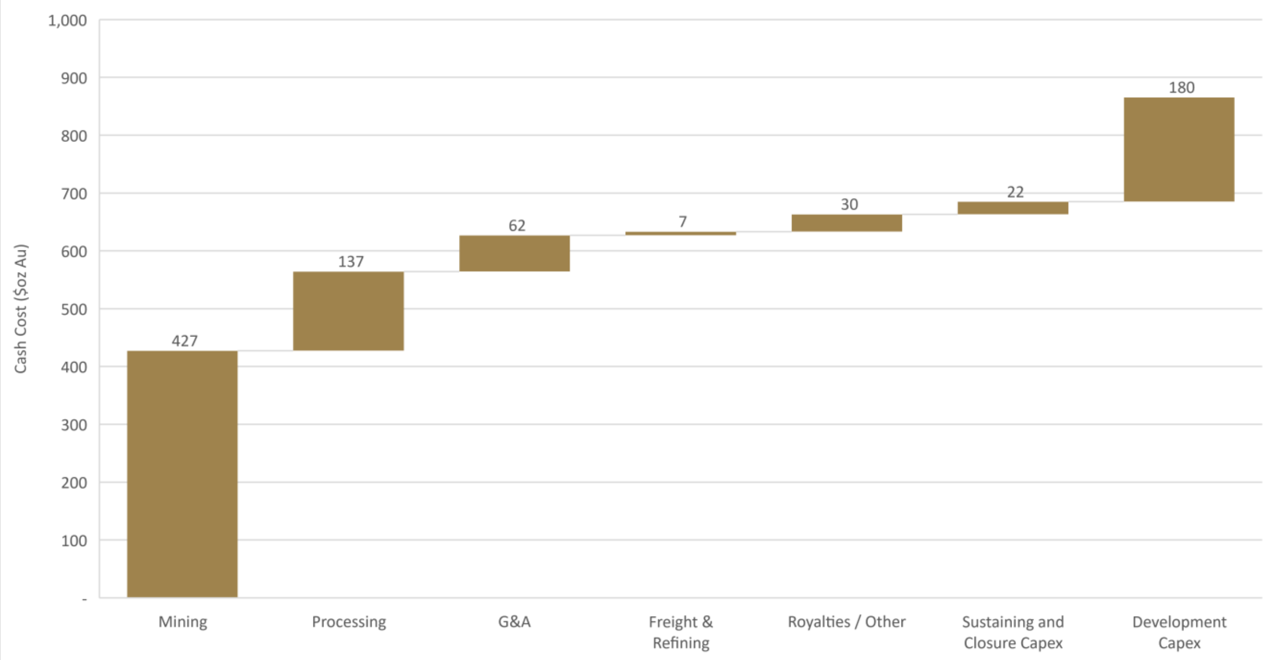


Figure 21-1. Cash-Cost Breakdown ($oz Au)

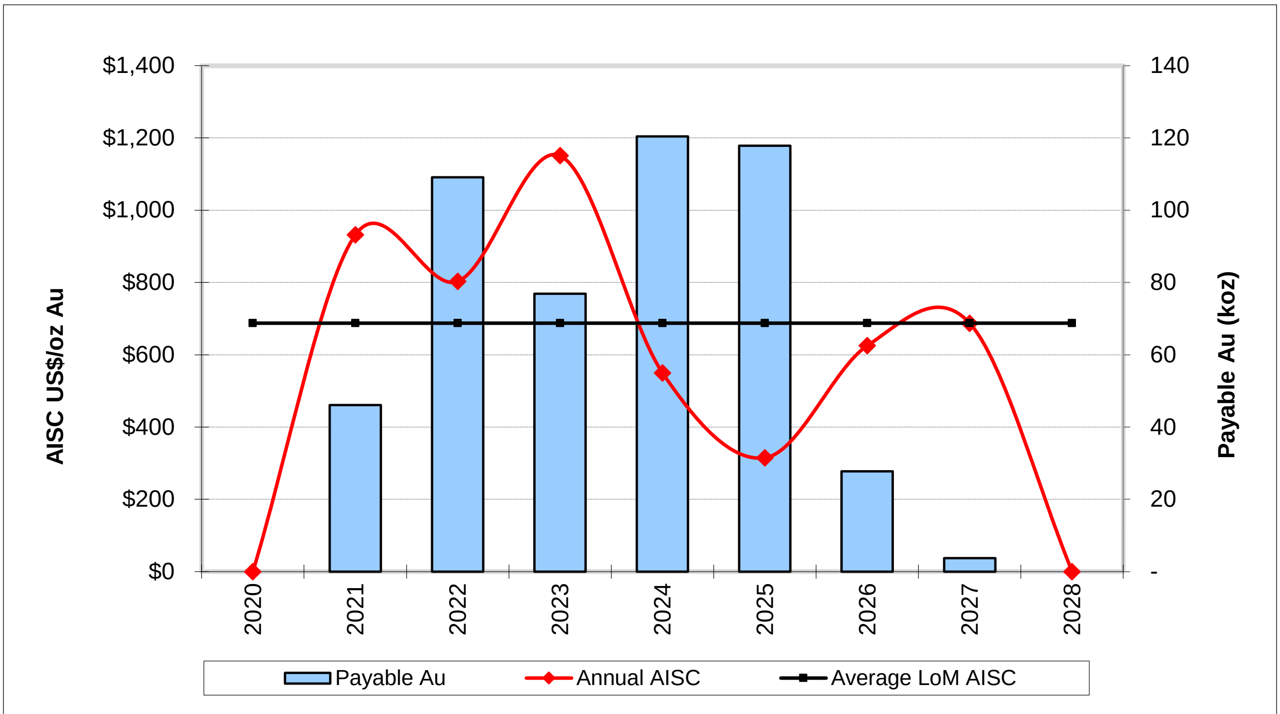


Figure 21-2. Annual LOM Cash-Cost

22 ECONOMIC ANALYSIS

22.1 ROYALTIES

Government royalties of Naira 5,400 (approximately US$14.89) are payable per ounce of gold. A private royalty of 3.0% of gross revenue is payable up to a total of US$7.5M. (DFS, RPA, Mar-2019)

22.2 REFINING

Doré payable factor at refinery is 99.9% Au. Doré transport and refining costs are estimated at $6.70 per ounce. (DFS, RPA, Mar-2019)

22.3 GOLD PRICE

An economic model was constructed using various macro assumptions. The forecast gold price of $1,600/oz Au for the life-of-mine was applied. This is believed to be somewhat conservative given the timing of when the operation will be expected to achieve first gold pour.

All figures assumed USD. Costs were therefore estimated in a USD exchange rate.

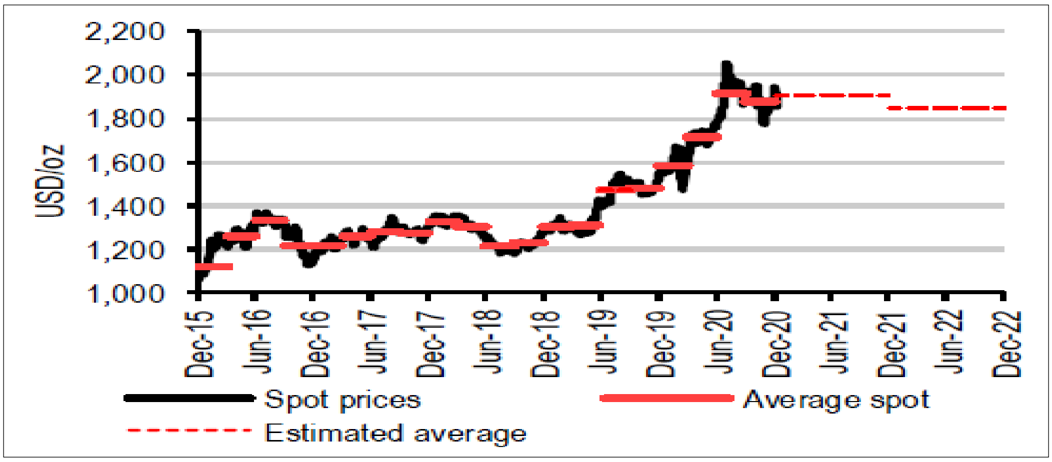


Figure 22-1. Gold Price Forecast (2022yr)

Source: Refinitiv Datastream, HSBC estimates

The long-term gold price compiled by HSBC was $1,600/oz USD.

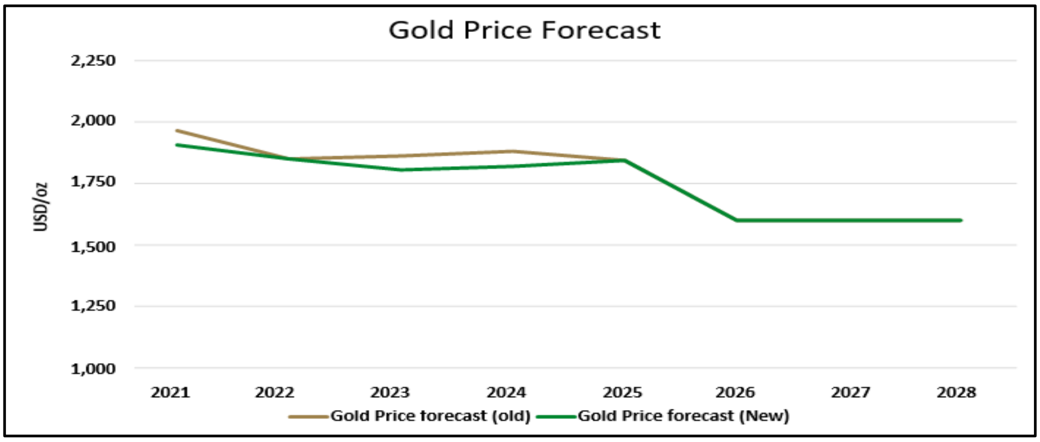


Figure 22-2. Long-Term Gold Price Forecast

Source: Refinitiv Datastream, HSBC estimates

22.4 CASHFLOW PROJECTION

The projected cashflow from the project based on the current reserve estimate and mine plan produces 502,000 ozs Au resulting in a total revenue of $803M. The life-of-mine Opex and Capex is $332.8M and $99.2M respectively. Dollars values are USD.

Table 22-1. Cashflow Projection

Item	Units	Sub Total	2020	2021	2022	2023	2024	2025	2026	2027	2028
Gold price	$/ oz		1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600
Ore mined	kt	4,007	-	500	622	780	1,381	725	-	-	-
Waste mined	kt	67,039	-	7,472	20,981	23,233	12,686	2,666	-	-	-
Total material mined	kt	71,046	-	7,972	21,603	24,013	14,067	3,392	-	-	-
Strip ratio	W:O	91	-	14.9	33.7	29.8	9.2	3.7	-	-	-
Ore milled	kt	4,007	-	323	715	715	715	715	715	109	-
Gold grade, milled	g/t	26	-	4.61	4.88	3.45	5.40	5.29	1.25	1.09	-
Contained gold, milled	koz	518	-	48	112	79	124	122	29	4	-
Recoverable gold, milled	koz	502	-	46	109	77	121	118	28	4	-
Average gold recovery	%	7	0	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	0
Payable gold sold	koz	502	0	46.4	108.8	76.9	120.4	117.9	27.8	3.7	-
Gold gross revenue	$M	803	-	74	174	123	193	189	44	6	-
Mining	$M	-214.3	-	-26.0	-60.1	-68.1	-44.6	-15.0	-0.4	-0.1	-
Processing	$M	-68.8	-	-6.7	-12.7	-12.7	-12.7	-12.7	-10.0	-1.3	-
G&A cost	$M	-31.3	-	-5.1	-5.4	-5.2	-5.2	-5.2	-4.9	-0.3	-
Infrastructure	$M	-	-	-	-	-	-	-	-	-	-
Dore freight/refining Cost	$M	-3.4	-	-0.3	-0.7	-0.5	-0.8	-0.8	-0.2	-0.0	-
Royalty	$M	-15.0	-	-2.9	-6.8	-1.2	-1.8	-1.8	-0.4	-0.1	-
Total cash costs after by-Product credits	$M	-332.8	-	-41.1	-85.9	-87.7	-65.1	-35.5	-15.9	-1.7	-
Operating margin	$M	470.1	-	33.1	88.2	35.3	127.5	153.1	28.6	4.3	-
Income tax	$M	-	-	-	-	-	-	-	-	-	-
Working capital	$M	-2.2	-0.9	-1.4	-	-	-	-	-	-	-
Operating cash flow	$M	467.9	-0.9	31.7	88.2	35.3	127.5	153.1	28.6	4.3	-
Initial capital	$M	-88.3	-32.3	-56.0	-	-	-	-	-	-	-
Sustaining capital	$M	-6.7	-	-1.1	-1.3	-1.3	-1.4	-0.5	-0.8	-0.2	-
Closure/reclamation capital	$M	-4.2	-	-	-	-	-	-1.2	-0.5	-	-2.5
Total capital	$M	-99.2	-32.3	-57.2	-1.3	-1.3	-1.4	-1.7	-1.3	-0.2	-2.5

22.5 LOM METRICS

The project operates commences production in 2021 through to 2028. The average LOM cash-cost and all-in sustaining cash costs (AISC) are $663.2 and $684.9 oz Au, respectively. The resultant net cashflow is $369M USD. With an NPV_5 of $311M USD and IRR of 88%.

Table 22-2. Economic Evaluation

	Units		2020	2021	2022	2023	2024	2025	2027	2028	2029
Annual discount factors	EOP 5%		1.00	1.00	0.95	0.91	0.86	0.82	0.78	0.75	0.71
a) Pre-Tax											
Free Cash Flow	$k	368.7	-33.1	-25.4	86.9	34.0	126.1	151.4	27.2	4.1	-2.5
Cumulative Free Cash Flow	$k		-33.1	-58.5	28.4	62.3	188.5	339.9	367.1	371.2	368.7
NPV @ 5%	$k	311.2	-33.1	-25.4	82.8	30.8	108.9	124.6	21.3	3.1	-1.8
Cumulative NPV	$k		-33.1	-58.5	24.2	55.1	164.0	288.6	309.9	313.0	311.2
IRR	%	88.4									
b) After-Tax											
Free Cash Flow	$0	368.7	-33.1	-25.4	86.9	34.0	126.1	151.4	27.2	4.1	-2.5
Cumulative Free Cash Flow	$0		-33.1	-58.5	28.4	62.3	188.5	339.9	367.1	371.2	368.7
NPV @ 5%	$0	311.2	-33.1	-25.4	82.8	30.8	108.9	124.6	21.3	3.1	-1.8
Cumulative NPV	$0		-33.1	-58.5	24.2	55.1	164.0	288.6	309.9	313.0	311.2
IRR	%	88.4									
Operating Metrics During Mining Phase											
Mine Life (Processing)	Years	6.2									
Maximum Daily Mining Rate	t/d mined	65,743	-	21,695	59,291	65,743	38,469	9,445	-	-	-
Maximum Daily Milling Rate	t/d milled	59,583	-	26,912	59,583	59,583	59,583	59,583	59,583	9,119	-
Average Mining Cost	$/t moved	3.0	-	3.3	2.8	2.8	3.2	4.4	-	-	-
Average Mining Cost	$/t milled	53.5	-	80.7	84.1	95.2	62.4	21.0	0.5	0.6	-
Processing Cost	$/t milled	17.2	-	20.6	17.8	17.8	17.8	17.8	14.0	11.7	-
G&A Cost	$/t milled	7.8	-	15.9	7.6	7.3	7.3	7.3	6.8	2.3	-
Infrastructure Cost	$/t milled	-	-	-	-	-	-	-	-	-	-
Offsite Costs	$/t milled	0.8	-	1.0	1.0	0.7	1.1	1.1	0.3	0.2	-
Royalty/Production Taxes	$/t milled	3.7	-	9.0	9.6	1.7	2.5	2.5	0.6	0.5	-
Total Cash Costs	$/t milled	83.0	-	127.2	120.1	122.7	91.0	49.6	22.2	15.3	-
Sales Metrics											
LOM Au Sales	koz	501.8	-	46	109	77	120	118	28	4	-
LOM Cash Cost	$0	332,772	-	41,086	85,866	87,697	65,083	35,485	15,885	1,669	-
LOM AISC	$0	343,671	-	42,230	87,167	89,017	66,524	37,155	17,205	1,850	2,522
LOM Cash Cost / oz Au	$/oz	663.2	-	886	789	1,141	541	301	572	449	-
LOM AISC / oz Au	$/oz	684.9	-	910	801	1,158	553	315	619	498	-
LOM Avg. Annual Au Sales	koz/yr	83.6									

22.6 SENSITIVITY ANALYSIS

The sensitivity analysis demonstrates that the project is robust with an NPV range from $222M USD to $400M USD based on a $1,400 oz Au to $1,800 oz Au gold price forecast. As demonstrated, the project is extremely sensitive to the price of gold. Given the advanced stage of construction and development, the Capex estimates are expected to be stable with no significant cost overruns.

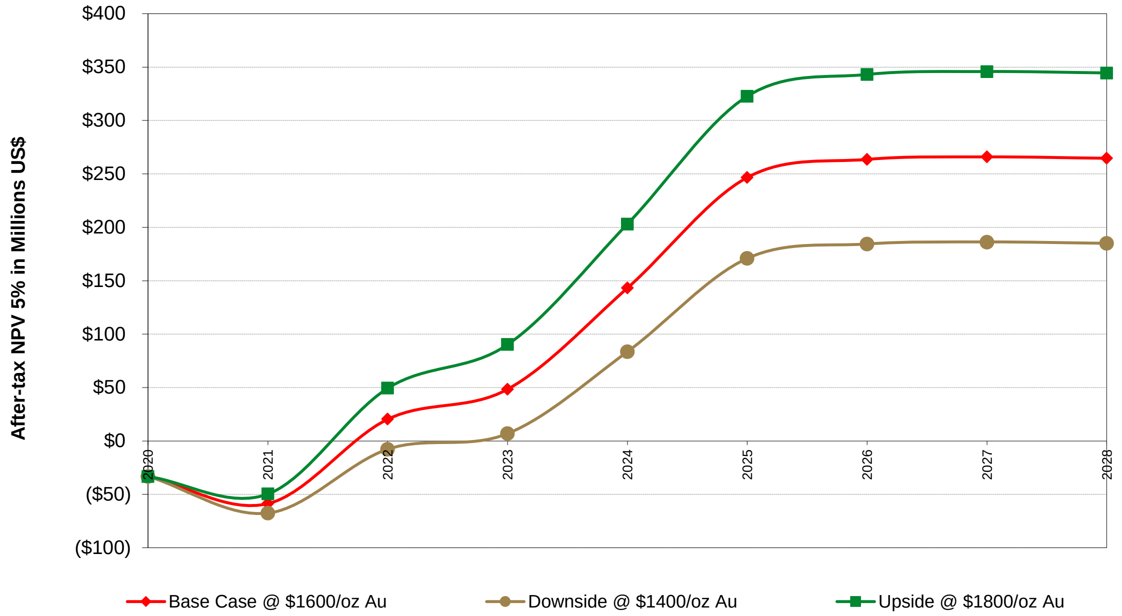


Figure 22-3. After-Tax Cumulative NPV Curve by Gold Price

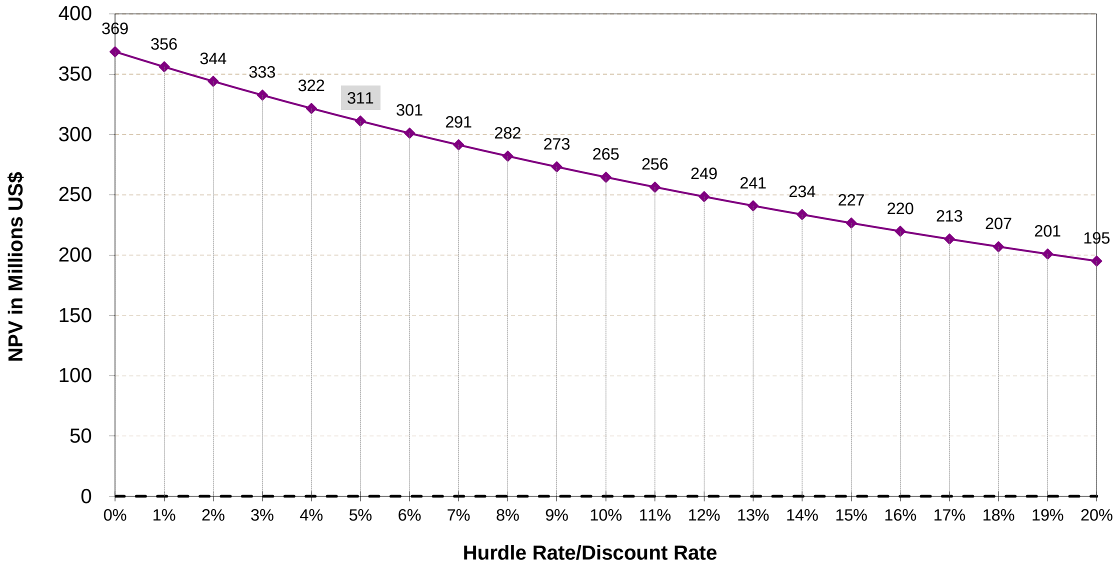


Figure 22-4. After-Tax NPV Profile by Discount Rate

23 ADJACENT PROPERTIES

Segilola is the largest known bedrock source of gold in the area. No bedrock exploration or development projects are in the near vicinity of the Segilola Project.

Significant alluvial-eluvial occurrences are known 15 km to 20 km west of the Project, particularly around Itagunmodi.

24 OTHER RELEVANT DATA AND INFORMATION

The authors are not aware of any other relevant data and information.

25 INTERPRETATION AND CONCLUSIONS

25.1 MINERAL RESOURCE AND MINERAL RESERVE

Segilola is an orogenic-style lode gold deposit within a regional scale shear zone. Primary gold mineralization commonly occurs in quartz veins within several lithologies. Sufficient exploration work has been completed to define a Mineral Resource as defined by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" (CIM, 2019).

Structural complexity associated with the cross-cutting dykes is identified in Lode 100. There may be additional unidentified dykes to the south as the southern lodes are less tightly drilled compared to the northern lode (Lode 100).

The offset between Lodes 100 and 200 is only defined by current drilling. Better definition of this offset (fault) should be a priority during grade control drilling and pit mapping.

The hanging wall lodes (Lode 400, 500) are less continuous than the main lodes within the Segilola mineralized zone. While these lodes add to the tonnes and grade available, they are not the main drivers for the pit shell. The lodes have been modelled to highlight the potential locations of structures carrying grade, areas that should be targeted with grade control drilling.

No significant Mineral Resource risks have been detected.

A review of the financial model supports the robustness of the reserve.

Minor design aspects that might pose operational risk (slope stability and rainy season pit dewatering) have been detailed in Item 16.

In view of the tight mining width of the various lodes in several areas, dilution in excess of the tight optimisation assumption of 12% is possible. This will have a marginal impact on production (head grade), and thus financials. This poses mainly a financial risk and less a Mineral Reserve risk.

The mining contract structure will require strict adherence to planned volumes. If these are not achieved, unit rates will increase, which places performance risk in the mining section with SROL. This poses a financial risk not a Mineral Reserve risk

No significant Mineral Reserve risks have been detected.

25.2 MINING

Slope design parameters especially on the eastern wall must be critically monitored, as batter angles are sub-parallel to foliation in, especially footwall contact areas.

The final bench height of 24 m on the final pit walls in the southern section of the pit could pose operational challenges. It should therefore be considered to halve these to 12 m heights, without flattening the overall angles – 12 m heights will enable all areas to have similar mining heights, berm positions and decrease toe loading (risk of failure).

In-pit dewatering holes will create more of a production nuisance than benefit over the relative short mine life. External dewatering holes in correct aquifers will add value.

The 6 month rainy season will have a significant impact in terms of water volumes collecting in the pit.

Current pit topography has overburden on high slopes, with the lowest part of the valley being on ore outcrop. Water ponding during the rainy season may hamper ore mining. Overburden pre-stripping will be critical to enable alternate water ponding and collection on waste areas, away from ore mining faces. The creation of sumps and installation of adequate pumping and piping infrastructure in the pit will be critical to ensure achievement of the mining volumes according to budget and plan.

The proximity of housing and other public infrastructure close to the mining activities will have to be approached with due care and supportive of a long-term relationship, as mining will impact on the communities with regards to safety, health, environment, and infrastructure. Issues may include the effects of blasting (fly rock, dust, noise, vibration), water quality for downstream users, lowering of groundwater levels potentially impacting water supply wells and boreholes in the surrounding community, and control of access to prevent ingress of people and livestock into areas where heavy equipment operates.

Based on the ore lode dimensions, dilution will remain a risk and achieving 12% or less dilution will require appropriate control and supervision over the ore mining operations

26 RECOMMENDATIONS

The offset between Lode 100 and 200 is only defined by current drilling. Better definition of this offset (fault) needs to be identified during grade control drilling and pit mapping.

The hanging wall lodes (Lodes 400, 500) should be targeted with grade control drilling.

The area between 10,800 to 11,000 mN has down dip gaps of 100 m between drill holes. These gaps in drilling coincide with the projected base of the pit. The surface above this target is steep country, and MA recommends early clearing (within the pit design) to enable additional drilling to optimally target the bottom of the pit. Approximately 6 holes for 1500 m are recommended.

Future grade control drilling should be optimised as grade continuity at Segilola is known to be erratic, which is expected in an orogenic gold deposit. MA suggests a 125 m strike length should be drilled to 12.5 m centres. The prime area for this optimisation experiment is north of the current tight drilling, 11775 mN, on Lode 100, (Iperindo Reef) where higher grades are predicted to extend to the surface.

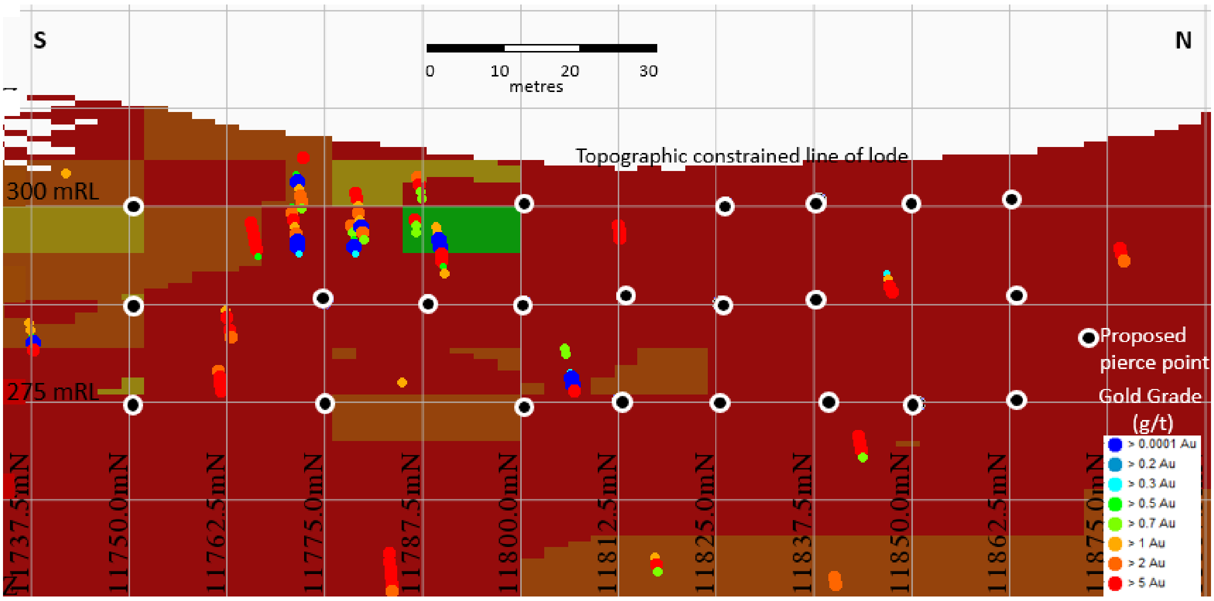


Figure 26-1. Suggested area for tight infill drilling to better define close spaced grade variance.

The close spaced drill program will be used to define better (higher resolution) variograms providing insights into the required drill spacing to define measured resources and optimise the grade control drill spacing required for accurate prediction of the feed grade on a daily or weekly basis.

Slope design parameters especially on the eastern wall must be critically monitored, as batter angles are sub-parallel to foliation, especially in footwall contact areas.

The final bench height of 24 m on the final pit walls in the southern section of the pit could pose operational challenges. It should therefore be considered to halve these to 12 m heights, without flattening the overall angles – 12 m heights will enable all areas to have similar mining heights, berm positions and decrease toe loading (risk of failure).

External dewatering holes rather than the use of in-pit dewatering will be of benefit.

Initial mining- and plant feed schedules should be reviewed to ensure realistic alignment.

For and on behalf of Mining Associates Pty Limited

IanTaylor BSc (Hons), G.Cert Geostats, AusIMM (CP), MAIG

Meiring. Burger M Sc (Geology), GDE (Mining Engineering), EIZ, SAIMM, SACNASP

Qualified Persons

Effective Date: 4 January 2021

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DATE AND SIGNATURE PAGE

This report titled "**INDEPENDENT TECHNICAL REPORT ON THE SEGILOLA GOLD PROPERTY ORUN PROVINCE, NIGERIA**

Effective date 4th January and dated 6th May 2021 was prepared and signed by the following authors:

Dated at Brisbane, Australia	Dated at George, South Africa
6 May 2021	6 May 2021
Ian Taylor	Meiring Burger
BSc (Hons), G.Cert Geostats, AusIMM (CP), MAIG.	M Sc (Geology), GDE (Mining Engineering), EIZ, SAIMM, SACNASP
Qualified Person	Qualified Person

CERTIFICATES OF QUALIFIED PERSONS

Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, M.AusIMM (CP)

I, Ian Andrew Taylor, B.Sc. (Hons), G.Cert. Geostats, do hereby certify:

- I am a Principal Geologist with Mining Associates with a business address at L6 445 Upper Edward Street Spring Hill Queensland 4004.
- This certificate applies to the technical report entitled "Independent Technical Report, Mineral Resource and Ore Reserve Estimate Segilola Gold Deposit Osun Province Nigeria" with the effective date of January 4, 2021 (the "Technical Report").
- I am a graduate of James Cook University (B.Sc. [Hons], 1993) and Edith Cowan University (Graduate Certificate in Geostatistics, 2013). I am a member and chartered professional in good standing of the Australasian Institute of Mining and Metallurgy (#110090). My relevant experience includes more than 25 years in the minerals industry. My work experience includes resource geology, production geology in open pit and underground mines, and exploration roles. I have worked more recently as a consulting geologist and have consulted primarily in relation to gold resource estimates including epithermal gold (high and low sulphur systems), orogenic gold, copper gold and gold-molybdenum (porphyries), skarn, VMS, and unconformity-related uranium resource projects in Australia, Indonesia, Papua New Guinea, Philippines, Fiji, Myanmar, Turkey, and Columbia.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I have not visited the Segilola Project property.
- I am independent of Thor Explorations Limited as defined by Section 1.5 of NI 43-101.
- I am responsible for Sections 1.1, 1.2, 1.3, 1.4, 1.8, 2.0, 3.0, 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0, 13.0, 14.0, 20.0, 23.0,24.0, 25.1, 26.0 and 27.0 of this Technical Report.
- I have had no prior involvement with the property that is the subject of the Technical Report.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 6th day of May 2021.

(Signed and Sealed) Ian Taylor

Ian Andrew Taylor, B.Sc. (Hons), G.Cert. Geostats, M.AusIMM (CP)

Principal Geologist

Mining Associates

Meiring Burger, (M Sc (Geology), GDE (Mining Engineering), EIZ, SAIMM, SACNASP

I, Albert Meiring Burger, (M Sc, GDE), do hereby certify:

- I am a Principal Consultant with Minexec Pty Ltd with a business address at 8 Plover Road, George, South Africa.
- This certificate applies to the technical report entitled "Independent Technical Report, Mineral Resource and Ore Reserve Estimate Segilola Gold Deposit Osun Province Nigeria" with the effective date of January 4, 2021 (the "Technical Report").
- I am a graduate of Free State University (B.Sc. [Hons] 1993), University of Johannesburg previously known as Rand Afrikaans University (M.Sc. (geology) 1995), North West University (B.Sc. (Hons) Environmental Management 1997) and University of the Witwatersrand (GDE Mining Engineering 2001).
- I am a member and chartered professional in good standing of the Engineering Institute of Zambia (#010802), South African Council for Natural Scientific Professions (#400006/97), South African Institute for Mining and Metallurgy (#707694). My relevant experience includes more than 25 years in the minerals industry. My work experience includes resource geology, production geology in open pit and underground mines, and exploration roles. Mine design, mining review and evaluation, mine planning and production and mine management roles in open pit and underground mines. I have worked more recently as a consulting mine engineer and have consulted primarily in relation to open pit copper and gold reserve estimates, mine valuations and mine due diligence studies in Burkina Faso, Zambia, Democratic Republic of the Congo, Botswana and Panama.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I visited the Segilola Project property between 7 and 10 April 2021.
- I am independent of Thor Explorations Limited as defined by Section 1.5 of NI 43-101.
- I have had no prior involvement with the property that is the subject of the Technical Report.
- I am responsible for Sections 1.5, 1.6, 1.7, 1.8, 15.0, 16.0, 17.0,18.0,19.0,21.0, 22.0, 24.0, 25.1, 25.2 and 26.0 of this Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for preparing in compliance with NI 43-101 and NI 43-101 F1.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 6th day of May 2021.

(Signed and Sealed) Meiring Burger

Albert Meiring Burger, M.Sc., GDE, EIZ, SAIMM, SACNASP

Principal Consultant

Minexec Pty Ltd (on behalf of: Mining Associates)

GLOSSARY OF TECHNICAL TERMS

This glossary comprises a general list of common technical terms that are typically used by geologists. The list has been edited to conform in general to actual usage in the body of this report. All units are metric units (SI units), except pounds (lb) and ounces (oz). However, the inclusion of a technical term in this glossary does not necessarily mean that it appears in the body of this report, and no imputation should be drawn. Investors should refer to more comprehensive dictionaries of geology in printed form or available on the internet for a complete glossary.

"Au"	chemical symbol for gold
"ARD"	Acid Rock Drainage
"BCM"	Bank Cubic Metre (i.e. in-situ volume)
"bench"	In mining, a bench is a narrow, strip of material cut into the side of an open-pit mine
"BMT"	Blast Movement Technologies, includes "BMI" Blast Movement Indicators
"bulk density"	The dry in-situ tonnage factor used to convert volumes to tonnage. Bulk density testwork is carried out on site and is relatively comprehensive, although samples of the more friable and broken portions of the mineralized zones are often unable to be measured with any degree of confidence, therefore caution is used when using the data.
"CIL"	Carbon in leach, the process of gold extraction in the plant
"cut-off grade"	The lowest grade value that is included in a resource statement. Must comply with JORC requirement 19 "*reasonable prospects for eventual economic extraction*" the lowest grade, or quality, of mineralized material that qualifies as economically mineable and available in a given deposit. May be defined based on economic evaluation, or on physical or chemical attributes that define an acceptable product specification.
"COG"	Break even cut of grade - Grade above which mineralisation is reported, see cut-off grade
"DGPS"	Differential Global Positioning System
"DFS"	Definitive feasibility Study
"diamond drilling, diamond core"	Rotary drilling technique using diamond set or impregnated bits, to cut a solid, continuous core sample of the rock. The core sample is retrieved to the surface, in a core barrel, by a wireline.
"Dollar" "$" "USD"	Dollars are quoted as US dollars
"down-hole survey"	Drillhole deviation as surveyed down-hole by using a conventional single-shot camera and readings taken at regular depth intervals, usually every 50 metres.
"drill-hole database"	The drilling, surveying, geological and analyses database is produced by qualified personnel and is compiled, validated and maintained in digital and hardcopy formats.

"flitch"	A proportion of the bench mined in one pass of the mining equipment
"g/t"	grams per tonne, equivalent to parts per million
"g/t Au"	grams of gold per tonne
"gold assay"	Gold analysis is carried out by an independent ISO17025 accredited laboratory by classical 'Screen Fire Assay' technique that involves sieving a 900-1,000 gram sample to 200 mesh (~75microns). The entire oversize and duplicate undersize fractions are fire assayed and the weighted average gold grade calculated. This is one of the most appropriate methods for determining gold content if there is a 'coarse gold' component to the mineralization.
"IRA"	Inter-ramp angle in open pit design
"ktpa"	Thousand tonnes per annum
"LOM"	Life of Mine
"m"	Scientific unit of length, metre, or meter
"m^3"	Unit of volume, a cubic metre
"M" "m"	Million, lower case when referring to dollar values
"mN"	Metres North
"mm"	Unit of length, millimetre. One thousandth of a metre
"micron (µ)"	Unit of length, one thousandth of a millimetre or one millionth of a metre.
"Mineral Resource"	A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics, and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories when reporting under CIM definitions and guidelines.
"MW"	Unit of Power, Megawatt (=million watts)
"NAF"	Non-Acid Forming
"NI43-101"	National Instrument 43-101 (Canadian reporting standards for mineral projects)
"OWA"	Overall wall angle, Angle from the upper crest to the toe of the slope at the pit bottom
"oz"	Troy ounce (= 31.103477 grams). koz thousand troy ounces. Moz = million troy ounces
"PAF"	Potential Acid Forming
"ppm"	A concentration of parts per million

<table>
<tr><td>“QAQC”</td><td>Quality Assurance/Quality Control. The procedures for sample collection, analysis and storage. Drill samples are despatched to ‘certified’ independent analytical laboratories for analyses. Blanks, Duplicates and Certified Reference Material samples should be included with each batch of drill samples as part of the Company’s QAQC program.</td></tr>
<tr><td>“RC”</td><td>Reverse Circulation drilling. A method of rotary drilling in which the sample is returned to the surface, using compressed air, inside the inner-tube of the drill-rod. A face-sampling hammer is used to penetrate the rock and provide crushed and pulverised sample to the surface without contamination.</td></tr>
<tr><td>“RL”</td><td>Reduced Level</td></tr>
<tr><td>“ROM”</td><td>Run of Mine, staging dump for mill feed material</td></tr>
<tr><td>“RQD”</td><td>Rock Quality Designation, RQD is a rough measure of the degree of jointing or fracture in a rock mass, measured as a percentage of the drill core in lengths of 10 cm or more, above 75% is good competent rock</td></tr>
<tr><td>“SMU”</td><td>Selective Mining Unit – The smallest block model block size that is considered practical for selective mining</td></tr>
<tr><td>“survey”</td><td>Comprehensive surveying of drillhole positions, topography, and other cadastral features is carried out by the Company’s surveyors using ‘total station’ instruments and independently verified on a regular basis. Locations are stored in both local drill grid and UTM coordinates.

The local grid is based on a two-point conversion,
<table>
<tr><th>Grid</th><th colspan="2">UTM WGS84 (31N)</th><th colspan="2">Local</th></tr>
<tr><th>Points</th><th>North</th><th>East</th><th>North</th><th>East</th></tr>
<tr><td>Point 1</td><td>820800.21</td><td>697073.05</td><td>0</td><td>2425</td></tr>
<tr><td>Point 2</td><td>834480</td><td>700740</td><td>14162.92</td><td>2425</td></tr>
</table>
Grid azimuths are rotated -15.4061 degrees from true north.</td></tr>
<tr><td>“SROL”</td><td>Segilola Resources Operating Limited, wholly owned subsidiary of Thor Exploration</td></tr>
<tr><td>“t”</td><td>Tonne (= 1 million grams)</td></tr>
<tr><td>“TMF”</td><td>Tails Management Facility</td></tr>
<tr><td>“UCS”</td><td>Uniaxial Compressive Strength, is the capacity of a material to withstand axially directed crushing force</td></tr>
<tr><td>“UTM”</td><td>Location data captured and located using the Universal Transverse Mercator format. Zone 31 North using World Geodetic System 1984 datum (WGS84)</td></tr>
<tr><td>“Whittle Shell”</td><td>An optimised pit shell using the common mining industry software package “Whittle”. The software uses the Lerch Grossmann algorithm.</td></tr>
<tr><td>“WRD”</td><td>Waste Rock Dump</td></tr>
</table>