THOR EXPLORATIONS LTD.

TECHNICAL REPORT ON THE SEGILOLA GOLD PROJECT FEASIBILITY STUDY, OSUN STATE, NIGERIA

NI 43-101 Report

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1 SUMMARY

EXECUTIVE SUMMARY

Roscoe Postle Associates Inc. (RPA) was retained by Thor Explorations Ltd (Thor) to prepare an independent Technical Report (the Technical Report) on the Segilola Gold Project (Segilola or the Project) located in the state of Osun, Nigeria. Auralia Mining Consulting Pty Ltd (Auralia), Norinco/CMGE, Yantai Orient Metallurgical Design and Research Institute (Yantai) and Knight Piésold Limited (KP) co-authored the Technical Report.

The purpose of this report is to disclose the results of a Definitive Feasibility Study (DFS) for an open pit and gold processing operation to exploit the Mineral Resources of the Segilola deposit. Additionally, a Preliminary Economic Assessment (PEA) based on Mineral Resources at depth for a proposed supplemental underground mine was completed. This Technical Report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

Thor is a Canadian mineral exploration company engaged in the acquisition, exploration and development of mineral properties located in Nigeria, Senegal, and Burkina Faso. Thor holds a 100% interest in the Segilola Gold Project located in Osun State Nigeria, a 70% interest in the Douta Gold Project located in south-eastern Senegal, and a 49% interest in the Bongui and Legue gold permits located in Houndé greenstone belt, south west Burkina Faso. Thor trades on the TSX Venture Exchange under the symbol "THX".

The Project is an advanced development project and is centred on the Segilola gold deposit which is currently the largest defined deposit in Nigeria. The deposit, which was formerly known as the Iperindo Reef project, was first discovered in 1945 as a result of alluvial workings in the area. In the 1980s, the Nigerian Mining Corporation (NMC) took an interest in the deposit and conducted a series of drill programmes. The Project was subsequently acquired from NMC by Tropical Mines Limited (TML), and after several ownership changes, in August 2016 by Thor.

During 2017, Thor undertook a 15-hole diamond drill programme to test the mineral potential beyond the defined Mineral Resource. Following the results of the exploration drilling, Thor undertook a Pre-Feasibility Study (PFS), which was prepared by Auralia in 2016/2017, and included an updated Mineral Resource estimate, effective 27 September

2017. The PFS proposed an open pit mining operation and a processing plant designed for a throughput of 500,000 tonnes per annum (tpa) of Run of Mine (ROM) ore. The outcome of the PFS was reported in a NI 43-101 Technical Report, with an effective date of 16 October 2017.

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 3.00 million tonnes (Mt) at an average grade of 4.20 g/t Au. The process plant is designed for a throughput of 650,000 tpa and gold recovery is projected to be 97%. The total gold recovered over the LOM is 393,000 oz Au, at an annual average rate of 66,000 oz Au.

In addition to the proposed open pit mine, there is potential to extract Mineral Resources below the open pit by underground methods. This has been evaluated in a separate PEA, completed by RPA in 2018/2019.

Table 1-1 summarises the open pit and underground Mineral Resource estimate, as at 1 December 2018. Mineral Resources are reported inclusive of Mineral Reserves. Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM (2014) definitions) were used for the estimate.

TABLE 1-1 SUMMARY OF MINERAL RESOURCE ESTIMATE AT 1 DECEMBER 2018
Thor Explorations Ltd. – Segilola Gold Project

Source	Classification	Tonnes (Mt)	Grade (g/t Au)	Contained Metal (000 oz Au)
Open Pit	Indicated	3.03	4.52	441
	Inferred	0.33	6.8	73
Underground	Indicated	0.09	9.39	28
	Inferred	0.35	7.9	90
Total	**Indicated**	**3.12**	**4.67**	**469**
	Inferred	**0.68**	**7.40**	**162**

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Open pit Mineral Resources are estimated by Auralia at a cut-off grade of 0.64 g/t Au and constrained within a pit optimisation shell using an Au price of $1,500/oz Au.
3. Underground Mineral Resources are estimated by RPA at a cut-off grade of 2.58 g/t Au and constrained within stope shapes using an Au price of $1,500/oz Au.
4. Open pit wireframes were defined by a nominal 0.50 g/t Au wireframing cut-off.
5. Underground wireframes were defined using a nominal 2.50 g/t Au wireframing cut-off and 2 m minimum mining width.
6. Open pit bulk density was interpolated using Inverse Distance Weighting squared.
7. Underground bulk density is 2.70 t/m^3.
8. High gold assays were capped to 40 g/t Au for open pit resources and 50 g/t Au for underground resources.
9. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability under the parameters used.
10. Numbers may not add due to rounding.

Neither Auralia or RPA are aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource and Mineral Reserve estimates.

Table 1-2 lists the Mineral Reserve estimate for the open pit mine, prepared by Auralia, as at 1 December 2018. The Mineral Reserve estimate for the open is based on Indicated Mineral Resources only. There are no Mineral Reserves defined for the proposed the underground mine.

TABLE 1-2 MINERAL RESERVES – 1 DECEMBER 2018
Thor Explorations Ltd. – Segilola Gold Project

	Category	Tonnage (Mt)	Grade (g/t Au)	Contained Metal (000 oz Au)
Open Pit	Probable	3.00	4.2	405
Total		**3.00**	**4.2**	**405**

Notes:
1. CIM (2014) definitions were followed for Mineral Reserves.
2. Mineral Reserves are estimated at a cut-off grade of 0.77 g/t Au.
3. Mineral Reserves are estimated using an average long-term gold price of $1,250 per ounce.
4. A minimum mining width of 10 m was used.
5. Mining dilution of 10% and mining recovery of 95% were applied
6. Numbers may not add due to rounding.

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

CONCLUSIONS

The following conclusions were made by Auralia, Norinco and RPA.

GEOLOGY AND MINERAL RESOURCES

- The Segilola Gold Project is an orogenic-style lode gold deposit which occurs within a regional-scale shear zone. The style of mineralisation is well understood.

- Drilling has delineated three steeply dipping lodes which form an elongate mineralised zone striking 010º and dipping 60º to 70º towards the west within a single robust shear zone, primarily in biotite gneiss. The known mineralised zone is approximately 2,000 m in strike length, between 70 m and 200 m in depth, and between two metres and 20 m in true thickness.

- The QPs consider that the drill hole data has been adequately validated with satisfactory quality control analysis. The quantity and quality of the geological, geotechnical, collar and downhole survey data is sufficient to support Mineral Resource estimation.

- In the opinion of the QPs, the analytical data is sufficiently reliable to support Mineral Resource estimation.

- The use of Ordinary Kriging (OK) to estimate the Mineral Resources is considered appropriate based on review of a number of factors, including the quantity and spacing of available data, the interpreted controls on mineralisation, and the style and geometry of mineralisation.

- The estimation was constrained with geological and mineralisation interpretations.

- At a cut-off grade of 0.64 g/t Au the Indicated Mineral Resources estimated for the open pit mine are 3.0 Mt at an average grade of 4.5 g/t Au containing 441 koz of contained gold. The Inferred Mineral Resources estimated for the proposed open pit mine are 0.3 Mt at an average grade of 6.8 g/t Au containing 73 koz of contained gold.

- At a cut-off grade of 2.50 g/t Au the Indicated Mineral Resources estimated for the underground mine are 0.09 Mt at an average grade of 9.4 g/t Au containing 28 koz of contained gold. The Inferred Mineral Resources estimated for the proposed underground mine are 0.3 Mt at an average grade of 7.9 g/t Au containing 90 koz of contained gold.

- The Mineral Resource estimate is consistent with the CIM (2014) definitions as incorporated by reference into NI 43-101.

MINING AND MINERAL RESERVES

- The Segilola deposit is amenable to conventional truck and shovel open pit mining methods and mining is proposed to be undertaken by a suitably qualified mining contractor.

- In the opinion of the QP, the data are sufficiently reliable to support a Mineral Reserve estimate.

- At a cut-off grade of 0.77 g/t Au, the Probable Mineral Reserves proposed to be mined by open pit methods at Segilola are estimated to be approximately 3.0 Mt at an average grade of 4.2 g/t Au, containing 405 koz Au.

- As contemplated in the DFS, the currently estimated Mineral Reserves for Segilola support an open pit mine life of approximately five and a half years at an average mill throughput of 625,000 tpa.

- The open pit operation will mine 49.0 Mt of waste over the life of mine, at an initial rate of approximately 1.4 million tonnes per month (Mtpm) for 25 months and a rate of 0.7 Mtpm for a further 19 months. The average LOM stripping ratio is 16.3.

- Mining ends in Month 46, however milling from a stockpile of ore continues for a further 14 months.

- LOM gold production averages approximately 66,000 oz Au per year.

MINERAL PROCESSING AND METALLURGICAL TESTING

- 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.

- The flowsheet selected is appropriate to the findings of the test work and the process design parameters are in agreement with the test work results.

- The QP considers the modelled recoveries for all ore sources and the process plant and engineering unit costs applied to the Mineral Resource and Ore Reserve process to be acceptable.

- The flowsheet is based on conventional, well-proven gold processing technology and equipment comprising crushing, grinding, gravity recovery and CIL.

- The grind size nominated of 106 microns is considered to be appropriate for optimal gold recovery versus energy input.

- The processing facility has been sized with a nameplate capacity of 650,000 tpa ore. This is adequate for the anticipated rate of mining ore from the open pit and provides additional operating flexibility and future expansion capacity with minimal additional capital expenditure.

INFRASTRUCTURE

- The mine site can be easily accessed by a sealed public road, which is linked to the national road system, however, there is insufficient power and water infrastructure to support the needs of the Project. Site power generation and local water sources will be provided for the Project.

- Planned Project infrastructure includes a processing plant, tailings management facility (TMF), power generation station, warehouses, workshops, reagent storage facility, administrative buildings, mining contractor facilities, accommodation camp and natural gas and diesel fuel storage tanks.

- There is adequate space available within the permitted land area for all of the planned Project infrastructure.

- Sufficient natural gas-fuelled generating capacity has been provided to meet the power needs of the Project processing plant and the other on-site infrastructure.

- Adequate provision has been made to meet the water supply needs of the Project through reclaimed water from the TMF and natural rainwater captured by the water supply dam (WSD). The TMF will supply between 40% and 100% of the monthly water demand. The WSD will make up any water deficit from the TMF, particularly during the dry season.

- Raw water from the WSD will be stored in the raw water pond. Raw water provides the majority of the make-up water for the process plant requirements.

- The TMF will be built 1.3 km southwest of the process plant and has sufficient capacity for the current LOM tailings.

ENVIRONMENTAL AND SOCIAL

- Thor has an in-country and local environmental and social team who are managing environmental and social aspects of the Project.

- The location of the Project is in a modified environment which has been logged for over 100 years and has remnants of gold mining from the 1950s.

- The mine area does not directly impact on the permanent dwellings of the nearest communities.

- Compensation for the loss of agricultural crops is being prepared via the Resettlement Action Plan (RAP) in line with Nigerian legislation and international standards. Compensation for damage to crops during exploration activities has been paid and documented.

- Thor has obtained two key environmental permits for the Project; the Environmental Impact Assessment (EIA) and the Certificate to Operate the Environment Protection and Rehabilitation Programme (EPRP).

- The EIA has been certified by the Federal Ministry of Environment, subject to standard conditions.

- Management plans have been prepared, or are in the process of preparation, to address environment and/or social impacts during exploration, construction, operation and closure phases.

- Thor has developed a stakeholder engagement plan for the consultation with identified key project stakeholders over the life of the Project.

ECONOMICS

- The Project as envisaged in the DFS, based on an open pit and gold processing operation and a Project operational life of approximately six years, demonstrates strong economics.

- The LOM total gold recovered is 393,000 oz Au, at an annual average rate of 66,000 oz of Au, generating a LOM revenue of $510.9 million using a gold price assumption of $1,300/oz Au.

- Pre-production initial capital is estimated to be $82.3 million.

- Capital and operating costs have been estimated to a level of accuracy commensurate with a DFS level of study.

- Average LOM All-in Sustaining Costs (AISC) are $638 per ounce.

- The Project is expected to benefit from being designated as a Pioneer Industry under Nigerian legislation, resulting in a tax shield for the entire duration of the current LOM.

- Undiscounted payback from the start of commercial production is approximately 1.4 years.

- The Project is most sensitive to gold grade, gold price, and gold recovery. Exchange rates have little impact as most costs and all revenue are in US dollars.

RISKS

Thor and RPA undertook a joint risk workshop to identify and assess the impact of the most salient Project related risks. The key risks identified were:

- Increases in costs for key materials and equipment when compared to the costs in the 2018 Feasibility Study estimates.

- Increases in labour and staff costs (particularly expatriate labour and staff) compared to base assumptions in the 2018 Feasibility Study estimates.

- Potential delays to the construction schedule due to delays encountered by the contractor in importing key fixed equipment (lead time as well as import delay at port).

- Delays to construction equipment delivery needed during the construction period.

- Influx of non-locals coming to and settling in surrounding villages or along the existing public road through mine site area.

- Community unrest and project delays caused by insufficient compensation for acquired land and crops prior to construction.

- Security and crime, major incident and petty pilfering of supplies and consumables and fuel.

RECOMMENDATIONS

RPA presents the following recommendations:

OVERALL

- RPA considers that the DFS demonstrates that the Project generates strong economics and Thor should consider advancing the implementation of the Project as envisaged.

- No significant environmental, social or permitting issues exist that would prevent the development of the Project as presently contemplated.

- Thor's Project implementation approach, based on the appointment of a qualified and experienced EPC contractor offering a fixed price lump sum EPC contract and the use of an experienced mining contractor, is a proven strategy in the global mining industry and elsewhere in Africa. Thor should advance the tender process and evaluation and engage in final negotiations for the EPC and mining contract with the selected contractors.

- The positive results of the PEA for the potential exploitation of the deeper resources by underground methods support the case for further exploration drilling of these resources and the preparation of a PFS.

GEOLOGY

- Generally, positive impacts on grades are observed with bigger sample sizes due to the nuggety nature, therefore, with the sample preparation now being undertaken in country, half-core sampling should be resumed in order to get a higher volume sample.

- Overall, there has been a positive impact on high grades through the use of Metallic Screen Fire Assay, although there are insufficient samples to define a Mineral Resource using them. Thor should undertake Metallic Screen Fire Assays on all samples >10 g/t Au going forward. When there is a sufficient number of analyses, these should be reviewed to determine whether the metallic screen fire assays are suitable to be used for future Mineral Resource estimates.

- The blank insertion rate should be increased to the same frequency as the standards and duplicates for future drilling campaigns. Blanks should also be inserted manually into areas of expected high-grade results to better test sample contamination during preparation.

- Re-instate Inter-laboratory checks (umpire analysis) for all future drilling programmes.

MINERAL RESOURCES

- Although known faults are limited these should be modelled in 3D and incorporated into the geological model to improve the usefulness of the model for future detailed mine planning.

- More bulk density samples should be collected, particularly in the underground resource domains to support the resource estimation.

- Optimised pit shells should be used to guide future drilling programmes to maximise the opportunities to upgrade mineral resource classification.

- A minimum mining width or minimum gram thickness should be applied to exclude any low-grade thin veins in future open pit Mineral Resource estimates.

GEOTECHNICAL STUDIES

- Further geotechnical investigations and analysis will be required as part of detailed mine design, and during open pit mining operations. The data and analysis from the twelve geotechnical drill holes provided a degree of confidence in the recommendations. However, only three holes extended into the east wall, so the analysis should be supplemented with additional triple-tube, oriented drill holes extended into the east wall of the pit and observational methods of design assessment during pit development. This will also provide essential data for the further engineering of the potential underground mine.

- Further hydrogeological studies should be completed to better understand the impact of groundwater on the open pit. This will also be essential input data for the further study of the underground mine.

- Geotechnical studies should be completed for the processing plant and related infrastructure, and for the locations of the waste dumps, if required.

MINING

- As part of the future detailed mine planning, the scheduling of waste mining and the quantity of ore being sent to the stockpiles should be further examined and optimised.

- The positive results of the PEA support further drilling and study work for the preparation of a pre-feasibility study.

PROCESSING

- Additional cyanide destruction test work is recommended to confirm the residence time requirements for effective cyanide destruction.

- A linear vibration screen and belt conveyor can be installed at the end of the SAG mill discharge port to return pebbles to the feed belt of SAG mill. Space for a pebble crusher should be reserved so that it can be installed at a later date if found to be required.

ECONOMIC ANALYSIS

A Cash Flow Projection has been generated from the LOM production schedule and capital and operating cost estimates, and is summarised in Table 1-5. All currency is in US dollars. A summary of the key criteria is provided below.

PRODUCTION

- The pre-production period starts in Q2 2019 with just over 17 months of construction and plant commissioning. The plant is anticipated to be ready for commercial production in September 2020.
- Mining starts in June 2020 with one month of pre-stripping and continues for 46 months at a maximum rate of 48,000 tpd mined and an average strip ratio of 16.3:1.
- Processing starts in July 2020 and operates for 58 months at an average ore throughput of 625,000 tpa or approximately 1,700 tpd during ROM operations.
- LOM ore production is 3.00 Mt at a grade of 4.20 g/t Au, with a constant gold recovery of 97%.

REVENUE

Metal recovery and sales over the LOM total 393,000 oz Au and average 66,000 oz Au per year. The metal price applied was $1,300 per troy ounce Au.

COSTS

Project pre-production initial capital totals $82.3 million. Due to the short mine life, no sustaining capital is included.

All maintenance costs are included in operating costs.

The average LOM unit operating costs over the mine life is $77.51/t milled. The initial capital currency split is expected to be 90% US$, 10% GBP, while the operating cost split is expected to be 90% US$, 10% NGN.

The following foreign currency exchange rates were used

- US$: GBP – 0.76 per GBP
- US$: NGN – 364.5 NGN per US$
- US$: EUR – 0.88 per EUR

ROYALTIES

Royalties total $13.4 million, or $4.45/t milled, over the LOM as shown in Table 1-3.

TABLE 1-3 ROYALTIES
Thor Explorations Ltd. – Segilola Gold Project

Item	$ 000s
Gov't – $14.89 per payable ounce gold	5,852
Private – 2.5% Gross Revenue (capped at $7.5M)	7,500
Total	**13,352**

INCOME TAXES, WORKING CAPITAL, AND CLOSURE COSTS

Income taxes/contributions, upfront working capital, and reclamation/closure costs are shown in Table 1-4. Withholding taxes on corporate dividends and interest payments are not incorporated into the Project economic analysis.

TABLE 1-4 INCOME TAXES, WORKING CAPITAL, AND OTHER
Thor Explorations Ltd. – Segilola Gold Project

Item	$ 000s
Corp. Income Taxes Paid	None *
Upfront Working Capital (Credit)	(653)
Mine/Plant Closure/Reclamation	2,172
Infrastructure Reclamation (in opex)	968
Salvage Value	0
Total	**2,487**

CORPORATE INCOME TAX

The Project economic analysis incorporates a five-year statutory income tax holiday from production start-up in July 2020, extended for an additional two years so the Project does not generate tax payable liabilities during the LOM.

In accordance with the mining industry pioneer status in Nigeria, Thor expects to receive a five-year exemption from customs duties, levies, and taxes including income tax. However, to estimate the risk of having tax payable liabilities due to the government, a proforma income tax calculation is included in the analysis based on general assumptions. Deductions from income for the purpose of estimating income subject to tax include the following items:

- Operating expenses are deducted 100% in year incurred.
- Depreciation is done with Units of Production (UoP) method.

- Net operating losses are carried forward indefinitely but not be used for prior tax years.
- No stockpile adjustments to operating deductions are included.
- Income tax rate of 30% applied to taxable income.

The LOM income tax payable liability to the Project if the tax holiday shield was not applied is estimated to be approximately $55 million with an effective tax rate of 21%.

UPFRONT WORKING CAPITAL

RPA proforma working capital assumptions include five days sales outstanding for accounts receivable, 14 days payable outstanding for labour, and 30 days payable outstanding for supplies, and 3% of cumulative annual balance of property, plant, and equipment (PP&E) for consumable inventories. Plant first fill and critical spares are included in the initial capital estimate.

With relatively small accounts receivable and inventories offset by a much larger accounts payable, an upfront working capital credit of $641,000 is generated in the analysis from start of construction in April 2019 through the first month of commercial production in September 2020. However, all working capital adjustments are recaptured at the end of mine life and so net to zero over the LOM.

RECLAMATION/CLOSURE COSTS

Reclamation/closure costs of the mining and mill areas are estimated at $2.175 million over five years starting in the first year after cessation of processing activities. In order to avoid starting the final reclamation work during the wet season (between May and October), the reclamation/final closure costs are assumed to start in January 2026, approximately eight months after the final processing of ore in June 2025, and will be complete in December 2026.

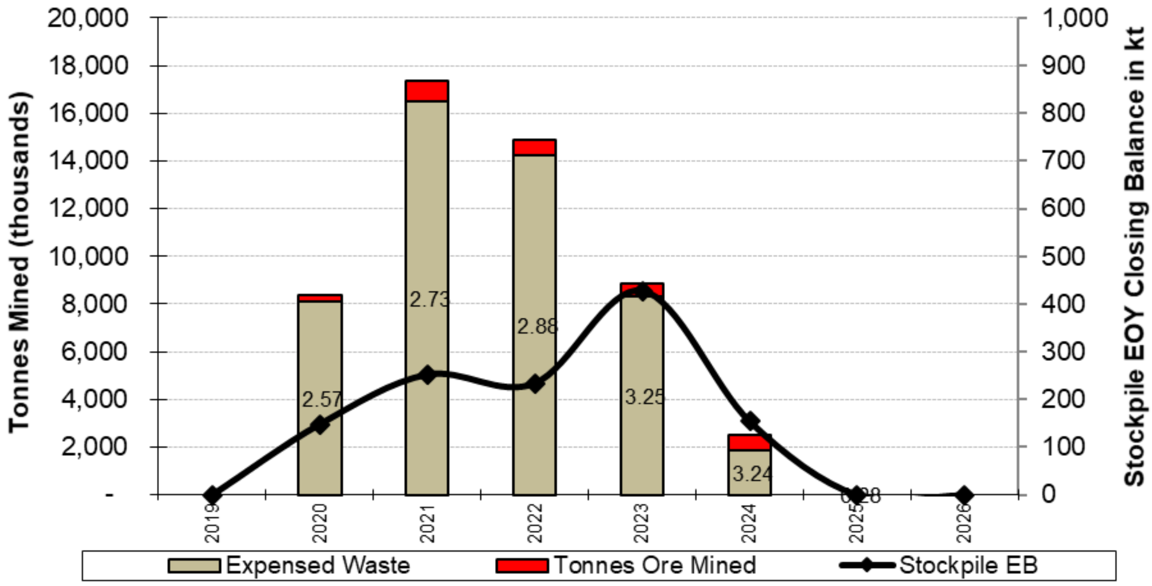
An additional $968,000 in post-closure reclamation costs for other infrastructure areas including the TMF ($825,000), the water storage dam ($110,000), and water management items ($33,000) are estimated for the Project. These costs have been included as operating expenses as part of the infrastructure cost.

No salvage credits were included in the economic evaluation.

CASH FLOW ANALYSIS

The LOM plan for the Project results in an average annual ore production of 625,000 tpa and includes significant variations in the ore and waste mining schedule and head grades over its planned five-year life. The base case includes some stockpiling of higher-grade ore in the early stages of mining, which will be processed later in the mine life. These variations are shown in Figures 1-1 and 1-2 and the resulting impact on the pre-tax free cash flow profile is shown in Figure 1-3.

FIGURE 1-1 MINE PRODUCTION PROFILE



Note: Numbers in chart are mining operating costs in $/t moved

FIGURE 1-2 MINED ORE VERSUS MILL PRODUCTION PROFILE

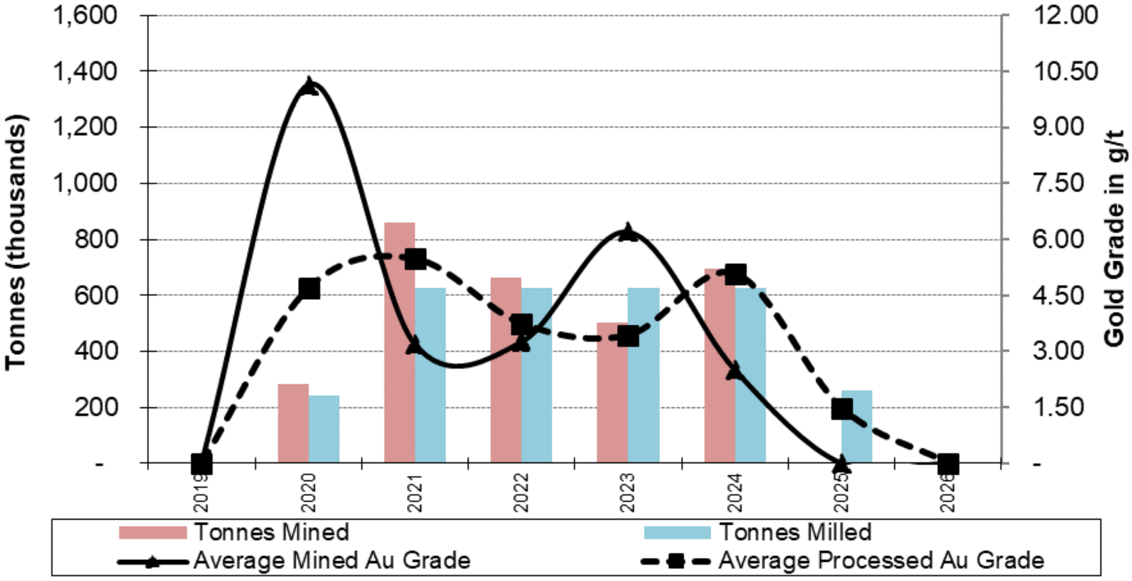


FIGURE 1-3 PROJECT PRE-TAX METRICS SUMMARY

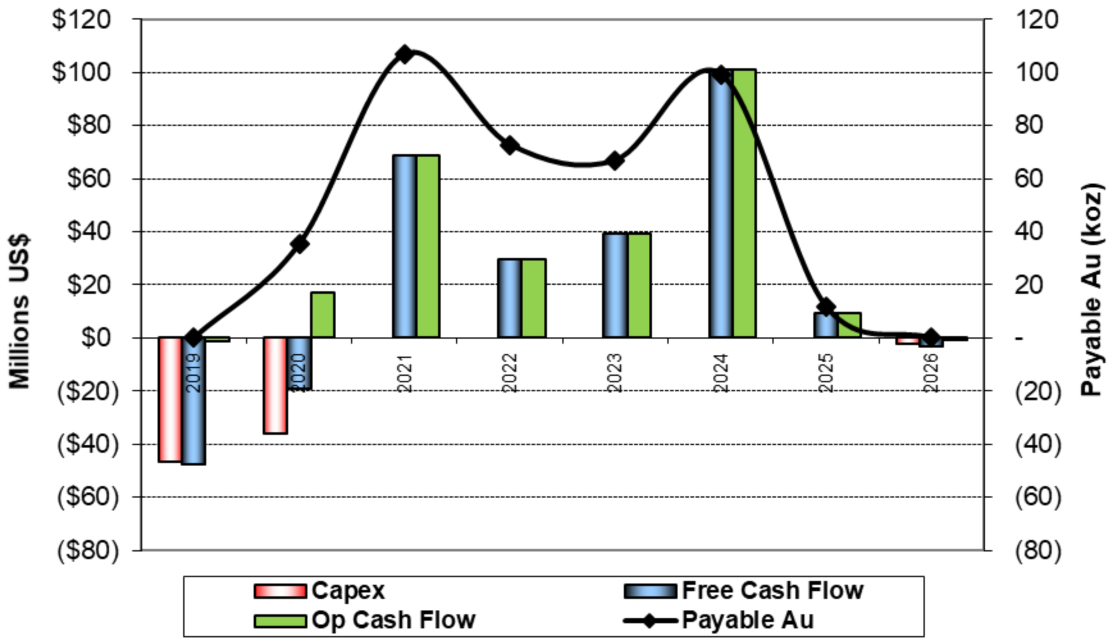


Table 1-5 shows the LOM total metrics for the Project as currently designed. The economic model was constructed on a 100% equity basis with end of period discounting. Whereas the data inputs and schedule are in a monthly format, the results are presented in both quarterly and annual formats with official metrics based on the annual format results.

TABLE 1-5 INDICATIVE PROJECT ECONOMICS
Thor Explorations Ltd. – Segilola Gold Project

Item	$ 000s
Gold Price ($/oz)	$1,300
Payable Gold (oz)	393,000
Total Gross Revenue	**510,900**
Mining Cost	(153,000)
Process Cost	(55,800)
G & A Cost	(17,300)
Infrastructure/Closure Cost	(6,500)
Site Operating Costs	**(232,700)**
Offsite Costs	(2,600)
NSR Royalty	(13,400)
Total Cash Costs	**(248,700)**
Operating Margin (EBITDA)	**262,300**
Operating Margin %	51%
Taxes	0
Working Capital*	(0)
Operating Cash Flow	**262,300**
Initial Capital	(82,300)
Sustaining Capital	0
Closure/Reclamation Capital	(2,200)
Total Capital	**(84,500)**
Pre-tax/Tax Shielded Free Cash Flow	**177,800**
Pre-tax/Tax Shielded NPV @ 5%	**138,000**
Pre-tax/Tax Shielded IRR	**50.5%**
Pre-tax/Tax Shielded Payback Period (Years)	**1.4**

*Note: An upfront working capital credit of $641,000 is generated in period April 2019 through September 2020. All working capital adjustments are recaptured at the end of mine life and so net to zero over LOM.

On a pre-tax (or tax holiday shielded) basis, the undiscounted cash flow totals $177.8 million over the mine life with a Net Present Value (NPV) at 5% of $138 million and the Internal Rate of Return (IRR) of 50.5%. The undiscounted payback from start of commercial production is approximately 1.4 years.

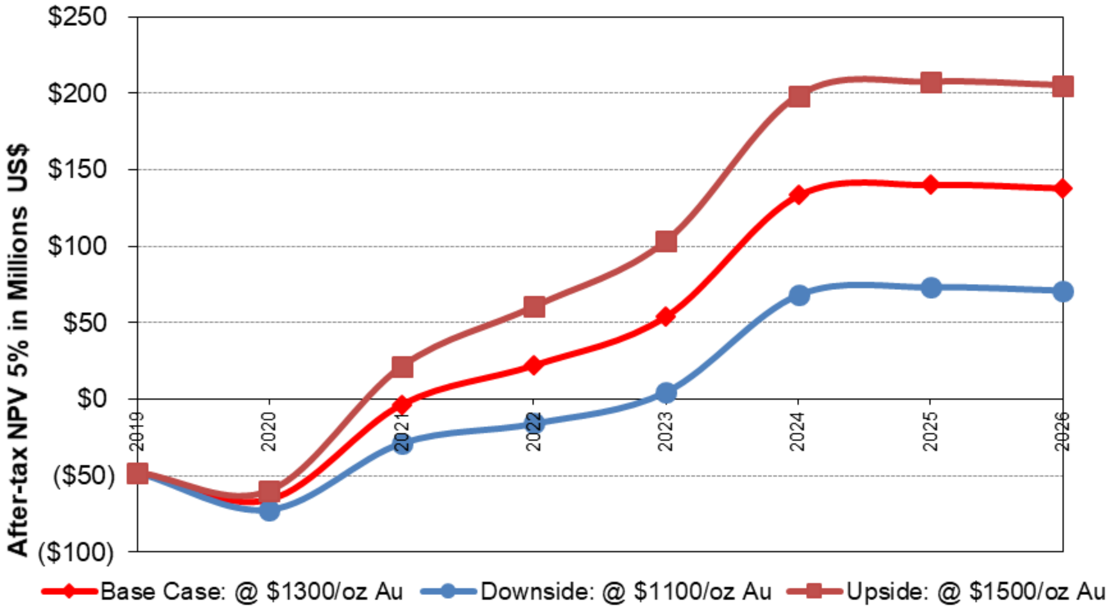
RPA also calculated the Project metrics on an after-tax basis using RPA generalized tax assumptions and without the statutory tax holiday shield to test the impact of the tax holiday on overall Project economics. The metrics included undiscounted cash flow of $123 million over the mine life with an NPV at 5% of $93 million and an IRR of 37.9%.

Figure 1-4 shows the cumulative NPV curves for the Project at a 5% discount rate at three different gold price points listed below:

- Base Case: $1,300 per ounce

- Upside: $1,500 per ounce
- Downside: $1,100 per ounce

FIGURE 1-4 PROJECT PRE-TAX CUMULATIVE NPV CURVES



Based on a pre-tax NPV at 5%, the breakeven gold price is estimated to be $887 per ounce. Each curve has a flatter slope in the period 2021 to 2022 that becomes steeper with increasing gold price.

Average LOM All-in Sustaining Costs (AISC) are $638 per ounce. The LOM cash and sustaining costs are shown in Table 1-6.

TABLE 1-6 ALL-IN SUSTAINING COSTS
Thor Explorations Ltd. – Segilola Gold Project

Item	$ 000s	$/oz Au
Contractor Mining	153,000	389
Thor Operations	55,800	142
Fixed G&A	17,300	44
Infrastructure/Closure	6,500	17
Subtotal Site Costs	**232,700**	**592**
Offsite Treatment	2,600	7
Total Operating Costs	**235,300**	**599**
Royalty	13,400	34
Total Cash Costs	**248,700**	**633**
Sustaining Capital Cost	0	0
Closure/Reclamation Capital	2,200	6

Item	$ 000s	$/oz Au
Corporate G&A	0	0
Off-Mine Exploration	0	0
Total Sustaining Costs	**2,200**	**6**
Total All-in Sustaining Costs	**250,800**	**638**
LOM Au Sold (oz)		393,000
LOM Average Au Sales per Year (oz)		66,000

SENSITIVITY ANALYSIS

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities:

- Head grade
- Gold recovery
- Gold price
- Exchange rates
- Operating costs
- Capital costs
- Discount rates

Pre-tax NPV and IRR sensitivities over the base case have been calculated for -20% to +20% variations for metal-related categories and exchange rates.

For gold recovery, due to the high overall recovery factor of 97%, the sensitivities have been calculated from -20% to +2.5%. For operating costs and capital costs, the pre-tax sensitivities over the base case have been calculated at -10% to +10% variation.

The sensitivities are shown in Table 1-7 and in Figures 1-5 and 1-6, respectively. Note that in this case, the pre-tax sensitivities are the same with the LOM tax holiday assumption in the economic model.

The analysis shows that the Project is most sensitive to gold grade, price, and recovery. Capital and operating costs have less effect than the metal-related categories mainly due to the small size and short life of the operation. Exchange rates have very little impact with most costs and all revenue being expended in US dollars.

TABLE 1-7 PRE-TAX SENSITIVITY ANALYSIS
Thor Explorations Ltd. – Segilola Gold Project

Factor Change	Head Grade (g/t Au)	NPV at 5% ($ 000s)	IRR (%)
0.80	3.36	$52,500	23.4%
0.90	3.78	$95,200	37.2%
1.00	**4.20**	**$138,000**	**50.5%**
1.10	4.62	$180,800	63.3%
1.20	5.04	$223,600	75.7%

Factor Change	Recovery (% Au)	NPV at 5% ($ 000s)	IRR (%)
0.80	77.60	$52,500	23.4%
0.90	87.30	$95,200	37.2%
1.00	**97.00**	**$138,000**	**50.5%**
1.03	99.43	$148,700	53.7%
1.03	99.43	$148,700	53.7%

Factor Change	Metal Price ($/oz Au)	NPV at 5% ($ 000s)	IRR (%)
0.80	$1,040	$51,100	22.9%
0.90	$1,170	$94,500	37.0%
1.00	**$1,300**	**$138,000**	**50.5%**
1.10	$1,430	$181,500	63.5%
1.20	$1,560	$225,000	76.1%

Factor Change	Exchange Rate (NGN: US$)	NPV at 5% ($ 000s)	IRR (%)
0.80	292	$137,100	50.2%
0.90	328	$137,600	50.3%
1.00	**365**	**$138,000**	**50.5%**
1.10	401	$138,500	50.6%
1.20	437	$138,900	50.7%

Factor Change	Operating Costs ($ 000s)	NPV at 5% ($ 000s)	IRR (%)
0.90	$225,000	$137,100	52.3%
0.95	$229,000	$137,600	51.4%
1.00	**$233,000**	**$138,000**	**50.5%**
1.05	$237,000	$138,500	49.5%
1.10	$241,000	$138,900	48.6%

Factor Change	Capital Costs ($ 000s)	NPV at 5% ($ 000s)	IRR (%)
0.90	$76,300	$146,100	57.1%
0.95	$80,400	$142,100	53.6%
1.00	**$84,500**	**$138,000**	**50.5%**
1.05	$88,600	$134,000	47.5%
1.10	$92,700	$130,000	44.8%

FIGURE 1-5 PRE-TAX NPV 5% SENSITIVITY ANALYSIS

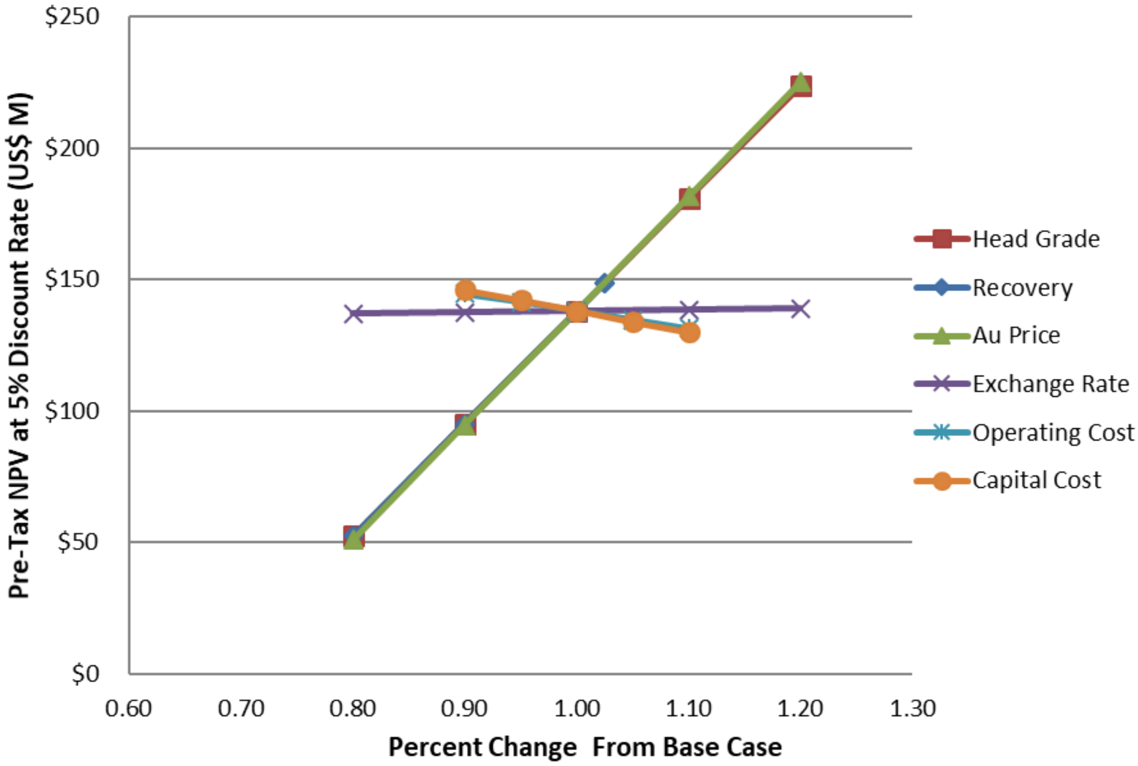
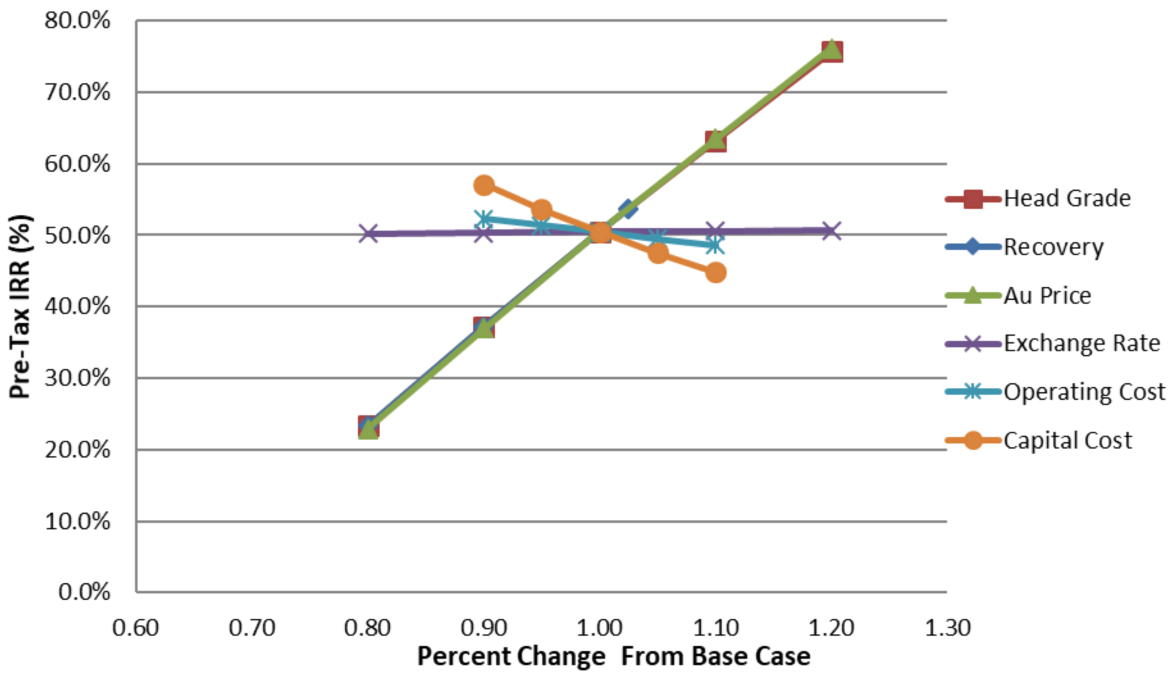
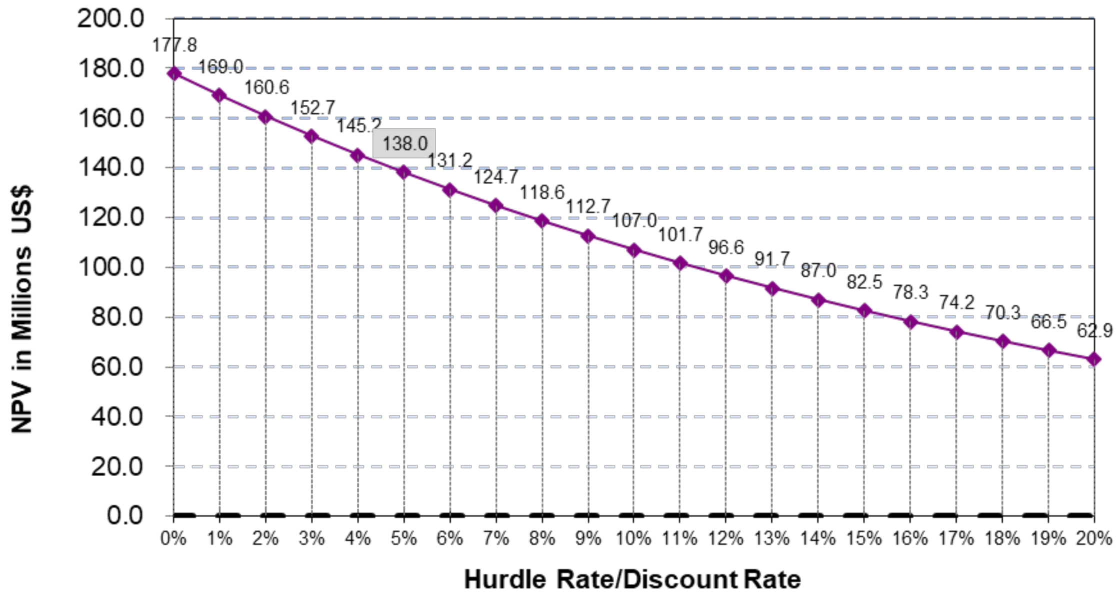


FIGURE 1-6 PRE-TAX IRR SENSITIVITY ANALYSIS



A sensitivity analysis of discount rates presented in Figure 1-7 shows that the Project as currently designed would have positive NPV/IRR even at a 20% discount rate.

FIGURE 1-7 PRE-TAX DISCOUNT RATE SENSITIVITY ANALYSIS



TECHNICAL SUMMARY

PROPERTY DESCRIPTION AND LOCATION

The Project site 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. It is situated between the town of Iperindo and Odo Ijesha village, in the Atakunmosa East Local Government constituency.

A small historic open pit and tailings area exists on the Project site; however, the Project is a greenfield project for which all necessary support facilities and infrastructure will be provided as part of the Project.

The Project site is situated close to a tarred road. Although there is a power supply grid in the region, there is insufficient grid capacity to provide electrical power to the Project. All power for the Project will be generated on site.

There is sufficient land area within the Project tenements to accommodate the proposed mining activities, waste dumps, processing plant, and tailings storage facilities.

Thor maintains an office, accommodation camp, and core logging and sample preparation facility, within a single secured compound on the outskirts of the town of Ilesha, located approximately 25 km north of the Project area.

LAND TENURE

The property comprises a mining licence (ML41), which covers an area of 1,720 ha (17.2 km^2) and is contained within the larger exploration licence (EL19066), which measures 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.

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. As of the date of this report, licence EL19066, belonging to SROL, is in good standing and all fees are fully paid.

Thor acquired a 100% interest in the Project 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.

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 and processed with a second-hand stamp mill together with a ball mill and tables acquired from Ghana. Presently, the small pit has mostly collapsed and filled with water. There are no reliable production records available for this period.

In 1970, the Property was acquired by Obokun Minerals Developments Limited (OMDC) which rehabilitated the plant, but operations ceased due to internal company problems.

During 1976, the Bureau de Recherches Géologiques et Minières (BRGM) completed mapping and geochemical surveys over the property. Polservice (Polish geologists and engineers) undertook a geological review, petrographic and metallurgical studies in 1981, and the following year Nigerian Mining Corporation (NMC) acquired the Project from OMDC and completed an eluvial drilling programme. Between 1983 and 1987, NMC undertook geological mapping, surveying, drilling, trenching, trench rehabilitation, and soil geochemistry. A new Exclusive Prospecting Licence (EPL) was issued to NMC in 1986.

During 1992, Pineridge Nigeria Ltd (PNL) carried out a detailed pre-investment study and data compilation and entered into a joint venture with NMC in 1994. Tropical Mines Ltd (TML) was incorporated as a joint venture company (owned 20% by NMC and 80% by PNL). NMC was issued with the Temporary Mining Licence (TMiL 19706) in 1995, which in 1996 was assigned to TML and approved for 21 years, together with a three-year temporary title.

In 1997, TML signed a joint venture agreement with Hansa through its consultancy company Hansa GeoMin Consult to form Ijesa GeoMin Mining Development Corporation Limited (IGMDC). IGMDC resurveyed 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, and completed petrographic and fluid inclusion studies. The TML-Hansa joint venture was terminated in 1999.

In 2007, Segilola Gold Ltd (SGL), then a wholly owned subsidiary of CGA Mining Limited (CGA), acquired the right to earn up to 51% undivided interest in the tenements. A detailed drilling of the known mineralised zone commenced. CGA generated a Mineral Resource estimate in 2009. SGL was transferred by CGA to its affiliate, Ratel Group Ltd (RGL), a Toronto Stock Exchange (TSX) listed entity. RGL completed a Feasibility Study (FS) in 2010 for internal purposes.

Between 2011 and 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 mineralisation. In 2012, a revised BFS 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.

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.

GEOLOGY AND MINERALISATION

Segilola is an orogenic-style lode gold deposit which occurs within a regional-scale shear zone within the Birimian Greenstone Belt.

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 (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.

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.

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 (those that cannot be differentiated into the above groups). The orthogneisses, which are considered to be the basement rocks, are located stratigraphically below the quartzite schists and meta-sediments. The stratigraphy trends north-northeast and generally dips steeply towards the west. There is minimal weathering and, as such, there is no distinction between weathered and fresh material.

The Segilola gold lodes are 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 the calc silicate and mylonitic biotite-garnet schists (footwall) in fractured pale to dark grey coloured smoky quartz veining, sheared pegmatites, and silica/chlorite/carbonate alteration.

The mineralisation is hosted in three steeply dipping vein sets or lodes; the Hanging Wall Lodes (Lodes 100 and 300) and the Footwall Lode (Lode 200). Together these form an elongate steeply dipping (60º to 70º) mineralised zone 2,000 m long striking at 010º. The entire mineralised zone is up to 200 m depth and between 2 m and 20 m in true thickness.

The Hanging Wall Sequence consists of a fractionated granodiorite unit that intrudes basement gneisses. Lode 100 is relatively discrete with sharp upper and lower contacts. Lode 300 is characterised by some of the highest gold grades with finely disseminated visible gold particles in vein material. The footwall lode (200) is characterised by a wider, more diffuse and lower-grade mineralisation developed around high-grade veins. Gold in these lodes is associated with quartz-feldspar-pegmatitic veins and coarse gold particles are usually associated with floating biotite flecks.

Higher gold grades and greater thicknesses are developed adjacent to a 5 m to 20 m thick zone that is thought to either the upper fractionated portion of the granodioritic intrusive or an associated zone of intense quartz-carbonate flooding located at the eastern margin of the Hanging Wall Sequence.

The onset of the footwall lithological sequence is marked by a high-grade metamorphic suite consisting of pale-grey silicified quartz-sillimanite schist, a narrow calc-silicate unit followed by the main sequence of garnet-biotite schist. calc-silicate unit and biotite schist.

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 trace element associations such as silver with gold. Minor sulphides, typically pyrite, are associated with the lodes.

EXPLORATION STATUS

Historical exploration activities on the Project have included geophysics, geological mapping, soil sampling, trench sampling, and drilling. Since acquisition of the Project in 2016, Thor has carried out soil and stream sediment sampling, tailings sampling, drilling, and light detection and ranging (LIDAR) surveys.

A total of 252 drill holes for approximately 28,000 m have been completed to date at the Project. Of these, 192 drill holes for 22,144 m were completed by CGA between 2008 and 2011 and by Thor in 2017 and 2018 and used in the current Mineral Resource estimate.

MINERAL RESOURCE ESTIMATE

The open pit Mineral Resource estimate for the Project has been prepared by Auralia and the underground Mineral Resource estimate for the Project has been prepared by RPA. CIM (2014) definitions were followed for Mineral Resource estimation. Indicated and

Inferred Mineral Resources have been classified based on drill hole spacing, grade continuity, and reliability of data.

All drill hole data available up until 1 December 2018 was used for the Mineral Resource estimates. The estimates are based on diamond drill core from CGA and Thor drill holes only. Table 1-1 summaries the open pit and underground Segilola Mineral Resources as of 1 December 2018.

OPEN PIT MINERAL RESOURCES

Geological cross-sections were interpreted to create three-dimensional wireframes of the mineralised zones (lodes) at a cut-off grade of 0.5 g/t Au. A maximum of 2.5 m (true width) of internal waste was included during wireframing and a lower cut-off grade was used in places to maintain geological continuity. The interpretation resulted in three wireframes for lodes 100, 200, and 300, all of which strike approximately 010° and dip 60° to 70° towards the west. The wireframes range in length from 580 m to 1,500 m and cover a total strike length of approximately 2,000 m, range in depth from surface to between 75 m and 250 m and in thickness from 0.5 m to 20 m.

The wireframes were used to identify assay samples within the mineralised zones which were then composited to a length of one metre and a top-cut value of 40 g/t Au was applied.

Geostatistical analysis and Quantitative Kriging Neighbourhood Analysis (QKNA) were completed on the Lode 200 domain. The variogram models and the search neighbourhood established from these analyses were then applied to the Lode 100 and Lode 300 domains. Grade interpolation for Au was carried out using OK with a search ellipse orientated at 020°, dipping at 70° towards the west, with a minimum of two and a maximum of 12 samples. A total of four passes with increasing search radii were used to interpolate all the blocks in the model.

Mineral Resource classification was guided by geological continuity, geological understanding, previous resource estimates, interpolation criteria and reliability, and drill hole spacing. Mineral Resource classification codes were assigned to the block model and these were reviewed to ensure that the classification zones were continuous.

Indicated Mineral Resource blocks were defined as those populated in estimation pass one or two, within an average distance of 30 m to nearest sample, and with a minimum of 12 samples. Inferred Mineral Resource blocks were those populated in estimation pass three, with an average distance to the nearest sample of 90 m or less, and a minimum of four samples.

In order to demonstrate that the estimated Mineral Resources have met the requirements of reasonable prospects of eventual economic extraction, Mineral Resources were constrained by and reported within an optimised pit shell using a gold price of $1,500/oz. The pit shell was limited in extent by a line which demarcated the planned underground mine. South of this line, the open pit Mineral Resources were reported inside the reserve pit shell. Mineral Resources were reported at a cut-off grade of 0.64 g/t Au.

UNDERGROUND MINERAL RESOURCES

Underground geological wireframes were prepared by Thor geologists at a cut-off grade of 2.5 g/t Au. RPA reviewed the data populations, and confirmed that a second grade population starting at 2.5 g/t Au is present which warranted the wireframing cut-off grade used. RPA reviewed the mineralised intercepts from these wireframes and regenerated underground wireframes using Seequent Leapfrog software. The wireframes were generated using a two-metre minimum mining width.

Assay samples were flagged with the mineralisation wireframes. The mineralised samples within the underground domains were capped at 50 g/t Au, and subsequently composited on one metre lengths. Geostatistical analyses were undertaken on the capped composites to determine the search and kriging parameters. Dynamic anisotropy was employed during interpolation to reflect local variations in the orientation of the mineralisation. The block model was interpolated using OK. A density of 2.70 t/m^3 was applied to all domains, based upon the average density of the underground mineralised material.

RPA considered a combination of drill hole spacing, distance to nearest sample, mineralisation continuity, minimum number of samples, and the search pass number to inform the Mineral Resource classification. For Indicated Resources, RPA outlined areas of the block model in each domain that were interpolated using a maximum drill spacing of 25 m by 25 m during the first and second pass (minimum of two holes and four samples). For the Inferred material, a 50 m by 50 m maximum drill spacing was employed with a minimum of two samples. Small areas of material that were below these minimum requirements were included to aid the classification continuity where the grade and geological continuity were observed to be consistent.

Mineral Resources are reported outside the Stage 3 Design Reserve open pit shape, south of a demarcation line that limits the open pit Mineral Resources, and inside stope shapes generated at $1,500/oz at a cut-off grade of 2.58 g/t Au.

MINERAL RESERVE ESTIMATE

The open pit Mineral Reserve estimate was prepared by Auralia and is summarised in Table 1-2 with an effective date of 1 December 2018. The Mineral Reserve excludes any Inferred Mineral Resources.

An economic cut-off grade of 0.77 g/t Au was determined for Mineral Reserves based on metal selling price and cost, processing cost and recovery, and general and administration (G&A) costs. A 10% mining dilution factor was applied to account for dilution that would be expected to occur during the course of mining due to the mixing of ore and waste material during blasting and excavation processes. A 95% mining recovery factor was applied to account for the amount of ore that is lost due to spillage and re-handling.

A final pit and two interim stage pits were designed based on optimisation shells created with the relevant cost, revenue, and physical input parameters. The final shell selected by Thor, shell 37, allowed for a Probable Mineral Reserve of over 400,000 oz with a strip ratio of 15.3.

No Mineral Reserves have been estimated for the proposed underground mine.

MINING METHOD

The Project is proposed to be a conventional open pit mining operation using 200 t hydraulic excavators and 90 t payload rear dump trucks. Mining will be carried out by a mining contractor(s), with all equipment provided by them. Ore blasting will be conducted on five metre benches, loaded in two flitches, and waste blasting will be on 10 m benches where possible. Narrow ore zones will require the implementation of careful blasting techniques to minimise dilution and maximise ore recovery. Blasting will be done with pumped emulsion as water is expected in the blast holes.

A total of 75% of the ROM ore is planned to be re-handled due to substantial stockpiling in the mining schedule. The ROM pad and stockpile area will be located adjacent to the processing plant at the southern end of the west waste dump. All waste rock from the pre-strip period and initial months of mining will be used to create the flat area for the ROM pad and ore stockpiles.

An accurate estimate of water inflow as the pit develops is yet to be completed. Current dewatering plans are based on an initial estimate of inflows of 3.6 m^3 to 18.6 m^3 per day, based on rainfall data, groundwater inflows, and a 95% runoff factor. Sub-horizontal drains will be used to promote passive drainage of the pit walls, typically installed from benches in the pit walls, with a horizontal spacing of 10 m to 30 m between drains. In addition, vertical dewatering wells pumped by submersible pumps will be installed if a significant concentration of groundwater seepage and inflow is observed in areas of the open pit.

Production is planned to commence in the Stage 1 pit towards the northern end of the deposit, targeting high-grade ore close to surface with mining progressing generally from north to south. The Stage 2 pit commences after nine months as ore supply from the Stage 1 pit is well established. From Month 24, the eastern access ramp in the Stage 2 pit is mined out to provide access to further ore, with the Stage 2 pit completed by Month 28. Stage 3 (the final pit design) commences in Month 15 upon the completion of Stage 1, and the final pit is completed in Month 46.

An average of 625,000 t of ore will be processed on an annual basis.

MINERAL PROCESSING

Diamond drilling in July 2018 provided a representative bulk sample for a metallurgical test work programme that was designed to build on historic test work results. Independent Metallurgical Operations Pty. Ltd. (IMO) completed the test work on a master composite sample and ten variability samples. The test work included comminution testing, GRG tests, cyanidation test work, static settling tests, and tailings characterisation.

Comminution test work included Crushing work index, semi-autonomous grind (SAG) Mill Comminution (SMC) testing, and Bond ball mill grinding test work. The results from the variability samples and master composite indicated that the samples were soft to medium hardness in terms of SMC testing (A x b ranged from 45.5 to 87.0), and that they were hard to very hard in terms of Bond ball mill work index (ranging from 17.9 kWh/t to 20.3 kWh/t).

GRG test work was conducted on a bulk sample of 20 kg taken from the master composite, for which the recovery was 77.2%. The results of the extended GRG (eGRG) test work were used by Consep Pty Ltd (Consep) for GRG modelling, indicating that, based on the composition of the master composite, the inclusion of a gravity circuit in the grinding circuit could recover 43% of feed gold. Additional GRG test work on the ten variability samples, undertaken at an intermediate P_{80} grind size of 300 µm to mimic cyclone underflow, resulted in recoveries ranging from 41% to 81%.

Leach test work was conducted on the recombined samples (gravity concentrate tailings) of the ten variability samples. The results indicated that while the majority of leaching was completed within 24 hours, leaching continued to the end of the tests at 48 hours. Leach recoveries after 48 hours ranged from 89.2% to 97.6% and averaged 93% for the ten samples. Overall recoveries (both leach and gravity concentration) at 48 hours ranged from 94.4% to 99.6%, with low to moderate cyanide consumption, and low lime consumption. Leaching conducted at P_{80} grind sizes of 150 µm, 106 µm, and 75 µm showed that slightly improved recoveries could be achieved at the finer grind, but that satisfactory recoveries were still achievable at 106 µm.

Overall, the metallurgical test work demonstrated that the mineralisation is amenable to recovery by gravity concentration and cyanide leaching.

Two static settling tests were conducted on leach residue from the master composite and these achieved final densities of 55% and 58% solids respectively.

Acid mine drainage characterisation of the master composite leach residue indicates a net acid producing potential of 6 kg H_2SO_4/t, resulting in a classification as potentially acid generating (PAG). Eight waste rock samples were tested for acid mine drainage characterisation, with results for net acid producing potentials of -496 kg H_2SO_4/t to 1 kg H_2SO_4/t, not acid generating (NAG).

A previous metallurgical test work programme was completed by AMMTEC Ltd. (AMMTEC) in 2010, for CGA, based on six composites of exploration core representing different zones of the orebody.

Most of the cyanide leach tests were conducted at a P_{80} grind size of 106 µm. Cyanide detoxification tests using the INCO SO_2/air process achieved effluents of <1 mg/L CN_{WAD} in less than 60 minutes from solutions containing 135.5 mg/L CN_{WAD}. Initial gravity tests on three of the composites provided mixed results, with Au recoveries between 33% and 70%.

The DFS flowsheet and plant design is based on the IMO test work results and consists of a conventional crushing circuit, two stage grinding, gravity, CIL, elution, electrowinning and smelting to produce gold doré.

The process plant will operate 365 day/year, 24 hours/day. The processing plant has been designed with a nameplate capacity of 650,000 tpa to cater for additional operating flexibility and future expansion capacity with minimal additional capital expenditure.

A ROM pad will be located at the southeast corner of the plant site. ROM ore will be crushed using a single-stage jaw crusher and the crusher circuit product will be transferred to the crushed ore stockpile. From the stockpile, the ore will be transferred by the mill feed conveyor to the semi-autogenous grinding (SAG) mill feed hopper.

The grinding circuit will consist of a SAG mill and ball mill in closed circuit with hydrocyclones, having a design throughput of 85 dry tph and grinding to a nominal cyclone overflow P_{80} of 106 µm.

One third of the cyclone underflow will gravitate to the Knelson concentrator scalping screen to remove +2 mm material. Screen oversize and the remaining cyclone underflow will gravitate into the ball mill feed chute and back into the mill for grinding. Screen undersize will report to the Knelson concentrator where free gold will be recovered from the slurry. The Knelson tails will gravitate back into the ball mill feed chute.

Gold and silver leaching will take place in a conventional CIL type circuit with four leach tanks (no carbon) and six adsorption tanks containing activated carbon. Total residence time for the combined leach and adsorption circuit will be 48 hours. The slurry leaving the last leach tank will report to the adsorption stage, consisting of six tanks having 20 hours total residence time. Dissolved gold and silver cyanide complexes will be adsorbed onto activated carbon contained within the adsorption tanks.

Slurry containing activated carbon will be pumped from the first adsorption tank and pass over the loaded carbon screen. The screened slurry will be discharged back to the adsorption tank and the recovered carbon will gravitate to the carbon storage tank for the elution circuit. Elution solution (eluate) containing dissolved precious metals will be passed through an electrowinning cell to plate out the contained gold.

Precious metal recovery and refining will be carried out in a secure building. Silver and base metals are removed from the sludge by leaching with nitric acid to form soluble nitrates. After leaching with nitric acid, the dried gold sludge is mixed with a range of fluxes and smelted at 1,100°C to produce bullion. Bullion will be stored in a vault prior to transport and refining off site.

PROJECT INFRASTRUCTURE

The planned Project infrastructure will consist of the following:

- Access to the site via a sealed access road which will service both the processing plant and mining compounds. Two internal mine roads will be built, 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. Plant site roads are internal roads between the administration area and plant site facilities.

- Buildings including a processing plant, warehouses, workshops, reagent storage facility, administrative buildings, mining contractor facilities, the accommodation, camp, fuel storage tanks; etc.

- Water. The total water consumption is estimated to be 3,251 m^3/day from the TSF and water supply dam, with fresh water consumption at 548 m^3/day, and the recovered water consumption at 2,703 m^3/day. The total fresh water consumption of 548 m^3/day includes water for processing at 406 m^3/day, water for dust suppression and gland seal water at 40 m^3/day, water for greening and spraying road at 12 m^3/day, water for domestic use at 41 m^3/day, and water for unforeseen purposes at 49 m^3/day. The ratio of water reuse processing is 83%. The water supply dam is replenished by rainfall.

- Sewage and Solid Waste Management. Effluent from all water fixtures in the process plant, mine services area and administration areas will drain to gravity sewerage systems. The gravity sewerage system for each area will drain to a sewage treatment plant system located to the north of the process plant. Wastes will be sorted and reused or recycled as far as the limited access to recycling facilities allows. General non-hazardous solid wastes will be deposited into a suitable mine waste dump and promptly covered to deter vermin and scavengers. Dangerous or hazardous waste will be collected and stored briefly before being transferred to a suitable permitted facility, either on-site or off-site depending on the specific materials and requirements.

- Power. The state grid corporation of Nigeria is unable to provide electricity for the Project, and as a result, generating sets will be used for power supply. A total of eight 1.2 MW generating sets will be installed and operated, all eight will be compressed natural gas (CNG) generator sets (10.5 kV/1,200 kW at 0.8 pf). Additionally, there will be one diesel generating unit for emergency power (400 V/800 kW at 0.8 pf).

- The TMF located 1.3 km southwest of the process plant. The TMF consists of a single valley style rock zoned embankment (north) and two small (less than 5 m) rock filled saddle dams (west and east). The TMF has been sized to accommodate 3.3 Mt while providing sufficient capacity for the operating pond and the Inflow Design Flood (IDF).

MARKET STUDIES

The principal commodity to be produced by the Project is gold, which is freely traded, at prices and terms that are widely known, so that prospects for sale of any production are virtually assured.

Metal prices used by Auralia and Thor are in line with the long-term consensus forecasts by independent banks and financial institutions. A gold price of $1,300 per ounce was used for the financial modelling.

No contracts for operations and/or metal supply have been negotiated or executed as yet.

ENVIRONMENTAL, PERMITTING AND SOCIAL CONSIDERATIONS

The Project covers a modified environment which has been logged for over 100 years and has remnants of gold mining from the 1950s. The land use now mostly consists of fruit trees, hunting and logging, providing income for communities to the north and south of the Project.

Thor has obtained two key environment permits for the Project; Environmental Impact Assessment (EIA) and the Certificate to Operate the Environment Protection and Rehabilitation Programme (EPRP).

The EIA for the development of the mine, processing and infrastructure facilities was certified by the Federal Ministry of Environment in March 2013, subject to standard conditions. The EIA approval has been enacted with ongoing work to address the required conditions as the Project moves through the project life cycle. Management plans have been prepared, or are in the process of preparation, to address environment and/or social impacts during exploration, construction, operation and closure phases.

The EPRP, which outlines key environment and social management objectives during operation and closure phases, was approved by the Ministry of Mines and Steel Development in December 2017.

Stakeholder consultation occurred during the EIA process and then intermittently until 2016 when Thor acquired the exploration and mining licences. A step up in consultation and participation occurred as part of the exploration phase and in the development of Community Development Agreements (CDAs), which have been developed for the three communities closest to the Project. Benefits to the communities as outlined in the CDAs have commenced with the construction of new boreholes and provision of local employment during the exploration stage. CDA committees with representations from each village have been established to aid in managing the CDA process. A stakeholder engagement plan has been developed by Thor, together with a grievance management plan.

The mine area does not directly impact on the permanent dwellings of the nearest communities. Compensation for the loss of agricultural crops is being prepared via the Resettlement Action Plan (RAP) in line with Nigerian legislation and international standards. Compensation for damage to crops during exploration activities has occurred and been documented.

The area has not experienced rapid change in its development in the past two decades. The existence of exploration licence and mining leases over a 70-year period has not triggered rapid development nor influx.

Communities in the villages of Imogbara, Odo Ijesha (including Ipole Ijesha), and Iperindo (combined population of 11,930) surrounding the mine development footprint are agriculture/market-based communities with low incomes. Each community has its own cultural governance system via an Oba, which is an appointed position.

CAPITAL AND OPERATING COST ESTIMATES

CAPITAL COST

The pre-production capital cost to bring the Project into production is estimated to be $87.6 million.

The pre-production capital costs for the process plant and infrastructure were estimated by Norinco using a combination of first principles, quotations and factored estimates. No capital costs are included for mining equipment which will be provided by the mining contractor.

Based on the level of project and engineering definition prepared for the DFS, the capital cost estimate is considered to be a Class 3 estimate (American Association of Cost Engineers (AACE) guidelines), with an overall accuracy level of ±10%.

A summary of the initial capital estimate for Project is shown in Table 1-8.

TABLE 1-8 PRE-PRODUCTION CAPITAL COST SUMMARY
Thor Explorations Ltd. – Segilola Gold Project

Description	$000s
Mining (ROM pad)	400
Process	19,800
Infrastructure	28,600
Indirects	13,000
Norinco EPC Contract Value	**61,800**
Powerline to TMF/WDS	400
Total EPC Contract	**62,000**
Total TMF	3,700
Total Contract Value	**65,900**
First Fills	1,900
Mine Contractor Mob / Haul road Est.	800
Owner Cost	11,000
VAT (on local purchases / services)	1,300
Duties (contingency)	1,500
Underground Development	0
Initial Capital	**82,300**
Pre-strip (Month 16)	2,900
Upfront Working Capital	2,400
Total Pre-Development Capital	**87,600**

No sustaining capital estimates were included in the Project. Estimates for raises to the initial tailings dam are incorporated under operating costs.

The following items were excluded from the capital cost estimate:

- Project financing and interest charges
- Escalation during construction
- Exchange rate variations

OPERATING COSTS

The average LOM operating costs are estimated to be $77.51/t, based on a contractor operated open pit mine and Owner operated processing plant.

A summary of the LOM operating costs is shown in Table 1-9.

TABLE 1-9 OPERATING COST SUMMARY
Thor Explorations Ltd. – Segilola Gold Project

Description	$/t milled
Contractor Mining ($2.85/t moved)	50.97
Process (aka Thor Operations)	18.58
Fixed G&A	5.78
Infrastructure	2.18
Total	**77.51**

The operating costs presented are based upon quotations obtained from mining contractors and first principle estimates for the Owner-operated process plant and site facilities. The Owner G & A costs include all non-mining operating, maintenance, and support personnel.

Operating costs for the Project have been estimated from first principles. Labour costs were estimated using Project specific staffing, salary, wage, and benefit requirements. Unit consumption of materials, supplies, power, water, and delivered supply costs were also estimated.

The Project's major consumable prices are diesel at a price of $0.675/l delivered to site, and a CNG generated power cost of $0.11/kWh.

OTHER RELEVANT DATA: UNDERGROUND PEA

A PEA was completed for the portion of the Mineral Resources at the Segilola deposit that were not included in the open pit design, in order to evaluate the potential economics of an underground operation.

An underground mine design was completed based on the underground Indicated and Inferred Mineral Resource using basic geotechnical and metallurgical data largely extrapolated from the open pit work. The proposed underground mine would operate concurrently with the open pit, with the higher-grade underground ore used to increase the volume and grade of the plant feed.

The proposed underground mine would be an underground operation, largely below the envisaged open pit, accessed by a surface ramp. The proposed mining method is Sub Level Open Stoping (SLOS) and cemented rock backfill, with sub-levels spaced at 15 m vertical intervals. It is intended that all mine development and production will be undertaken by a qualified and experienced mining contractor.

The life of the underground operation mine is estimated to be approximately six years, at a steady state production rate of 156,000 tpa. The LOM average mined grade is estimated to be 6.33 g/t Au, with a total Au recovery of 103,000 oz.

The infrastructure proposed in the DFS will be used by the underground operations, including the processing plant, tailings dam, offices, workshop areas, power supply and waste dumps. In the PEA the plant is planned to be used to its full design capacity of 650,000 tpa, while in the open pit only scenario in the DFS, the plant is expected to be operated at 625,000 tpa. The increase in throughput will begin in the 3rd month after commissioning the 650,000 tpa plant as open pit material will be available and will not need to be stockpiled to the same extent as in the open pit only situation.

Underground development begins in the third month after commissioning of the process plant with first mill feed to the plant occurring ten months later. The first 34,000 tpa of underground mill feed can be processed with no impact on the open pit mill feed schedule, however the remaining underground mill feed will displace lower grade open pit material, which will be stockpiled for processing later. During ROM operations, the open pit and underground operations will provide an average of 545,000 tpa and 105,000 tpa of mill feed respectively.

The pre-production capital cost for the underground mine is estimated to be $12.5 million, which includes the initial development and excludes provision of mining equipment, which will be supplied by the contractor.

The estimated operating costs are based on a combination of DFS estimates, benchmarking and factored costs and is estimated to be $125/tonne of mineral mined. The cost estimates are considered to be to an accuracy level ±50%, commensurate with a PEA level of study.

Based on the results of the PEA, it is projected that the underground mine would generate an accretive NPV of $34.7M at a 5% discount rate, with a payback period of 1.1 years.

The economic analysis contained in the PEA is based, in part, on Inferred Mineral Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which the PEA is based will be realised.

2 INTRODUCTION

Roscoe Postle Associates Inc. (RPA) was retained by Thor Explorations Ltd. (Thor) to prepare an independent Technical Report (the Technical Report) on the Segilola Gold Project, (Segilola or the Project), located in the state of Osun, Nigeria. The purpose of this report is to disclose the results of a Definitive Feasibility Study (DFS) for a proposed open pit mine and gold processing operation to exploit the Mineral Resources of the Segilola deposit. Additionally, Preliminary Economic Assessment (PEA) based on Mineral Resources at depth for a proposed supplemental underground mine was completed.

The Mineral Resource and Mineral Reserve estimates have been prepared according to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014. This Technical Report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

Thor is a Canadian mineral exploration company engaged in the acquisition, exploration and development of mineral properties located in Nigeria, Senegal, and Burkina Faso. Thor holds a 100% interest in the Segilola Gold Project located in Osun State Nigeria, a 70% interest in the Douta Gold Project located in south-eastern Senegal, and a 49% interest in the Bongui and Legue gold permits located in Houndé greenstone belt, south west Burkina Faso. Thor trades on the TSX Venture Exchange under the symbol "THX".

The Project was the subject of a NI 43-101 Preliminary Feasibility Study (PFS) prepared by Auralia Mining Consulting Pty Ltd (Auralia) based on drilling completed up to July 2017 (Auralia, 2017). An updated Mineral Resource estimate has been prepared in consideration of additional drill results, a recent re-interpretation of geological and mineralized domains, and revised cut-off grades. The open pit block model has been constrained with a Whittle pit shell to demonstrate reasonable prospects for eventual economic extraction for resource reporting purposes. The underground block model has been constrained with potential mining shapes to demonstrate reasonable prospects for eventual economic extraction for resource reporting purposes.

The DFS envisages an open pit mine and the construction of a 650,000 tpa nameplate capacity processing plant, which would consist of a conventional crushing circuit, two stage grinding, gravity, Carbon in Leach (CIL), elution, electrowinning and smelting to produce gold doré. The DFS envisions a construction start date in Q2 2019 and a 17-month construction period with a five-year mine life.

The PEA for the underground mine envisages an initial four year underground operation which can be brought into production during the open pit mine life to supplement the open pit ore with higher grade underground plant feed. The PEA is based, in part, on Inferred Mineral Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realised.

The deposit remains open at depth below the resources considered in the underground Project.

SOURCES OF INFORMATION

Thor employed a number of independent consultants for the preparation of the DFS:

- RPA – Underground Mineral Resources and Economics, Project Management and Report Compilation
- Auralia – Geology, Open Pit Mineral Resources, Mineral Reserves, and Mining
- Norinco/CMGE Yantai Orient Metallurgical Design and Research Institute (Yantai) - Mineral Processing and Infrastructure
- Knight Piésold Limited (KP) - Tailings Management Facility by Knight Piésold Limited (KP)
- SRK Consulting Ltd and Peter Clifton and Associates – hydrogeological assessment
- Peter O'Bryan and Associates – geotechnical assessment for the open pit
- GR Engineering Services Ltd. (GRE) – metallurgical and processing consulting services
- Independent Metallurgical Operations Pty Ltd (IMO) – metallurgical test work

Site visits were carried out to the property by David JF Smith, CEng, RPA Principal Mining Engineer, between 19th and 21st September 2018, and by Jack Lunnon, CGeol., RPA Senior Geologist, on 9th and 10th February 2019. While at site, Mr. Smith visited the historical open pit, the proposed locations of the open pit, process plant site and tailings disposal facility and observed the surrounding infrastructure. Mr. Lunnon visited the historical open pit, observed drill hole collar locations, examined drill core at Thor's Exploration Camp and Core Processing Facility and reviewed the sampling and QA/QC procedures used by Thor.

Auralia site visits were carried out by Anthony Keers (MAusIMM, CP) Director, in May 2017 and by Chris Speedy, MAIG, on 9th and 10th February 2019. Mr. Keers and Mr. Speedy inspected the historic excavations, drill core, surface conditions and existing infrastructure.

Marion Thomas (Pr.Sci.Nat), Environmental and Social Consultant, and Richard Elmer (MIMMM), Director and Principal Geotechnical Engineer, of Knight Piésold Limited, UK, visited the site on 9th and 10th February 2019 and inspected the various project locations and the neighbouring communities.

During the site visit discussions were held with the following personnel from Thor:

- Mr. Segun Lawson, Chief Executive Officer, President and Director.
- Mr. James Philip, Vice President, Corporate Development.
- Mr. Alfred Gillman, FAUSIMM (CP), Group Exploration Manager.
- Ms. Louise Porteus, Environmental and Social Manager.
- Mr. Ayoyb Nyenzi, Senior Geologist.
- Mr. Oyewole Akinloye, Senior Geologist.
- Mr. Olufemi Olubi, Project Geologist.
- Mr Kunle Laogun.
- Mr. Alayande Bolayi Temborun, Geological Assistant.

Various independent consultants, appointed by Thor, assisted in the preparation of the FS, including:

Qualified Person (QP) responsibility for this report is outlined below:

- Mr. Smith of RPA is responsible for overall preparation of the report and related disclosure in Sections 1 to 3, 21, 22, and 25 to 29.

- Mr. Lunnon of RPA is responsible for parts of Sections 1, 14, and 25 to 29.

- Mr. Keers of Auralia, is responsible for Sections 15 and 16 and for parts of Sections 1, and 25 to 29.

- Mr. Speedy of Auralia, is responsible for Sections 4 to 12, and parts of Sections 1, 14, and 25 to 29. The preparation of Sections 5, 6, 7, 8, 9 and 10 was carried out with the assistance of Alf Gillman of Thor.

- Mr. Qiang Ji of CGME is responsible for Sections 13 and 17, and parts of Section 1, 18, and 25 to 29 of the Technical Report.

- Mr. Richard Elmer of Knight Piésold UK is responsible for parts of Sections 1, 18, and 25 to 29 of the Technical Report.

- Ms. Marion Thomas of Knight Piésold UK is responsible for Sections 20 and parts of Sections 1, and 25 to 29 of the Technical Report.

- Mr Malensek of RPA is responsible for parts of Section 22 and parts of Section 1, 21, 24 and 25 of the Technical Report.

The documentation reviewed, and other sources of information, are listed at the end of this report in Section 27 References.

LIST OF ABBREVIATIONS OF MEASURES

Units of measurement used in this report conform to the metric system. All currency in this report is US dollars (US$) unless otherwise noted.

μ	micron
μg	microgram
a	annum
A	ampere
bbl	barrels
Btu	British thermal units
°C	degree Celsius
C$	Canadian dollars
cal	calorie
cfm	cubic feet per minute
cm	centimetre
cm^2	square centimetre
d	day
dia	diameter
dmt	dry metric tonne
dwt	dead-weight ton
g	gram
G	giga (billion)
g/L	gram per litre
g/t	gram per tonne
gr/m^3	grain per cubic metre
ha	hectare
hp	horsepower
hr	hour
Hz	hertz
J	joule
k	kilo (thousand)
kcal	kilocalorie
kg	kilogram
km	kilometre
km^2	square kilometre
km/h	kilometre per hour
kPa	kilopascal
kVA	kilovolt-amperes
kW	kilowatt
kWh	kilowatt-hour
L	litre
lb	pound
L/s	litres per second
m	metre
M	mega (million); molar
m^2	square metre
m^3	cubic metre
MASL	metres above sea level
m^3/h	cubic metres per hour
mi	mile
min	minute
μm	micrometre
mm	millimetre
mph	miles per hour
MVA	megavolt-amperes
MW	megawatt
MWh	megawatt-hour
oz	Troy ounce (31.1035g)
oz/st, opt	ounce per short ton
ppb	part per billion
ppm	part per million
psia	pound per square inch absolute
psig	pound per square inch gauge
RL	relative elevation
s	second
t	metric tonne
tpa	metric tonne per year
tpd	metric tonne per day
$	United States dollar
V	volt
W	watt
wmt	wet metric tonne
wt%	weight percent
yr	year

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by RPA for Thor Explorations Ltd. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to RPA at the time of preparation of this report,
- Assumptions, conditions, and qualifications as set forth in this report.

For the purpose of this report, RPA has relied on ownership information provided by Thor Explorations Ltd. RPA 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.

RPA has relied on Thor Explorations Ltd. for guidance on applicable taxes, royalties, and other government levies or interests, applicable to revenue or income from the Segilola Gold Project.

Except for the purposes legislated under provincial securities laws, any use of this report by any third party is at that party's sole risk.

4 PROPERTY DESCRIPTION AND LOCATION

The Segilola Gold Project site 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 Universal Transverse Mercator (UTM) co-ordinates for the approximate centre of the Project are 700987 mE, 832281 mN (UTM Zone 31N).

Thor acquired a 100% interest in the Project 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.

LAND TENURE

The property comprises a mining licence (ML41), which covers an area of 1,720 ha (17.2 km^2 / 81 Cadastral units) and is contained within the larger exploration licence (EL19066), which measures 2,700 ha (27.0 km^2 / 135 Cadastral units). 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, Thor 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.

Figure 4-1

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Location Map

March 2019

Source: Map Resources, 2019.

RPA

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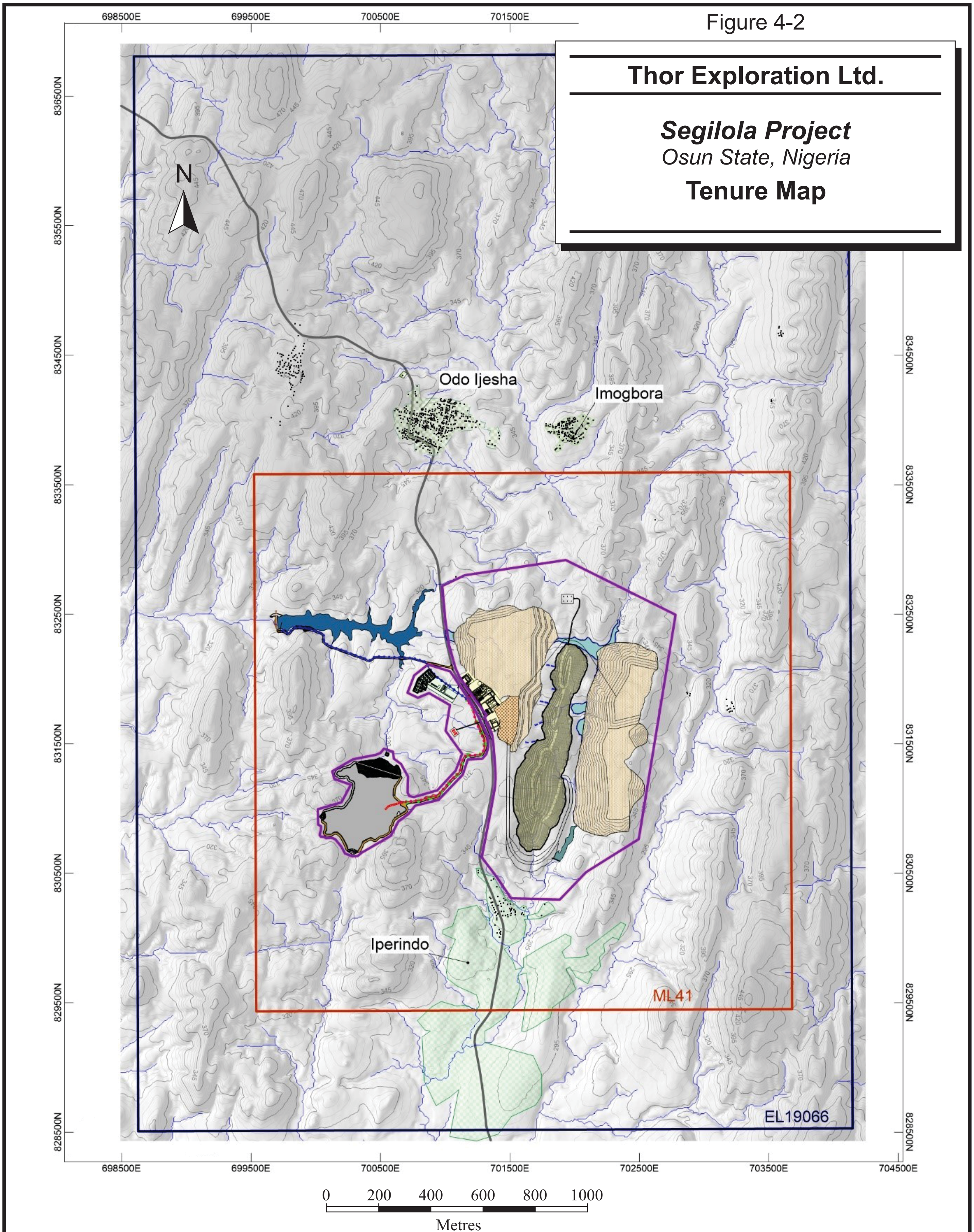


March 2019

Source: Thor, 2019.

Obligations required to maintain the mining licence includes:

- Meet 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 pa (approximately $22,300).

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:

- Meet the prescribed reporting requirements in line with Schedule 5 of the Nigerian Minerals and Mining Regulations 2011.

- Annual service fee of N2,000 per cadastre unit (approximately $5.5 per cadastre unit) which amounts to N270,000 (approximately $745).

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

SURFACE RIGHTS

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.

PERMITS AND APPROVALS

RPA is not aware of any environmental liabilities on the property. Thor has all required permits to conduct the proposed work on the property. RPA is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the property.

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

ENCUMBRANCES, 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 in the course of mining or exploration are subject to the payment of 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:

- a tax exemption for the first three years of operation, which period 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, nationalisation, or acquisition by any government of the federation unless the act is in the national interest or for a public purpose and under a law that makes provision for payment of fair and adequate compensation and a right of access to the courts for the determination of the investors' interest or right and the amount of compensation to which 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.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

ACCESSIBILITY

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.

The surrounding settlements, which consist of three villages, are inhabited by approximately 11,000 people. The village inhabitants are mainly farmers with family holdings which are linked by minor roads and footpaths.

CLIMATE

The area has humid tropical climate, with an average annual rainfall of approximately 1,800 mm, concentrated in the wet season from May to October. The two season regimes (dry and wet) are influenced by two dominant wind currents prevailing in the Project area.

More than 80% of the total annual rainfall is received between the months of May and October with the mean annual total in excess of 2,500 mm. The mean maximum and minimum temperatures across the Project region are over 32°C (in the month of March) and 24°C (in the month of August) respectively. The highest relative humidity range within the Project area is between 81% and 91% corresponding to the wettest months (June through August).

Operations are possible year-round.

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 sheet metal roofs. There are no utility provisions (e.g., electricity, potable water supply, or wastewater collection system), hence there is a reliance on streams and boreholes for water and gas/wood for energy supply.

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

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

INFRASTRUCTURE

The only infrastructure present at the site is the historic open pit and tailings. Currently there is no electrical power or water utilities at the site. The Project is, however, situated close to the existing local road infrastructure which connects the Project to the local townships and communities.

There is sufficient space within the tenements to enclose future possible mining activities, waste dumps, processing plant, and tailings storage facilities.

Thor 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 Ilesha, located approximately 25 km north of the Project area.

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 30 m.

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

6 HISTORY

PRIOR 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 Thor'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 Ilesha S. E. (Sheet 243 S.E.) of the Nigeria 1:50,000 Series was compiled from aerial photography and ground control by the Government of Canada as part of an aid program with the Government of Nigeria. The map sheet has a contour interval of 50 ft. This appears to have been the topographic control 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 15 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 and a three-year temporary title.

- 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. A detailed drilling of the known mineralised zone commenced.

- 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 mineralisation. 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.

- 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.

EXPLORATION HISTORY

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

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 mineralisation 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).

SOIL SAMPLING

As part of the exploration carried out on the property by various companies, a total of 2,898 recorded soil samples have been collected from the Project area (Table 6-1).

TABLE 6-1 HISTORICAL SOIL SAMPLING
Thor Explorations Ltd – Segilola Gold Project

Company	Number of Au analyses completed
Hansa/IGMDC	1,882
CGA	1,016
Total	**2,898**

Hansa/IGMDC collected a total of 1,882 samples. 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 (Figure 6-1).

A focussed soil sampling program was carried out by CGA, following the northern and southern strike projections of the known mineralisation (Figure 6-2). 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 mineralisation (Figure 6-1 and Figure 6-2).

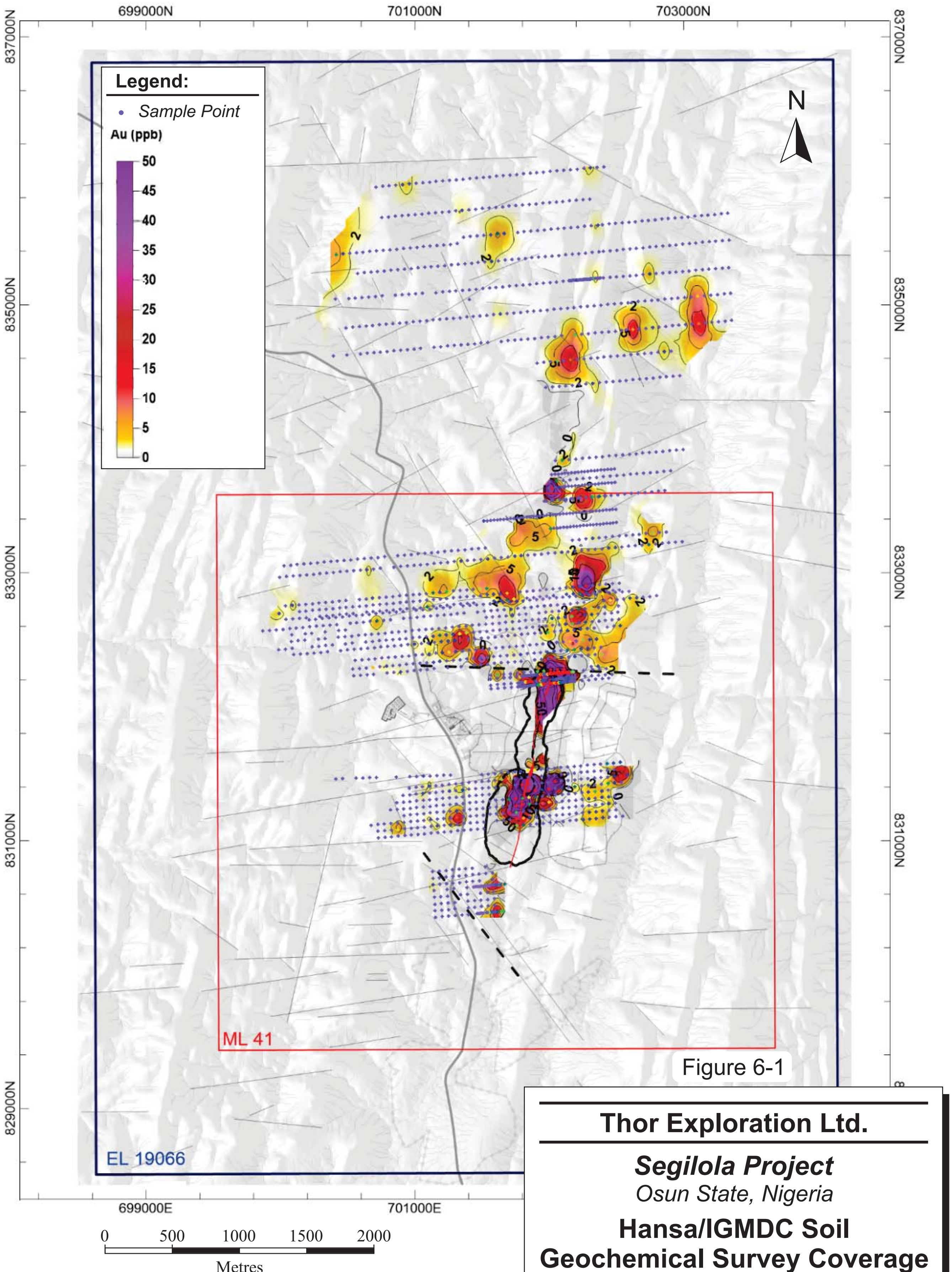


Figure 6-1

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Hansa/IGMDC Soil Geochemical Survey Coverage

March 2019

Source: Thor, 2019.

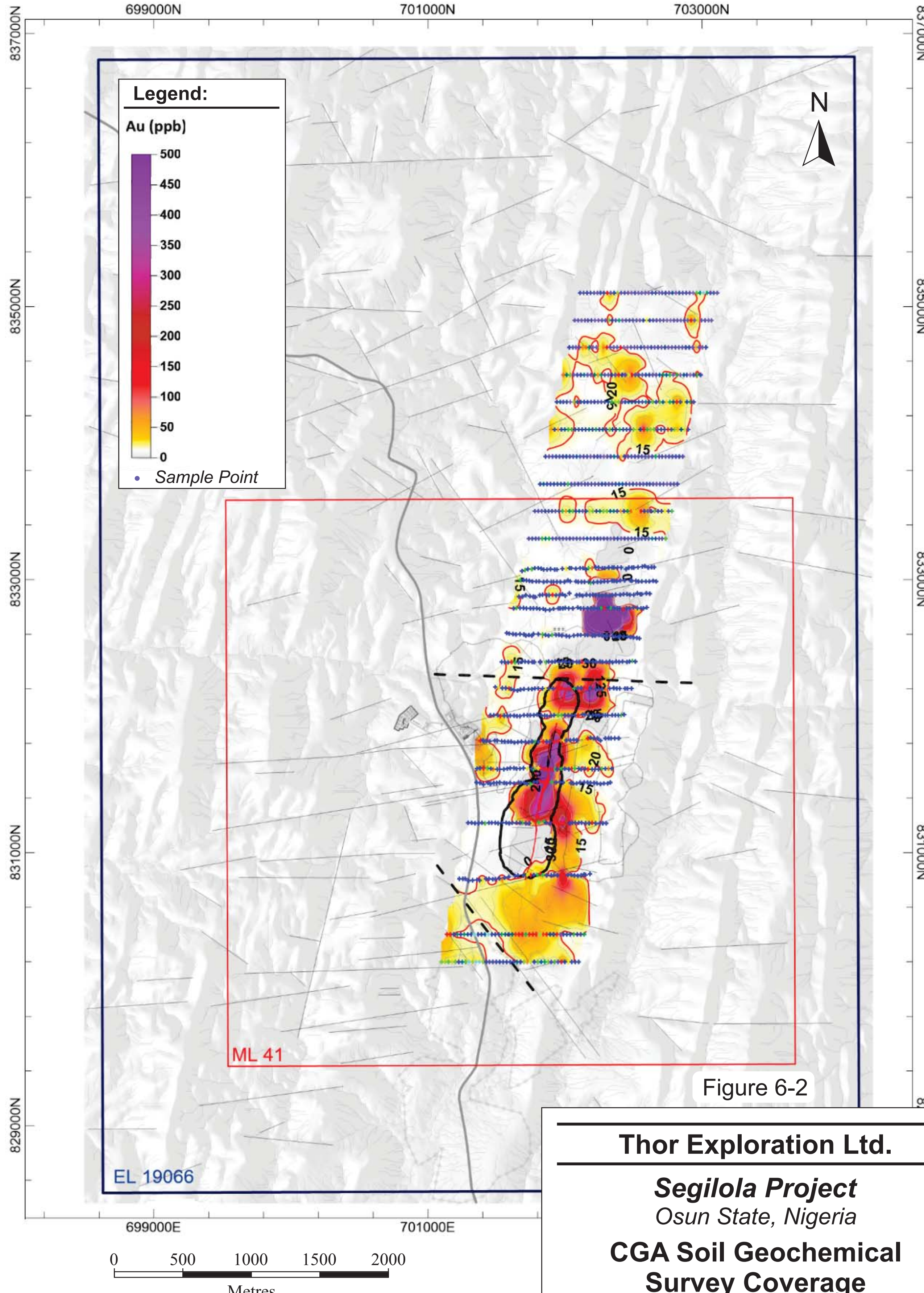


Figure 6-2

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

CGA Soil Geochemical Survey Coverage

March 2019

Source: Thor, 2019.

TRENCHING

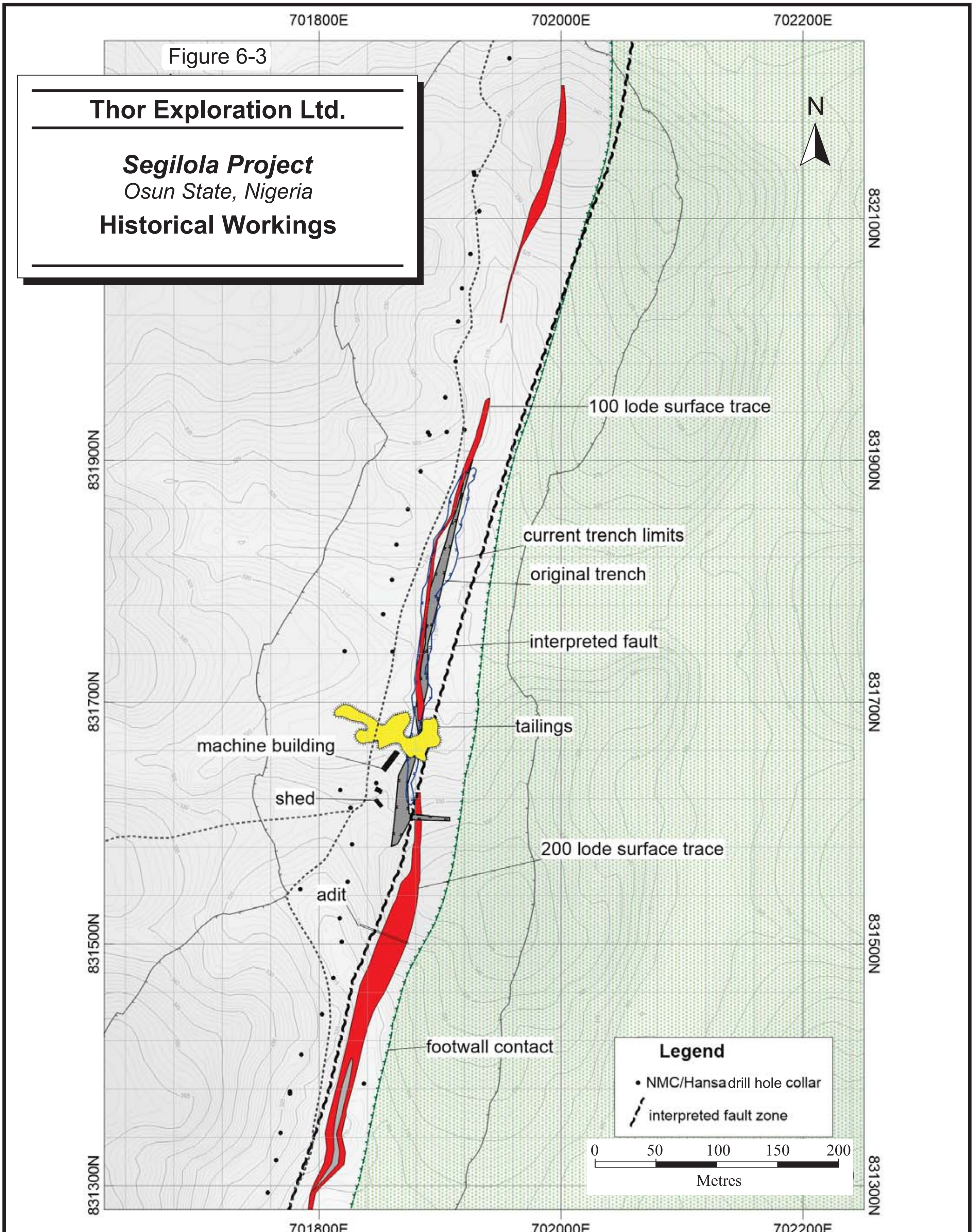
Both Hansa/IGMDC and CGA undertook several trenching programs. These trenches were typically between 0.5 m and 1.5 m cm deep and were focussed mostly on the outcrop of the gold-bearing vein system. The mineralised vein system is well exposed in a trench in the central part of the resource area.

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 6-3). The operators processed the ore with a second-hand stamp mill together with a ball mill and tables acquired from Ghana.

Presently, the trench has mostly collapsed and filled with water. At the southern end of the trench, it appears that the vein pinched out at the surface and a small cross-cut was made towards the east, presumably to explore for the vein. This pinch-out is consistent with the southerly plunge of the high-grade shoot at this point.

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



March 2019

Source: Thor, 2019.

HISTORICAL RESOURCE ESTIMATES

Historical mineral resource estimates prepared by previous owners prior to Thor's acquisition in 2016 are listed in Table 6-2. 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 6-2 HISTORICAL MINERAL RESOURCE ESTIMATES
Thor Explorations Ltd – Segilola Gold Project

Company	Date	Indicated Mineral Resources			Inferred Mineral Resources		
		Tonnes (Mt)	Grade (g/t Au)	Contained Gold (oz)	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (oz)
Pineridge Ltd	1992				1.06	10.1	347,000
Hansa	1991				1.4	6.0	270,000
Odessa Resources Pty Ltd	2009[1]	3.66	4.40	522,000	0.82	4.10	106,000

Notes:
1. In 2009, Odessa Resources Pty Ltd produced an updated Mineral Resource estimate for CGA based on drilling completed during 2009.

7 GEOLOGICAL SETTING AND MINERALISATION

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 mineralisation 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 of the ISZ, where the Project is located, and the rocks to the west.

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.

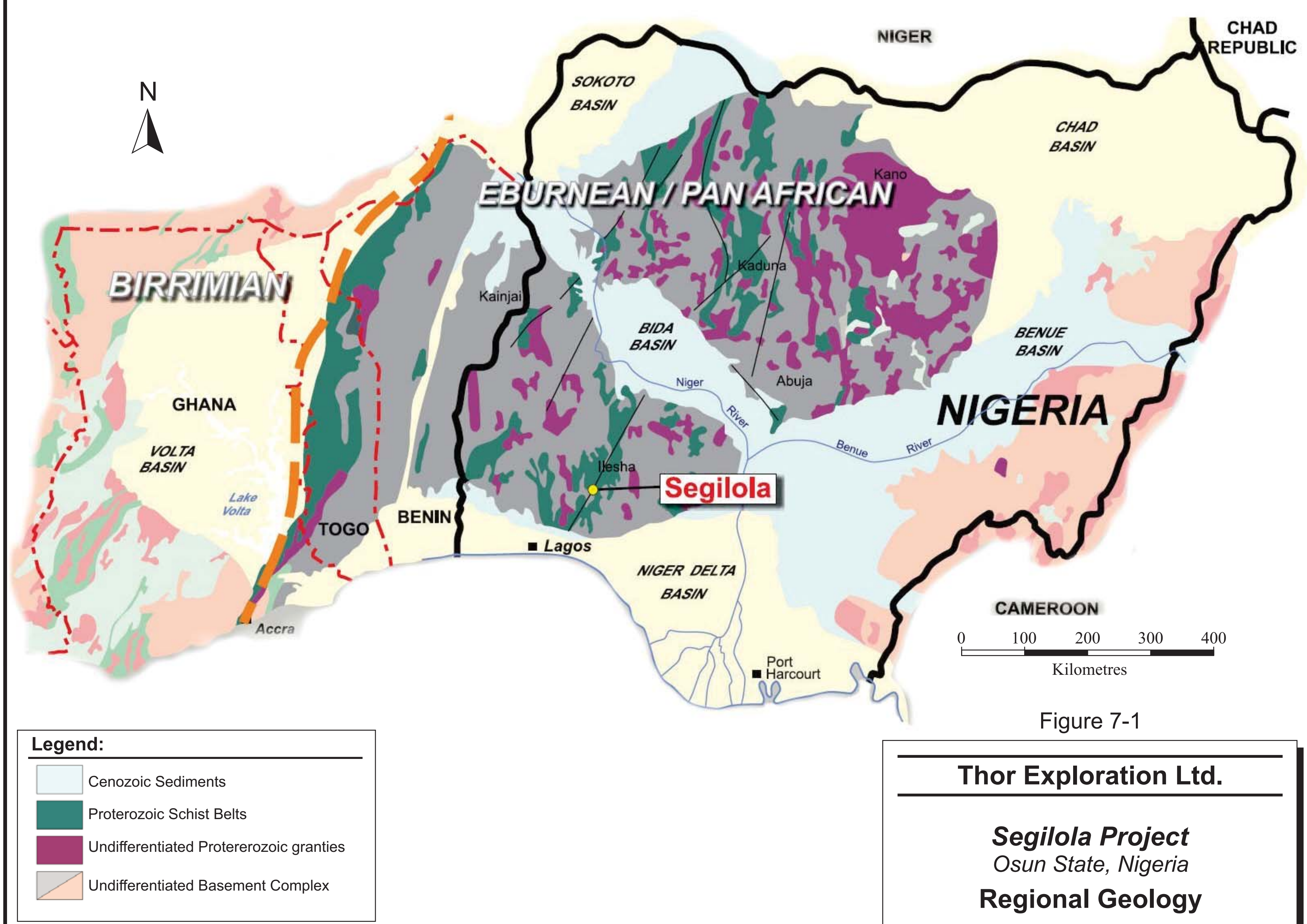


Figure 7-1

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Regional Geology

March 2019

Source: Thor, 2019.

LOCAL GEOLOGY

LITHOLOGY

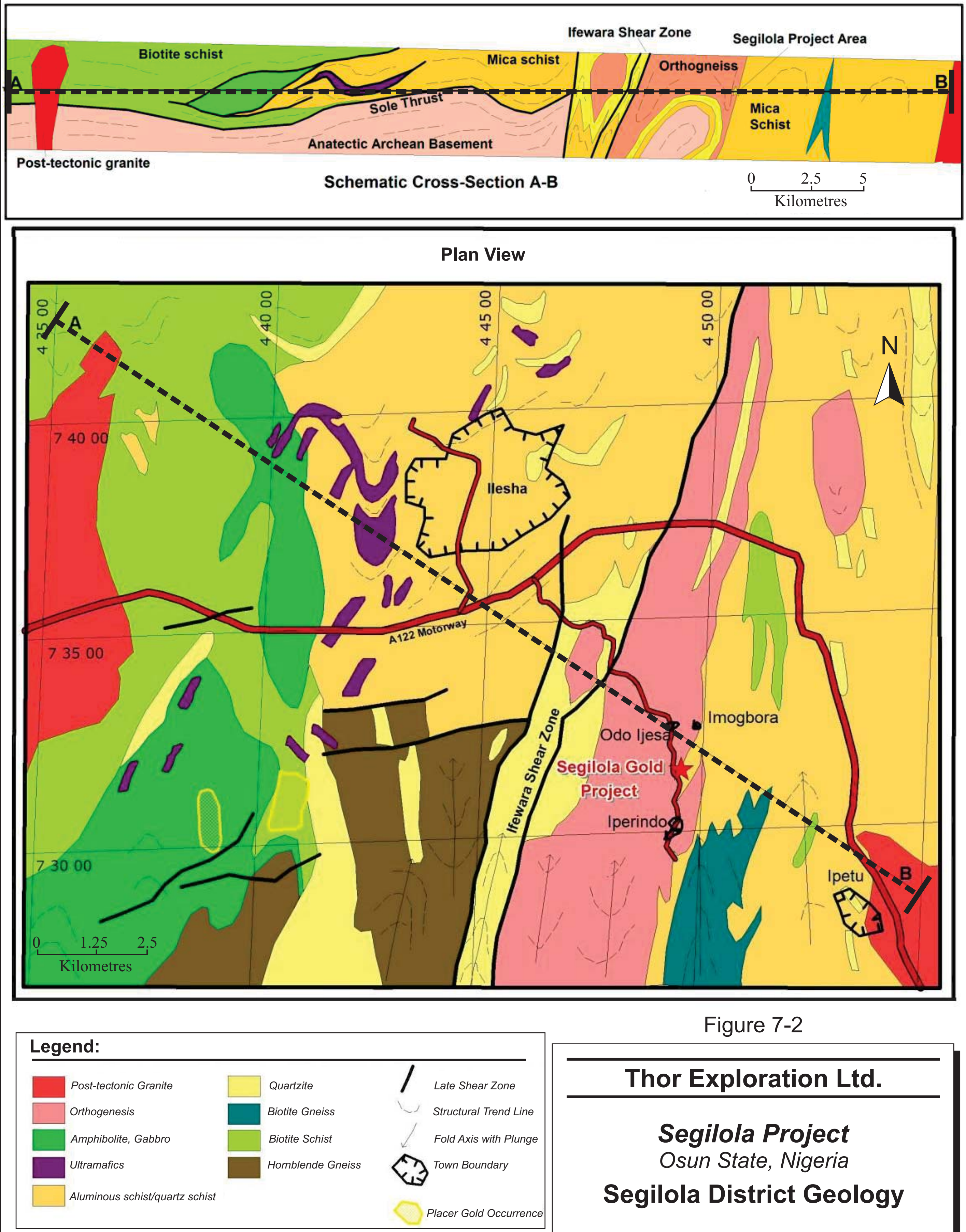
According to Caby and Boesse (2001), an Archaean basement (U-Pb zircon ages of 2,600 Ma) outcrops to the west of the area (Figure 7-2). It is typically composed of grey gneiss with lenses of orthogneiss of tonalitic-granodioritic composition, along with some lenses of amphibolite. It is extensively intruded 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.

STRUCTURE

Caby and Boesse (2001) distinguish two main deformation events; D1 and D2, both of Pan-African age. D1 generally produced recumbent (flat-lying) high temperature foliations with fold axes typically trending from 120º to 150º. 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 040º to 080º, which is interpreted as the overall sense of tectonic movement.



March 2019

Source: Thor, 2019.

Figure 7-2

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Segilola District Geology

To the east of the ISZ, the D2 event produced upright refolding of the D1 foliation at various scales, with sub- horizontal fold-axes trending 010º to 030º. Steep, ductile, north-northeast trending shear zones were also formed during this event and host a range of mylonitic rock types and Pan-African pegmatites. To the west of the ISZ, the D2 event is represented by large basin and dome structures deforming the D1 foliation. Mylonites within the ISZ exhibit a consistent flat dextral sense of shear indicators.

Caby and Boesse (2001) suggest that although it may have been initiated as a lateral ramp during the D1 thrusting event, the shear zone was mainly active during the D2 event, under conditions of decreasing temperature, from sillimanite to greenschist (retrogressive) facies metamorphic conditions. It is locally cut by Phanerozoic east-northeast trending brittle faults, with minor dextral offsets.

PROPERTY GEOLOGY

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.

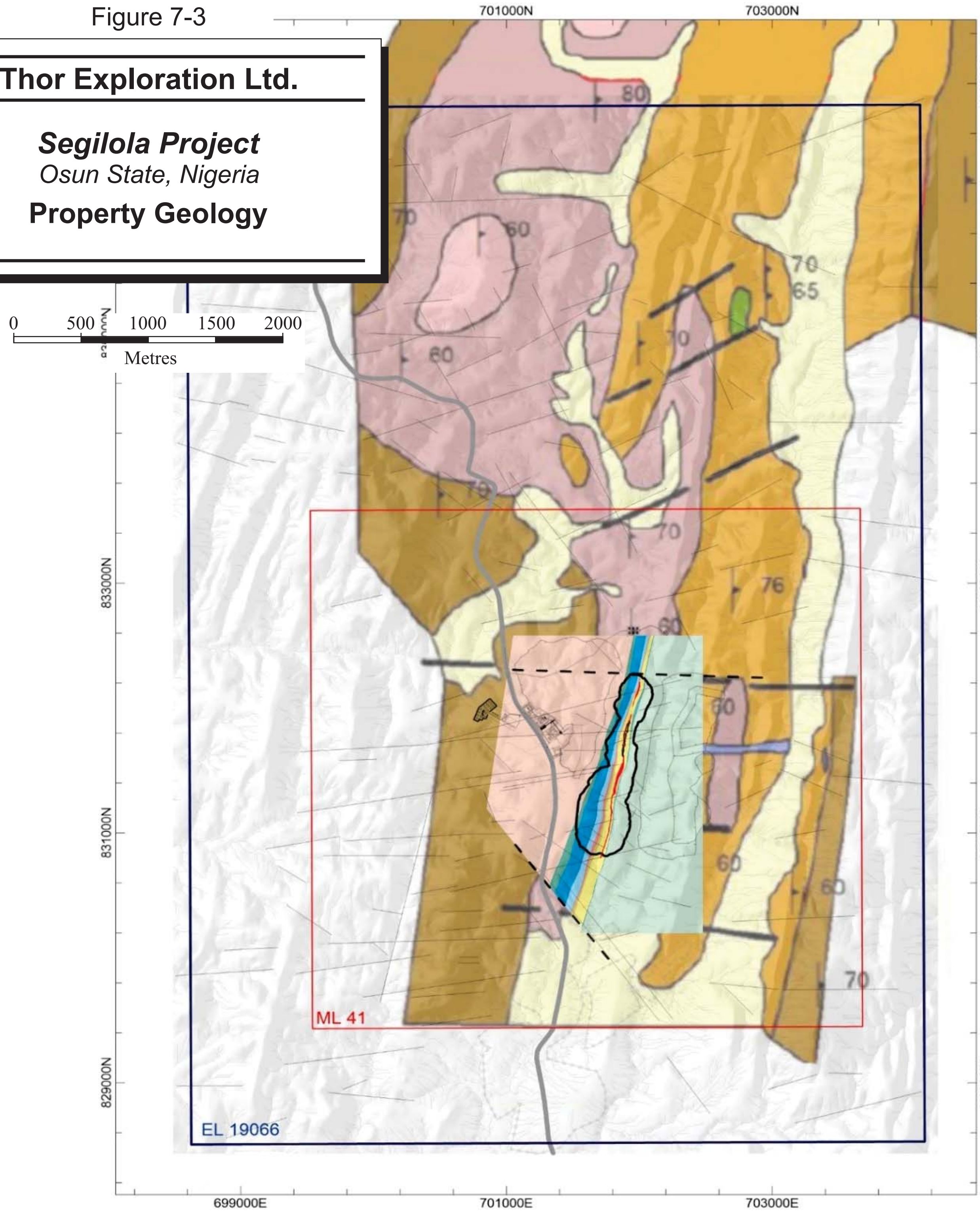
Figure 7-3

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Property Geology



Legend:

Legend (Hansa)	*Alluvium*	**Legend (Thor)**	*Granite/Gneiss*		
	Amphibolite		*Biotite Schist*		
	Biotite Gneiss		*GDM1 Granodiorite*		
	Dolerite Dyke		*GDM2 Granodiorite*		
	Gneiss		*GDS Alteration Zone*		*Mineralisation*
	Orthogenesis		*DGS Biotite Schist*		*Main Fault (Interpreted)*
	Quartzite, quartz mica schist		*SZQ1/2 Altered Schist*		*Minor Fault (Interpreted)*
			Calc Silicate		*DFS Pit Perimeter*
			Footwall Biotite Schist		

March 2019 **Source: Thor, 2019.**

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 a more granitic one, but a metamorphic overprint 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. Another outcrop of garnet bearing schists/gneisses is located 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. are skarns). They are locally sulphide-bearing and so have some potential for gold-(copper) mineralisation, however, the drilling has not identified gold mineralisation in the footwall rocks within the Project area.

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 290º or from 090° to 110º. Mapped joints, however, cluster with steep dips towards 005º and 185º. In measurements from the oriented core there is a preponderance of foliations dipping steeply towards 315º or 135º (20 measurements) and fractures and joints dipping steeply towards 320º and 140º (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.

MINERALISATION

Drilling results demonstrate that gold mineralisation occurs in fractured pale to dark grey coloured smoky quartz veining, sheared pegmatite, and silica/chlorite/carbonate alteration. The mineralisation is hosted in three steeply dipping vein sets or lodes; the Hanging Wall Lodes (Lodes 100 and 300) and the Footwall Lode (Lode 200). Together these form an elongate mineralised zone striking 010º and dipping 60º to 70º towards the west within a single shear zone, primarily in biotite gneiss. The currently drilled mineralised zone is approximately 2,000 m in strike length, between 70 m and 200 m in depth, and between 2 m and 20 m in true thickness.

SEQUENCES

The mineralisation 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 mineralisation 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 7-4). The depth of weathering varies from one metre to two metres in the west to five metres 10 m in the mineralised shear zone.

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
Thor Explorations Ltd – Segilola Gold Project

Lode	Description	Estimated Average True Width (m)	Interpretation	Average Au in uncut 1m Composites (g/t)
100	High Grade Hanging Wall Lode	4.0	Northern continuation of Lode 200	6.62
200	Main Footwall Lode	10.0	Developed only southeast of Oblique Strike-Slip Fault	3.55
300	Southern Hanging Wall Lode	3.0	Developed only southeast of Oblique Strike-Slip Fault	11.52

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 mineralisation 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 7-5 and Figure 7-6, respectively.

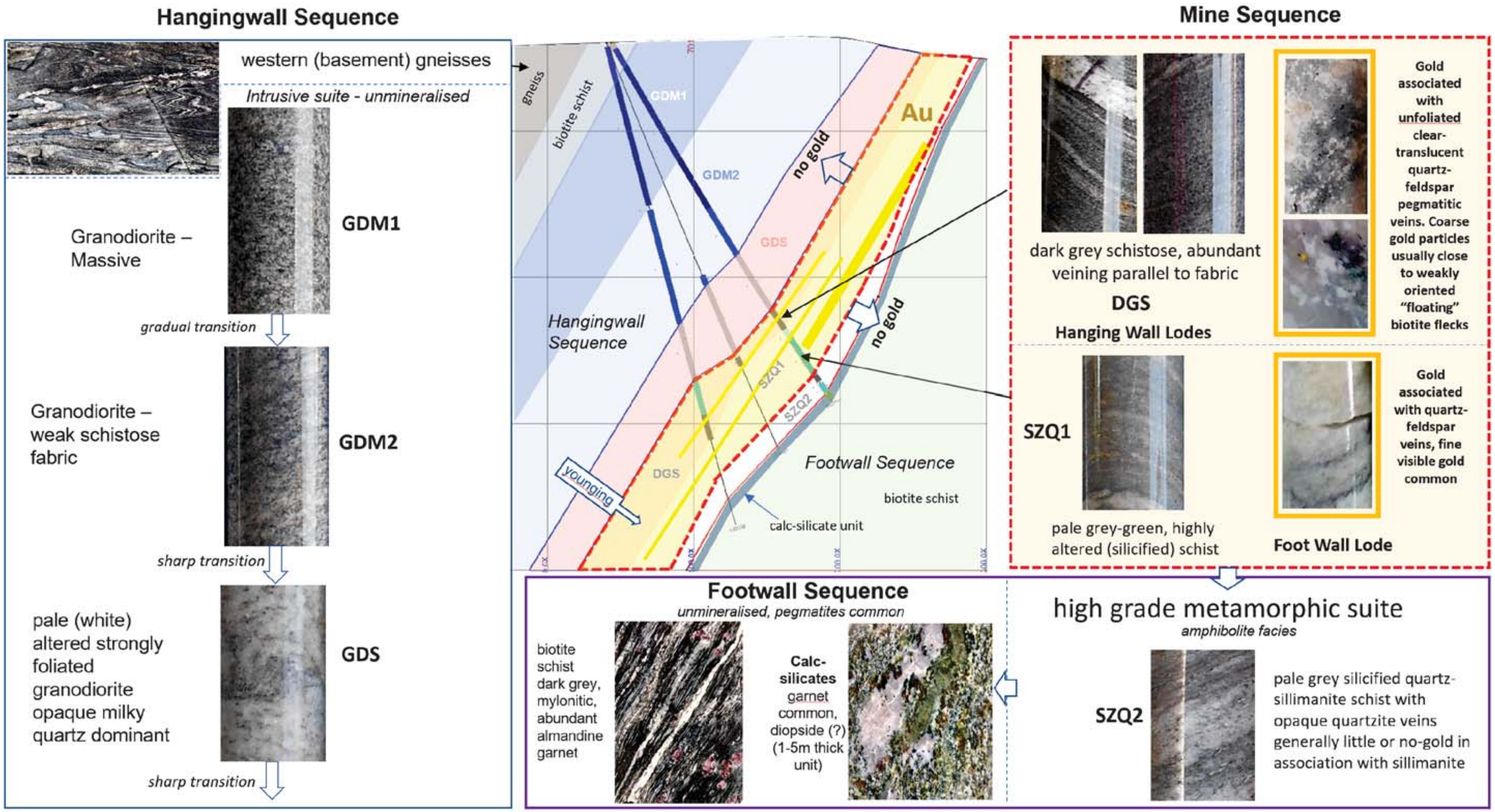


Figure 7-4

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Mine Sequence

Source: Thor, 2019.

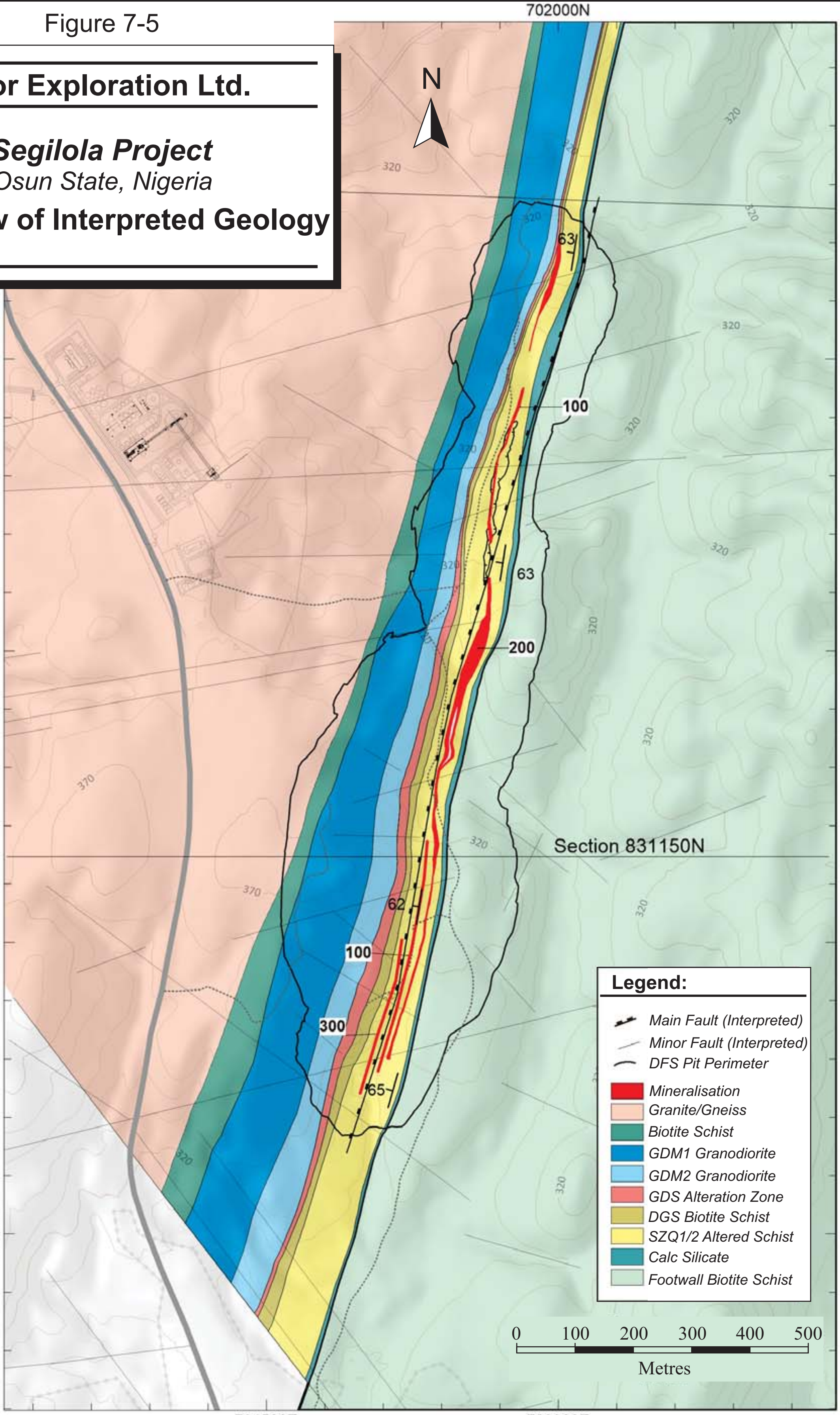
Figure 7-5

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Plan View of Interpreted Geology



March 2019

Source: Thor, 2019.

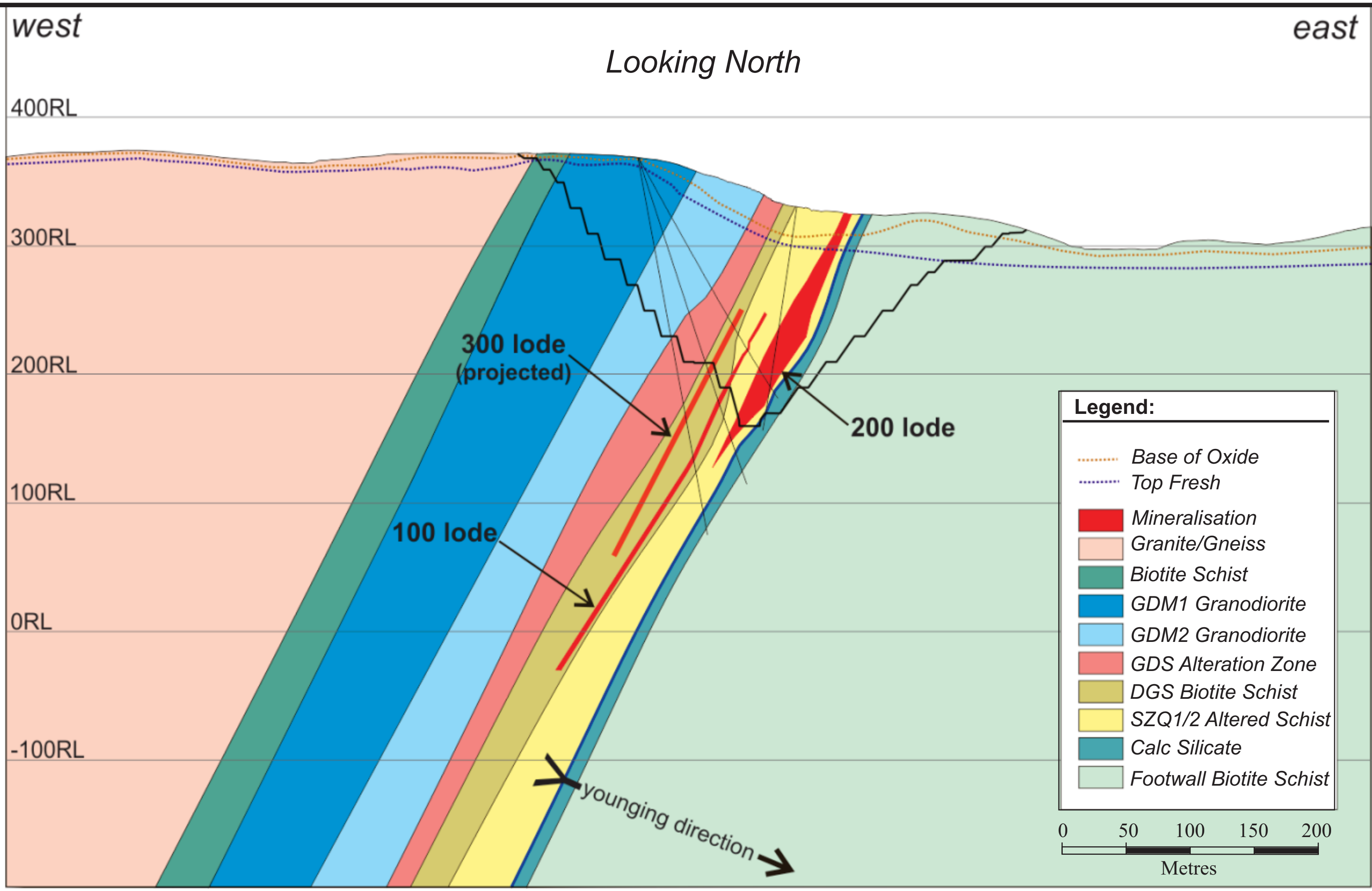


Figure 7-6

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Cross Section 831,150 N

March 2019

Source: Thor, 2019.

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

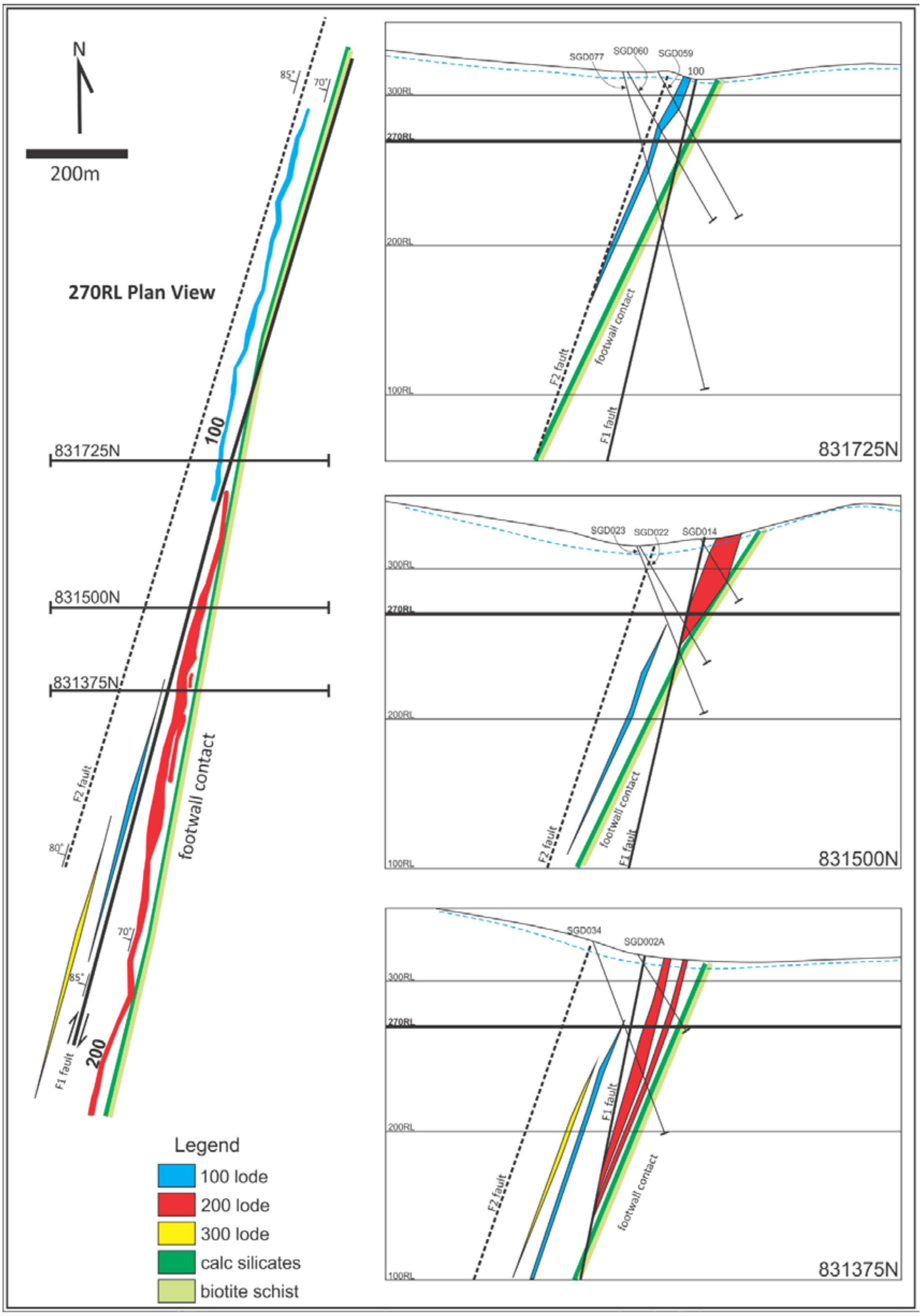
Strike slip faults occur along the axis of the mineralisation which, itself, may be developed either side of the faults and is only lost within the fault itself, over several metres, in the north-central part of the deposit.

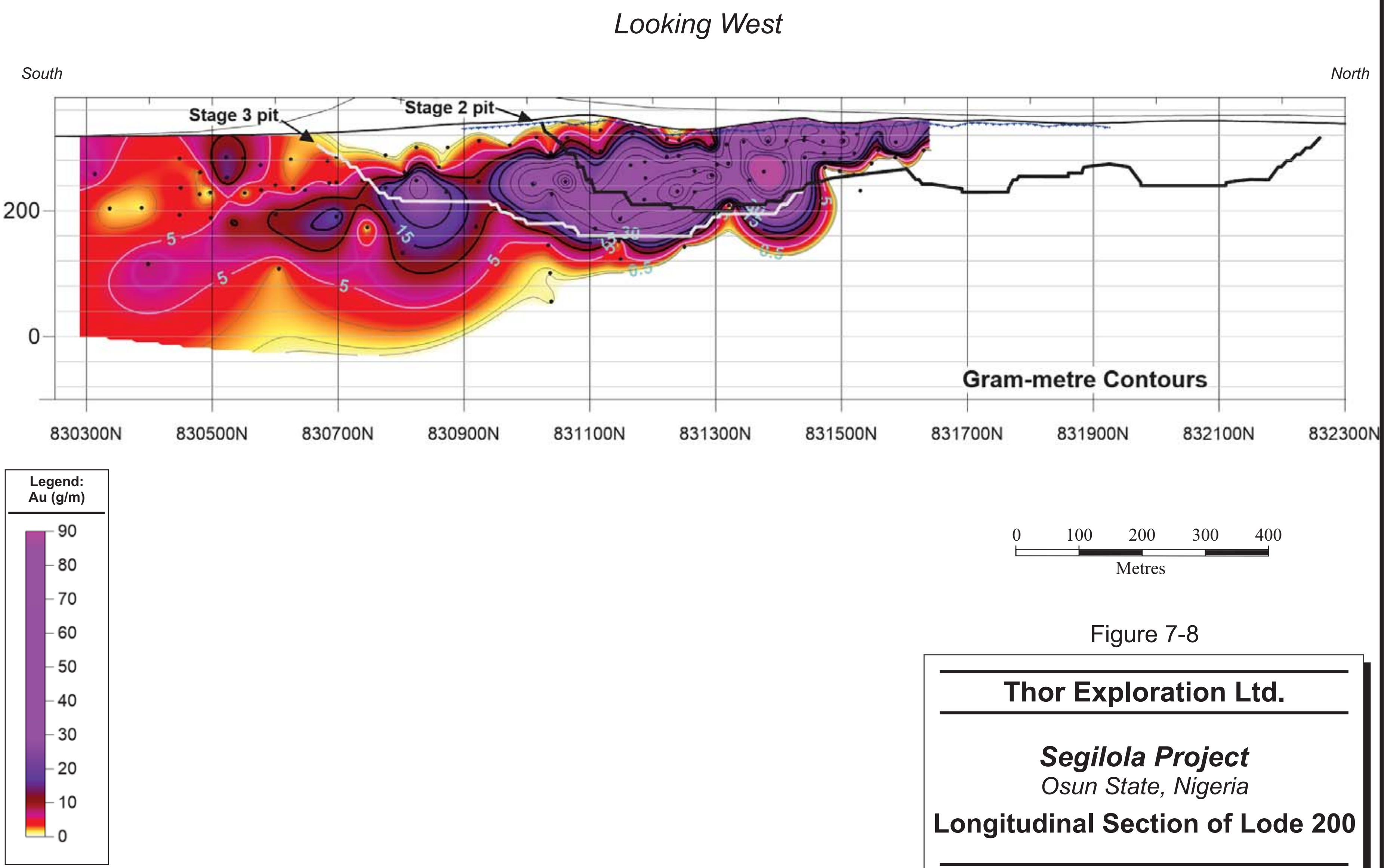
To the north, the two features converge with the fault passing into a mylonitic footwall. In this area, the main Footwall Lode 200 is absent as it has been faulted out.

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 mineralisation.

Lode 200 has an apparent southerly plunge (Figure 7-8).

FIGURE 7-7 SCHEMATIC CROSS SECTIONS ALONG STRIKE (AT 270RL) SHOWING STRUCTURAL INTERACTION OF THE LODES, FOOTWALL, AND SUB-VERTICAL STRIKE-SLIP FAULTS





March 2019

Source: Thor, 2019.

Figure 7-8

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Longitudinal Section of Lode 200

This plunge is not thought to be due to a plunging shoot but appears to be structurally controlled by the intersection of a sub-vertical fault and the westerly dipping footwall.

These two surfaces diverge slightly towards the south thus creating a shallow south-plunging intersection vector. The mineralisation above the intersection of the surfaces is robust and predictable in its location.

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 mineralisation 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' mineralisation within a characteristically grey-green, strongly silicified zone of biotite schists and gneisses (SZQ1).

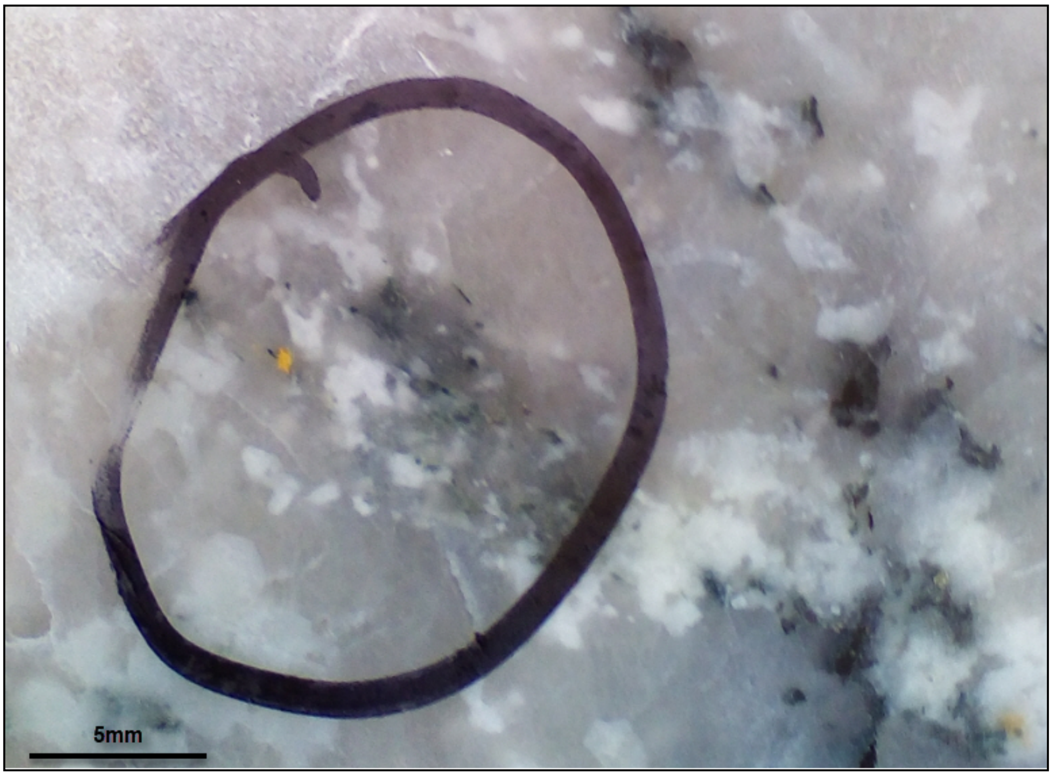
The mineralised lodes generally comprise highly silicified fine-grained, foliated biotite gneiss typically intruded by both discordant and concordant pegmatitic quartz-feldspar veins. Typical specimens of these rocks are shown in Figure 7-9.

FIGURE 7-9 PEGMATITIC QUARTZ-FELDSPAR VEIN (SEG001) AND BIOTITE-QUARTZ (QUARTZITE) SCHIST (SEG002) FROM DRILL HOLE SGD155

Source: Thor, 2019

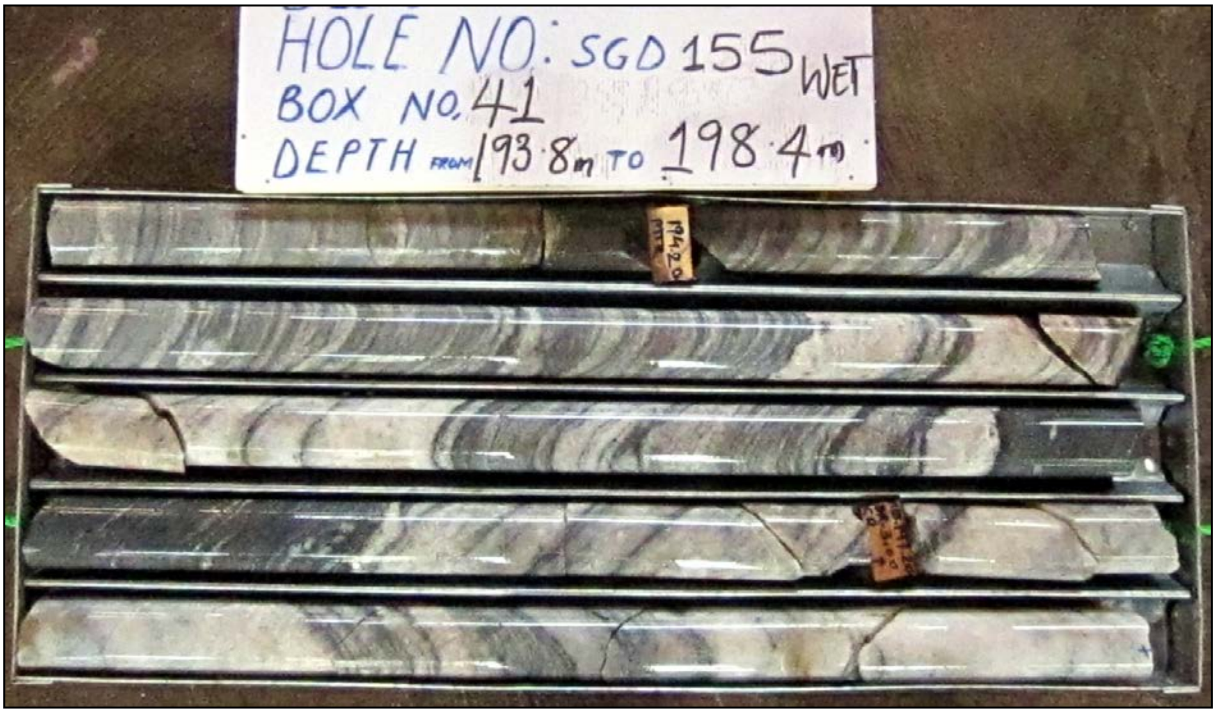
Shearing, fracturing, and alteration influence the location of gold mineralisation. This relationship has generated multiple zones of gold mineralisation hosted by shears now represented by chlorite and calcite alteration, together with quartz veining and pyrite development (Figure 7-10 and Figure 7-11).

FIGURE 7-10 VISIBLE GOLD WITHIN PEGMATITIC QUARTZ-FELDSPAR VEIN – FROM DRILL HOLE SGD155



Source: Thor, 2019

FIGURE 7-11 INTENSE SILICA-CARBONATE ALTERATION OF MINERALISED ZONE IN DRILL HOLE SGD155



Source: Thor, 2019

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 mineralisation, or to remobilisation of sulphides.

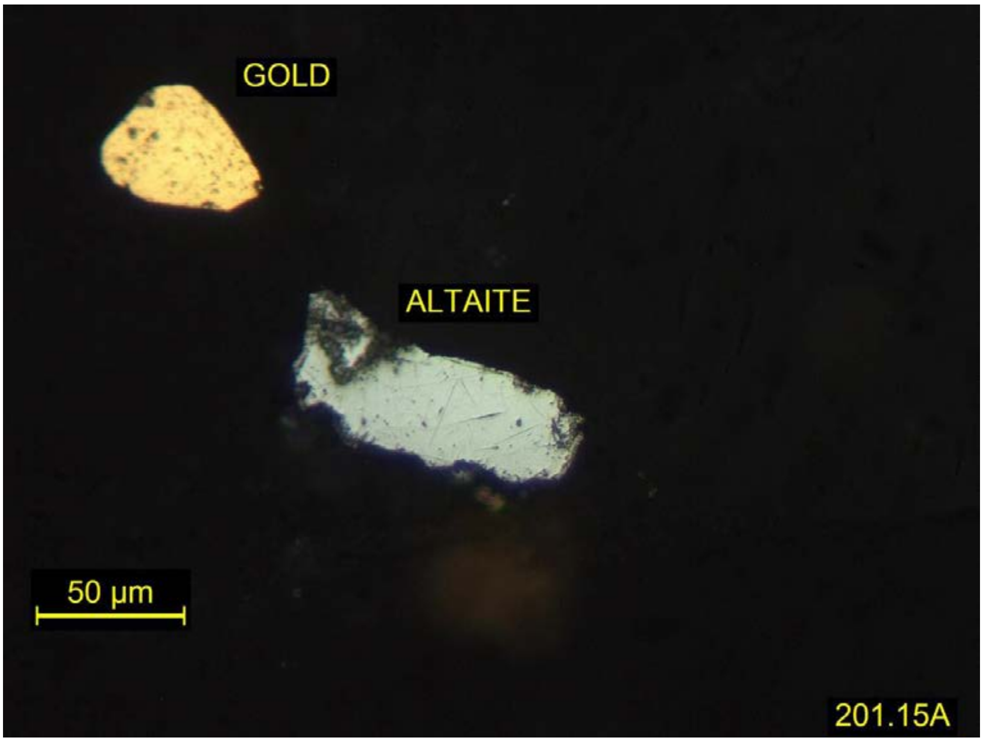
Native gold is visible in both altered wall rock and in quartz-feldspar veins. It occurs with petzite (a silver gold telluride), within pyrite, and quartz veins. The typical size of native gold blebs is approximately 10 µm. Gold, either as native grains, flakes and blebs occurs together with gold-on-pyrite in alteration zones, along tension gashes, hair-like fractures, joints, and minor faults.

In 2017, two samples, one of hanging wall quartz-feldspar vein mineralisation (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 and conducted by Townend Mineralogy Laboratory (Perth).

The sample from SDG156 was composed of a very coarse, slightly deformed and slightly recrystallised quartz containing around 10% fresh microcline. Two slides were produced from this sample (Figure 7-12 and Figure 7-13). These contained only traces of sulphides present as angular linear grans of pyrite or clusters with chalcopyrite. There were at least four tellurides detected, usually in close association with gold. The main non-gold telluride was altaite (PbTe). Coarsest grains to 200 µm may be in contact with the gold telluride calaverite, only rarely was altaite in contact with gold.

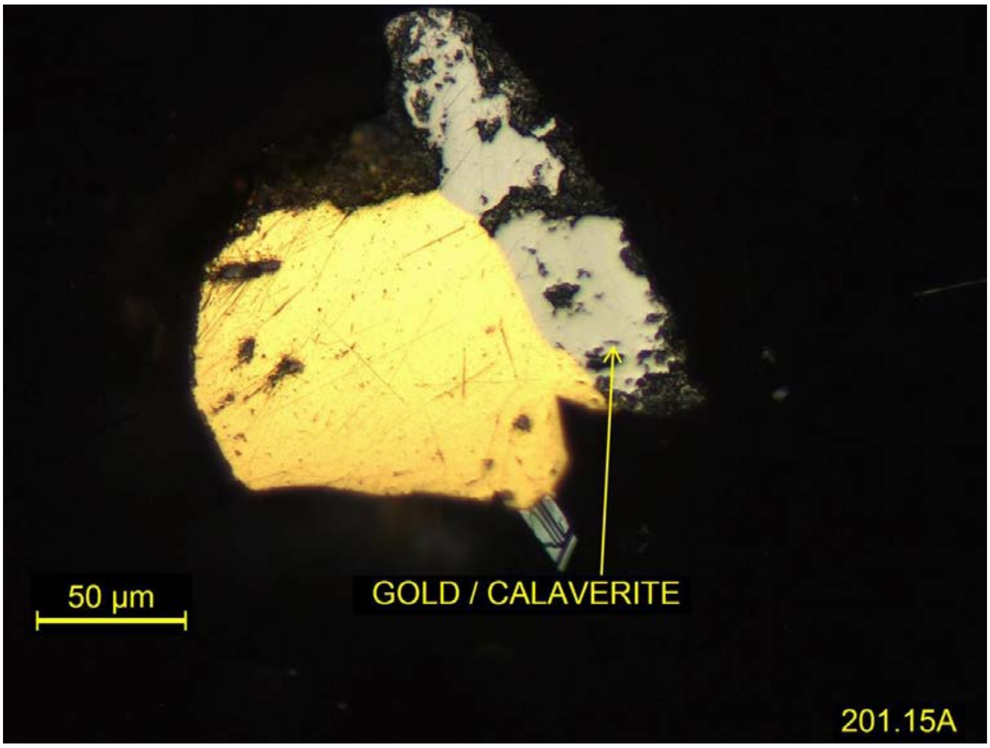
The sample from SGD155 (Figure 7-14) contained vein quartz and quartzite-host rock that is essentially identical to the sample from SGD156. The gneiss had a dominant quartz feldspar mineralogy and contained well oriented fresh biotite and muscovite, as well as oriented fibrous sillimanite. The slide produced from this sample contained sulphides and rate magnetite but no gold. Pyrite was the most common sulphide present as anhedral single grains, and also associated with chalcopyrite, pyrrhotite, and sphalerite plus a vein like mass. The other sulphides were single occurrences.

FIGURE 7-12 POLISHED SECTION 1 – SGD156



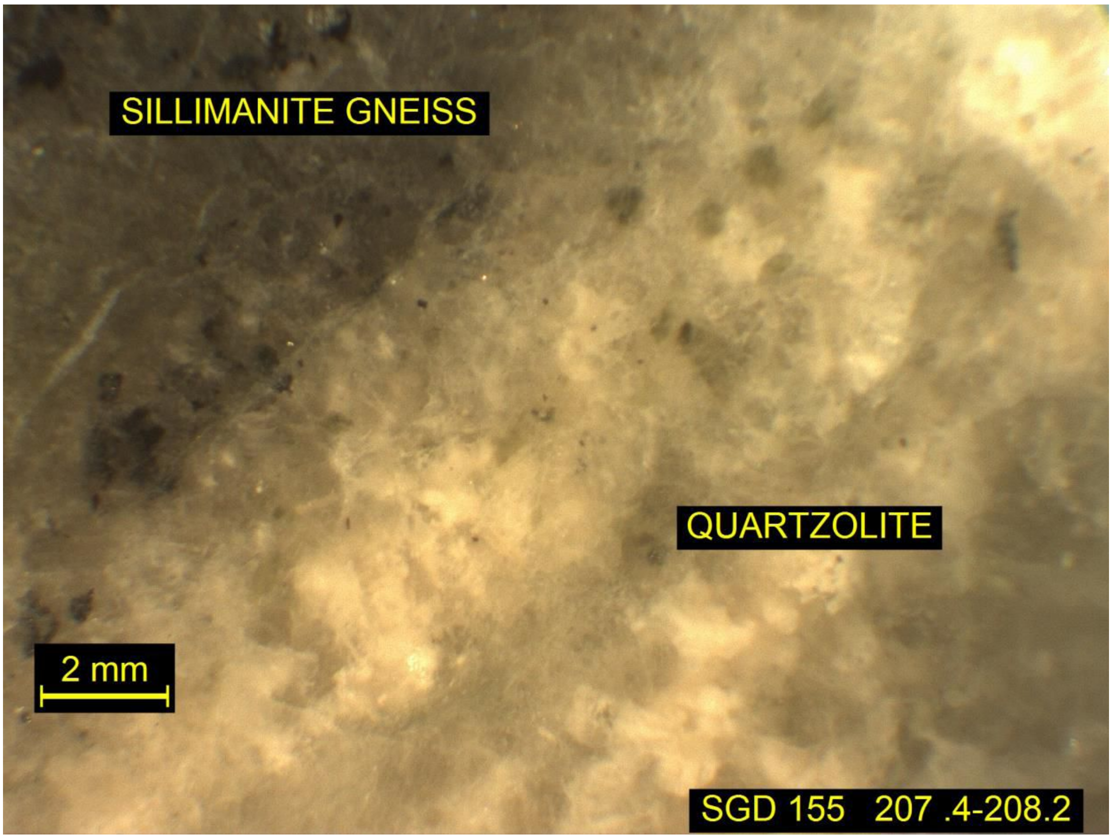
Source: Thor, 2019

FIGURE 7-13 POLISHED SECTION 2 – SGD156



Source: Thor, 2019

FIGURE 7-14 POLISHED SECTION 3 – SGD155



Source: Thor, 2019

TRACE ELEMENTS AND DELETERIOUS MINERALS

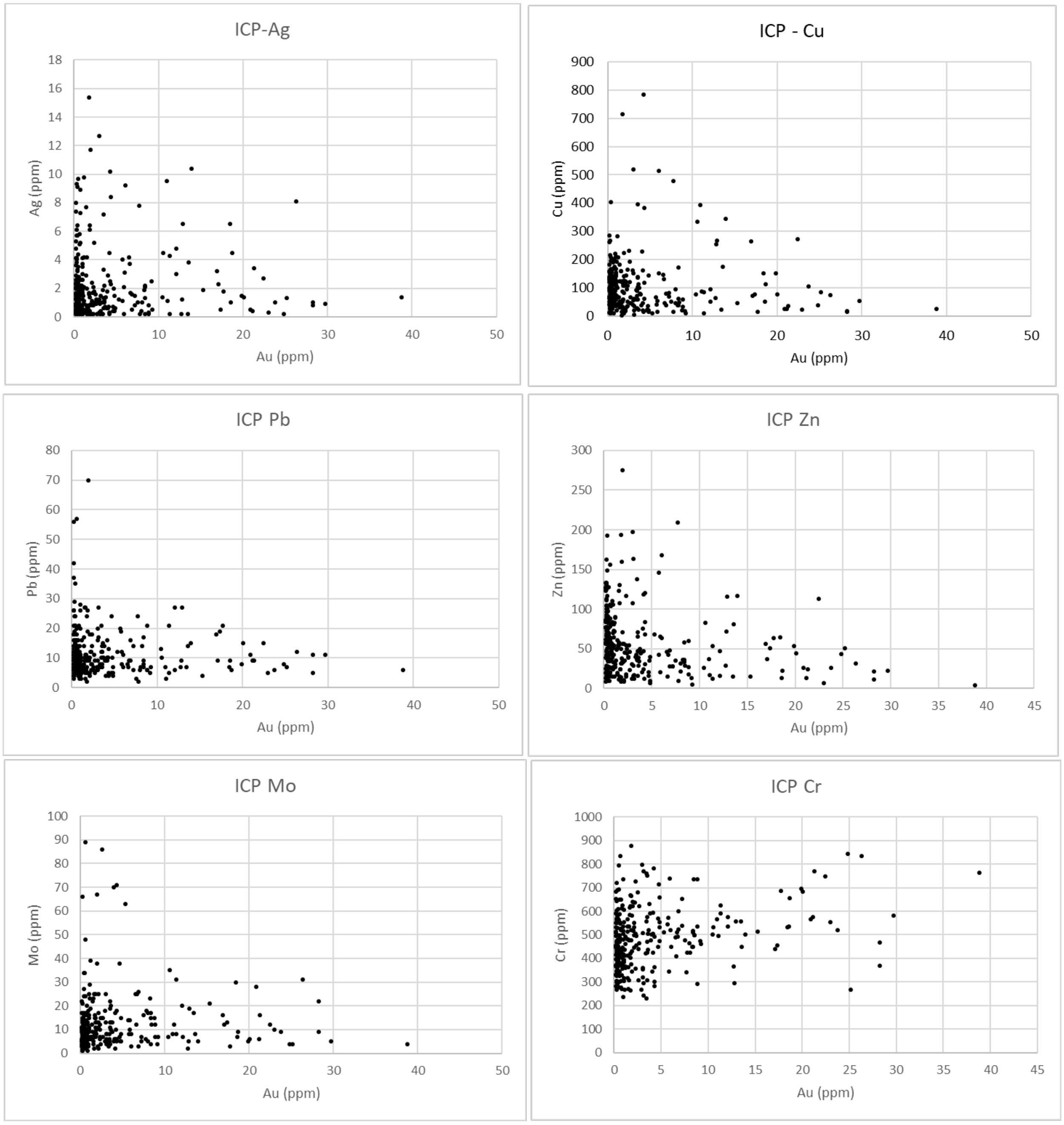
Geochemical analysis of 310 samples indicates an absence of deleterious elements (Table 7-2). It also shows that there is no correlation between gold and silver or any base metal elements (Figure 7-15).

TABLE 7-2 TRACE ELEMENT CONCENTRATIONS AT VARIOUS GOLD GRADE CUT-OFFS
Thor Explorations Ltd – Segilola Gold Project

Method	Units	Element	>0.2 g/t Au Cut Off	>0.5 g/t Au Cut Off	Variation	Variation %	>1.0 g/t Au Cut Off	Variation	Variation %
FA	ppm	Au	4.82	6.74	1.92	40%	8.51	3.69	55%
ICP	ppm	Ag	2.34	2.54	0.20	9%	2.39	0.05	2%
ICP	ppm	Cu	101.26	103.95	2.69	3%	105.16	3.90	4%
ICP	ppm	Pb	11.34	11.01	-0.33	-3%	10.77	-0.56	-5%
ICP	ppm	Zn	55.19	50.50	-4.70	-9%	49.40	-5.79	-11%
ICP	ppm	Mo	19.12	23.49	4.37	23%	21.87	2.75	12%
ICP	ppm	As	2.01	2.01	0.00	0%	2.01	0.01	0%
ICP	%	Fe	1.49	1.45	-0.04	-3%	1.46	-0.03	-2%
ICP	%	S	0.33	0.32	-0.01	-2%	0.32	-0.01	-4%
ICP	ppm	Sb	3.65	3.68	0.03	1%	3.75	0.11	3%
ICP	ppm	Te	10.06	10.09	0.03	0%	10.11	0.05	1%
ICP	%	Al	0.54	0.45	-0.08	-15%	0.44	-0.10	-21%
ICP	ppm	B	10.13	10.15	0.02	0%	10.17	0.04	0%
ICP	ppm	Ba	46.59	40.68	-5.91	-13%	39.66	-6.93	-17%
ICP	ppm	Be	0.54	0.52	-0.02	-3%	0.53	-0.01	-2%
ICP	ppm	Bi	2.15	2.22	0.07	3%	2.08	-0.07	-3%
ICP	%	Ca	0.21	0.19	-0.03	-12%	0.20	-0.01	-8%
ICP	ppm	Cd	0.71	0.64	-0.06	-9%	0.63	-0.08	-13%
ICP	ppm	Co	4.82	4.80	-0.02	0%	4.77	-0.05	-1%
ICP	ppm	Cr	474.09	491.17	17.08	4%	498.85	24.76	5%
ICP	ppm	Ga	10.36	10.35	-0.01	0%	10.43	0.07	1%
ICP	ppm	Hg	1.00	1.00	0.00	0%	1.00	0.00	0%
ICP	%	K	0.27	0.25	-0.02	-8%	0.24	-0.03	-13%
ICP	ppm	La	16.19	14.48	-1.71	-11%	13.83	-2.36	-16%
ICP	%	Mg	0.19	0.17	-0.02	-9%	0.16	-0.02	-13%
ICP	ppm	Mn	153.45	148.68	-4.77	-3%	150.32	-3.13	-2%
ICP	%	Na	0.03	0.03	0.00	-11%	0.03	0.00	-11%
ICP	ppm	Ni	19.48	20.18	0.70	4%	20.32	0.84	4%
ICP	ppm	P	110.40	99.43	-10.97	-10%	94.78	-15.61	-16%
ICP	ppm	Sc	2.41	2.32	-0.09	-4%	2.32	-0.09	-4%
ICP	ppm	Sr	10.30	8.22	-2.08	-20%	8.46	-1.84	-22%
ICP	ppm	Th	8.75	8.30	-0.45	-5%	8.25	-0.50	-6%
ICP	%	Ti	0.05	0.05	0.00	-9%	0.04	-0.01	-14%
ICP	ppm	Tl	10.00	10.00	0.00	0%	10.00	0.00	0%
ICP	ppm	V	13.55	13.63	0.08	1%	14.07	0.52	4%
ICP	ppm	W	10.02	10.00	-0.02	0%	10.00	-0.02	0%
ICP	ppm	Zr	5.17	5.20	0.04	1%	5.26	0.10	2%
		Count	310	216			167		

Note. FA- fire assay; ICP – inductively coupled plasma

FIGURE 7-15 TRACE ELEMENT SCATTER PLOTS RELATIVE TO GOLD



Source: Thor, 2019

8 DEPOSIT TYPES

Segilola is an orogenic-style lode gold deposit which occurs within a regional-scale shear zone.

Host rocks comprise an overturned sequence of high-grade amphibolite-facies meta-sediments intruded by a large, possibly differentiated, granodiorite sill-like body. The mineralisation is developed within a series of steeply dipping, tabular, very continuous, late-stage quartz-pegmatite veins that do no exhibit any form of significant deformation such as folding or faulting. The geological and mineralogical characteristics of the mineralised 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.

SURFACE SAMPLING

SOIL SAMPLING

From 2016 to 2018, Thor 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 (Figure 9-1). Multi-element ICP analyses were carried out on a total of 1,338 samples. No trace element associations with gold were detected, however, 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 mineralisation (Exploration Target Area 2 in Figure 9-3).

STREAM SEDIMENT SAMPLING

During 2018, Thor 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. Figure 9-2 shows the location of the sample points together with the significant results.

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 (Figure 9-3).

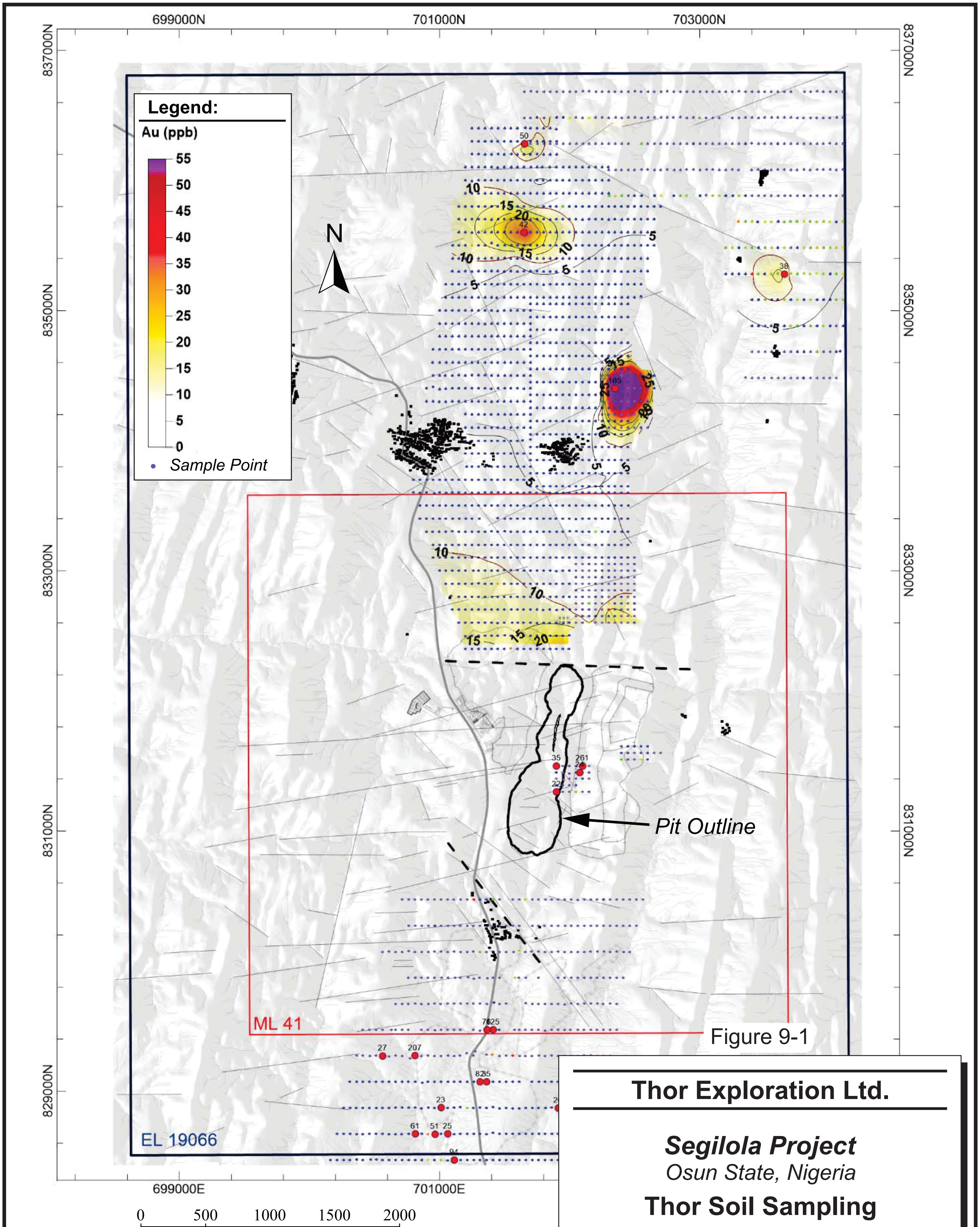


Figure 9-1

Thor Exploration Ltd.

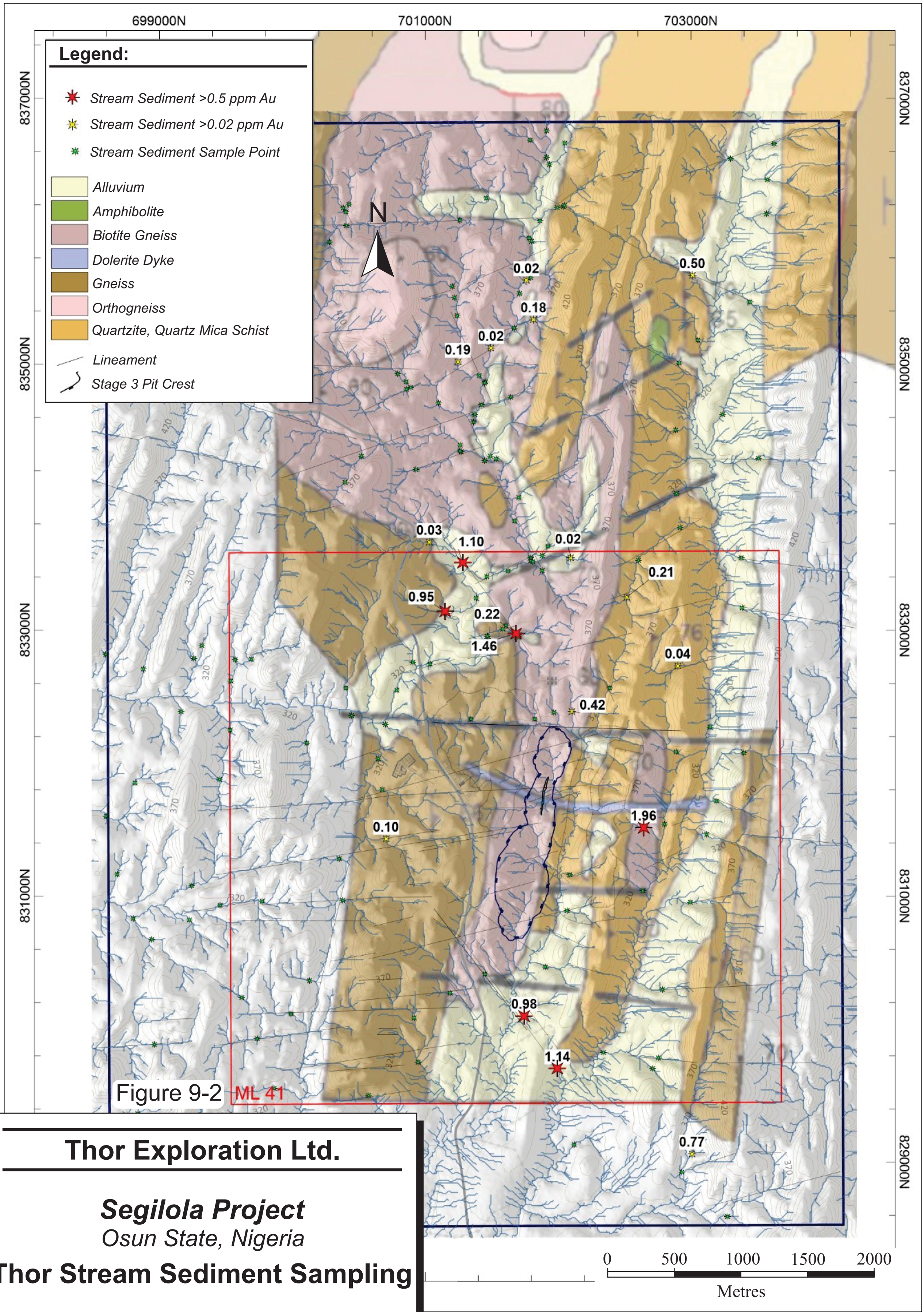
Segilola Project

Osun State, Nigeria

Thor Soil Sampling

March 2019

Source: Thor, 2019.



Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Thor Stream Sediment Sampling

March 2019

Source: Thor, 2019.

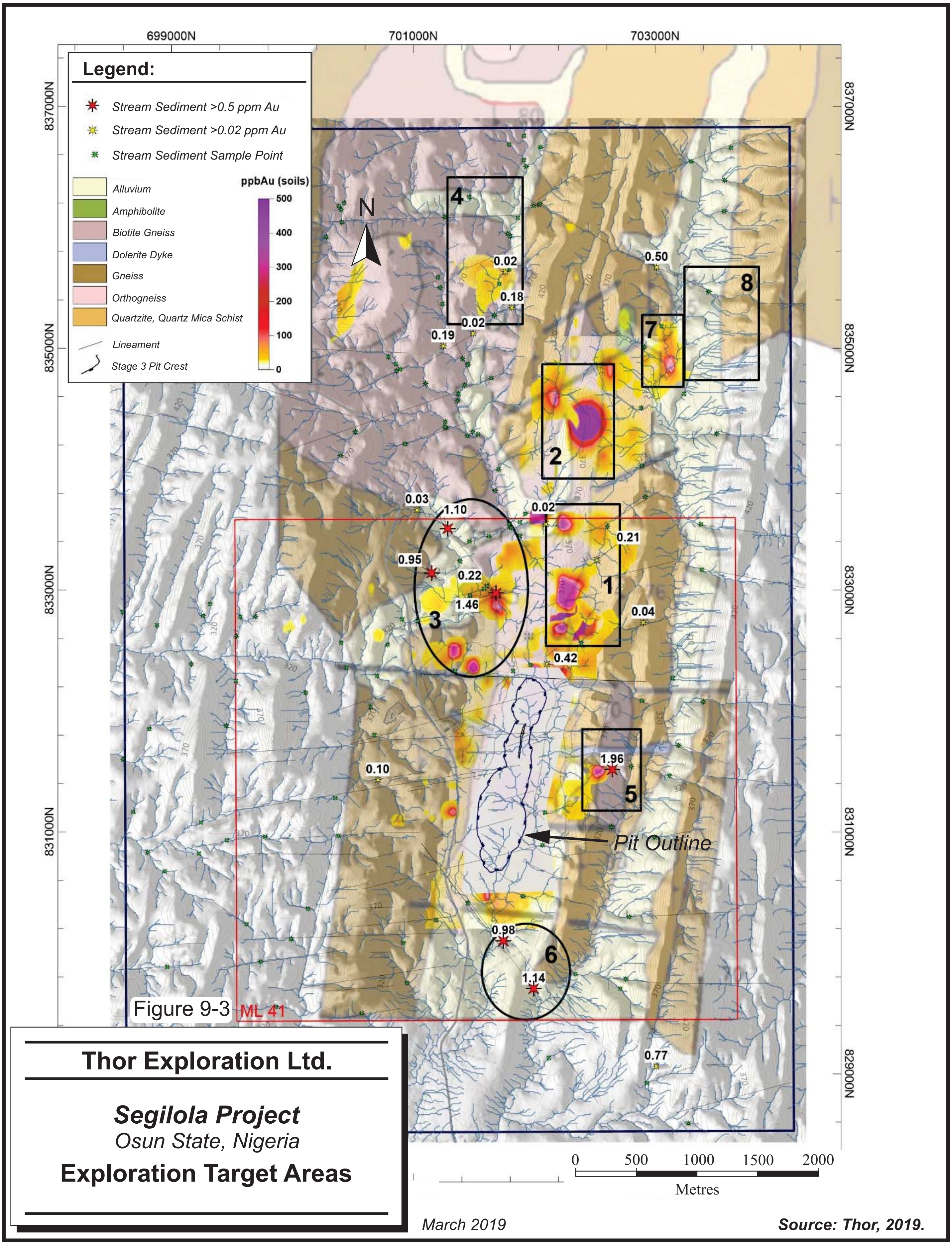


Figure 9-3

Thor Exploration Ltd.

Segilola Project
Osun State, Nigeria

Exploration Target Areas

March 2019

Source: Thor, 2019.

TARGET GENERATION

A total of eight Exploration Target Areas have been delineated from surface sampling and will be assessed in forthcoming exploration programmes (Figure 9-3, above).

TOPOGRAPHIC SURVEY

In 2018, Thor 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.

GEOPHYSICS

In 2018, Thor 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.

Figure 9-4 shows the total magnetic intensity in the vicinity of the Project. From this image 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.

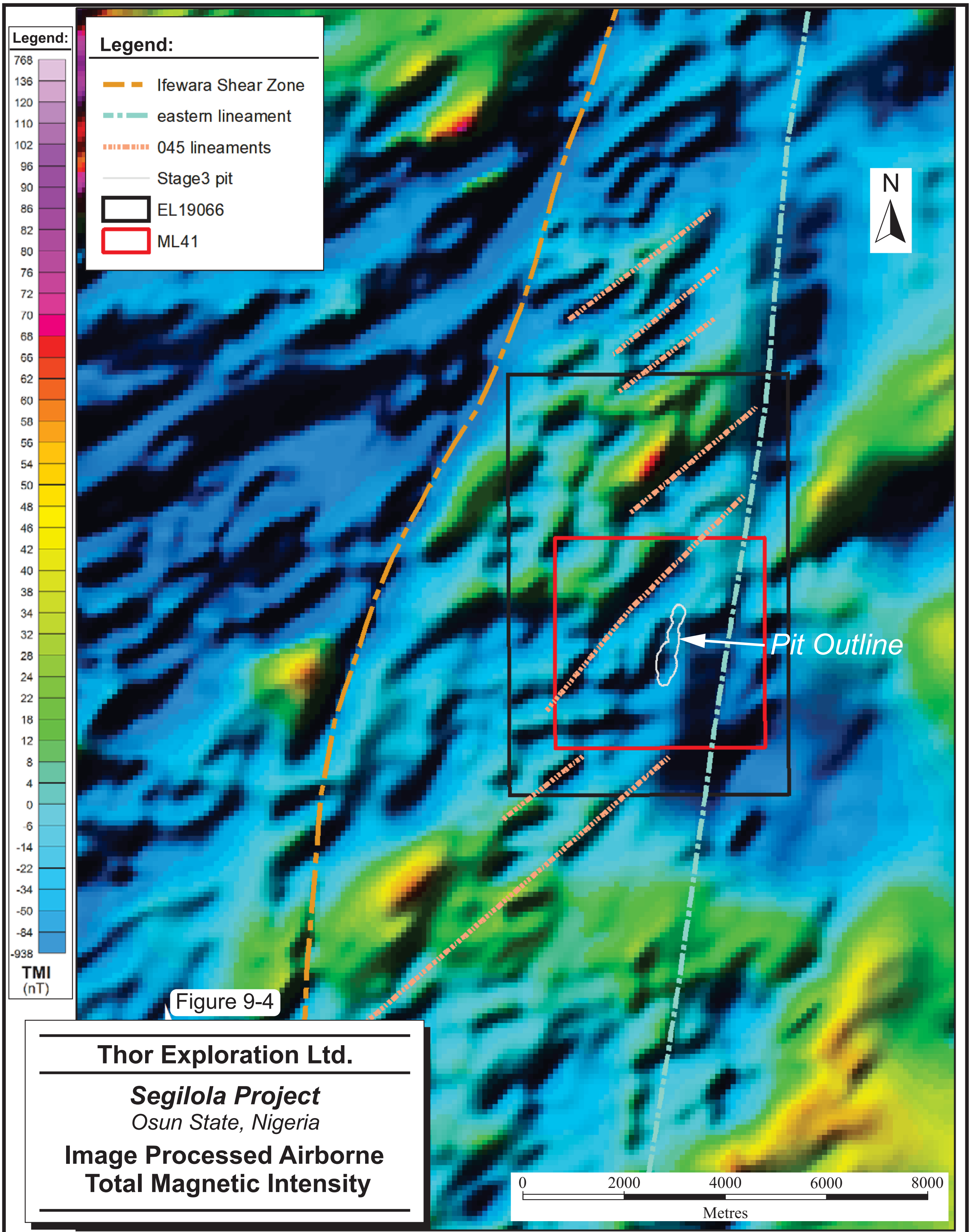


Figure 9-4

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Image Processed Airborne Total Magnetic Intensity

March 2019

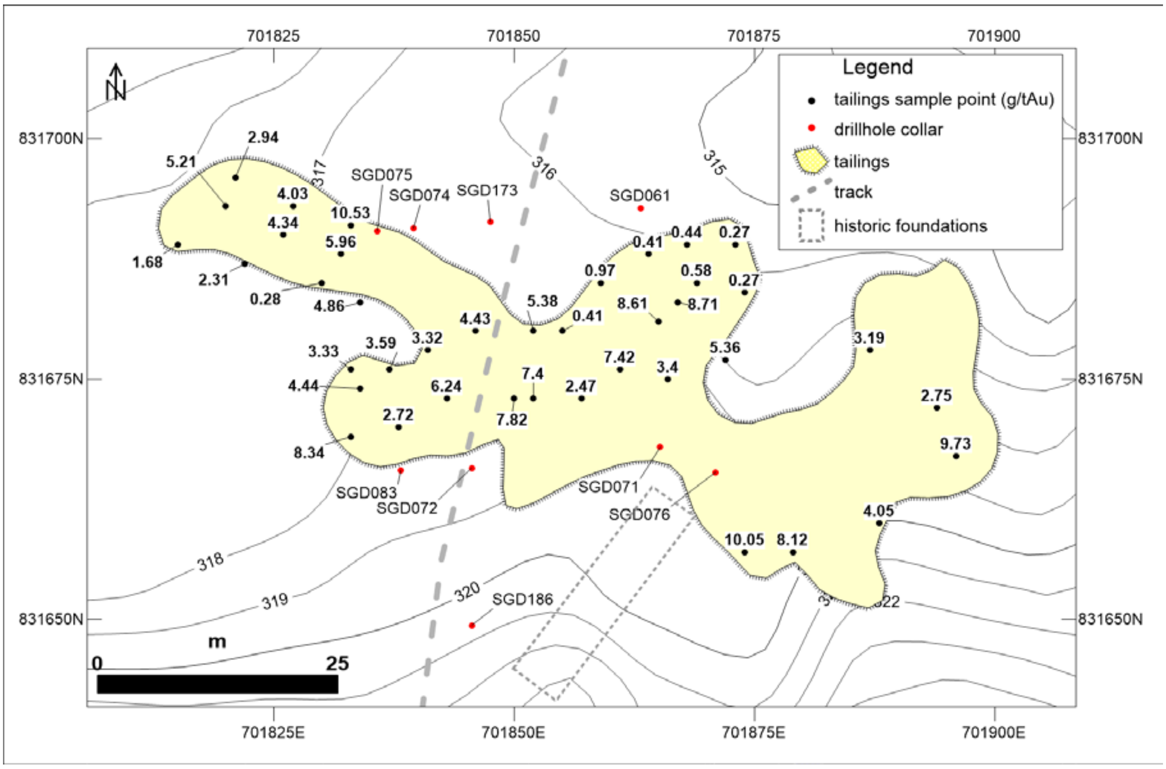
Source: Thor, 2019.

TAILINGS

In 2018, a total of 40 samples was collected from an area of tailings located north of the concrete foundations of the machine buildings (Figure 9-5). 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 (Figure 9-5).

FIGURE 9-5 TAILINGS SAMPLE LOCATIONS



10 DRILLING

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 mineralisation mostly with the objective of producing a Mineral Resource estimate. 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 mineralisation (Table 10-1). The holes were drilled approximately 25 m apart along the strike of the mineralisation, with just a single hole on each section. The drilling intersected gold mineralisation 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 (Table 10-1). Drilling was completed by Geo Core Drillers (GCD) and the state-owned NMC.

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 mineralisation and intersect the ore body at depth.
- Two exploration holes to step out from previous drilling and clarify the lateral extension of the mineralisation.

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

Between 2008 and 2011, CGA undertook three resource definition drilling programmes which comprised 159 holes over 15,987 m (Table 10-1). 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 (QA/QC) data, a lack of verifiable downhole survey data, and the lack of verifiable core intersections due to full-core sampling.

CURRENT DRILLING

The most recent drilling on the Project was conducted by Thor in 2017 and 2018 mainly with the intention of testing the down dip extension of the mineralised zone. Thor's exploration concept is 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 over 8,359 m were completed by Thor in 2017 and 2018. Of these holes, seven were for metallurgical purposes and seven were for geotechnical purposes (Table 10-2).

DRILLING SUMMARY

The Segilola deposit has been drilled over a strike length of over 2,000 m. 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° in order to intersect the lodes. Gold mineralisation is developed within a linear vein that dips at 65° to 70° towards the west. The vein varies in thickness from 3 m to 15 m true width. The dominant sample length is one metre. This sample interval is considered to be optimum to accurately define the mineralised envelope.

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

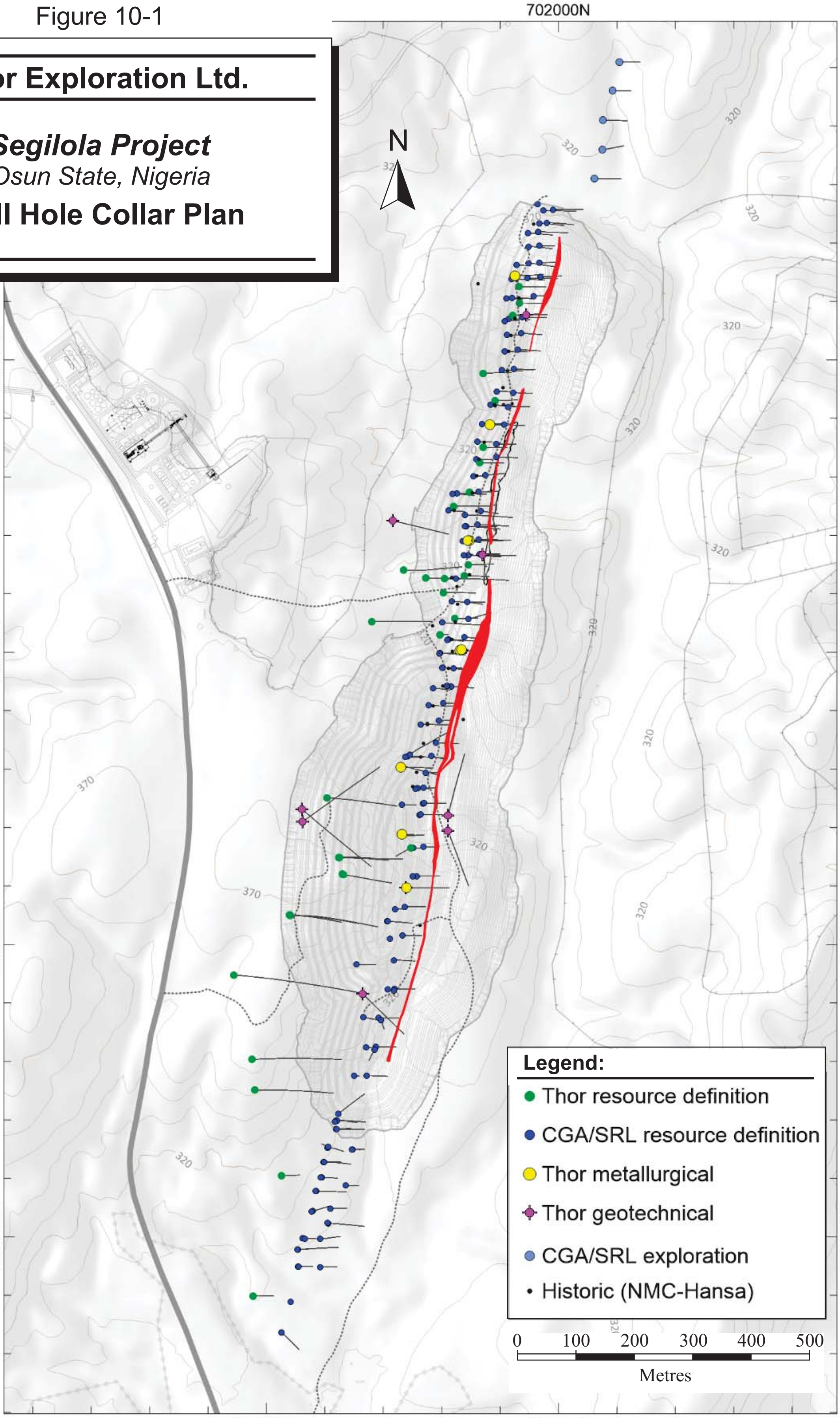
Figure 10-1

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Drill Hole Collar Plan



March 2019

Source: Thor, 2019.

TABLE 10-1 DRILLING SUMMARY BY YEAR
Thor Explorations Ltd – Segilola Gold Project

Year	NMC		Hansa		CGA/SGL		Thor		Total	
	No. Holes	Metres Drilled	No. Holes	Metres Drilled	No. Holes	Metres Drilled	No. Holes	Metres Drilled	No. Holes	Metres Drilled
1984-87	33	2,962							33	2,962
1997-98			7	895					7	895
2008					34	2,638			34	2,638
2009					89	9,644			89	9,644
2011					36	3,705			36	3,705
2017							16	4,129	16	4,129
2018							37	4,230	37	4,230
Total	**33**	**2,962**	**7**	**895**	**159**	**15,987**	**53**	**8,358**	**252**	**28,202**

TABLE 10-2 DRILLING SUMMARY BY PRIMARY PURPOSE
Thor Explorations Ltd – Segilola Gold Project

Type	No. Holes	Metres Drilled
Resource (CGA/Thor)	187	21,434
Metallurgical	9	694
Geotech	9	1,377
Exploration	7	840
Historical (Hansa/NMC)	40	3,857
Total	**252**	**28,202**

DRILLING PROCEDURES

The procedures discussed in this section refer only to the drilling completed by CGA and Thor. There is insufficient information on the procedures used for the NMC and Hansa drilling, as such it is not documented here and the data from this drilling has not been incorporated into the current Mineral Resource estimate.

Thor's drilling programmes largely continued with the industry-standard drilling, logging and QA/QC protocols and procedures established by CGA.

DRILLING METHOD

Table 10-3 outlines the number of holes and meters drilled by each drilling company.

TABLE 10-3 DRILLING SUMMARY BY DRILLING COMPANY
Thor Explorations Ltd – Segilola Gold Project

Drilling Company	CGA/SGL		Thor		Total	
	No. Holes	Metres Drilled	No. Holes	Metres Drilled	No. Holes	Metres Drilled
GeoHydro	36	3,705			36	3,705
Spektra Geotek	117	12,024			117	12,024
Tanylag	6	257			6	257
CMC			53	8,358	53	8,358
Total	**159**	**15,986**	**53**	**8,358**	**212**	**24,344**

CGA drilling was completed by three different contractors (Table 10-3) using predominately HQ core (63.25 mm diameter) and NQ (47.6 mm diameter) sized core.

All Thor 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.

The majority of the core from Thor holes were orientated using the Reflex orientation system.

SURVEY COORDINATE SYSTEM AND CONTROL

The coordinate system used for all data collection and surveying is the Universal Transverse Mercator (UTM) projection, Zone 31 North, using the World Geodetic System 1984 datum (WGS84). Table 10-4 summarises the contractors used for surveying.

TABLE 10-4 SURVEY CONTRACTORS
Thor Explorations Ltd – Segilola Gold Project

Survey Company	Equipment	Collars Surveyed
Survey Services (Ghana)	Trimble R4 Dual Frequency GPS in the RTK mode, Topcon GTS–722	SGD001-SGD154
Sphero Grid Survey (Nigeria)	Total Station Digital Electronic theodolite and GPS Pro Mark 2 and Pro Mark 3.	SGD155-SGD198 GTFS002-01 to GTFS002-13

Various control points with iron pegs set in concrete have been established on the Project and validation checks of the control points were routinely undertaken together with some earlier surveyed holes. Approximately one in ten existing drill holes were re-surveyed with the same instrument as a check on prior field surveys. Results indicate that the accuracy of the collar locations is within ±0.3 m for easting, northing, and RL.

Drill hole collar elevations were adjusted to the DTM as the LIDAR data, being accurate to the centimetre, are superior to the conventional survey elevations. Northings and eastings were not adjusted.

Visually the CGA drill holes plot correctly compared to the Thor holes, however, the location of CGA drill holes should be surveyed with a DGPS for verification purposes.

Although it is unlikely to materially affect the collar elevations, further work should be undertaken to confirm the observed discrepancy between the surveyed collar elevations and LIDAR collar elevations.

COLLAR LOCATION AND SURVEY

Drill hole sites were initially located using a hand-held GPS. Once the sites were located, the qualified surveyors accurately positioned the planned drill location using a ProMark 2 GPS and Kolida Digital Total Station DGPS (Table 10-4).

The system utilises a base receiver set-up on a control point with a separate rover receiver used for the survey of the drill hole collar. The data from both instruments is post processed in laptops using Astech Solutions, KOLIDA Downloading, and AutoCAD software.

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

DOWNHOLE SURVEY

Downhole survey measurements were made to determine the spatial position and bottom of each hole on completion.

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

A magnetic declination correction of -1° 37' is applicable for the Project area.

DRILL CORE HANDLING PROCEDURES

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.

A geologist determined the Rock Quality Designation (RQD) at the core shed. Prior to logging, the core pieces were fitted together on rails in order to check the depth markers placed by the drillers and possible core mix-ups and to calculate the core recovery. The reference line of the oriented core was drawn from Reflex Ezy-Mark tool, and the geotechnical parameters were logged. The core was placed back in the boxes to clearly display the line of reference or the angle between the dominant fabric (bedding, foliation) and the core axis. Subsequently, the lithological contacts, structural features, mineralised zones, and the samples are marked up.

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 mineralisation.
- 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 non-geological 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.

RECOVERY AND RQD

Core recovery during the drilling programme was measured systematically on all core.

Core recovery is very good ranging from 86.8% from the holes completed by CGA to 96.6% in the holes completed by Thor.

RQD percentages vary from 40.3% for the CGA holes to 79.0% for Thor's holes. This is most likely a reflection of two factors; the core diameter used and the average total depth of hole. CGA predominantly used NQ diameter core, whilst Thor used HQ and NQ diameter core, and the HQ core appears to show a better recovery in oxide material. The average depth of the CGA holes is 100 m and the average of the Thor holes is 157 m. The CGA holes have a greater relative amount of weathered material per hole, 29% versus 18%.

DENSITY MEASUREMENT

The deposit has a relatively shallow weathering profile. The depth of the top of fresh rock over the mineralised zone is generally approximately 20 m to 25 m below the surface and reduces to 1 m to 2 m either side.

Densities are based upon specific gravity measurements completed by CGA and Thor (carried out internally) and on behalf of Thor (carried out by MS Analytical laboratory) (Tables 10-5 and 10-6, Figures 10-2 and 10-3). Both used industry standard core immersion techniques.

Specific gravity determinations were taken by CGA for 1,402 individual full-core samples. Core from every hole was selected, with the determinations carried out on site using the immersion method, whereby weight in air and weight in water are used to determine the displacement and then density. An average density of 2.67 was obtained for both lodes 100 and 200. Only five specific gravity (SG) determinations were carried out historically on Lode 300 material and thus the average SG of 2.35 may not be truly representative. The variance in SG between the lodes and host rock is minimal.

Thor completed 39 drill core specific gravity determinations on drill core using water immersion methods. Core samples were dried, and wax was used to seal the core prior to immersion. Thor also submitted 271 samples to MS Analytical for drill core density determinations using the water immersion method from core samples mostly obtained at depth and outside the indicated resource limits. These samples averaged 2.74 within the mineralised lodes and 2.69 for waste/country rock.

TABLE 10-5 SPECIFIC GRAVITY STATISTICS
Thor Explorations Ltd – Segilola Gold Project

Lode	Thor		CGA		Total	
	No.	Mean	No.	Mean	No.	Mean
100	12	2.94	79	2.63	91	2.67
200	67	2.71	137	2.65	204	2.67
300	-	-	5	2.35	5	2.35
Mean Lode	**79**	**2.74**	**221**	**2.64**	**300**	**2.67**
Waste/Country Rock	231	2.69	1,181	2.68	1,412	2.68
Total	**310**		**1,402**		**1,712**	**2.68**

TABLE 10-6 SPECIFIC GRAVITY STATISTICS BY WEATHERING CODE
Thor Explorations Ltd – Segilola Gold Project

All Material	Thor		CGA		Total	
	No.	Mean	No.	Mean	No.	Mean
Oxide	9	1.99	67	2.56	76	2.49
Transition	11	2.47	80	2.65	91	2.63
Fresh	248	2.74	1,297	2.68	1,545	2.69
Total	**268**	**2.70**	**1,444**	**2.67**	**1,712**	2.68

Figure 10-2 shows the distribution of samples on which density determinations were made. There is little variation of density with depth which is consistent with the shallow weathering profile and the general proximity of fresh rock close to surface (Figure 10-3).

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

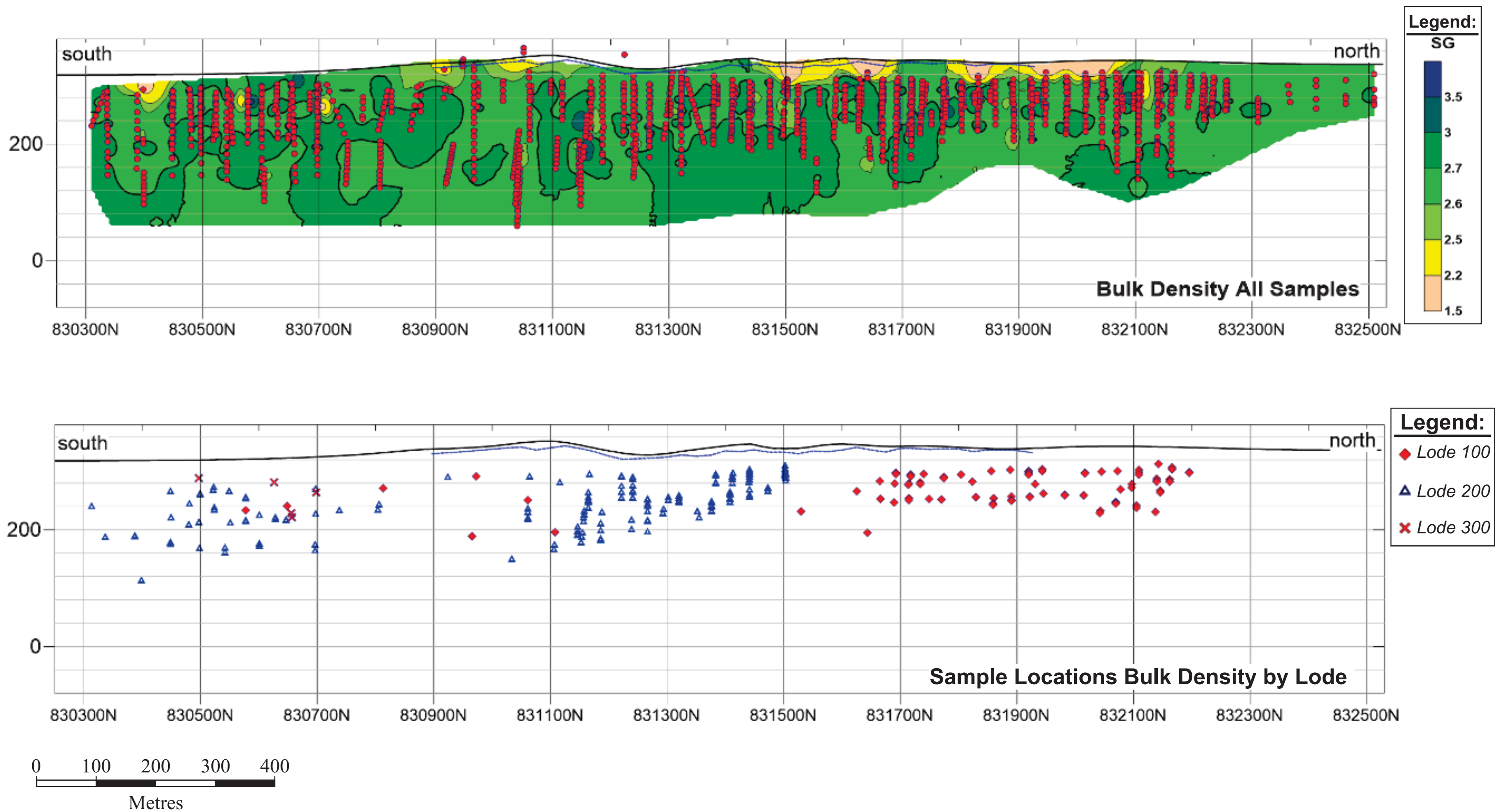


Figure 10-2

Thor Exploration Ltd.

Segilola Project

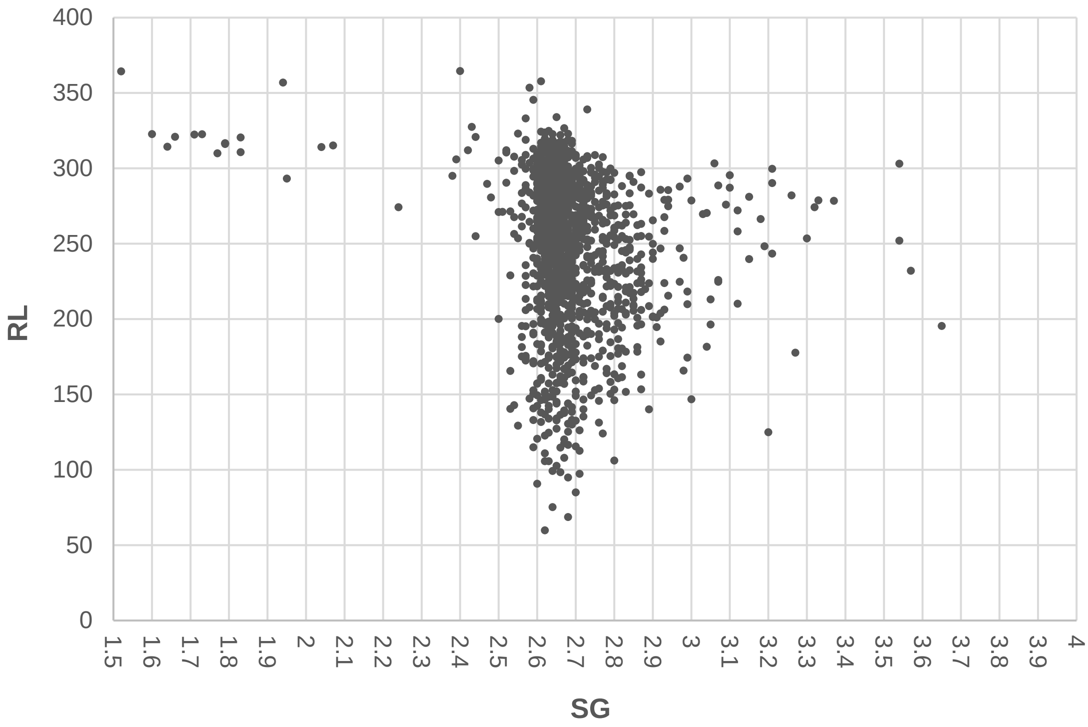
Osun State, Nigeria

Distribution of Bulk Density Sampling

March 2019

Source: Thor, 2019.

FIGURE 10-3 VARIATION OF ALL DATA BULK DENSITY WITH DEPTH



COMMENTS

A total of 252 drill holes have been completed on the Project, with 187 drill holes used to define the mineralisation. The QP notes that mineralisation is predictable to the south at depth, based on current wide-spaced drilling, and that further infill drilling between 830600N and 830950N is warranted in the future.

In the opinion of QP, the quantity and quality of the geological, geotechnical, collar, and downhole survey data collected by the past and present operators on the Project are sufficient to support Mineral Resource estimation for the following reasons:

- Drilling procedures and core logging meet industry standards.
- Recovery data from drill core data are acceptable.
- Collar surveys have been performed using industry-standard instrumentation.
- Downhole surveys were collected at the time of the programmes using industry standard instrumentation.
- Drill orientations are generally appropriate for the mineralisation style and are optimal for the orientation of the mineralisation for the bulk of the resource areas.

- Drill spacing has been adequate to first outline, then infill and define mineralised zones.

No factors were identified with the data collection from the drill programmes that could materially affect resource estimation accuracy or reliability.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

SAMPLING METHOD AND APPROACH

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

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. Sampling has predominantly been half-core NQ or quarter core HQ (Table 11-1).

TABLE 11-1 SUMMARY OF CORE SIZE SAMPLING
Thor Explorations Ltd – Segilola Gold Project

Core Sample	HQ	NQ	Total
Half-core	596	8,686	9,282
Quarter-core	1,613	199	1,812
Total	2,209	8,885	11,094

Quarter-core sampling was carried by Thor during 2016 to reduce sample weights for air-freighting to Vancouver. The QP recommends that all future sampling should be completed on half core in order to obtain a more representative sample.

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 one hole were consolidated at site and the sample numbers were 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 sacks.

SAMPLE PREPARATION

For CGA drilling, the sample preparation was completed at the analytical laboratory; SGS in Tarkwa, Ghana (Table 11-2).

For Thor drilling, sample preparation was completed in two different locations. Before 2017 the samples were prepared at the analytical laboratory; MS Analytical in Vancouver, BC, 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, BC for analysis.

All Thor 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).

SAMPLE ANALYSIS

ANALYTICAL LABORATORIES

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

TABLE 11-2 ANALYTICAL LABORATORY SUMMARY
Thor Explorations Ltd – Segilola Gold Project

Operator	Laboratory	Location	Time Period	Sample Type Analysed
CGA	SGS	Tarkwa, Ghana	2008 - 2011	Soil Samples
CGA	SGS	Tarkwa, Ghana	2008 - 2011	Drill Hole Samples
Thor	MS Analytical	Vancouver, BC, Canada	2017 - 2018	Surface Samples
Thor	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.

Both laboratories are independent of CGA or Thor.

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

SAMPLE ANALYSES

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).

Thor 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
Thor Explorations Ltd – Segilola Gold Project

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	

METALLIC SCREEN FIRE ASSAY

From 2017 onwards, 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 11-1 and Table 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 through the use of Metallic Screen Fire Assay, although there are insufficient samples to define a Mineral Resource using them. RPA recommends that Thor undertake Metallic Screen Fire Assays on all samples greater than 10 g/t Au going forward. When there is a sufficient number of analyses, these should be reviewed to determine whether the Metallic Screen Fire Assays are suitable to be used for future Mineral Resource estimates.

FIGURE 11-1 FIRE ASSAY AND METALLIC SCREEN FIRE ASSAY QQ PLOT

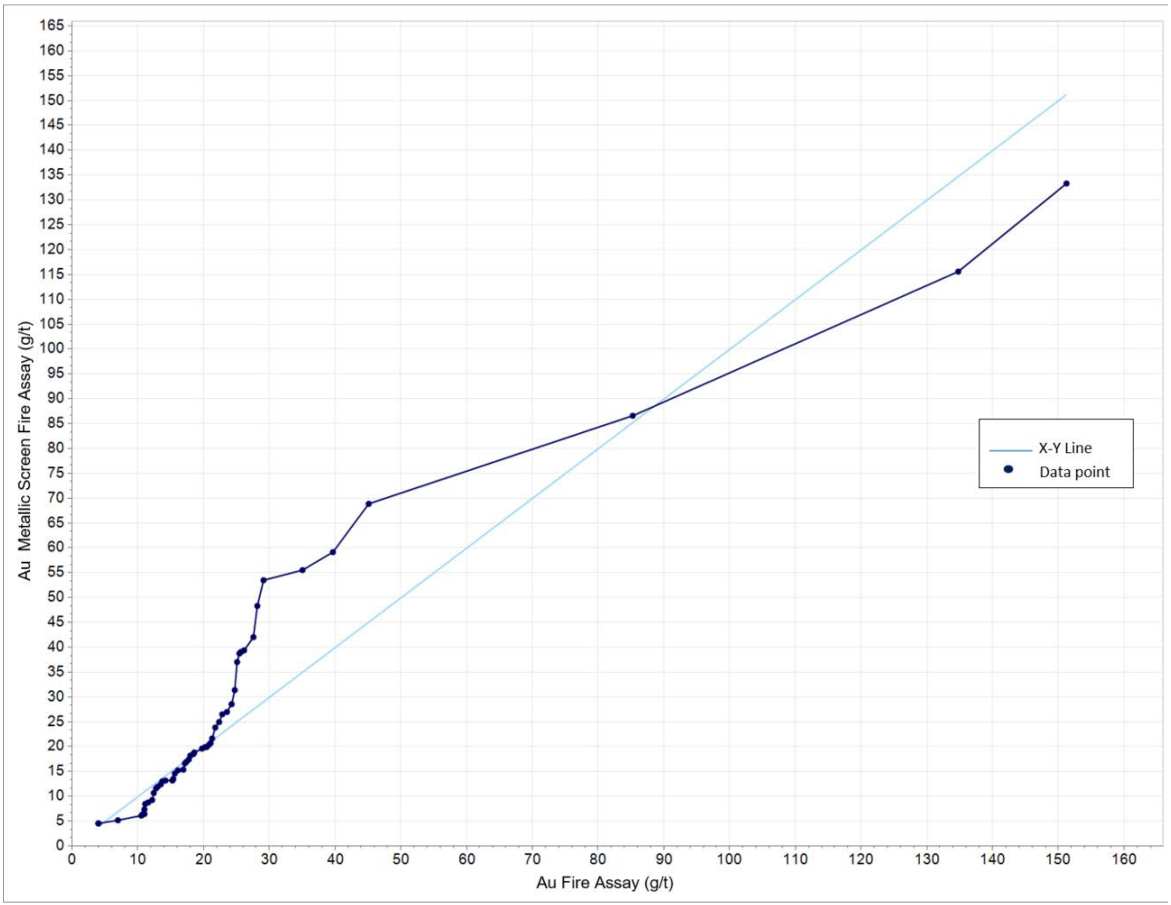


TABLE 11-4 COMPARISON BETWEEN FIRE ASSAY AND METALLIC SCREEN FIRE ASSAY (SORTED BY FIRE ASSAY VALUE)
Thor Explorations Ltd – Segilola Gold Project

Hole ID	Samples ID	Fire Assay (FAS221) Grade (Au g/t)	Metallic Screen Fire Assay (MSC150) Grade (Au g/t)	Difference (Au g/t)	Variance (%)
SGD195	SX082032	3.98	11.68	7.7	193%
SGD179	SX080975	10.45	6.39	-4.06	-39%
SGD195	SX082037	10.57	6.6	-3.97	-38%
SGD195	SX082033	10.97	8.41	-2.56	-23%
SGD157	SX079011	10.99	8.68	-2.31	-21%
SGD195	SX082031	11.02	10.2	-0.82	-7%
SGD195	SX082006	11.36	5.92	-5.44	-48%
SGD177	SX080751	12.1	18.82	6.72	56%
SGD155	SX078902	12.3	15.3	3	24%

Hole ID	Samples ID	Fire Assay (FAS221) Grade (Au g/t)	Metallic Screen Fire Assay (MSC150) Grade (Au g/t)	Difference (Au g/t)	Variance (%)
SGD182	SX081244	12.81	18.19	5.38	42%
SGD178	SX080883	12.91	12.93	0.02	0%
SGD188	SX081667	13.46	32.1	18.64	138%
SGD177	SX080756	13.6	13.21	-0.39	-3%
SGD195	SX082010	13.95	12.3	-1.65	-12%
SGD189	SX081728	15.3	6.39	-8.91	-58%
SGD163	SX079877	15.31	4.47	-10.84	-71%
SGD155	SX078903	15.53	18.54	3.01	19%
SGD155	SX078901	15.95	16.55	0.6	4%
SGD188	SX081664	16.93	19.97	3.04	18%
SGD195	SX082023	17.09	19.67	2.58	15%
SGD179	SX080951	17.37	16.82	-0.55	-3%
SGD178	SX080879	17.72	13.1	-4.62	-26%
SGD155	SX078915	17.99	15.27	-2.72	-15%
SGD195	SX082036	18.43	17.45	-0.98	-5%
SGD193	SX081864	18.57	21.73	3.16	17%
SGD192	SX081834	19.87	13.17	-6.7	-34%
SGD157	SX078997	20.23	8.89	-11.34	-56%
SGD157	SX078996	20.48	20.72	0.24	1%
SGD171	SX080481	20.88	61.57	40.69	195%
SGD171	SX080486	21.16	25.14	3.98	19%
SGD179	SX080993	21.3	11.5	-9.8	-46%
SGD167	SX080198	21.93	19.93	-2	-9%
SGD192	SX081832	22.46	14.44	-8.02	-36%
SGD189	SX081720	22.98	38.81	15.83	69%
SGD193	SX081866	23.76	26.8	3.04	13%
SGD173	SX080565	24.84	51.09	26.25	106%
SGD182	SX081241	25.18	55.04	29.86	119%
SGD155	SX078914	25.66	24.14	-1.52	-6%
SGD156	SX078937	25.72	39.53	13.81	54%
SGD178	SX080894	26.33	29.07	2.74	10%
SGD188	SX081666	28.24	20.4	-7.84	-28%
SGD171	SX080482	28.25	74.75	46.5	165%
SGD171	SX080483	29.71	55.86	26.15	88%
SGD173	SX080563	38.8	38.79	-0.01	0%
SGD156	SX078936	40.43	43.36	2.93	7%
SGD157	SX078995	48.9	39.19	-9.71	-20%
SGD173	SX080564	117.63	97.03	-20.6	-18%
SGD174	SX080619	151.21	133.27	-17.94	-12%

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. The Mineral Resource estimate relies predominantly on CGA data and metallic screen fire assays were not carried out during the CGA programmes.

In the QP's opinion, metallic screen fire assays should be completed on all samples with grades above 10 g/t Au. When there is a sufficient number of analyses, these should be reviewed to determine whether the metallic screen fire assays are suitable to be used for future Mineral Resource estimates.

The metallic screen fire assay data indicates the presence of a coarse gold fraction as confirmed by subsequent metallurgical testing.

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 company personnel transported the samples to DHL couriers in Lagos for delivery to SGS Laboratory, Tarkwa, Ghana for the CGA samples and to MS Analytical Laboratories, Vancouver, BC, Canada via air freight for the Thor samples. For samples that were prepared at the MS Analytical preparation facility in Abuja, the samples were collected and transported from the exploration camp by laboratory staff.

QUALITY ASSURANCE AND QUALITY CONTROL

To ensure the reliability of the assay data, Thor has put into place a set of QA/QC procedures. This section details the QA/QC from Thor's 2017 to 2018 drilling programmes; all previous QA/QC was documented in the PFS completed by Auralia. The discussion of QA/QC is divided into 'Field' samples submitted by Thor and the 'Laboratory' samples internally submitted by MS Analytical.

FIELD QA/QC

Table 11-5 summarises the field QA/QC data.

TABLE 11-5 QA/QC SUMMARY
Thor Explorations Ltd – Segilola Gold Project

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

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).

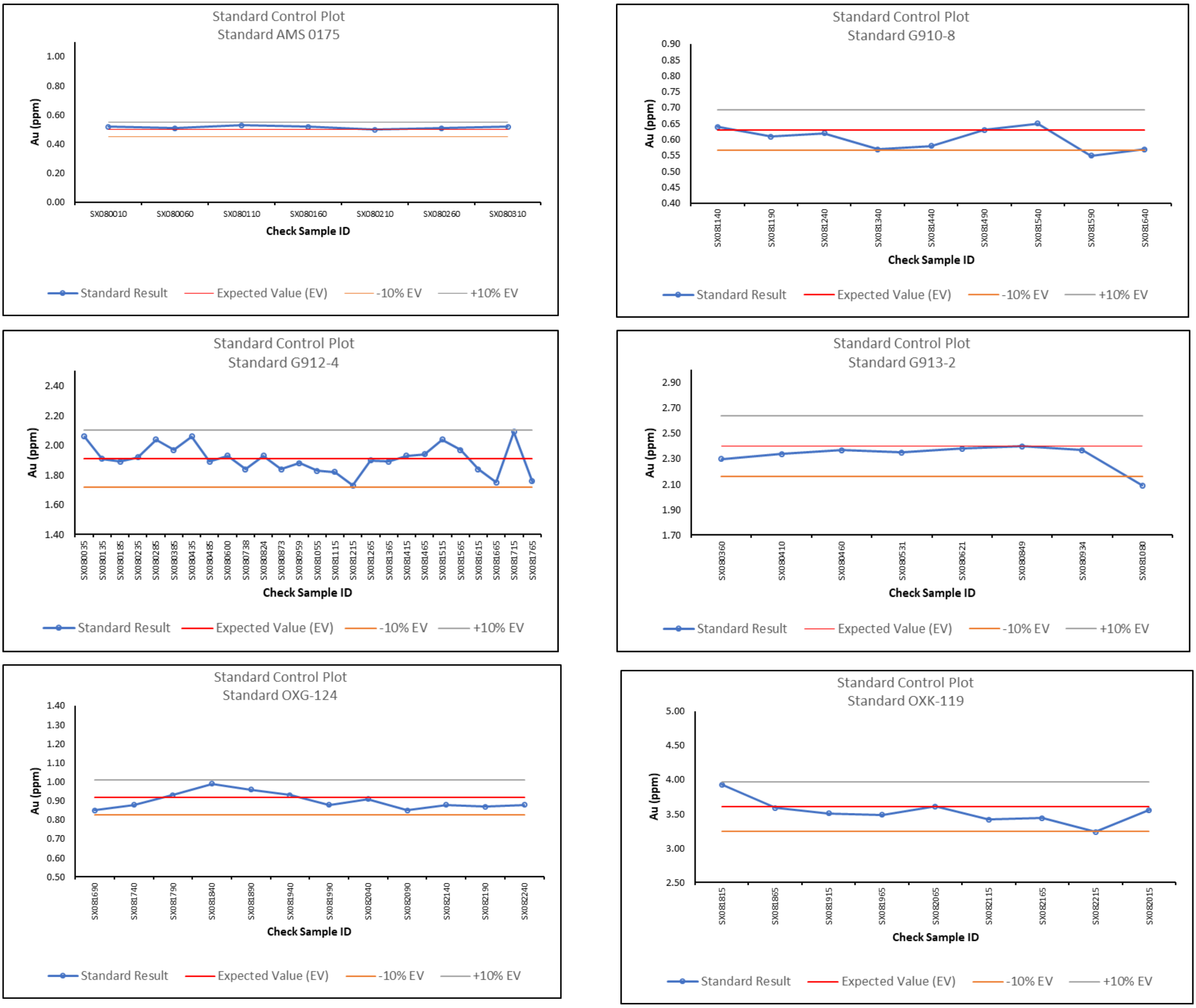
Thor used standards supplied by both Geostats Pty Ltd and African Mineral Standards, South Africa (AMS). All the standards were supplied in jars with the exception of AMS 0175, which was supplied in pre-measured 50 g packets. The standards supplied within jars were weighed into a small bag by on-site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

Table 11-6 and Figure 11-2 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 11-6 LIST OF STANDARDS
Thor Explorations Ltd – Segilola Gold Project

Standard	No of Assays	Expected Value (g/t Au)	Expected Value Range (g/t Au)	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	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

FIGURE 11-2 FIELD STANDARD CONTROL PLOTS



BLANKS

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

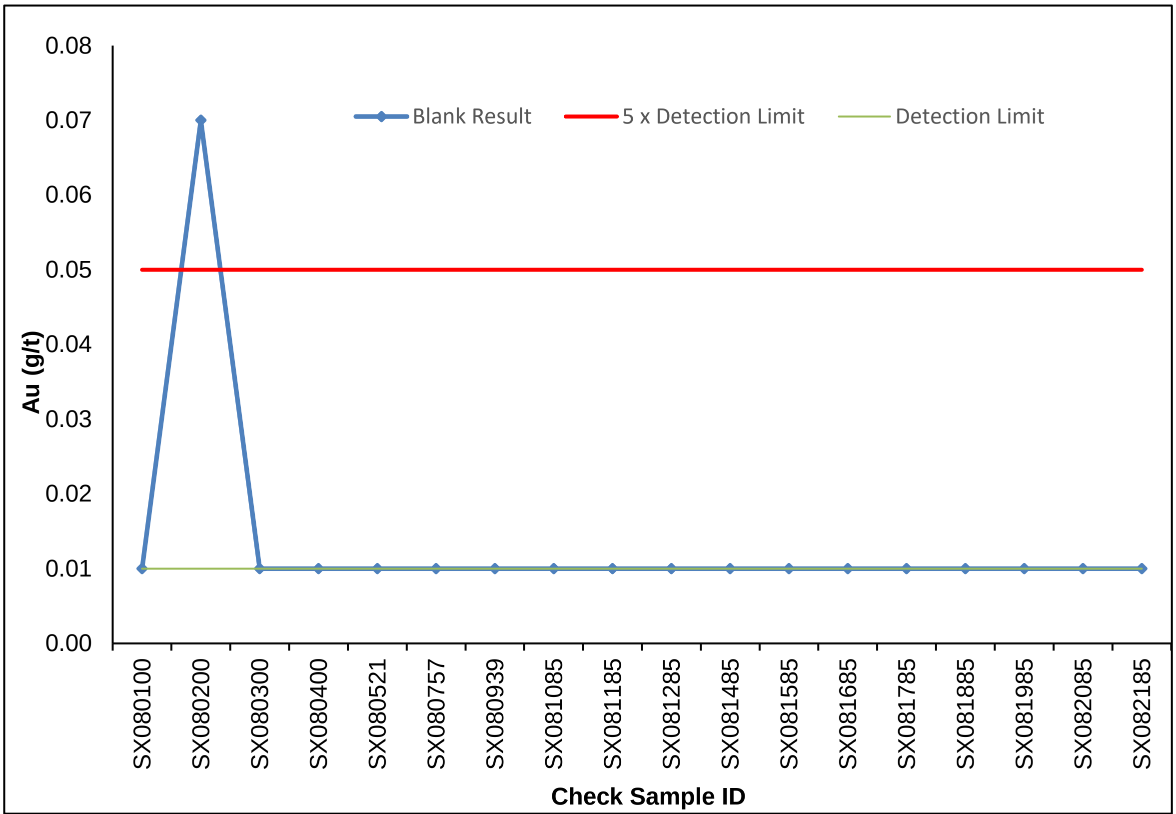
Certified laboratory blanks supplied by AMS were used. These coarse blanks (Blank No. 0166) were made from homogenised silica quartz and had a gold content of less than 0.001 g/t Au

Table 11-7 and Figure 11-3 show the results for the blank sample analysis. In the QP's opinion, the blank results are acceptable, with only one failure.

TABLE 11-7 FIELD BLANKS
Thor Explorations Ltd – Segilola Gold Project

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

FIGURE 11-3 FIELD BLANK CONTROL PLOT



The QP recommends that the blank insertion rate be increased to the same frequency as the standards and duplicates for future drilling campaigns. Blanks should also be inserted manually into areas of expected high-grade results to better test sample contamination during preparation.

DUPLICATES

Field duplicates are used to determine sampling error and also 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 (Table 11-5). The duplicate samples were produced from both half and quarter core.

Figure 11-4 shows a scatter plot of duplicates versus original values and Figure 11-5 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 mineralisation. In general, the data indicates reasonable precision for the sampling method given the nuggety nature of the mineralisation.

FIGURE 11-4 FIELD DUPLICATE SCATTER PLOT

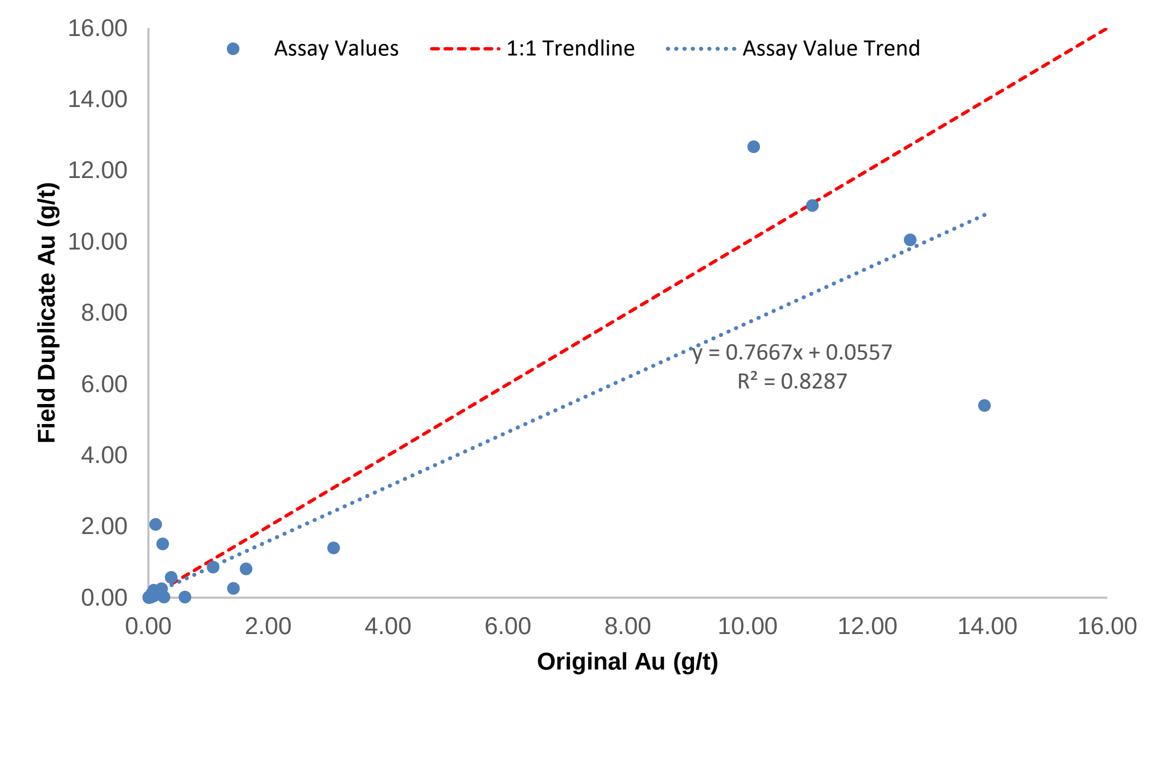
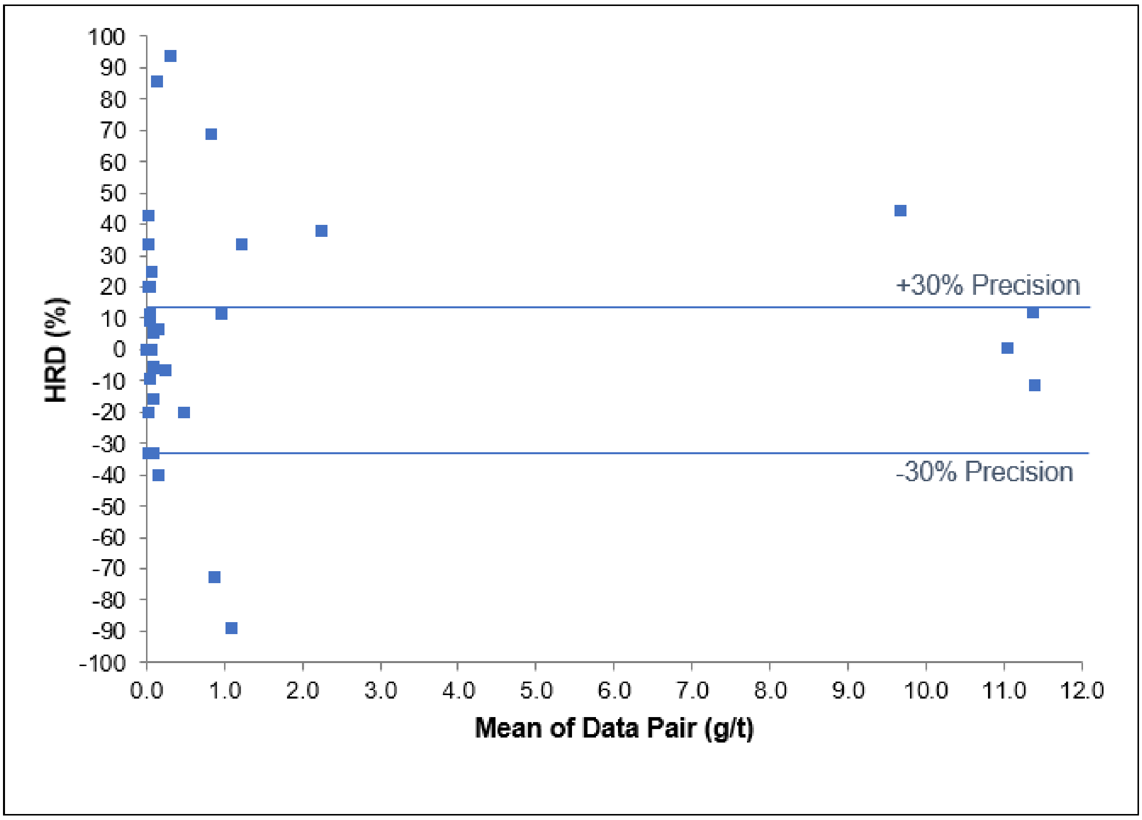


FIGURE 11-5 DUPLICATE MEAN VS HRD PLOT



The QP recommends that Thor insert duplicates of coarse material (coarse reject duplicates). After the initial crush, these should be taken and analysed to ensure that the laboratory is homogenising the material sufficiently. Currently this is only undertaken by the laboratory.

LABORATORY QA/QC

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. At the time of writing this report, Thor has yet to receive the final report regarding this issue.

BLANKS

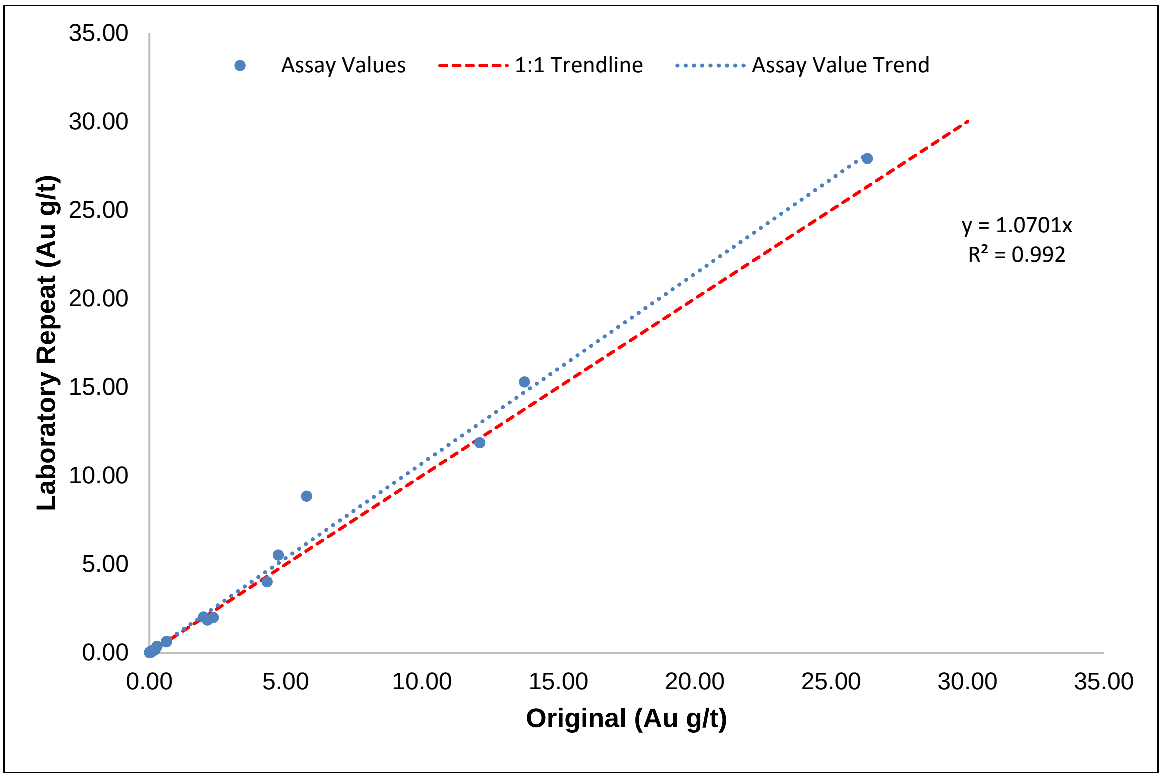
MS Analytical laboratory inserted 151 blanks to assess contamination within the process flow. No failures, i.e. values above the control limit of 0.03 ppm, were present for the 151 observations.

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.

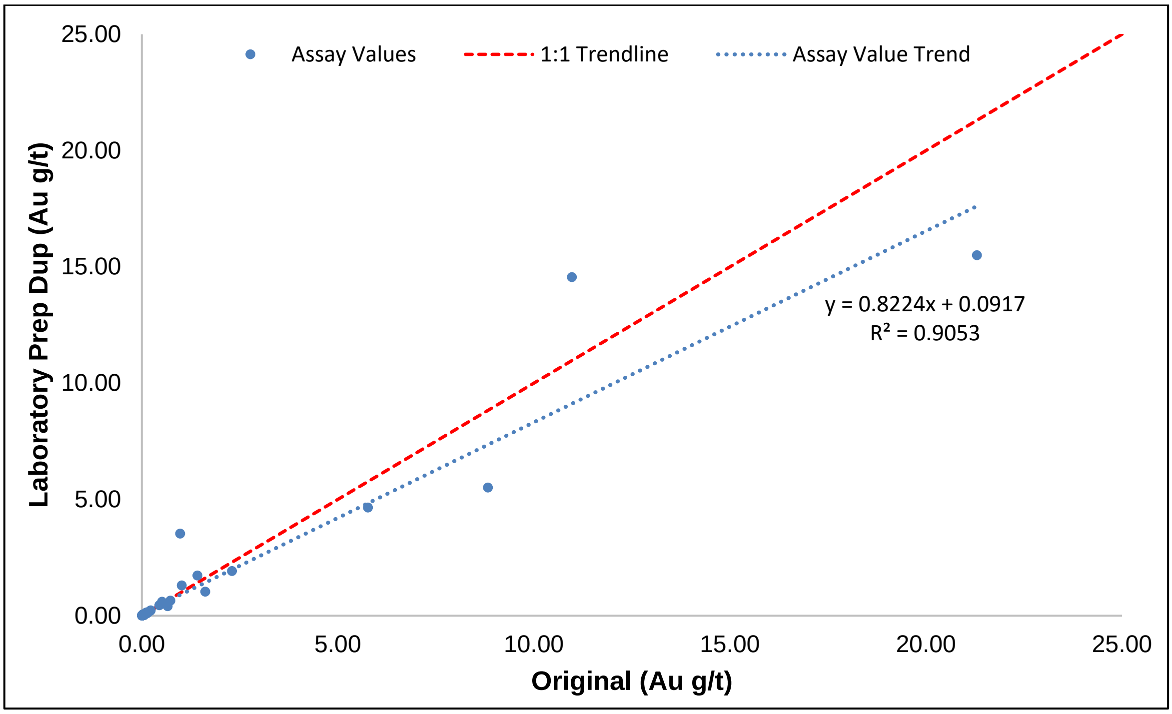
The pulp (analytical) duplicates were generally inserted every 40th sample. A total of 48 were analysed and the duplicate results were plotted against the original results in Figure 11-6. The visual check and the calculated R^2 value of 0.992 indicate a good correlation between the original and duplicate samples.

FIGURE 11-6 PULP DUPLICATE VS ORIGINAL SCATTERPLOT



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 11-7. The visual check and the calculated R^2 value of 0.9053 indicate an acceptable correlation between the original and duplicate samples.

FIGURE 11-7 COASE REJECT DUPLICATE VS ORIGINAL SCATTERPLOT

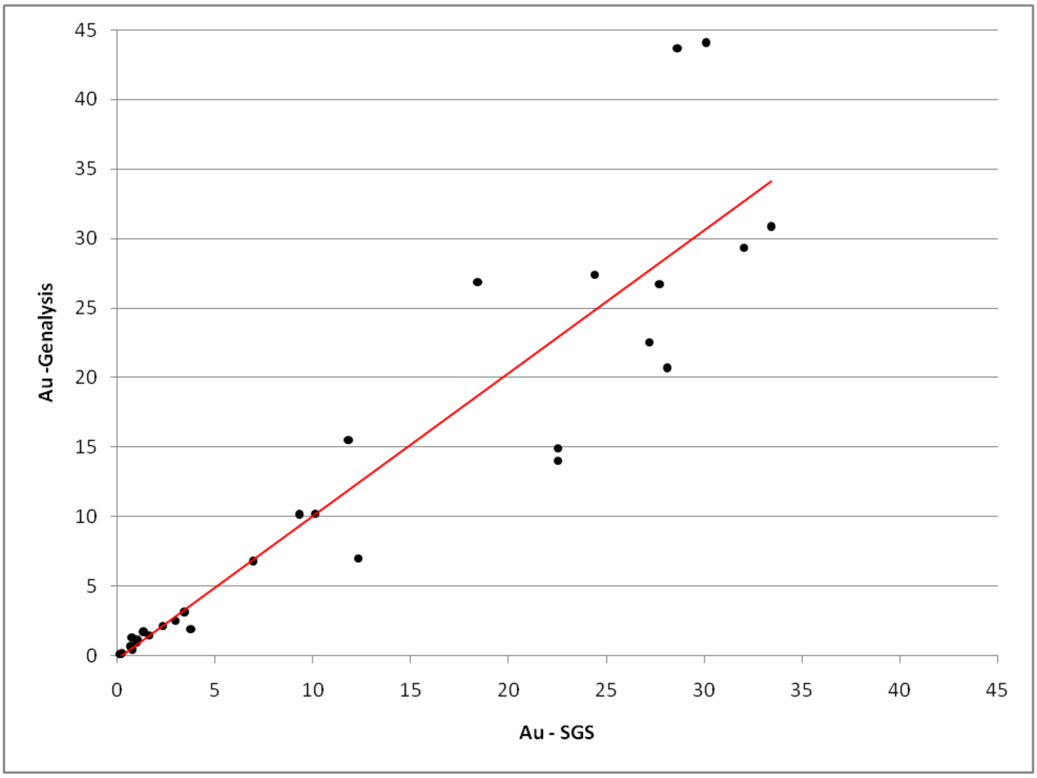


UMPIRE LABORATORY ANALYSIS

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 11-8).

No inter-laboratory checks were completed by Thor. The QP recommends that inter-laboratory checks be performed for all future drilling programmes.

FIGURE 11-8 INTER-LABORATORY CHECK ASSAYS SCATTERPLOT



COMMENTS

SAMPLE PREPARATION, ANALYSES AND SECURITY

Auralia is of the opinion that the sample collection, preparation, analysis, and security used by Thor were generally performed in accordance with exploration best practices and industry standards and are suitable for use in Mineral Resource estimation.

- Sample collection and preparation is in line with industry-standard methods for the gold mineralisation.
- All sample preparation and analyses were carried out at independent laboratories in Tarkwa, Ghana and Vancouver, BC, Canada with additional sample preparation being undertaken in Abuja, Nigeria. No aspect of laboratory sample preparation or analysis was conducted by an employee, officer, director, or associate of Thor.
- Samples have been kept secure and were always attended to by drill crews or Project staff while at the Project site or logging facilities, and delivered to the laboratory either directly by Project staff or commercial trucking companies.
- Current sample storage procedures and storage areas are consistent with industry standards.
- The sample data collected was validated before importing into the master database.

The QA/QC procedures and management are consistent with industry standard 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.

- Analysis of the CRMs and blanks show that the laboratories produced reliable assays with no evidence of significant and systemic contamination or bias.
- Samples were analysed at a reputable and certified laboratory.
- No material sample bias was identified during the review of the drill data and assays.
- Observation of the drill core during the site visits and inspection and validation of the data collected satisfied the QP that the drill data is adequate for the estimation of Inferred and Indicated Mineral Resources.

Although there are no observed issues, the QP recommends that the QA/QC results for standards should be reviewed to three standard deviations going forward, as the current +/- 10% can be overly lenient at high grades and overly harsh at low grades.

Although the QA/QC procedures are well understood by site geologists, the QP recommends that they be formalised into an auditable Standard Operating Procedures (SOP) document for reference purposes.

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

12 DATA VERIFICATION

DATA VALIDATION

Drilling data is stored in Microsoft Excel spreadsheet format and all of the key fields and tables were audited as part of this study. The data verification was undertaken independently by Auralia using in-built validation tools in Surpac and by importing and interrogating the database in Microsoft Access.

COLLAR TABLE

Drill collar locations were audited in three dimensions using Surpac software. The collar elevations had been adjusted to the DTM as the LIDAR data, accurate to the centimetre, are superior to the conventional survey elevations. Northings and eastings were not adjusted.

The drill collars were checked to ensure there were no projection issues prior to modelling.

Although it is unlikely to materially affect the collar elevations, further work should be undertaken to confirm the observed discrepancy between the surveyed collar elevations and LIDAR collar elevations.

DOWNHOLE SURVEY TABLE

The downhole survey table from the Microsoft Excel master database was checked by examining the changes from one downhole survey record to the next in all holes for both azimuth and dip. No anomalies were noted.

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 Excel master assay sheet and values on the assay certificates. A spatially representative selection of drill holes was chosen from all resource zones, covering different years of drilling and assaying. The number of assays checked is given in Table 12-1.

TABLE 12-1 ASSAY VALIDATION SUMMARY
Thor Explorations Ltd – Segilola Gold Project

No. Holes	No. Holes Checked	Total Number of Assays	No. of Assays Checked	% Checked
187	160	11,082	9,851	89%

Overall, the database was found to be 'clean'. Three instances of crossing samples and one sample of zero thickness were found. Eight samples were not correctly flagged as duplicate samples. Minor transposing of weight values was identified and has been rectified by the Thor.

GEOLOGICAL DATA VERIFICATION AND INTERPRETATION

Interrogation of the lithology table depths found one invalid depth record. Validation of lithology rock codes found a number of inconsistencies with the use of lithology codes.

Interrogation of the colour, texture, alteration tables found only one invalid depth record (comma instead of decimal point) with no inconsistencies with the dictionary codes. Interrogation of the weathering table dictionary found some minor inconsistencies in the logging of weathered material. It is recommended that the CGA logged holes be converted to the current Thor weathering codes. A small number of records were found in the preceding tables without any logged code, colour (22), texture (8), and alteration (2).

Interrogation of the table veins noted a small number of inconsistencies with the dictionary code (qtz v QTZ v VQZ) as well as a number of blank records (24) without any vein code. No inconsistencies were found with the mineralogy/sulphide fields of the vein table.

Geology data verification was completed by checking that the lithology designation was correct in each sample interval. For each hole, all relevant data was compiled into graphic logs to check and review the various downhole datasets.

Aurelia is of the opinion that database verification procedures for the Project comply with industry standards and are adequate for the purposes of Mineral Resource estimation.

The QP recommends that the data be transferred into a relational geological database, with data validation capabilities to ensure a clean dataset.

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.

The AMMTEC test work was performed on six composites of drill core from zones within the orebody identified as North, Central, and South. Testing completed included:

- Head assays.
- Grinding parameter determination.
- Gravity testing.
- Leach testing.
- Cyanide detoxification testing.
- Settling test (by Outotec).

Gold assays for the composites ranged from 2.5 g/t Au to 9.7 g/t Au, and are shown in Table 13-1 together with the comminution testing results.

TABLE 13-1 SMC AND BOND MILL WORK INDEX TEST RESULTS SUMMARY
Thor Explorations Ltd – Segilola Gold Project

Sample	Au		Ai	ta	DWi	A	b	A x b	BWi
	g/t	g/t							kWh/t
North	7.28	6.25	0.387	0.56	4.59	75.8	0.76	57.6	18.0
Central	9.29	6.90	0.359	0.66	3.90	77.3	0.87	67.2	18.1
South	9.67	8.78	0.332	0.69	3.74	70.3	1.02	71.7	18.8
Blend	4.93	8.36							
Composite 1+2	2.47		0.351	0.64	4.02	71.4	0.91	64.9	17.5
Composite 3+4	3.32	2.54	0.303	0.61	4.22	69.6	0.90	62.6	14.1

Notes:
1. Source: Characterisation Test work conducted on Segilola Samples for CGA Mining Limited, AMMTEC Ltd, 2010

Cyanide leaching test work was completed at three different grind sizes (K_{80} of 150 µm, 106 µm, and 75 µm) for the North, Central, and South composites to evaluate the effect on gold recovery. Average gold recoveries after 48 hours were 96.9%, 98.2%, and 99.2% at 150 µm, 106 µm, and 75 µm respectively.

Gravity test work was completed on the North, Central, and South composites. This consisted of Knelson centrifugal concentration, followed by hand-panning. Gold recoveries to the pan concentrates were 35.4%, 69.5%, and 33.1% for the three composites respectively.

Samples of detoxified gold tailings slurry and un-detoxified damp solids were provided to Outotec for settling test work. The samples were combined and turned into slurry using Perth tap water. Efforts to produce a thickened slurry were unsuccessful due the coarse particles packing together and preventing their being pumped out of the test equipment and properly sampled, and the fine particles remaining in the overflow.

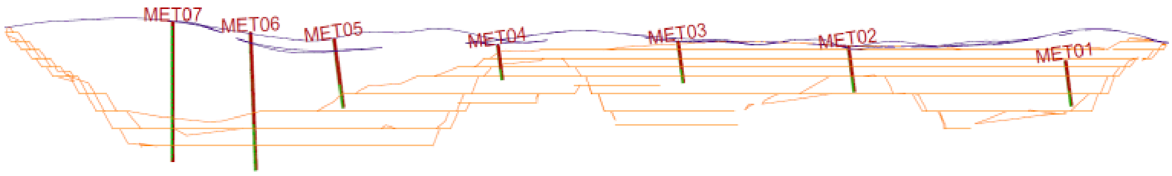
The IMO test work program involved the generation of a master composite sample, and 10 variability composites to undergo the proposed test work. Testing undertaken included:

- Head assays.
- Comminution test work
- Gravity recoverable gold test work and modelling
- Cyanide leach test work.
- Settling test work.

Thor provided IMO with interval samples from a recent drilling campaign conducted at the Segilola Gold Project. The intervals from each hole were utilised to generate the master and variability composites for metallurgical test work.

The drilling campaign undertaken by Thor consisted of nine drill holes, labelled MET01 through to MET07, including two twinned holes MET03B and MET04B. Intervals were provided as half HQ drill core, pre-composited in assay intervals. The twinned holes were provided by Thor as whole core with no interval assays. The drill hole long section plan undertaken by Thor is shown in Figure 13-1.

FIGURE 13-1 LONGITUDINAL SECTIONS SHOWING METALLURGICAL DRILL HOLE SAMPLE LOCATIONS



All nine drill holes were selected for use during compositing with their individual interval details summarised in Table 13-2.

TABLE 13-2 DRILL HOLE INTERVAL SUMMARY
Thor Explorations Ltd – Segilola Gold Project

Drill Hole	#	From (m)	To (m)	Interval (m)	Au (ppm)
MET01	SGD171	69.0	83.0	14.0	6.66
MET02	SGD172	40.0	47.0	7.0	1.20
MET03	SGD173	30.0	45.0	15.0	9.31
MET03B	SGD176	34.0	42.0	8.0	-
MET04	SGD174	18.0	42.0	24.0	6.68
MET04B	SGD175	23.0	35.5	12.5	-
MET05	SGD177	71.0	95.0	24.0	2.19
MET06	SGD178	83.0	112.0	29.0	2.94
MET07	SGD179	88.0	128.0	40.0	1.82

The variability composite make-up is detailed in Table 13-3. All of the composites, except composite 11, underwent variability test work.

TABLE 13-3 COMPOSITE INTERVAL SUMMARY
Thor Explorations Ltd – Segilola Gold Project

Composite	Drill Hole	From (m)	To (m)	Interval (m)	Au (ppm)
Composite 1	MET01	69.0	83.0	14.0	6.66
Composite 2	MET03	30.0	45.0	15.0	1.20
Composite 3	MET03B	34.0	42.0	8.0	9.31
Composite 4	MET04	18.0	42.0	24.0	-
Composite 5	MET04B	23.0	35.5	12.5	6.68
Composite 6	MET05	71.0	95.0	24.0	-
Composite 7	MET06	83.0	100.0	17.0	2.19
Composite 8	MET06	100.0	112.0	12.0	2.94
Composite 9	MET07	88.0	107.7	19.7	1.82
Composite 10	MET07	107.7	128.0	20.3	6.66
Composite 11	MET02	40.0	47.0	7.0	1.20

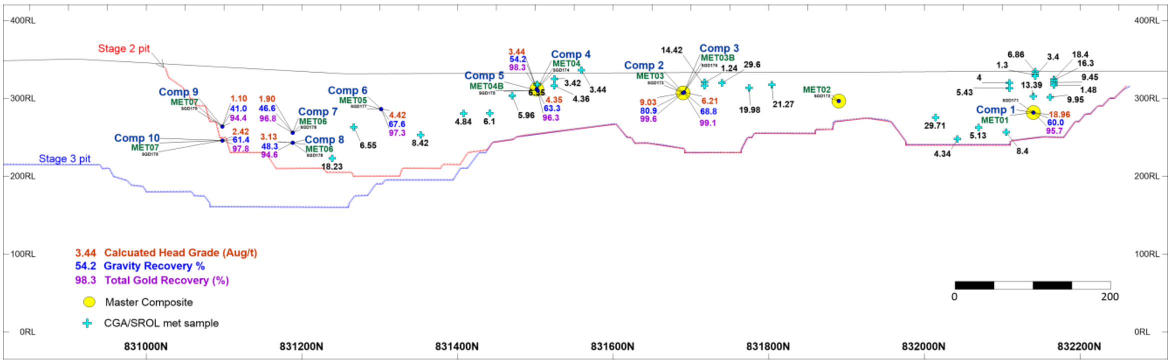
Selected composites were utilised to generate the master composite, detailed in Table 13-4. The mass ratios of the individual intervals within the composite samples were determined based on their respective presence within the deposit and are equal to the distribution of the interval meterage ratios, indicating no bias to any of the variability composites.

TABLE 13-4 MASTER COMPOSITE INTERVAL SUMMARY
Thor Explorations Ltd – Segilola Gold Project

Composite	Drill Hole	From (m)	To (m)	Interval (m)	Au (ppm)	Mass Ratio (%)	Meterage Ratio (%)
Composite 1	MET01	69	83	14	6.66	23.2	23.2
Composite 2	MET03	30	45	15	9.31	24.7	25.0
Composite 4	MET04	18	42	24	6.68	40.0	40.0
Composite 11	MET02	40	47	7	1.20	12.1	11.7
Master Composite	-	-	-	60	6.68	100.0	100.0

The master composite is representative of approximately the first three years of mine life with weighting towards the payback period (1 to 1.5 years). In the current model, most of the ore is from the Stage 3 pit in approximately Month 30 (2.5 years). The samples location diagram is shown in Figure 13-2, and there is very little variation throughout the deposit.

FIGURE 13-2 LONGITUDINAL SECTIONS SHOWING METALLURGICAL SAMPLE LOCATIONS



COMMINUTION TEST WORK

SAMPLES

Eight core samples were selected by IMO in agreement with Thor to undergo uniaxial compressive strength (UCS) test work. The samples were selected from varying depths across drill holes MET01 to MET05.

Two groups of samples were selected to undergo Crusher Work Index (CWi) testing with Group 1 representing drill holes MET01 to MET03, and Group 2 representing MET04 to MET05.

Samples representing each of the variability composites and the master composite were selected and submitted to JK Tech for SMC testing.

TEST RESULTS

Comminution test results are presented in Tables 13-5 to 13-7, indicating:

- UCS test values ranging from 71.7 MPa to 190.6 MPa.
- CWi testing conducted on group 1 samples indicating an average work index of 16.0 kWh/t.
- CWi testing conducted on group 2 samples indicating an average work index value of 11.6 kWh/t.
- SMC results indicate A x b values ranging from 45.5 to 87.0, characterising the ore as medium to soft resistance to impact breakage.
- SAG Circuit Specific Energy (SCSE) values for each of the composites agree with the A x b values, categorising the ore as medium to soft with values ranging from 7.2 kWh/t to 9.3 kWh/t.
- The t_a values for each of the composites ranged from 0.4 to 0.9, with the ore varying from medium to soft resistance to abrasion grinding.
- BWi testing utilised a closing screen size of 106 µm, resulting in BWi ranging from 17.9 kWh/t to 20.3 kWh/t, categorising all composites as very hard.

TABLE 13-5 UCS RESULTS
Thor Explorations Ltd – Segilola Gold Project

Drill Hole	Hole ID	From (m)	To (m)	UCS (MPa)
MET01	SGD171	73.5	73.8	90.73
MET01	SGD171	74.3	74.6	134.37
MET02	SGD172	41.2	41.6	138.14
MET02	SGD172	40	40.4	114.74
MET03	SGD173	35.6	35.9	86.60
MET03	SGD173	33.6	33.85	71.68
MET04	SGD174	32.9	33.3	157.86
MET05	SGD177	76.8	77.2	190.57

TABLE 13-6 CWI RESULTS
Thor Explorations Ltd – Segilola Gold Project

Variable	Units	Group 1	Group 2
Average Work Index	kWh/tonne	16.00	11.60
Maximum Work Index	kWh/tonne	23.80	25.20
Minimum Work Index	kWh/tonne	7.50	6.80
Median Work Index	kWh/tonne	17.35	10.80
75th Percentile	kWh/tonne	19.03	12.83
85th Percentile	kWh/tonne	20.02	15.02
95th Percentile	kWh/tonne	22.76	20.17
Average Impact Energy	Joules	43.52	31.16
Maximum Impact Energy	Joules	60.80	67.90
Minimum Impact Energy	Joules	20.50	16.30
Average SG of Specimens	SG	2.59	2.62

TABLE 13-7 COMMINUTION PARAMETER TEST RESULTS
Thor Explorations Ltd – Segilola Gold Project

	A x b	ta	SCSE	SMC Parameters(kWh/t)				BWi
			(kWh/t)	Mia	Mib	Mih	Mic	kWh/t
Composite 1	47.8	0.47	9.04	17.00	25.55	12.10	6.20	18.68
Composite 2	47.2	0.46	9.08	17.30	25.93	12.40	6.40	18.86
Composite 3	56.0	0.55	8.44	15.10	26.71	10.40	5.40	19.36
Composite 4	75.0	0.74	7.54	11.90	27.71	7.70	4.00	19.81
Composite 5	87.0	0.85	7.15	10.60	28.38	6.70	3.40	20.27
Composite 6	45.5	0.44	9.26	17.60	26.11	12.70	6.60	19.00
Composite 7	48.4	0.47	9.01	16.90	28.00	12.10	6.20	20.16
Composite 8	51.6	0.50	8.77	16.10	24.67	11.30	5.80	17.87
Composite 9	47.7	0.47	9.04	17.10	26.89	12.20	6.30	19.58
Composite 10	58.1	0.56	8.37	14.60	27.18	10.00	5.20	19.85
Master Composite	54.1	0.53	8.57	15.40	26.30	10.70	5.50	19.10
Maximum	87.0	0.85	9.26	17.60	28.38	12.70	6.60	20.27
Minimum	45.5	0.44	7.15	10.60	24.67	6.70	3.40	17.87
Average	56.2	0.55	8.57	15.42	26.67	10.75	5.55	19.32
Median	51.6	0.50	8.77	16.10	26.71	11.30	5.80	19.36
75th Percentile	57.05	0.56	9.04	17.05	27.44	12.15	6.25	19.83
85th Percentile	66.55	0.65	9.06	17.20	27.85	12.30	6.35	20.01
95th Percentile	81.00	0.80	9.17	17.45	28.19	12.55	6.50	20.22

MASTER COMPOSITE TEST WORK

HEAD ASSAY

Master composite head assay results are presented in Table 13-8 indicating an average gold grade of 12.61 g/t Au, significantly greater than the calculated 6.66 g/t Au based on the interval assays. Results indicated low concentrations of potentially deleterious elements such as Sb and organic carbon.

TABLE 13-8 MASTER COMPOSITE HEAD ASSAY RESULTS BASED ON THE IMO REPORT
Thor Explorations Ltd – Segilola Gold Project

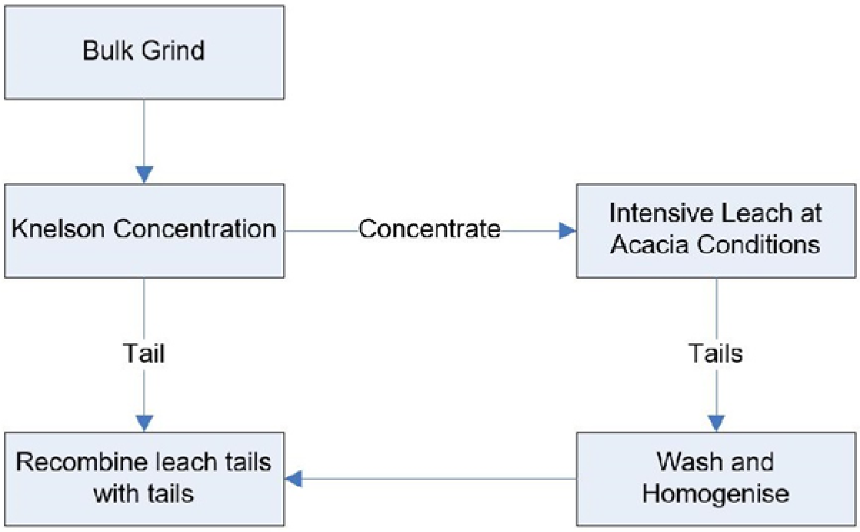
Element	Unit	LDL	Master Composite
Forecast Au	g/t		6.66
Au Average	g/t	0.005	12.61
Au	g/t	0.005	10.41
Au	g/t	0.005	14.82
Ag	ppm	2	2
As	ppm	20	<20
Cu	ppm	5	368
Sb	ppm	0.5	<0.5
Te	ppm	0.5	<0.5
S	%	0.01	0.46
Sulphide	%	0.01	0.46
Sulphate	%	0.01	<0.01
Total Carbon	%	0.01	0.03
Non-Carbonate Carbon	%	0.01	0.03
Fe	%	0.01	1.47
Mg	%	0.01	0.26
Pb	ppm	20	80
Zn	ppm	5	43

GRAVITY RECOVERY

Gravity recovery test work, as presented in Figure 13-3, was conducted on a bulk master composite sample to allow:

- a consistent feed to the leach optimisation tests;
- determine cyanide leach kinetics of a gravity tailings, as would be expected in a processing plant;
- determine a gravity recovery from a single Knelson concentration step (the Knelson concentrate was not panned to reduce the mass of the gravity concentrate).

FIGURE 13-3 MASTER COMPOSITE GRAVITY TEST WORK BLOCK FLOW DIAGRAM



Gravity gold recovery presented in Table 13-9 was back-calculated from the intensive leach solution assays and average calculated head grades from the gravity tailings leach tests. Summarised results indicate a mass recovery of 0.48% and high gravity recoverable gold of 77.6%.

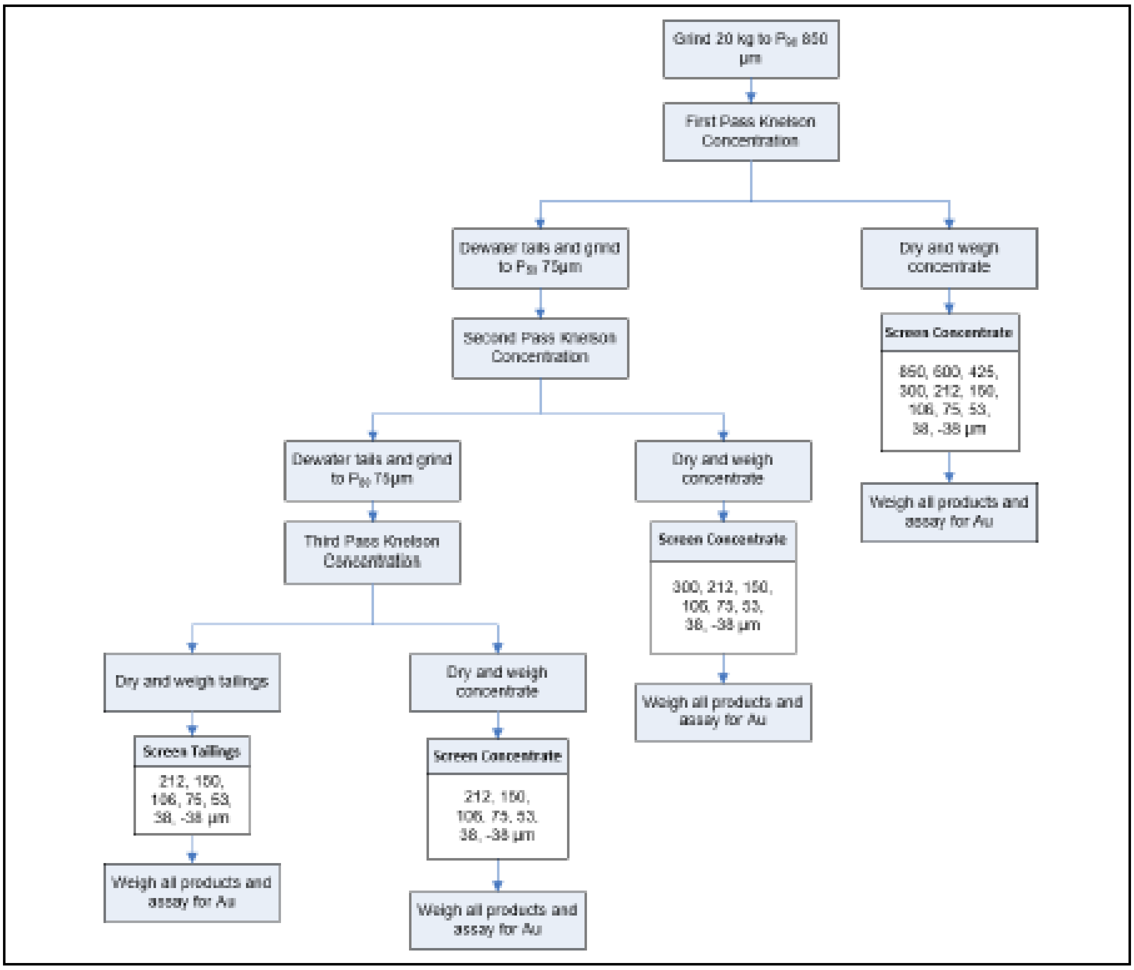
TABLE 13-9 MASTER COMPOSITE GRAVITY RECOVERY RESULTS
Thor Explorations Ltd – Segilola Gold Project

	Units	Master Composite
Concentrate Mass Recovery	%	0.48
Calculated Head Grade	g/t	8.9
Gravity Gold Recovery	%	77.6%
Gravity Gold Grade	g/t	6.9
Leach Feed Grade	g/t	1.99

MASTER COMPOSITE GRAVITY RECOVERABLE GOLD MODELLING

A comprehensive Gravity Recoverable Gold (GRG) test was conducted on a bulk 20 kg master composite sample, with the test work results submitted to Consep Pty Ltd for GRG modelling. Gravity test work block flow diagram is shown in Figure 13-4.

FIGURE 13-4 GRAVITY RECOVERABLE GOLD TEST WORK BLOCK FLOW DIAGRAM



The GRG modelling conducted by Consep indicated the following results:

- 49.7% of the feed Au is recovered in the first stage of Knelson concentration.
- Modelled processing gravity recovery of 43.0%, equating to approximately 55.7% of the total GRG.
- Final expected gravity concentrate grade of approximately 7,461 g/t Au.
- It was necessary to linearly extrapolate values for two assays for the size by size analysis of the second stage concentrate as there was insufficient sample to assay; this introduces some uncertainty to the modelling results.

CYANIDE LEACHING

Cyanide leaching of the master composite was conducted on 1 kg sub-samples of the recombined gravity concentration tailings. Three stages of leach tests were conducted on the samples, targeting:

- optimisation of the leach feed grind size;

- cyanide concentrations; and
- dissolved oxygen concentrations.

Each stage of master composite leach optimisation is discussed in the following sections.

GRIND SIZE OPTIMISATION

Grind size optimisation of the master composite involved three cyanide leach tests conducted at P_{80} of 75 µm, 106 µm, and 150 µm. Conditions utilised throughout the leach tests were as follows:

- 500 ppm NaCN initial, maintained at 300 ppm.
- Dissolved oxygen maintained between 6-8 mg/L by air addition.
- 40% solids in Perth tap water.
- pH maintained at 9.8 with lime.
- Kinetic points at 2, 4, 8, 24, and 48 hours.

Results for the grind size optimisation leach tests indicate overall gold recoveries ranging from 98.3% to 99.5%. Summarised results and kinetic leach curves are presented in Table 13-10 indicating the following:

- Gravity recoveries average 77.4%.
- Overall gold recoveries of 98.4%, 99.3% and 99.5% for LT1 (150 µm), LT2 (106 µm), and LT3 (75 µm) respectively, indicating increasing recoveries with reducing grind size.
- Reduced leach kinetics at 150 µm (90.2%, 8-hour recovery), whilst 106 µm resulted in the fastest leach kinetics (95.6%, 8-hour recovery).
- Increased residue grades of 0.15 g/t Au for 150 µm compared to 0.06 g/t Au and 0.05 g/t Au for 106 µm and 75 µm grinds respectively.

TABLE 13-10 GRIND SIZE OPTIMISATION LEACH TEST RESULTS
Thor Explorations Ltd – Segilola Gold Project

Grind Size P_{80}	Units µm	LT1 150	LT2 106	LT3 75
Gravity Recovery	%	77.12	78.44	76.74
Overall 8 Hour Recovery	%	90.20	95.55	93.33
Overall 24-Hour Recovery	%	97.32	99.49	99.19
Overall 48 Hour Recovery	%	98.35	99.32	99.47
Gravity Recovery	g/t	6.84	6.86	6.79
Leach Recovery	g/t	1.88	1.83	2.01
Total Recovery	g/t	8.73	8.69	8.8
Calculated Head Grade	g/t	8.87	8.75	8.84
Assayed Head Grade	g/t	12.61	12.61	12.61
Residue Grade	g/t	0.15	0.06	0.05
24 Hour Cyanide Cons	kg/t	0.04	0.08	0.08
48 Hour Cyanide Cons	kg/t	0.11	0.12	0.15
24-hour Lime Cons	kg/t	0.51	0.29	0.3

Cyanide consumptions were low, ranging from 0.11 kg/t to 0.15 kg/t for a 48-hour duration. A trend of increased cyanide consumption with reducing grind size can be observed indicating the liberation of cyanide consumers such as sulphide and copper from the finer grind size.

Lime consumption was very low, ranging from 0.29 kg/t to 0.51 kg/t for 24 hours with no increase for 48 hours. Further testing should be conducted in site water to determine any buffering effects.

CYANIDE OPTIMISATION

Following completion of the leach grind size optimisation, two leach tests were conducted on the master composite samples to determine the effect of cyanide concentration on the leach kinetics and overall recovery. The cyanide dosages used for the two tests were as follows:

- LT4 - 300 ppm initial, 150 ppm maintained.
- LT5 - 750 ppm initial, 400 ppm maintained.

The conditions maintained throughout the two tests were as follows:

- Dissolved oxygen maintained between 6 mg/L to 8 mg/L.
- 40% solids in Perth tap water.
- pH maintained at 9.8 with lime.
- Grind size P_{80} 106 µm.
- Kinetic points at 2, 4, 8, 24, and 48 hours.

Cyanide optimisation leach tests resulted in gold recoveries ranging from 98.9% to 99.4%. Summarised results and kinetic leach curves are shown in Table 13-11 with the inclusion of LT2 (CN 500 – 300 ppm) as a comparison, indicating the following:

- Gravity recoveries averaging 78.3%.
- Overall 48-hour recoveries of 98.9%, 99.3%, and 99.4% for LT4 (300 ppm), LT2 (500 ppm), and LT5 (750 ppm) respectively, indicating decreased recovery at a reduced CN concentration.
- Slower leach kinetics at a reduced CN concentration of 300 ppm, with a 6.5% reduction in 8-hour leach recovery compared to 500 ppm initial cyanide concentration.
- Increased residue grade by 0.04 g/t Au due to the reduced CN concentrations from 0.06 g/t Au to 0.10 g/t Au.

Calculated head grades for the leach tests ranged from 8.73 g/t Au to 8.93 g/t Au, consistent with those reported in the first round of grind size optimisation leaching.

TABLE 13-11 CYANIDE OPTIMISATION LEACH TEST RESULTS SUMMARY
Thor Explorations Ltd – Segilola Gold Project

CN Concentration	Units ppm	LT2 500-300	LT4 300-150	LT5 750-400
Gravity Recovery	%	78.44	77.2	79.3
Overall 8 Hour Recovery	%	99.55	89.08	94.54
Overall 24-Hour Recovery	%	99.49	97.02	99.23
Overall 48-Hour Recovery	%	99.32	98.89	99.35
Gravity Recovery	g/t	6.86	6.89	6.92
Leach Recovery	g/t	1.83	1.94	1.75
Total Recovery	g/t	8.69	8.83	8.67
Calculated Head Grade	g/t	8.75	8.93	8.73
Assayed Head Grade	g/t	12.61	12.61	12.61
Residue Grade	g/t	0.06	0.1	0.06
24 Hour Cyanide Cons’	kg/t	0.08	0.1	0.21
48 Hour Cyanide Cons’	kg/t	0.12	0.16	0.34
24 Hour Lime Cons’	kg/t	0.29	0.3	0.3

Cyanide consumptions were low, ranging from 0.12 kg/t to 0.34 kg/t for a 48-hour duration, increasing as expected with increased cyanide tenor. Lime consumptions were low, ranging from 0.29 kg/t to 0.30 kg/t for 24 hours with no increase for 48 hours. Further testing should be conducted in site water to determine any buffering effects.

The overall results discussed above led to the selection of an initial CN concentration of 300 ppm, maintained at 150 ppm to be utilised throughout the remainder of the test work program.

CARBON IN LEACH OPTIMISATION

One Carbon In Leach (CIL) test was conducted on the master composite following determination of the optimum conditions as follows:

- Dissolved oxygen maintained between 6-8 mg/L.
- 300 ppm initial, 150 ppm maintained.
- 40% solids in Perth tap water.
- pH maintained at 9.8.
- Grind size P_{80} 106 µm.
- 20 g/L Carbon.

Results for the master composite CIL test are presented in Table 13-12, including results from LT4 (optimum conditions) indicating the following:

- Gravity recoveries of 77.2% to 80.0% for LT4 and LT8 respectively.
- Minor 0.1% variation in overall Au recovery.
- Residue grades of 0.10 g/t Au and 0.11 g/t Au for LT4 and LT8 respectively indicating constant recovery for the two tests.

TABLE 13-12 CARBON IN LEACH TEST RESULTS SUMMARY
Thor Explorations Ltd – Segilola Gold Project

	Units	LT4 Optimum	LT8 CIL
Gravity Recovery	%	77.20%	80.80%
Overall 48 Hour Recovery	%	98.90%	98.80%
Gravity Recovery	g/t	6.89	6.93
Leach Recovery	g/t	1.94	1.54
Total Recovery	g/t	8.83	8.47
Calculated Head Grade	g/t	8.93	8.58
Assayed Head Grade	g/t	12.61	12.61
Residue Grade	g/t	0.1	0.11
24 Hour Cyanide Cons	kg/t	0.1	0.28
48 Hour Cyanide Cons	kg/t	0.16	0.39
24 Hour Lime Con	kg/t	0.3	0.15

VARIABILITY COMPOSITES TEST WORK

HEAD ASSAY

Summarised variability composite head assay results are presented in Table 13-13 revealing average gold grades ranging from 1.2 g/t Au to 18.6 g/t Au. In general, the composites indicate a close correlation between the calculated and assay feed grades with the exception of composites 1, 2, and 4.

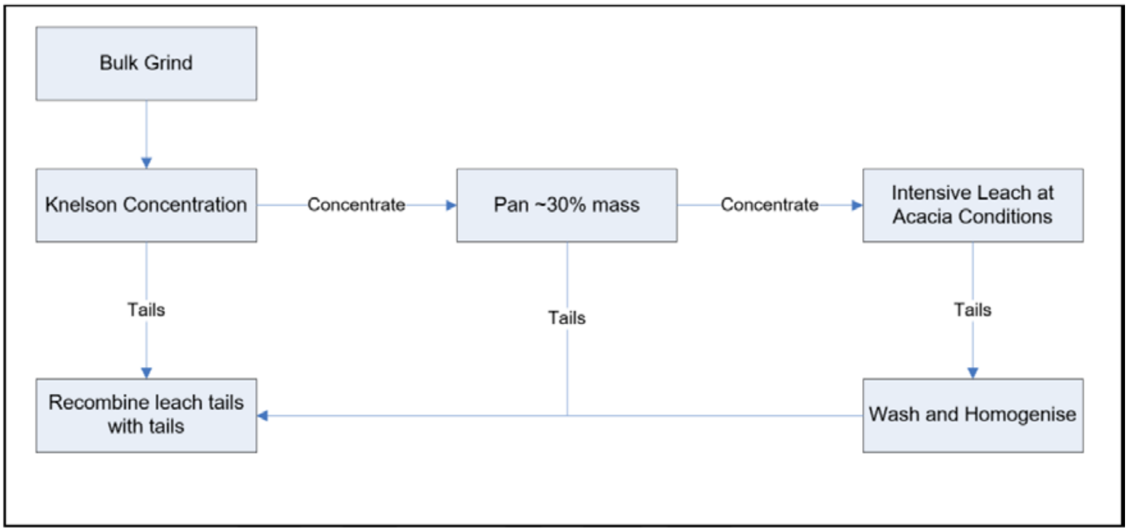
TABLE 13-13 VARIABILITY COMPOSITE HEAD ASSAY RESULTS
Thor Explorations Ltd – Segilola Gold Project

Element	Unit	LDL	Variability Composite									
			1	2	3	4	5	6	7	8	9	10
Forecast Au	g/t	-	6.66	9.31		6.68		2.19	2.62	3.39	1.23	2.4
Au Average	g/t	-	18.56	5.04	2.32	3.22	4.48	2.54	1.61	2.33	1.17	1.85
Au	g/t	0.005	16.35	4.77	2.05	3	4.53	2.78	1.44	2.77	1.15	1.53
Au (dup)	g/t	0.005	20.77	5.32	2.6	3.44	4.42	2.3	1.78	1.9	1.2	2.18
Ag	ppm	2	<2	<2	<2	<2	<2	<2	<2	4	<2	<2
As	ppm	20	<20	<20	<20	<20	<20	<20	<20	<20	<20	<20
Cu	ppm	5	63	73	56	73	61	180	84	131	96	122
Sb	ppm	0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	0.8	<0.5	<0.5
Te	ppm	1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1
S	%	0.01	0.55	0.54	0.34	0.31	0.22	0.46	0.23	0.5	0.3	0.35
Sulphide	%	0.01	0.55	0.52	0.34	0.31	0.22	0.46	0.23	0.5	0.3	0.35
Sulphate	%	0.01	<0.01	0.02	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Total Carbon	%	0.01	0.08	0.05	0.05	0.04	0.02	0.09	0.06	0.08	0.07	0.08
Non-Carbonate Carbon	%	0.01	0.03	0.02	0.03	0.03	0.03	0.03	0.02	0.03	0.03	0.03

GRAVITY RECOVERY

Gravity recoverable gold of the ten variability composites (Figure 13-5) was assessed by grinding a 5 kg charge to P_{80} 300 µm to mimic a cyclone underflow recycle stream and passed through a laboratory scale 3" Knelson concentrator. The Knelson concentrate was then panned to remove approximately 70% of the mass with the concentrate subsequently leached for 20 hours at 10% solids with a standard assay tab. The leach residue was thoroughly washed and blended back together with the Knelson tailings.

FIGURE 13-5 VARIABILITY COMPOSITE GRAVITY TEST WORK BLOCK FLOW DIAGRAM



Gravity gold recovery of the variability composites presented in Tables 13-14 and 13-15 was back-calculated from the intensive leach solution assays and averages of the calculated head grades from the gravity tailings leach tests.

Gravity test work results indicate mass recoveries ranging from 0.12% to 0.25% and high gravity recoverable gold ranging from 40.97% (0.45 g/t Au) to 80.88% (7.30 g/t Au), compared to 77.6% (6.9 g/t Au) for the master composite.

TABLE 13-14 VARIABILITY COMPOSITE 1 TO 5 GRAVITY RECOVERY RESULTS
Thor Explorations Ltd – Segilola Gold Project

	Units	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5
Concentrate Mass Recovery	%	0.22	0.17	0.12	0.2	0.16
Calculated Head Grade	g/t	18.96	9.03	6.21	3.44	4.42
Gravity Gold Recovery	%	59.99	80.88	68.8	54.17	63.27
Gravity Gold Recovery	g/t	11.37	7.3	4.27	1.86	2.75
Leach Feed Grade	g/t	7.58	1.73	1.94	1.58	1.6

TABLE 13-15 VARIABILITY COMPOSITE 6 TO 10 GRAVITY RECOVERY RESULTS
Thor Explorations Ltd – Segilola Gold Project

	Units	Comp 6	Comp 7	Comp 8	Comp 9	Comp 10
Concentrate Mass Recovery	%	0.17	0.22	0.25	0.14	0.24
Calculated Head Grade	g/t	4.42	1.9	3.13	1.1	2.42
Gravity Gold Recovery	%	67.58	46.57	48.29	40.97	61.43
Gravity Gold Recovery	g/t	2.98	0.88	1.51	0.45	1.49
Leach Feed Grade	g/t	1.43	1.01	1.62	0.65	0.93

CYANIDE LEACHING

Cyanide leaching of the ten variability composites was conducted on 1 kg subsamples of the recombined gravity concentration tailings.

Two stages of leach tests were conducted on the samples, the first utilising optimum conditions determined from the master composite test work, and the second to optimise the leach recoveries based on the response from the first round of leaches.

VARIABILITY LEACHING - ROUND 1

The first round of leach tests conducted on the variability composites utilised the optimum conditions determined from the master composite test work and is as follows:

- Air sparging, targeting a dissolved oxygen concentration of 6 ppm to 8 ppm.
- Cyanide concentrations of 300 ppm initial, 150 ppm maintained.
- 80% passing grind size of 106 µm.
- 40% solids in Perth tap water.
- pH maintained at 9.8 with lime.

Variability leach test kinetic recoveries are presented in Tables 13-16 and 13-17. These results indicate the following:

- Gravity gold recoveries ranging from 41.7% to 82.5% (0.45 g/t Au to 7.32 g/t Au).
- Overall Au recoveries ranging from 94.4% to 99.6% (1.02 g/t Au to 18.12 g/t Au).
- Increased leach kinetics for composites possessing lower gravity recoveries, indicating the presence of coarse gravity gold within the leach feed.
- Residue grades ranging from 0.04 g/t Au to 0.8 g/t Au.

TABLE 13-16 VARIABILITY COMPOSITE 1 TO 5 ROUND 1 LEACH TEST RESULTS
Thor Explorations Ltd – Segilola Gold Project

	Units	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5
Gravity Recovery	%	60.01	82.48	71.27	52.61	64.51
Overall 8 Hour Recovery	%	76.13	93	83.6	75.73	76.85
Overall 24-Hour Recovery	%	90.47	98.87	94.84	92.05	89.51
Overall 48-Hour Recovery	%	95.68	99.58	99.13	98.35	96.26
Calculated Head Grade	g/t	18.94	8.88	6.08	3.54	4.28
Assayed Head Grade	g/t	18.56	5.04	2.32	3.22	4.48
Residue Grade	g/t	0.82	0.04	0.05	0.06	0.16
24 Hour Cyanide Cons’	kg/t	0.08	0.08	0.01	0.04	0.02
48 Hour Cyanide Cons’	kg/t	0.12	0.09	0.06	0.08	0.12
24 Hour Lime Cons’	kg/t	0.2	0.25	0.28	0.36	0.27

TABLE 13-17 VARIABILITY COMPOSITE 6 TO 10 ROUND 1 LEACH TEST RESULTS
Thor Explorations Ltd – Segilola Gold Project

	Units	Comp 6	Comp 7	Comp 8	Comp 9	Comp 10
Gravity Recovery	%	68.23	45.82	46.76	41.69	61.07
Overall 8 Hour Recovery	%	82.7	74.41	69.75	77.98	76.23
Overall 24-Hour Recovery	%	94.15	95.74	89.41	92.54	92.64
Overall 48-Hour Recovery	%	97.26	96.85	94.6	94.37	97.75
Calculated Head Grade	g/t	4.4	1.93	3.28	1.08	2.45
Assayed Head Grade	g/t	2.54	1.61	2.33	1.17	1.85
Residue Grade	g/t	0.12	0.06	0.18	0.06	0.06
24 Hour Cyanide Cons’	kg/t	0.14	0.08	0.17	0.12	0.13
48 Hour Cyanide Cons’	kg/t	0.26	0.21	0.24	0.23	0.2
24 Hour Lime Cons’	kg/t	0.09	0.11	0.05	0.04	0.09

The reduced gravity recovery achieved for Composite 1 may have been due to significantly high head grade of 18.9 g/t Au, which may have caused over-loading of the bench Knelson concentrator resulting in restricted further Au gravity recovery.

Calculated head grades for composites 1 to 10 ranged from 1.08 g/t Au to 18.94 g/t Au, similar to those reported in the head assay results. However, composite 2 and 3 both reported calculated head grades approximately 3.8 g/t Au higher than the original assays, due to the coarse spotty nature of gold occurrence (‘nugget effect’) in the ore.

Overall variability leach test results indicate high recoveries across each of the composites with final 48-hour Au recoveries ranging from 94.4% to 99.6%. Leach recovery increases between 24 and 48 hours ranged from 0.7% to 6.8% with an average of 4.0%, markedly increased compared to the 1.9% increase for the master composite.

Cyanide consumptions were low to moderate, ranging from 0.06 kg/t to 0.26 g/t for a 48-hour duration. Lime consumption was low, ranging from 0.04 kg/t to 0.36 kg/t for 24 hours with no increase for 48 hours.

Further testing should be conducted in site water to determine any buffering effects.

VARIABILITY LEACHING - ROUND 2

The second round of variability composite leach tests were conducted at an increased cyanide concentration, targeting improved kinetics and reduced residence times noting the 4.0% average increase in gold extraction for phase 1. Conditions maintained throughout the tests were as follows:

- NaCN 500 ppm initial, maintained at 300 ppm.
- Air sparging, targeting a dissolved oxygen concentration of 6 ppm to 8 ppm.
- 80% passing grind size of 106 µm.
- 40% solids in Perth tap water.
- pH maintained at 9.8 with lime.

Variability leach test kinetic curves are presented in Tables 13-18 and 13-19. These results indicate the following:

- Gravity recoveries ranging from 40.2% to 79.3% (0.45 g/t Au to 7.27 g/t Au).
- Overall Au recoveries ranging from 96.5% to 99.6% (1.07 g/t Au to 18.58 g/t Au).
- Minor variation in leach recoveries resulting from increased cyanide concentrations.
- Residue grades ranging from 0.04 g/t Au to 0.4 g/t Au.

Calculated head grades for composites 1 to 10 ranged from 1.11 g/t Au to 18.97 g/t Au, similar to those reported in the first round of variability leach testing.

The variation in gold recovery between 24-hour and 48-hour retention times for the variability composites is summarised in Table 13-20. These results indicate that the Round 1 leach tests achieved an average increase in recovery of 3.96% in recovery between 24-hour and 48-hour leach residence times. The increased cyanide concentrations utilised in the Round 2 resulted in an average recovery increase of 3.62% between the 24-hour to 48-hour leach times; marginally lower than the Round 1 results.

Conducting a CIL test on each of the variability composites may potentially increase the leach kinetics and improve the 24-hour gold recoveries.

TABLE 13-18 VARIABILITY COMPOSITE 1 TO 5 ROUND 2 LEACH TEST RESULTS
Thor Explorations Ltd – Segilola Gold Project

	Units	Comp 1 LT11	Comp 2 LT12	Comp 3 LT13	Comp 4 LT14	Comp 5 LT15
Gravity Recovery	%	59.98	79.27	66.33	55.73	62.04
Overall 8 Hour Recovery	%	78.97	91.52	82.28	79.31	80.38
Overall 24-Hour Recovery	%	91.47	97.78	94.34	94.17	92.63
Overall 48-Hour Recovery	%	97.9	99.6	99.03	98.23	98.25
Calculated Head Grade	g/t	18.97	9.17	6.34	3.34	4.42
Assayed Head Grade	g/t	18.56	5.04	2.32	3.22	4.48
Residue Grade	g/t	0.4	0.04	0.06	0.06	0.08

	Units	Comp 1 LT11	Comp 2 LT12	Comp 3 LT13	Comp 4 LT14	Comp 5 LT15
24 Hour Cyanide Cons'	kg/t	0.08	0.11	0.09	0.1	0.09
48 Hour Cyanide Cons'	kg/t	0.22	0.28	0.21	0.09	0.22
24 Hour Lime Cons'	kg/t	0.1	0.1	0.11	0.13	0.12

TABLE 13-19 VARIABILITY COMPOSITE 6 TO 10 ROUND 2 LEACH TEST RESULTS SUMMARY
Thor Explorations Ltd – Segilola Gold Project

	Units	Comp 1 LT11	Comp 2 LT12	Comp 3 LT13	Comp 4 LT14	Comp 5 LT15
Gravity Recovery	%	59.98	79.27	66.33	55.73	62.04
Overall 8 Hour Recovery	%	78.97	91.52	82.28	79.31	80.38
Overall 24-Hour Recovery	%	91.47	97.78	94.34	94.17	92.63
Overall 48-Hour Recovery	%	97.9	99.6	99.03	98.23	98.25
Gravity Recovery	g/t	11.38	7.27	4.21	1.86	2.74
Leach Recovery	g/t	7.2	1.86	2.07	1.42	1.6
Total Recovery	g/t	18.58	9.14	6.28	3.28	4.34
Calculated Head Grade	g/t	18.97	9.17	6.34	3.34	4.42
Assayed Head Grade	g/t	18.56	5.04	2.32	3.22	4.48
Residue Grade	g/t	0.4	0.04	0.06	0.06	0.08
24 Hour Cyanide Cons'	kg/t	0.08	0.11	0.09	0.1	0.09
48 Hour Cyanide Cons'	kg/t	0.22	0.28	0.21	0.09	0.22
24 Hour Lime Cons'	kg/t	0.1	0.1	0.11	0.13	0.12

TABLE 13-20 VARIABILITY COMPOSITE RECOVERY DIFFERENCE FROM 24 HOUR TO 48 HOUR LEACH RETENTION TIME
Thor Explorations Ltd – Segilola Gold Project

	Round 1	Round 2
Minimum	0.72%	-0.48%
Maximum	6.75%	6.44%
Average	3.96%	3.62%
Median	4.70%	3.82%
Standard Deviation	2.15%	1.94%

TEST EVALUATION

- The test work programme is considered to be appropriate for development of a process flowsheet.
- The flowsheet selected is appropriate to the test work findings.
- The design parameters are in agreement with the test work findings.

- The Bond work indices and abrasion indices are slightly higher than the average Western Australian 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

INTRODUCTION

The open pit Mineral Resource estimate for the Segilola Gold Project has been prepared as at 1 December 2018 by Mr. Christopher Speedy, MAIG, of Auralia, who is a QP under NI 43-101.

The underground Mineral Resource estimate for the Project has been prepared as at 1 December 2018 by Mr. Jack Lunnon, CGeol, of RPA, who is a QP under NI 43-101.

All gold grade estimation was completed using Ordinary Kriging (OK). This estimation approach is considered appropriate based on review of a number of factors, including the quantity and spacing of available data, the interpreted controls on mineralisation, and the style and geometry of mineralisation. The estimation was constrained with geological and mineralisation interpretations.

Table 14-1 summarises the open pit and underground Mineral Resources as at 1 December 2018. Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM (2014) definitions) were used for Mineral Resource classification.

TABLE 14-1 SUMMARY OF MINERAL RESOURCE ESTIMATE – 1 DECEMBER 2018
Thor Explorations Ltd. – Segilola Gold Project

Source	Classification	Tonnes (Mt)	Grade (g/t Au)	Contained Metal (000 oz Au)
Open Pit	Indicated	3.03	4.52	441
	Inferred	0.33	6.8	73
Underground	Indicated	0.09	9.39	28
	Inferred	0.35	7.9	90
Total	**Indicated**	**3.12**	**4.67**	**469**
	Inferred	**0.68**	**7.4**	**162**

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Open pit Mineral Resources are estimated by Auralia at a cut-off grade of 0.64 g/t Au and constrained within a pit optimisation shell using an Au price of $1,500/oz Au.
3. Underground Mineral Resources are estimated by RPA at a cut-off grade of 2.58 g/t Au and constrained within stope shapes using an Au price of $1,500/oz Au.
4. Open pit wireframes were defined using a nominal 0.50 g/t Au wireframing cut-off.

5. Underground wireframes were defined using a nominal 2.50 g/t Au wireframing cut-off and 2 m minimum mining width.
6. Open pit bulk density was interpolated using Inverse Distance Weighting squared.
7. Underground bulk density is 2.70 t/m^3.
8. High gold assays were capped to 40 g/t Au for open pit resources and 50 g/t Au for underground resources.
9. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability under the parameters used.
10. Numbers may not add due to rounding.

The QPs are not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimates.

OPEN PIT MINERAL RESOURCE ESTIMATE

The open pit Mineral Resource estimate for the Segilola Gold Project has been prepared as at 1 December 2018 by Mr. Christopher Speedy, MAIG, of Auralia, who is a QP under NI 43-101.

Auralia estimated the open pit Mineral Resources using all drill hole data available up until 1 December 2018.

CIM (2014) definitions were used for Mineral Resource classification. Table 14-2 summarises the open pit Mineral Resources for the Project.

TABLE 14-2 SUMMARY OF OPEN PIT MINERAL RESOURCES – 1 DECEMBER 2018
Thor Explorations Ltd – Segilola Gold Project

Category	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Indicated	3,032	4.52	441
Inferred	331	6.8	73

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Open Pit Mineral Resources are reported within an optimised pit shell, as defined by the parameters shown in Table 14-19 at a cut-off grade of 0.64 g/t Au.
3. Estimation constrained by wireframes defined using a nominal 0.5 g/t Au lower cut-off.
4. The Mineral Resource estimate has been prepared by Mr Christopher Speedy of Auralia Mining Consulting Pty Ltd, who is a qualified person under NI 43-101.
5. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.
6. Thor has adopted industry-standard procedures for sampling, data verification, compiling, interpreting and processing the data used to estimate Mineral Reserves and Mineral Resources.
7. Mineral Resources are inclusive of Mineral Reserves.
8. Numbers may not sum due to rounding.

Auralia is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the open pit Mineral Resource estimates.

DATABASE VALIDATION

Auralia undertook the following data validation steps:

- Checked for missing collar co-ordinates, missing hole depths, missing downhole surveys; mis-matched collar, survey or assay depths; and overlapping intervals.
- Checked for missing or overlapping intervals for geology and assay interval data.
- Checked for negative and null assays

The validity of the database used for the Mineral Resource estimation has been confirmed with checks for internal consistency and accuracy. As a result of these checks, the QP considers that the drill hole data has been adequately validated with satisfactory quality control analysis. The QP is of the opinion that the data is suitable for use in the estimation of Indicated and Inferred Mineral Resources.

SUMMARY OF DATA USED IN ESTIMATE

The area of the resource was drilled using diamond core holes on a nominal 25 m to 30 m by 25 m grid spacing or closer. A total of 197 diamond core holes (22,361m) were used in the geological model and resource estimation process. The majority of drill hole azimuths were 090° magnetic at declinations of between -50° and -60°, to optimally intersect the mineralised zones. A small number of drill holes (11) were drilled vertically due to restricted rig access. Resource database statistics are shown below in Table 14-3.

TABLE 14-3 RESOURCE DATABASE STATISTICS
Thor Explorations Ltd – Segilola Gold Project

Item	No.
Resource Holes (Includes 1 Geotech and 7 Metallurgical Drill Holes)	194
Total Drilled (m)	22,361
Shallowest Hole (m)	18
Deepest Hole (m)	341
Average Depth (m)	113.5
Assay Intervals	10,657
Lithology Intervals	1,954
Weathering Intervals	545

INTERPRETATION AND MODELLING

MINERALISATION INTERPRETATION

The mineralisation wireframes were generated by Thor and were provided to Auralia for use in the Mineral Resource estimate. Auralia checked the reasonableness of the wireframes and made minor modifications where appropriate.

The mineralised 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 locality of gold mineralization. This relationship has generated multiple zones of gold mineralisation hosted within a shear zone now represented by chlorite and calcite alteration, together with quartz veining and pyrite development.

The resource wireframes represent multiple mineralised structures that trend 010°, dip between 80° and 70° towards the west and extend over a continuous strike length of 2 km (Figures 14-1 and 14-2).

Lode 100 is 1,770 m in strike length, has an average thickness of 2 m, and on average extends 140 m down dip up to a maximum of 280 m. Lode 200 is 1,500 m in strike length, varies in true width from 3 m to 18 m, and extends up to 430 m down dip, with an average of 250 m. Lode 200 is the main footwall lode. Lode 300 is 580 m in strike length, has an average thickness of 1.5 m, and extends to a maximum of 180 m down dip with an average of 130 m.

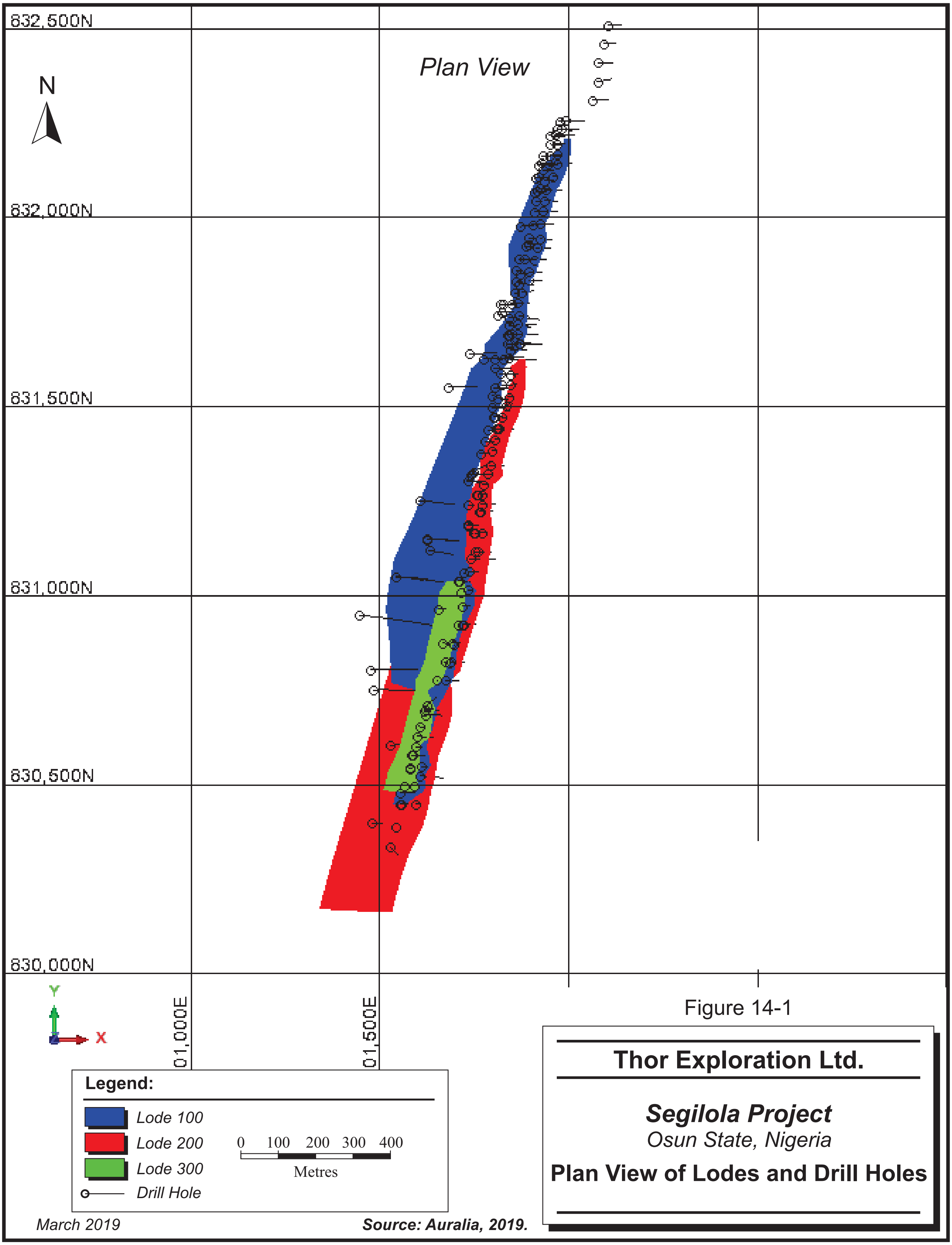


Figure 14-1

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Plan View of Lodes and Drill Holes

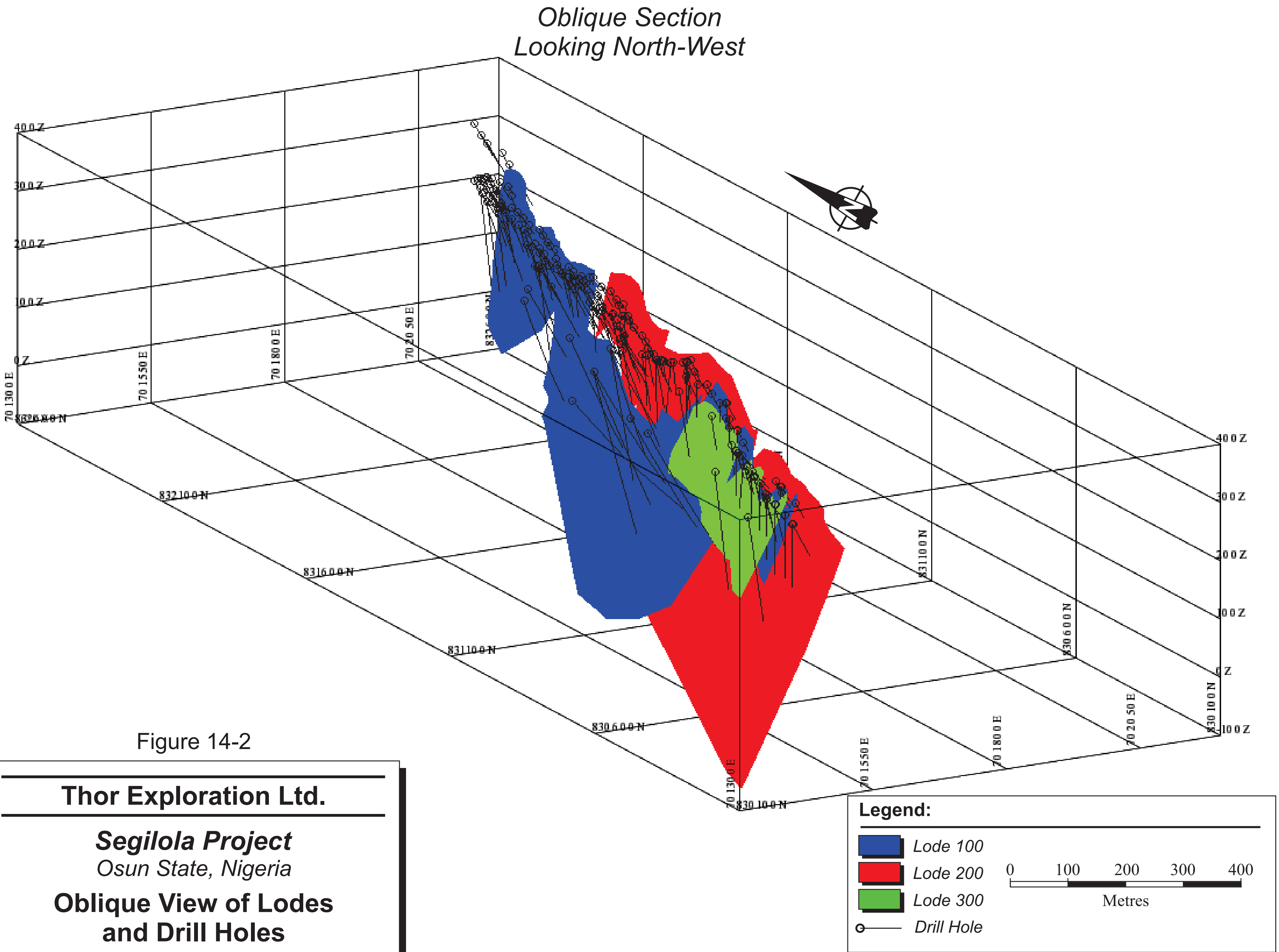


Figure 14-2

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Oblique View of Lodes and Drill Holes

March 2019

Source: Auralia, 2019.

PREPARATION OF WIREFRAMES

The wireframes were constructed on approximately 25 m spaced cross sections orientated perpendicular to drilling. A nominal 0.5 g/t Au cut-off grade was used to define the overall shear zone mineralisation. This cut-off grade was chosen as there is a sharp transition between background or below detection levels to >0.5 g/t Au. Good continuity of the mineralised envelopes was observed at the scale of the sectional spacing with moderate grade variability typical for deposits of this type.

The interpretation was designed to capture the broad mineralisation halo that encompasses the geological vein system and was not intended to constrain individual veins or vein clusters. This interpretation is appropriate for the estimation technique for the major domains, and for the proposed open pit mining method.

Due to the varying hole inclinations, the allowed amount of internal dilution was set nominally at 2.5 m true width or less. No minimum thickness was applied.

The following techniques were generally employed while interpreting the mineralisation:

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

The QP reviewed the wireframes provided by Thor and noted some inconsistencies:

- At times the wireframes were not consistent in how far they were extended beyond the first and last interpreted cross section, and between sections. There was no issue with the overall interpreted direction and dip of the envelopes which were maintained across the deposit length.

- There were instances of internal dilution occurring in some of the larger mineralisation intercepts which can lead to the block model estimation method smoothing the grade in localised areas.

- A minimum thickness was not applied to the wireframes which has resulted in the wireframes "pinching" to accommodate thin sample intervals (approximately 0.5 m). To accommodate this pinching, the block model was sub-blocked to a relatively small size which can cause issues in estimation.

- The QP recommends that a minimum width be applied to the wireframes. Alternatively, a minimum grade multiplied by thickness cut-off should be applied when reporting the Mineral Resource.

Despite the above inconsistencies, the QP is of the opinion that the wireframes are acceptable for the resource estimation.

Weathering surfaces were generated for base of oxidation and the top of fresh rock by Laplace gridding of downhole points extracted from the weathering database. A surface for the competent/semi-competent rock was developed from manual interpretation of the logging data and core photography.

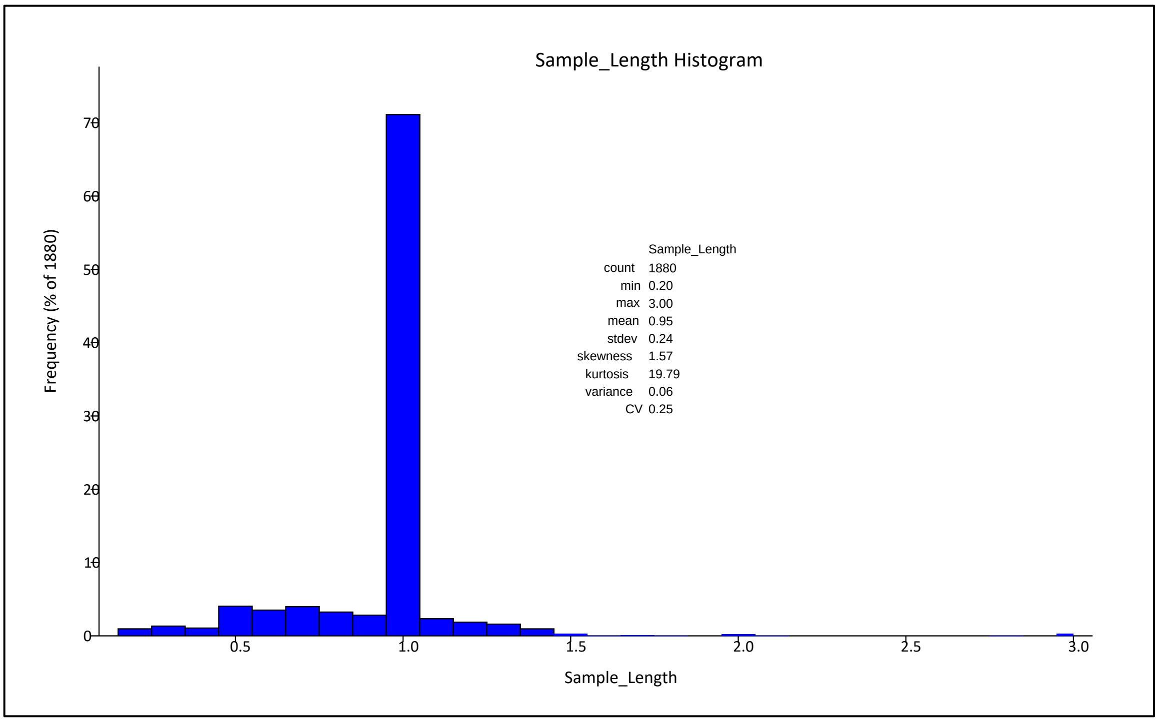
DATA FLAGGING AND COMPOSITING

Assays were flagged according to mineralised domain. Each domain was assigned a unique numerical code to allow the application of hard boundary domaining if required during grade estimation.

The samples were then composited as a means of achieving uniform sample support. It should be noted, however, that equalising sample length is not the only criterion for standardising sample support. Factors such as angle of intersection of the sampling to mineralisation, sample type and diameters, drilling conditions, recovery, sampling/sub-sampling practices and laboratory practices all affect the 'support' of a sample. Exploration and mining databases which contain multiple sample types and/or sources of data provide challenges in generating composite data with equalised sample support, and uniform support is frequently difficult to achieve.

The lengths of the samples were statistically assessed prior to selecting an appropriate composite. The majority (70%) of the samples within the resource wireframes are sampled at 1.0 m (Figure 14-3).

FIGURE 14-3 SAMPLE LENGTH INSIDE RESOURCE WIREFRAMES



After consideration of relevant factors relating to geological setting and mining, including likely mining selectivity and bench/flitch height, a regular 1.0 m composite was selected as the most appropriate composite interval. A total of 149 samples greater than 1.0 m were split to make the 1.0 m composites. Composites of less than 1.0 m were retained by addition to the previous composite resulting in a composite file containing composites between 1.0 m and 1.7 m.

These composites were used for subsequent statistical, geostatistical, and grade estimation investigations

STATISTICAL ANALYSIS

Summary descriptive statistics were generated for all domains (Table 14-4). A high maximum grade of 135 g/t Au was noted for Lode 300. The grade distributions are typical for gold deposits of this style and show a positive skew or near lognormal behaviour (Figures 14-4 to 14-6). The coefficient of variation is moderately high, consistent with the presence of high outlier grades that potentially require cutting (capping) for grade estimation.

TABLE 14-4 SUMMARY STATISTICS GROUPED BY LODE FOR 1 M COMPOSITES OF UNCUT GOLD GRADE
Thor Explorations Ltd – Segilola Gold Project

Item	Lode 100	Lode 200	Lode 300
Count	391	1,159	68
Minimum (g/t Au)	0.02	0.01	0.01
Maximum (g/t Au)	117.63	75.87	135.00
Mean (g/t Au)	6.62	3.55	11.52
Std. Dev	11.98	6.31	33.83
CV	1.81	1.81	2.94

FIGURE 14-4 HISTOGRAM AU LODE 100

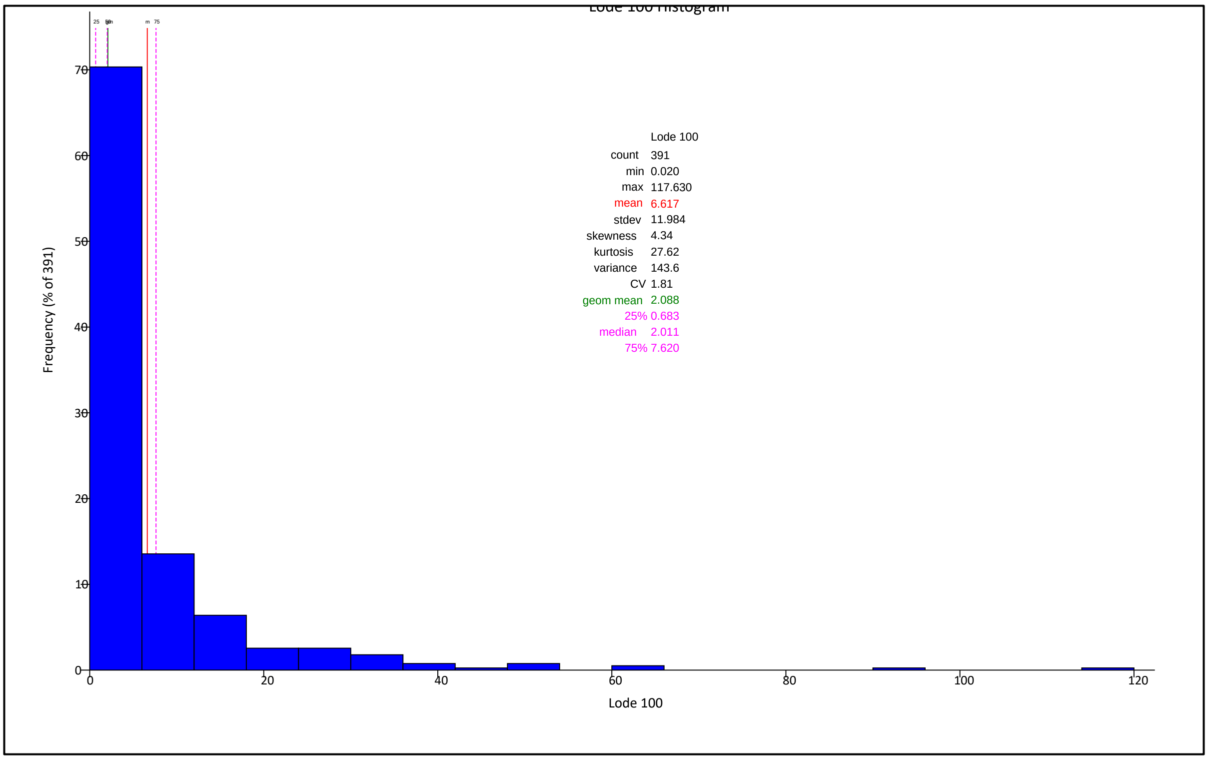


FIGURE 14-5 HISTOGRAM AU LODE 200

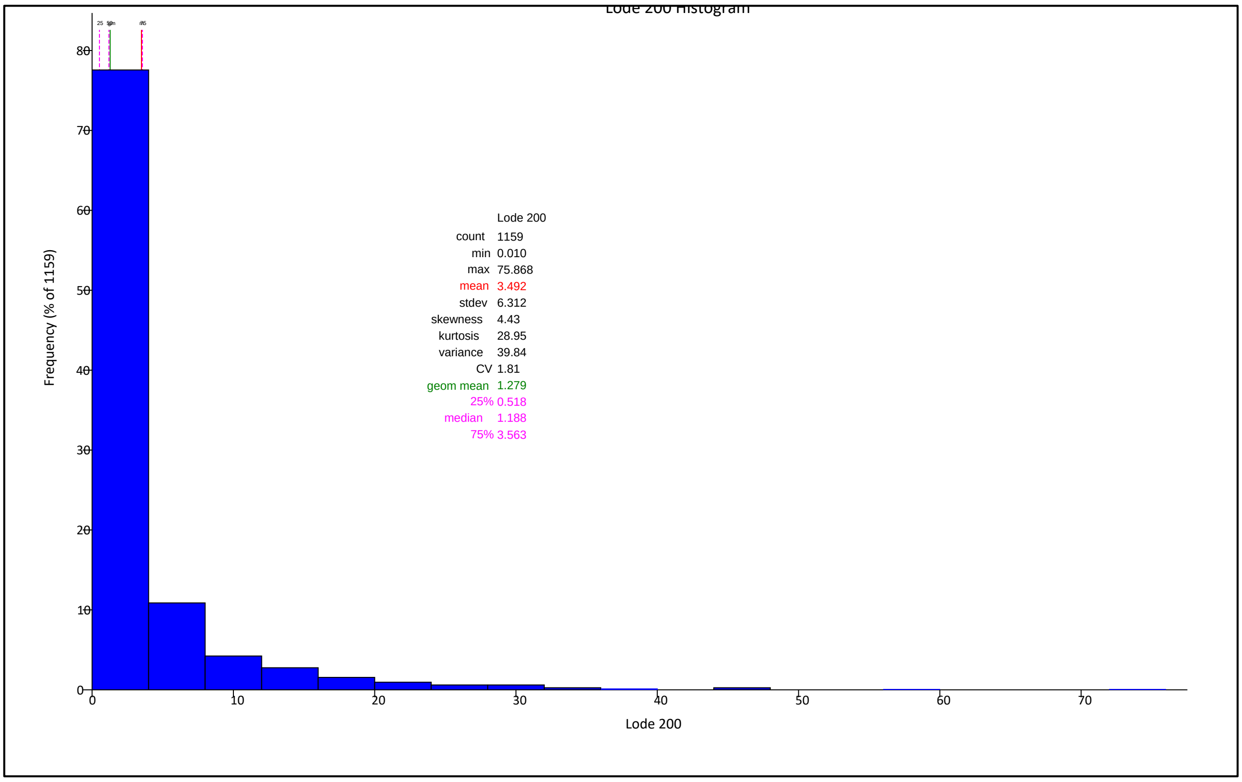
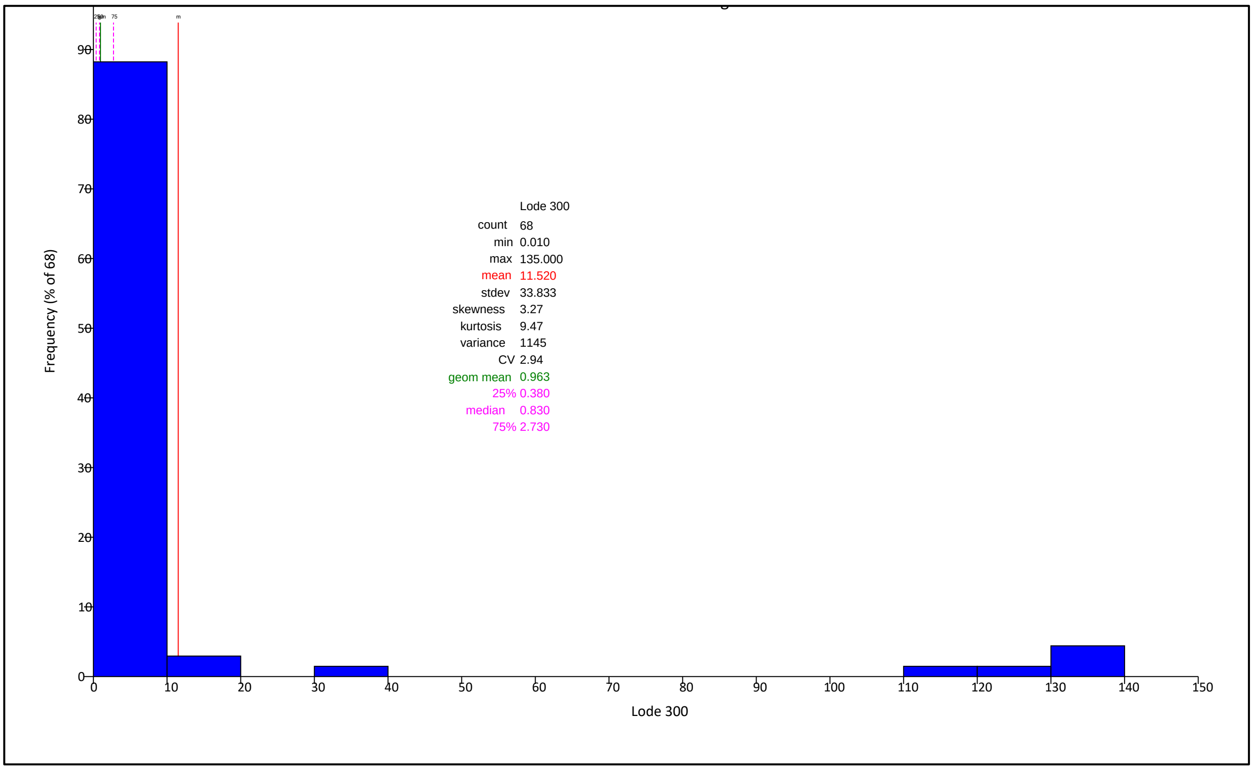


FIGURE 14-6 HISTOGRAM AU LODE 300



CAPPING ANALYSIS

The gold grades for each domain are characterised by moderately high CV values, indicating that high-grade values may contribute significantly to the mean grades. Capping analysis was reviewed on both a domain by domain basis as well as all domains combined. Domain 200 is the dominant domain and contains the greatest number of samples, therefore more weight was given to this domain during capping analysis. A single cap was applied to all domains.

The effects of the highest-grade composites on the mean grade and standard deviation of each domain was investigated by compiling and reviewing histograms and probability plots (Figure 14-7 to 14-10). Composite data was viewed in 3D to observe the spatial distribution of the highest grades observed in each domain to assess the appropriateness of the top-cut grade. Clustering of the highest grades in one or more particular areas may indicate that the grades do not require to be capped and need to be dealt with in a different way.

FIGURE 14-7 LOG PROBABILITY PLOT FOR LODE 100

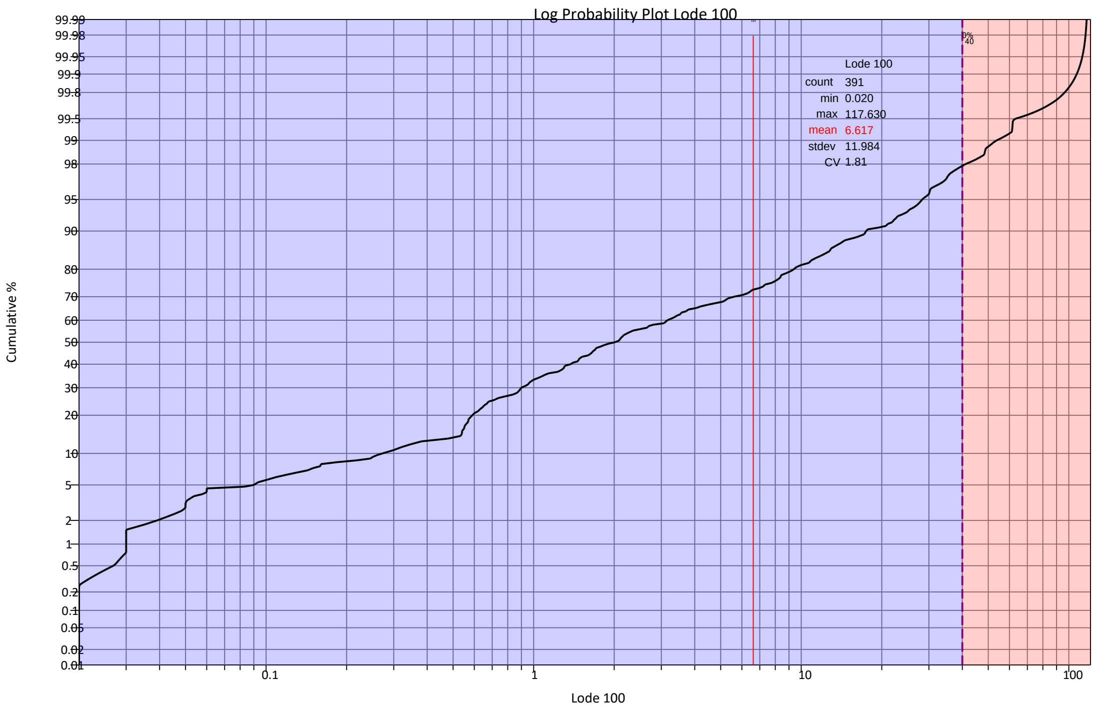


FIGURE 14-8 LOG PROBABILITY PLOT FOR LODE 200

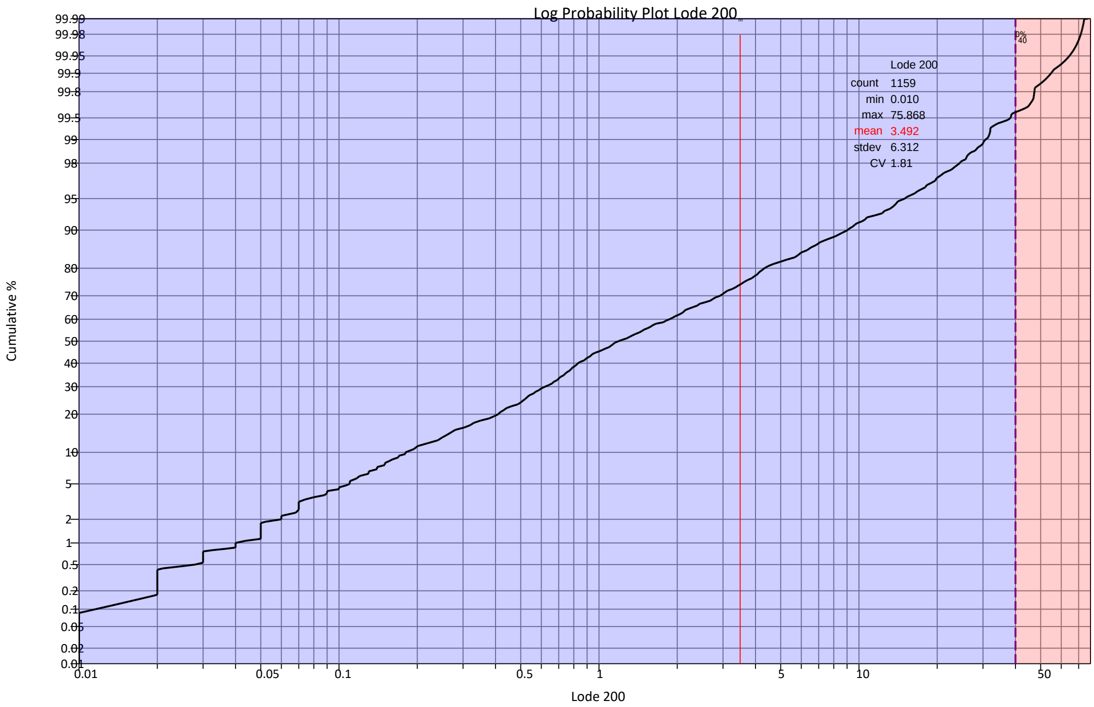


FIGURE 14-9 LOG PROBABILITY PLOT FOR LODE 300

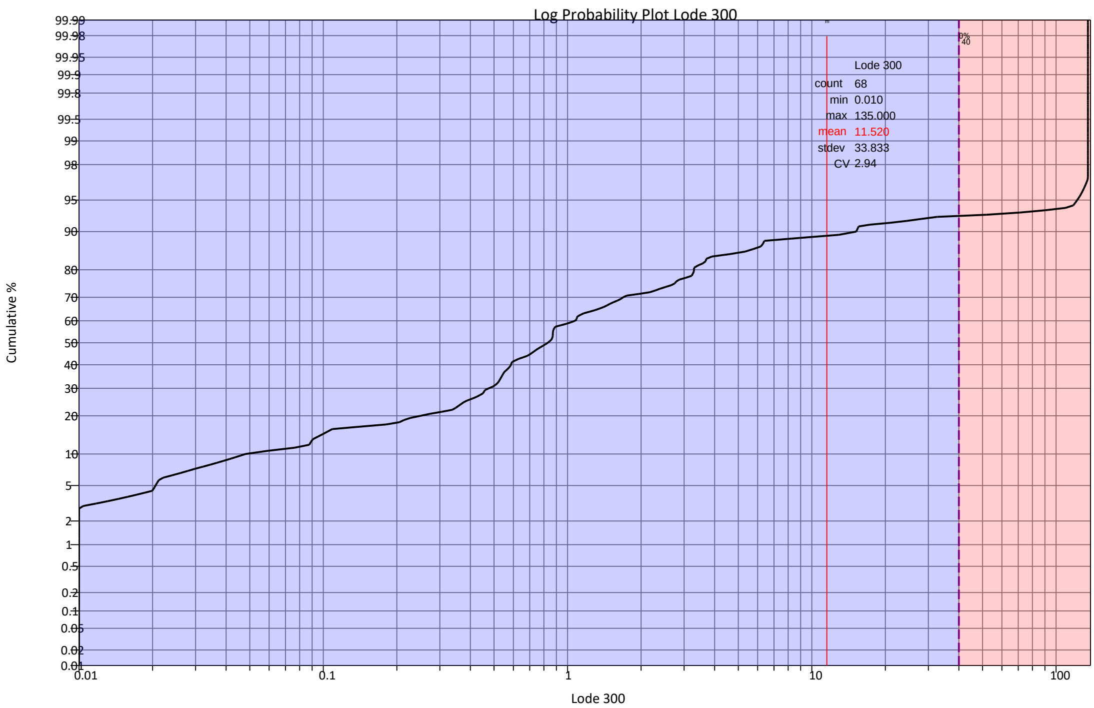
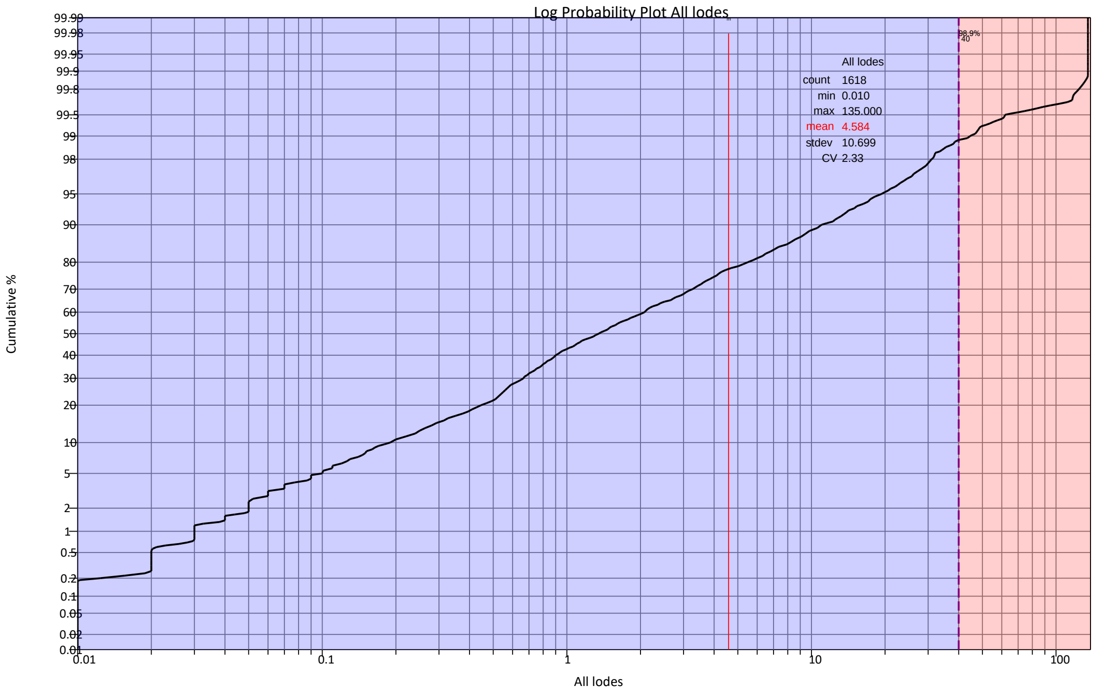


FIGURE 14-10 LOG PROBABILITY PLOT FOR ALL LODES



A range of different capping values were considered (Table 14-5). Ultimately, a capping value of 40 g/t Au was selected for all domains.

TABLE 14-5 COMPOSITE CAPPING STATISTICS
Thor Explorations Ltd – Segilola Gold Project

Lode	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (Au g/t)	Std. Dev	Variance	CV	Number of Samples Cut	% Reduction Mean Grade
100	391	0.02	135	6.62	11.98	143.60	1.81	-	-
			100	6.57	11.59	134.40	1.76	1	0.76
			80	6.49	11.02	121.50	1.70	2	1.96
			60	6.38	10.37	107.60	1.63	4	3.63
			40	**6.09**	**9.09**	**82.72**	**1.49**	**9**	**8.01**
			20	5.11	6.28	39.43	1.23	37	22.81
200	1,159	0.01	135	3.49	6.31	39.84	1.81	-	-
			100	3.49	6.31	39.84	1.81	-	-
			80	3.49	6.31	39.84	1.81	-	-
			60	3.48	6.17	38.08	1.77	1	0.29
			40	**3.43**	**5.82**	**33.91**	**1.70**	**7**	**1.72**
			20	3.17	4.63	21.46	1.46	44	9.17
300	68	0.01	135	11.52	33.83	1145.00	2.94	-	-
			100	9.36	26.13	682.60	2.79	5	18.75
			80	7.89	20.97	439.60	2.66	5	31.51
			60	6.42	15.88	252.00	2.47	5	44.27
			40	**4.95**	**10.94**	**119.70**	**2.21**	**6**	**57.03**

Lode	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (Au g/t)	Std. Dev	Variance	CV	Number of Samples Cut	% Reduction Mean Grade
			20	3.31	5.92	35.06	1.79	7	71.27
All	1,618	0.01	135	4.58	10.69	114.5	2.33	-	-
			100	4.48	9.59	91.98	2.14	6	2.18
			80	4.40	8.84	78.22	2.01	7	3.93
			60	4.30	8.09	65.38	1.88	10	6.11
			40	**4.14**	**7.11**	**50.50**	**1.72**	**22**	**9.61**
			20	3.64	5.20	27.02	1.43	88	20.52

The impact of the chosen capping value on each domain is summarised in Table 14-6.

TABLE 14-6 SUMMARY STATISTICS GROUPED BY DOMAIN OF CAPPED GOLD GRADE (AU G/T)
Thor Explorations Ltd – Segilola Gold Project

	Lode 100		Lode 200		Lode 300	
Item	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped
Count	391	391	1,159	1,159	68	68
Minimum (g/t Au)	0.02	0.02	0.01	0.01	0.01	0.01
Maximum (g/t Au)	117.63	40.00	75.87	40.00	135.00	40.00
Mean (g/t Au)	6.62	6.09	3.55	3.43	11.52	4.95
Std. Dev	11.98	9.09	6.31	5.82	33.83	10.94
CV	1.81	1.49	1.81	1.70	2.94	2.21

SPATIAL ANALYSIS

VARIOGRAPHY

The variography was investigated and modelled using Snowden Supervisor on the composited and cut data. The rotations are tabulated as dip and dip direction of major, semi-major, and minor axes of continuity.

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

Experimental normal score semi-variograms were then generated to investigate the direction of maximum grade continuity in three directions (major, semi-major, and minor). There were insufficient sample numbers to generate satisfactory variograms for the smaller domains 100 and 300 but it was possible to generate suitable variograms for the Lode 200 domain (Figures 14-11 to 14-13). Lode 100 and 300 display identical mineralisation and geometry to Lode 200 and so the variogram model for the Lode 200 domain was applied to all domains (Table 14-7).

FIGURE 14-11 NORMAL SCORE VARIOGRAM AND MODEL MAJOR DIRECTION

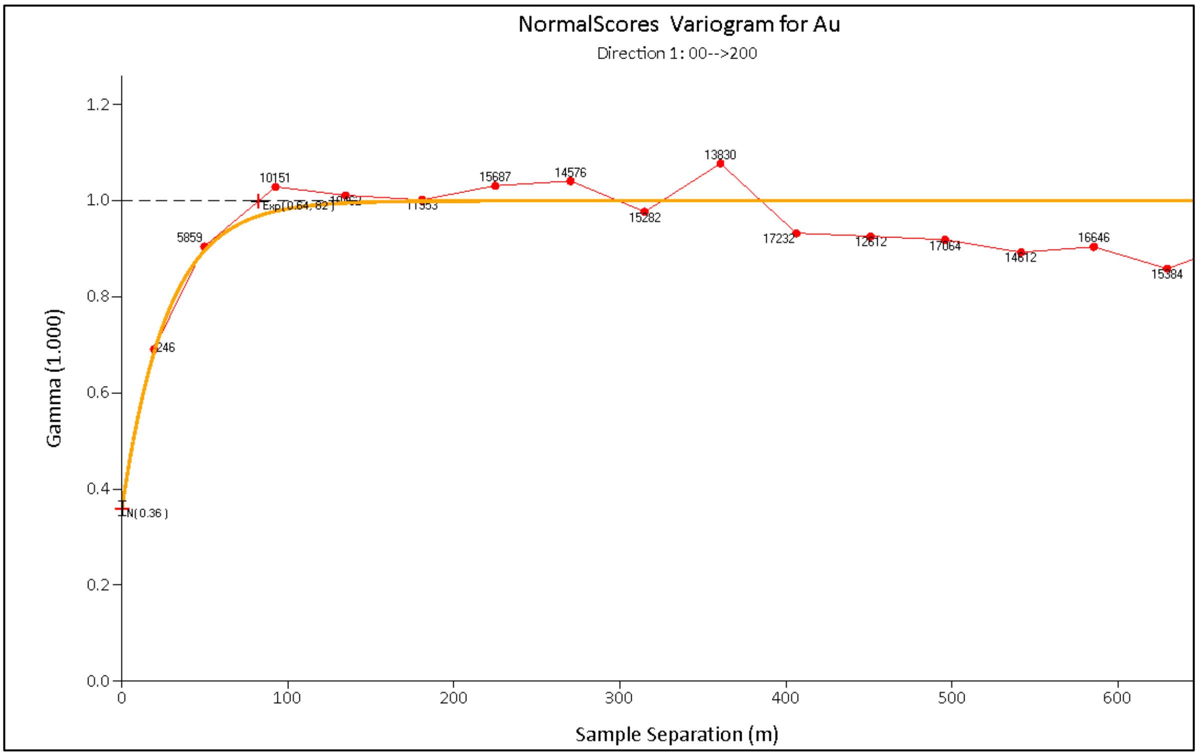


FIGURE 14-12 NORMAL SCORE VARIOGRAM AND MODEL SEMI MAJOR DIRECTION

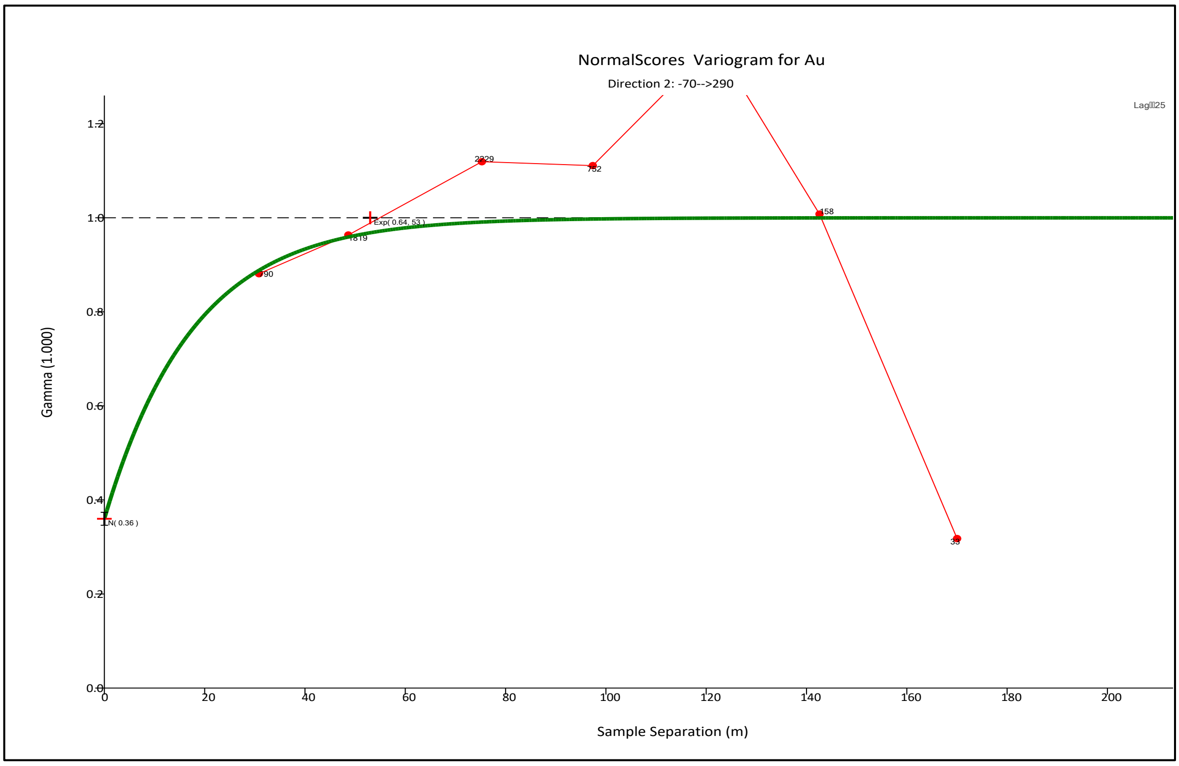


FIGURE 14-13 NORMAL SCORE VARIOGRAM AND MODEL MINOR DIRECTION

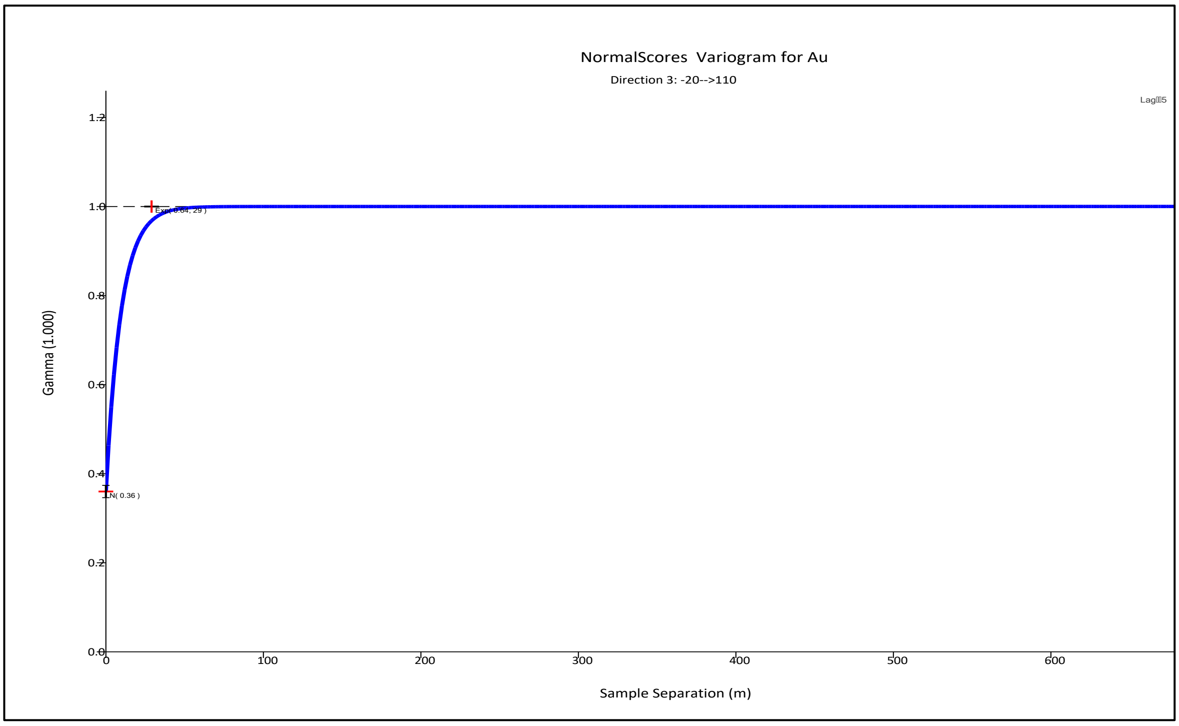


TABLE 14-7 VARIOGRAM PARAMETERS
Thor Explorations Ltd – Segilola Gold Project

Lode	Direction	Dip	Azimuth	Nugget	Sill	Range (m)
	Major-Along Strike	0	200	0.36	0.64	82
200	Semi-Major Across Strike	70	110	0.36	0.64	53
	Minor- Down Dip	-20	110	0.36	0.64	29

QUANTITATIVE KRIGING NEIGHBOURHOOD ANALYSIS

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

A general summary of the main steps is provided:

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

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

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

- Block size.
- Search ellipse dimensions.
- Minimum and maximum sample numbers in a search plan.

A summary of the results is provided in Table 14-8. Sample spacing and lode geometry also had an impact on the choices selected below as well as the style of mineralisation.

TABLE 14-8 QKNA SUMMARY
Thor Explorations Ltd – Segilola Gold Project

Parameter	Results
Block Size	4 m (X) by 20 m (Y) by 8 m (Z)
Minimum and Maximum Samples	2 and 12
Search Ellipse Dimensions	41 m by 29 m by 5 m

BLOCK MODELLING

A 3D block model was created in the National grid (WGS84/UTM 31N) using Surpac software. The parent block size was selected on the basis of the average drill spacing as well as enabling the best representation of the lode geometry and taking into account the values generated by the QKNA. Sub-blocking was used to ensure that the block model volume adequately represents the mineralisation. A parent block size of 4 m by 20 m by 8 m was selected with sub-blocking to 1 m by 5 m by 2 m (Table 14-9).

TABLE 14-9 BLOCK MODEL COORDINATES AND BLOCK SIZE
Thor Explorations Ltd – Segilola Gold Project

Item	X	Y	Z
Minimum Coordinates (m)	701,080	830,000	-300
Maximum Coordinates (m)	702,600	832,740	452
Maximum Block Size (m)	4	20	8
Minimum Block Size (m)	1	5	2
Rotation		None	

The main block model parameters are summarised below in Table 14-10.

TABLE 14-10 BLOCK MODEL PARAMETERS
Thor Explorations Ltd – Segilola Gold Project

Attribute Name	Type	Decimals	Background Value	Description
Au_ok_cut	Float	2	0	Estimated Grade OK Cap
Au_id2_cut	Float	2	0	Estimated Grade Inverse Distance Squared Cap
Avg_ani_dtns	Float	2	0	Average Anisotropic Distance to Samples
Bv_au_ok	Float	2	0	Block Variance
Cbs_au_ok	Float	2	0	Conditional Bias Slope
kani_dtns	Float	2	0	Anisotropic Distance to Nearest Sample
Density	Real	2	0	Combine Ore and Waste Densities
Ke_au_ok	Float	2	0	Kriging Efficiency
Kvar_au_ok	Float	2	0	Kriging Variance
Lgm_au_ok	Float	2	0	Lagrange Multiplier
Modelled_sg	Real	2	0	Modelled Specific Gravity for Ore
Negwts_au_ok	Integer	-	0	Negative Weights
Nums_au_ok	Integer	-	0	Number of Informing Samples
Rock_Code	Character	-	-	Rock Code for Geotechnical Purposes
Sg	Float	2	2.70	Specific Gravity for Waste
Weath	Integer	-	2	Weathering (air =0, Oxide = 1, Transitional =2, Fresh =3)
Wfm_code	Integer	-	0	Coding Blocks with Wireframe Number
OreWaste	Integer	-	Waste	Ore or Waste
Pass	Integer	-		Pass 1 – 4
Resclass	Integer	-		Resource Category (1 – Measured, 2 – Indicated, 3 – Inferred, 4 – Mineral Potential)

A visual review of the wireframe solids and the block model indicated correct flagging of the block model. Additionally, a check was made of coded volume versus wireframe volume which confirmed the above.

SEARCH STRATEGY AND GRADE ESTIMATION

OK was used for grade estimation and was completed using Surpac software and is considered to be an appropriate method for grade estimation in gold deposits such as Segilola when adequate consideration is given to restricting the influence of high-grade data.

The search strategy and parameters were tailored to account for the various geometrical, geological, and geostatistical characteristics of the deposit. The search strategy is summarised in Table 14-11.

TABLE 14-11 SEARCH PARAMETERS
Thor Explorations Ltd – Segilola Gold Project

Domain	All lodes			
Pass	**1**	**2**	**3**	**4**
Y Search Radius (m)	41	61.5	82	492
X Search Radius (m)	29	43	58	173
Max Vertical Distance (m)	5	12	23	252
Minimum Samples	2	2	2	2
Maximum Samples	12	12	12	12
Search Azimuth (°)	020	020	020	020
Search Dip (°)	70	70	70	70
Search Plunge (°)	0	0	0	0

The size and orientation of the search ellipsoid for the estimation process was based on the variogram parameters. A minimum of two samples and maximum of 12 samples were selected for the block grade estimations. No other restrictions, such as a minimum number of informed octants, a minimum number of holes or a maximum number of samples per hole were applied to the estimation process to avoid over smoothing of the model.

A four-pass search strategy was used to estimate the grade of the parent blocks (i.e. the sub-blocks of each parent block all have the same grade). The estimation parameters of the second and third passes are the same except the search ellipsoid was enlarged by 1.5 times and 3 times the dimensions of the first pass, respectively. In this case, priority was given to estimates from the first pass, followed by estimates from the second pass for un-estimated blocks from the first pass, and finally the estimates of the third pass for un-estimated blocks from the first and second passes. Pass four was used to ensure all blocks coded with the attribute 'ore' were estimated. Only blocks within the block attribute 'ore' was estimated.

TABLE 14-12 GRADE ESTIMATE PARAMETERS
Thor Explorations Ltd – Segilola Gold Project

Pass	Blocks Informed (%)	Average Grade (Au g/t)	Average Anisotropic Distance to samples (m)
1	21	4.02	26.80
2	20	3.89	42.81
3	16	3.99	59.09
4	43	3.67	216.34

A hard boundary was used during grade interpolation to ensure that grades were only interpolated using assays from the requisite domain.

BULK DENSITY DATA

A bulk density database has been supplied containing a total of 300 data points and these were flagged by the various lithological and oxidation modelling wireframes.

Bulk densities were estimated into material designated as oxide, transitional, and fresh rock in the block model via inverse distance weighting squared (IDW^2) estimation method, no hard boundaries were used between the different oxidation zones. Waste densities were assigned by weathering due to little no variation in density values.

One estimation pass only was used, and the estimation parameters are tabulated in Table 14-13.

TABLE 14-13 BULK DENSITY SAMPLE SEARCH CRITERIA
Thor Explorations Ltd – Segilola Gold Project

Attribute	Value
Y Search Radius (m)	450
X Search Radius (m)	450
Max Vertical Distance (m)	40
Minimum Samples	2
Maximum Samples	40
Search Azimuth (°)	020
Search Dip (°)	70
Search Plunge (°)	0

A summary of the average mineralised density and assigned waste densities are provided in Table 14-14.

TABLE 14-14 SUMMARY OF DENSITIES
Thor Explorations Ltd – Segilola Gold Project

Weathering	Average Mineralisation Density (t/m^3)	Assigned Waste Density (t/m^3)
Oxide	2.59	1.74
Transitional	2.66	2.47
Fresh	2.72	2.69

BLOCK MODEL VALIDATION

All relevant statistical information was recorded to enable validation and review of the OK estimates. The recorded information included:

- Number of samples used per block estimate.
- Average distance to samples per block estimate.
- Estimation flag to determine in which estimation pass a block was estimated.

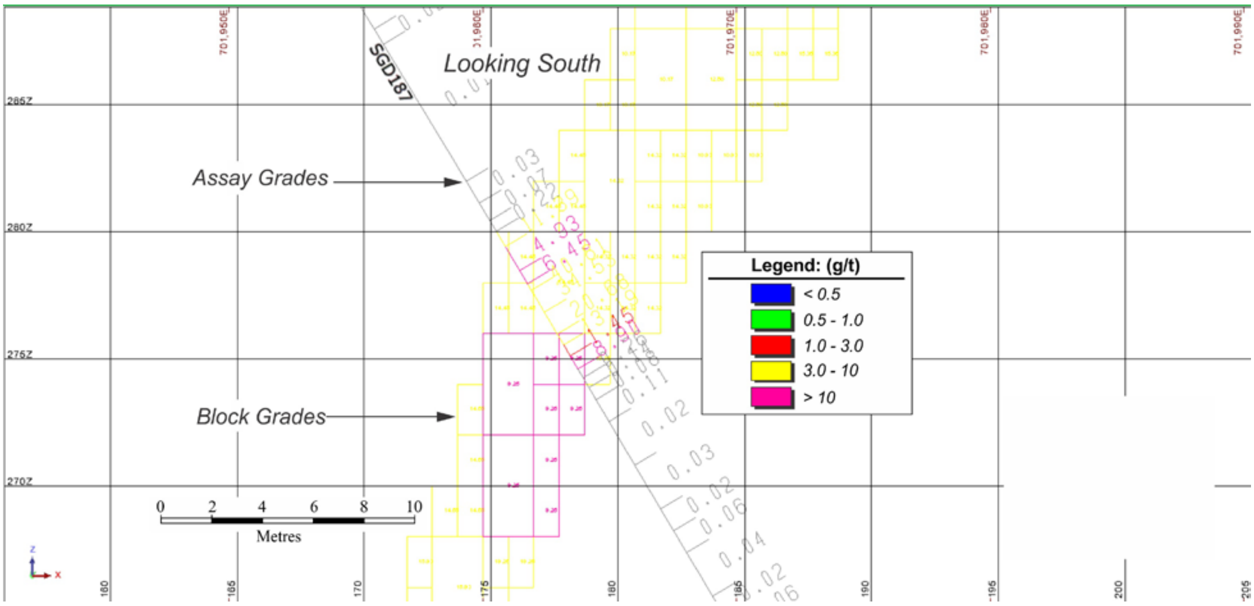
Visual and statistical checks were made:

- Visual checks of cross sections, long sections, and plans.
- Comparison of block and wireframe volume.
- Comparison of the estimate mean versus the composite mean, including weighting where appropriate to account for data clustering.
- Comparison of different estimation methods versus the input composite data.

VISUAL VALIDATION

The block model grades in each domain were visually checked in cross section against the composited data used for estimation. The validation sections showed a strong correlation between the block and the composite grade (Figure 14-14).

FIGURE 14-14 VISUAL VALIDATION



COMPARISON OF WIREFRAME AND BLOCK MODEL VOLUME

No major volume discrepancies were found when the volume of the wireframe was compared to the volume of the block model (Table 14-15).

TABLE 14-15 COMPARISON OF WIREFRAME VS MODEL VOLUME
Thor Explorations Ltd – Segilola Gold Project

Lode	Wireframe Volume (m^3)	Model Volume (m^3)	Percentage Difference (%)
100	991,138	990,710	-0.04
200	2,353,897	2,355,160	0.05
300	101,464	101,400	-0.06
Total	**3,446,499**	**3,447,270**	**0.02**

OTHER ESTIMATION CHECKS

An IDW^2 estimate was used as a validation check of the OK estimate. Comparing the results of the passes against one another that are classified in the Mineral Resource (Table 14-16) Pass 3 produces slightly higher grades compared to the OK estimation. The grade estimation check between IDW^2 and OK shows very similar patterns in all directions and confirms that the multiple search pass strategy is suitable.

TABLE 14-16 COMPARISON OF IDW^2 VS OK ESTIMATION
Thor Explorations Ltd – Segilola Gold Project

Pass	IDW^2 Grade (g/t Au)	OK Grade (g/t Au)	Percentage Difference IDW^2 v OK
Pass 1	4.02	4.02	0.00
Pass 2	3.85	3.89	-1.04
Pass 3	3.94	3.99	-1.27
Average Grade	3.94	3.97	-0.71

SWATH PLOT VALIDATION OF INTERPOLATED GRADES

The swath plots (Figures 14-15 to 14-17) show a reasonable correlation between the composite grades and the OK block model grades. The broad trends demonstrated by the raw data are honoured by the block model, and the interpolated grades are generally lower than the composite values, where the raw cut grade exceeds 10 g/t Au. The comparison illustrates the effect of the interpolation, which results in smoothing of the block grades compared to the raw grades. Overall, the comparison between the OK and IDW^2 swath plots show a reasonably close correlation.

FIGURE 14-15 ELEVATION SWATH PLOT COMPOSITE CUT GRADE (ORANGE) VERSUS IDW^2 BLOCK MODEL GRADE (BLACK) VERSUS OK BLOCK MODEL GRADE (BLUE)

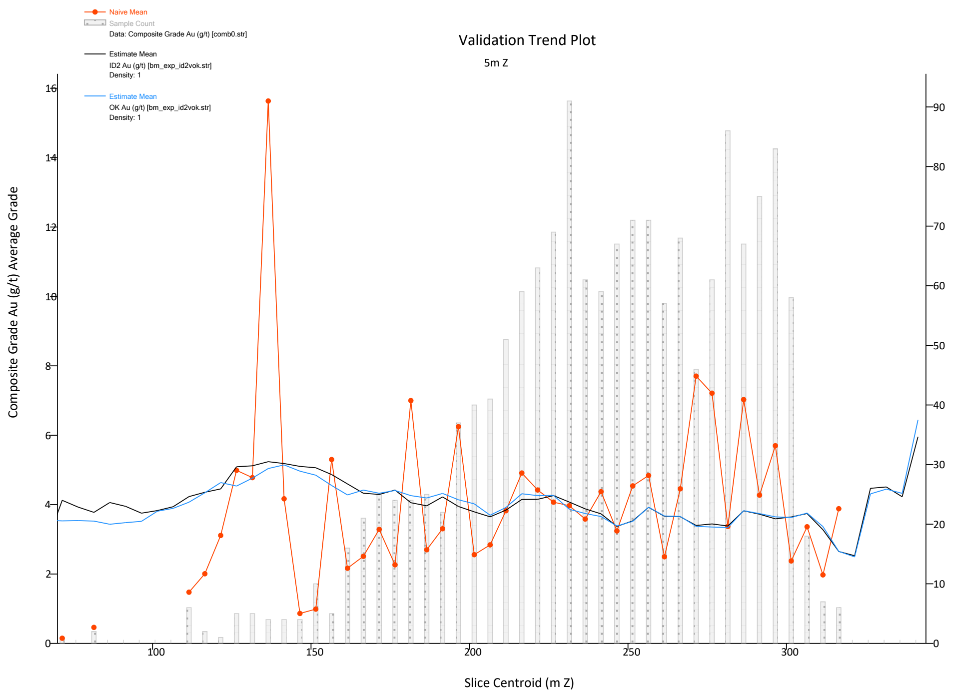


FIGURE 14-16 ALONG-STRIKE SWATH PLOT COMPOSITE CUT GRADE (ORANGE) VERSUS IDW^2 BLOCK MODEL GRADE (BLACK) VERSUS OK BLOCK MODEL GRADE (BLUE)

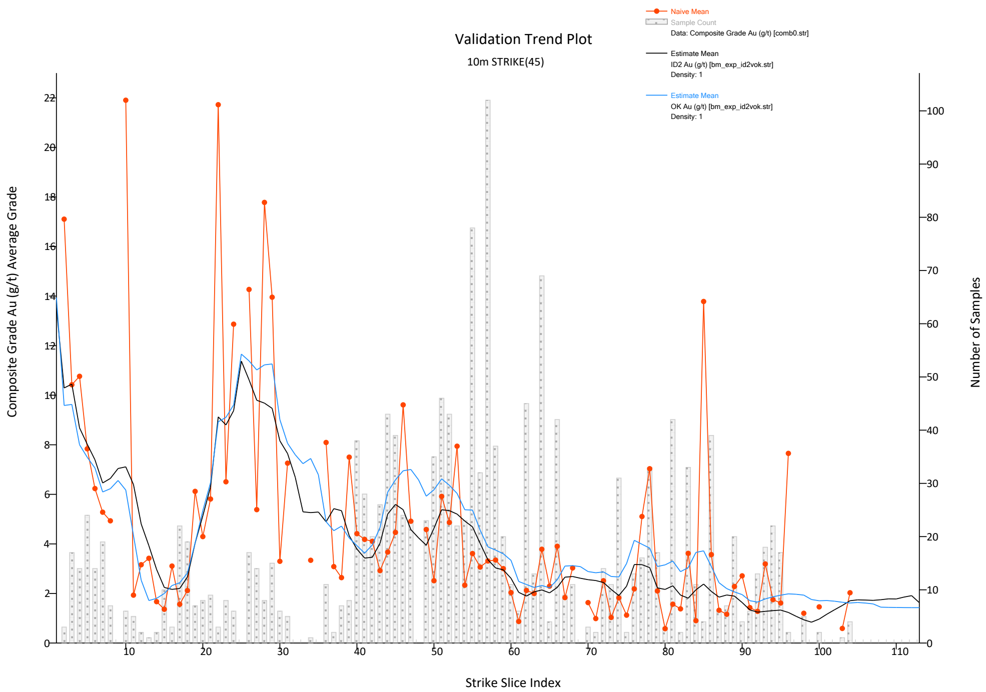
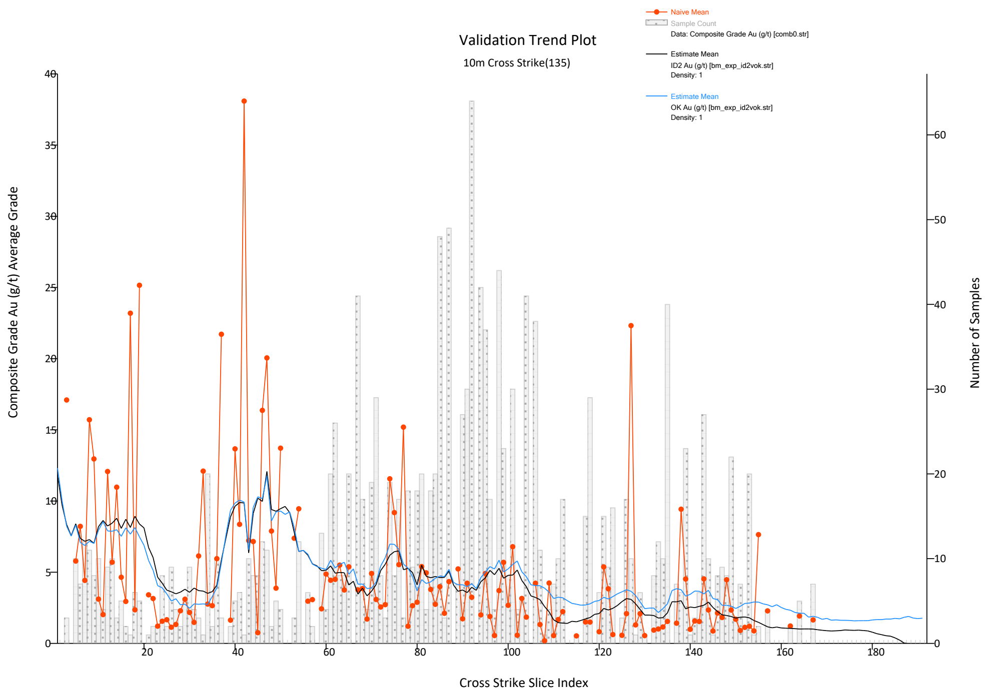


FIGURE 14-17 CROSS-STRIKE SWATH PLOT COMPOSITE CUT GRADE (ORANGE) VERSUS IDW^2 BLOCK MODEL GRADE (BLACK) VERSUS OK BLOCK MODEL GRADE (BLUE)



GRADE TONNAGE CURVE

Grade tonnage curves are a visual representation of the impact of cut-off grades on Mineral Resources and Mineral Reserves. The grade tonnage curve displays the tonnage above the cut-off grade and average grade of a deposit relative to cut-off grade. As the criteria for ore classification becomes more selective, the tonnage above the cut-off grade of the deposit decreases. Conversely, as the cut-off grade is lowered, the tonnage of the deposit increases because the standard used to distinguish between ore and waste has become less selective. As the cut-off grade increases, so does the average grade of the ore mined. The curves ultimately show how the average grade and tonnage of a material delivered to a certain process are dependent on the cut-off grade selected.

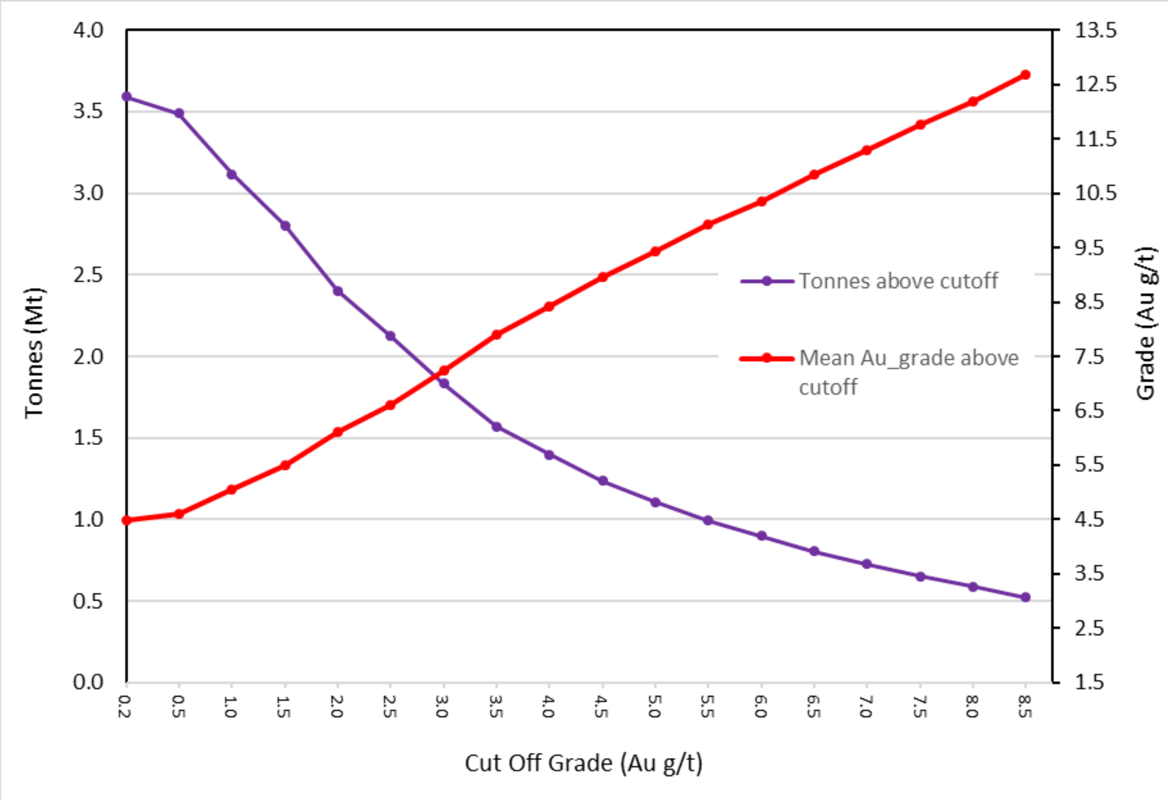
The Segilola Mineral Resource grade tonnage curve (Table 14-17 and Figure 14-18) indicates that the block model is performing well, with flattening of the curves below 0.5 g/t Au, which reflects the wireframing cut-off used.

TABLE 14-17 GRADE AND TONNAGE AT VARIOUS CUT OFF GRADES
Thor Explorations Ltd – Segilola Gold Project

Cut-off Grade Au g/t	Tonnes (Mt)	Average Grade (Au g/t)
0.00	9.15	3.98
0.25	9.10	4.00
0.50	8.94	4.06
0.75	8.55	4.22
1.00	8.10	4.41
1.25	7.68	4.58
1.50	6.99	4.90
1.75	6.29	5.27
2.00	5.54	5.73
2.25	5.14	6.01
2.50	4.83	6.25
2.75	4.51	6.50
3.00	4.25	6.73
3.25	4.05	6.90
3.50	3.69	7.25
3.75	3.42	7.53
4.00	3.05	7.97
4.25	2.83	8.27
4.50	2.67	8.51
4.75	2.49	8.80
5.00	2.38	8.98
5.25	2.23	9.23
5.50	2.07	9.52
5.75	1.90	9.88
6.00	1.81	10.09
6.25	1.72	10.30
6.50	1.61	10.55
6.75	1.54	10.73
7.00	1.49	10.86
7.25	1.45	10.98
7.50	1.39	11.13
7.75	1.34	11.25
8.00	1.30	11.35
8.25	1.24	11.51
8.50	1.15	11.76
8.75	1.07	11.99
9.00	1.00	12.20
9.25	0.81	12.93
9.50	0.71	13.44
9.75	0.68	13.62
10.00	0.63	13.88
10.25	0.59	14.12
10.50	0.57	14.30
10.75	0.52	14.62
11.00	0.47	14.99
11.25	0.45	15.23
11.50	0.43	15.37

Cut-off Grade Au g/t	Tonnes (Mt)	Average Grade (Au g/t)
11.75	0.41	15.52
12.00	0.37	15.98
12.25	0.35	16.15
12.50	0.34	16.33
12.75	0.33	16.45
13.00	0.31	16.60
13.25	0.30	16.79
13.50	0.27	17.19
13.75	0.26	17.30
14.00	0.24	17.60
14.25	0.22	17.94
14.50	0.17	18.86
14.75	0.16	19.26
15.00	0.15	19.53
15.25	0.14	19.88
15.50	0.13	20.16
15.75	0.12	20.41
16.00	0.11	20.95
16.25	0.11	21.12
16.50	0.10	21.42
16.75	0.10	21.62
17.00	0.09	21.77
17.25	0.09	22.00
17.50	0.08	22.23
17.75	0.08	22.48
18.00	0.08	22.72
18.25	0.07	23.10
18.50	0.07	23.23
18.75	0.06	23.49
19.00	0.06	23.92
19.25	0.06	24.26
19.50	0.05	24.41
19.75	0.05	24.50
20.00	0.05	24.88

FIGURE 14-18 GRADE TONNAGE CURVE



DEPLETION FOR MINING ACTIVITY

A small amount of historical mining has Small depletion of the deposit has occurred. The wireframe and block model are clipped to the digital terrain model (DTM). The DTM includes the historic pits.

RESOURCE CLASSIFICATION

The Segilola Gold Mineral Resource has been classified and reported in accordance with the CIM (2014) definitions as incorporated in NI 43-101. Resource classification is based on confidence in the geological domaining, drill spacing, and geostatistical measures. The initial classification process was based on an interpolation distance and minimum samples within the search ellipse.

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

- Geological continuity.
- Geology sections plan and structural data.
- Previous resource estimates and assumptions used in the modelling and estimation process.

- Interpolation criteria and estimate reliability based on sample density, search and interpolation parameters, not limited to kriging efficiency, kriging variance, and conditional bias.

- Drill hole spacing.

Applying these confidence levels, resource classification codes were assigned to the block model using the following criteria as shown in Figures 14-19 to 14-21:

- Indicated Resource:
    - Blocks are predominately estimation pass 1 or 2.
    - Average distance to nearest data of 30 m or less.
    - Minimum of 12 samples.

- Inferred Resource:
    - Blocks are predominately estimation pass 3.
    - Average distance to nearest data of 90 m or less.
    - Minimum of 4 samples.

Some material that was outside of these criteria were included to ensure continuity.

Once the criteria above were applied, shapes were generated around contiguous lodes of classified material which was used to flag the block model to ensure continuous zones of classification. The resource estimate for the Segilola deposit has been classified as Indicated and Inferred Mineral Resources based on the confidence levels of the key criteria as presented in Table 4-18.

TABLE 14-18 CONFIDENCE LEVELS BY KEY CRITERIA
Thor Explorations Ltd – Segilola Gold Project

Items	Discussion	Confidence
Drilling Techniques	Diamond - Industry Standard approach.	Moderate/High
Logging	Standard nomenclature has been adopted but not used in entire database.	Moderate
Drill Sample Recovery	Recoveries are not recorded in entire database. Review of current drilling suggests diamond core recoveries are of acceptable standard.	Moderate/High
Sub-sampling Techniques and Sample Preparation	Diamond core sampling conducted by industry standard techniques.	Moderate/High
Quality of Assay Data	Appropriate quality control procedures are available. They were reviewed on site and considered to be of industry standard.	Moderate/High
Verification of Sampling and Assaying	Sampling and assaying procedures have been assessed and are considered of appropriate industry standards.	Moderate

Items	Discussion	Confidence
Location of Sampling Points	Survey of all collars conducted with accurate survey equipment. Investigation of downhole survey indicates appropriate behaviours.	Moderate/High
Data Density and Distribution	Majority of regions defined on a notional 25 to 30 m E by 25 m N drill spacing.	Moderate
Audits or Reviews	Data collection assessed during site review.	Moderate/High
Database Integrity	Assay certificates have been verified and no issues were identified.	Moderate/High
Geological Interpretation	Mineralization controls are well understood. The mineralization constraints are robust but relatively broad and therefore of moderate confidence.	Moderate
Estimation and Modelling Techniques	Ordinary Kriging is considered to be appropriate given the geological setting and grade distribution.	High
Cut off Grades	OK is independent of cut-off grade although the mineralization constraints were based on a notional 0.5 g/t Au lower cut-off grade. A 0.64g/t lower cut-off grade is considered appropriate for reporting.	Moderate/High
Mining Factors or Assumptions	Not Applied	N/A
Metallurgical Factors or Assumptions	Not Applied	N/A
Tonnage Factors (In-situ Bulk Densities)	Localised data collected diamond core in waste rock and ore rock in oxide, transitional and fresh material.	Moderate/High

FIGURE 14-19 RESOURCE CLASSIFICATION LODE 100

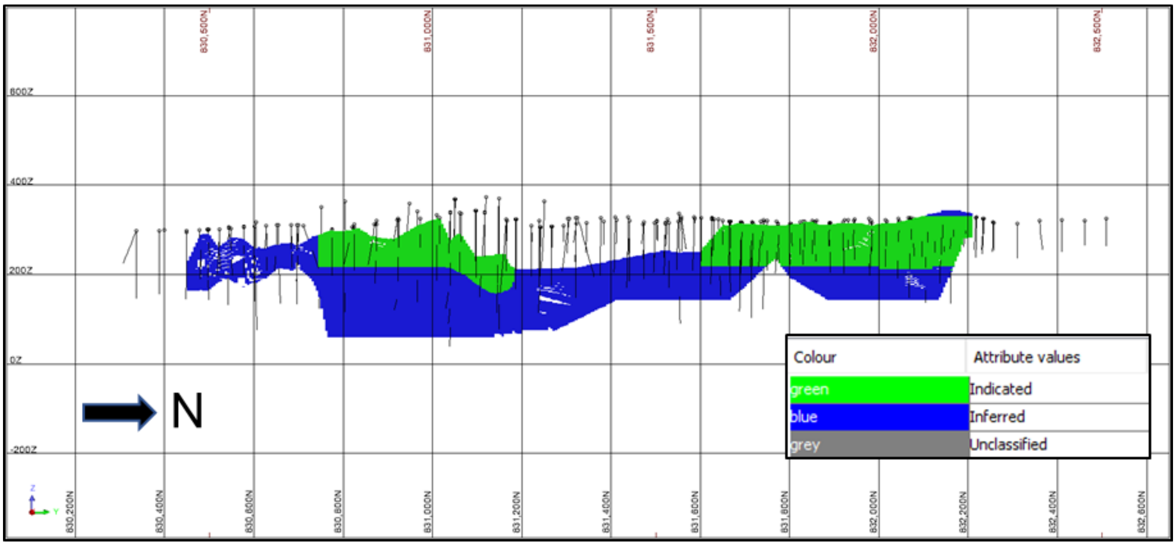


FIGURE 14-20 RESOURCE CLASSIFICATION LODE 200

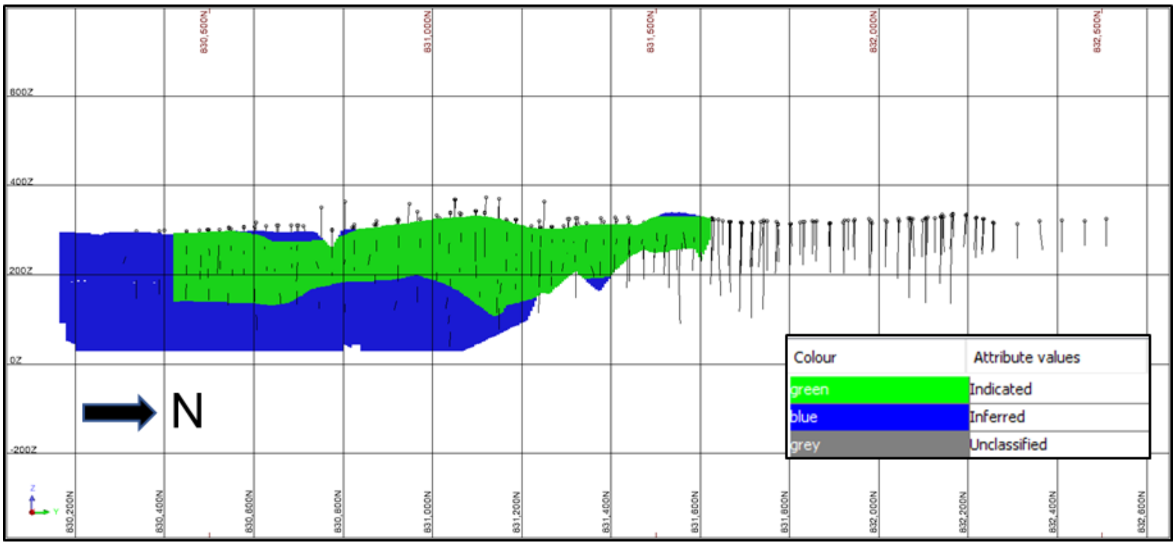
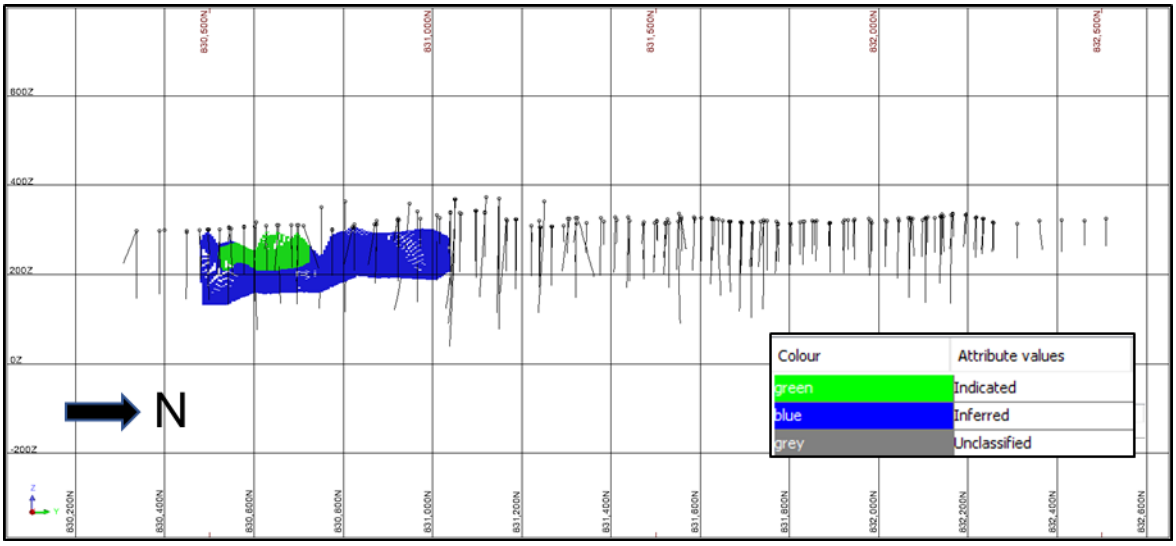


FIGURE 14-21 RESOURCE CLASSIFICATION LODE 300



OPEN PIT MINERAL RESOURCE REPORTING

Based on the 'reasonable prospects of eventual economic extraction' test as required by the CIM, the Mineral Resources are reported inside an optimised pit shell (Table 14-19).

TABLE 14-19 OPTIMISATION PARAMETERS APPLIED FOR MINERAL RESOURCE REPORTING
Thor Explorations Ltd – Segilola Gold Project

Whittle Input Parameter	Value	Unit
Overall Pit Slope	50W/42E	degrees
Surface Mining Cost (Waste)	$2.67	$/t
Mining Dilution	10%	%

Whittle Input Parameter	Value	Unit
Mining Recovery	95%	%
Processing Cost	$19.40	$/t ore
Processing Recovery (Au)	97%	%
G&A Cost	$5.77	$/t ore
Grade Control	$0.34	$/t ore
Re-handle	$0.65	$/t ore
Refining	$0.88	$/t ore
Discount Rate	8%	%
Metal Price Gold	$1,500.00	$/oz
Selling Cost/Royalties*	$14.88	$/oz

The reporting of the open pit Mineral Resource takes into account the underground Mineral Resource prepared by RPA. This involves using a dividing plane between the underground and open pit resource areas, along with the optimised Whittle resource pit shell resulting from the optimisation parameters shown in Table 14-19, and the Mineral Reserve pit design. These surfaces are shown in Figure 14-22.

In summary, the open pit Mineral Resource is reported as follows:

- Inside the pit optimisation shell north of the dividing plane.
- Above the reserve pit design south of the dividing plane.
- At a 0.64 g/t Au cut-off grade.

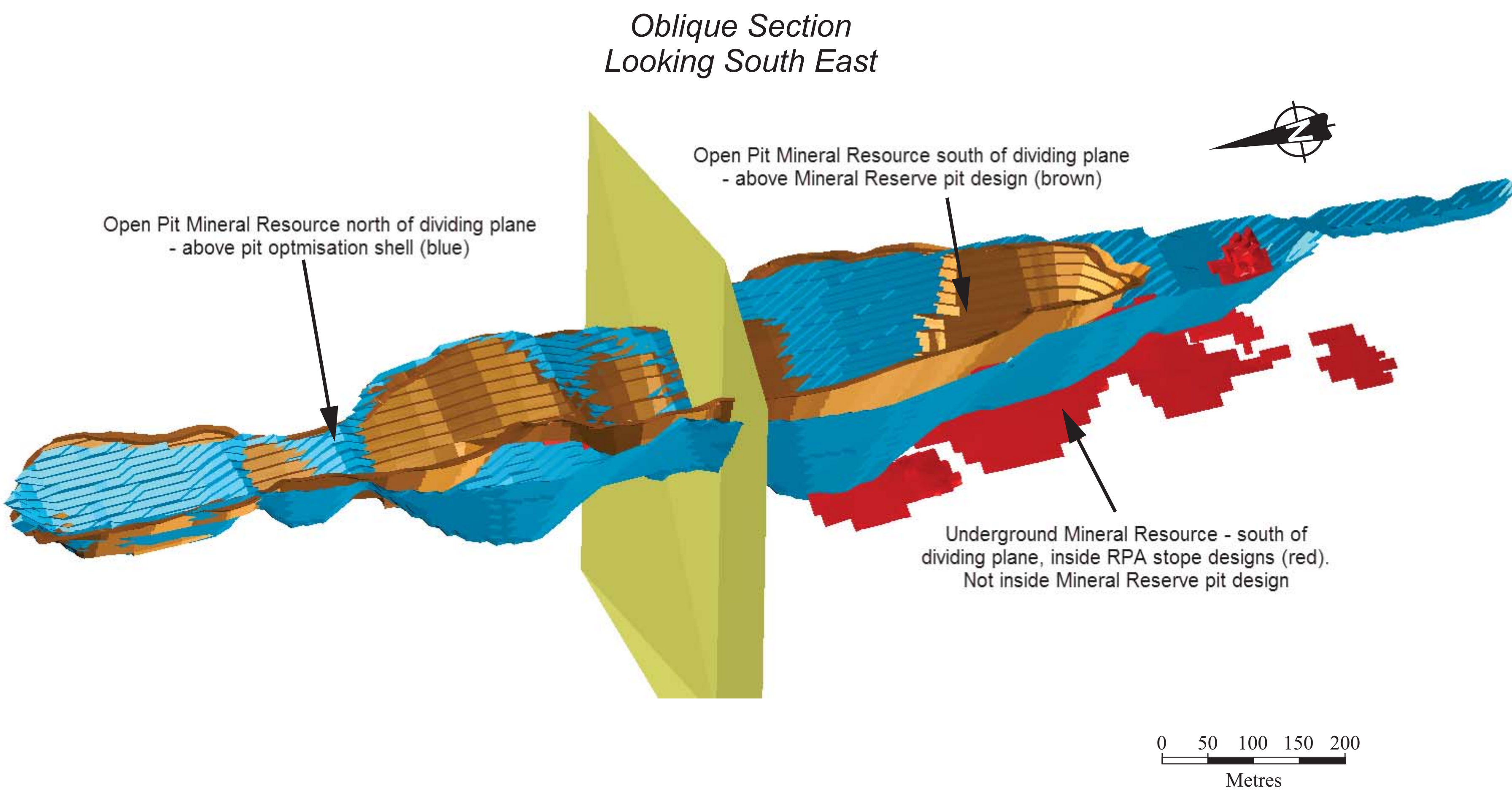


Figure 14-22

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Mineral Resource Reporting Diagram

March 2019

Source: Auralia, 2019.

The open pit Mineral Resource is inclusive of all Indicated and Inferred material within the constraints above and is summarised in Table 14-20.

TABLE 14-20 OPEN PIT MINERAL RESOURCE STATEMENT EFFECTIVE 1 DECEMBER 2018
Thor Explorations Ltd – Segilola Gold Project

Classification	Open Pit Domain	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Indicated	100	705	6.05	137
	200	2,327	4.06	304
	300	0	1.17	0
	Total	**3,032**	**4.52**	**441**
Inferred	100	251	6.6	53
	200	33	6.1	7
	300	47	8.6	13
	Total	**331**	**6.8**	**73**

Notes:
1. Open Pit Mineral Resources are reported within an optimised pit shell, as defined by the parameters shown in Table 14-19 at a cut-off grade of 0.64 g/t Au.
2. Estimation constrained by wireframes defined using a nominal 0.5 g/t Au lower cut-off.
3. The Mineral Resource estimate has been prepared by Mr Christopher Speedy of Auralia Mining Consulting Pty Ltd, who is a qualified person under NI 43-101.
4. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.
5. Thor has adopted industry-standard procedures for sampling, data verification, compiling, interpreting and processing the data used to estimate Mineral Reserves and Mineral Resources.
6. Mineral Resources are inclusive of Mineral Reserves.
7. Numbers may not sum due to rounding.

COMPARISON WITH PREVIOUS RESOURCE ESTIMATES

The previous resource estimates are presented in Table 14-21. Differences exist in the classification of the resources between the 2017 and 2018 models, and these are discussed further below.

TABLE 14-21 PREVIOUS OPEN PIT RESOURCE ESTIMATES
Thor Explorations Ltd – Segilola Gold Project

Year	Company	Indicated Tonnes (Mt)	Indicated Grade (g/t/ Au)	Indicated Contained Gold (oz)	Inferred Tonnes (Mt)	Inferred Grade (g/t/ Au)	Inferred Contained Gold (oz)
2016	Odessa Resources Pty Ltd [1]	4.58	3.80	555,000	-	-	-
2017	Auralia [2]	3.93	4.30	539,000	0.84	5.10	137,000

Notes:
1. In 2016, Odessa Resources Pty Ltd produced an updated Mineral Resource estimate for Thor based on existing data in order to support the execution of a share purchase agreement.

2. In 2017, Auralia Consulting Pty Ltd updated the Mineral Resource estimate for the Segilola Project to support the PFS. The Mineral Resources are reported within a Whittle shell amenable to open pit extraction. Resources are reported at a 0.64 g/t Au cut-off grade. CIM (2014) definitions were followed for Mineral Resources.

The change from the 2017 estimate is explained by:

- Changes to the reporting Whittle shell for open pit Mineral Resources, and the reporting of Mineral Resources only within the reserve pit shell south of the demarcation line (shown in Figure 14-22).

- Drilling down dip has resulted in an increase in grade in the Inferred category.

OPEN PIT MINERAL RESOURCE COMMENTS

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

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

- Use optimised pit shells as a guide to creating drilling programmes that maximise the upgrading from lower to higher resource classification and reduce mining risk attributed to data density and quality.

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

- Set the mineralisation/wireframe width to a minimum of 2 m or the thickness deemed to be suitable for the minimum mining width or apply a minimum grade multiplied by thickness cut-off.

- Although limited faulting is observed to intersect the geological model, the faults should be modelled in 3D and incorporated into the geological model.

UNDERGROUND MINERAL RESOURCE ESTIMATE

An underground Mineral Resource estimate was generated by RPA, which comprises the higher-grade mineralisation located below the Stage 3 Reserve Open pit.

RPA estimated the underground Mineral Resources using all drill hole data available up until 1 December 2018.

Thor supplied a database that had been cleaned and validated by Aurelia for the open pit Mineral Resources. RPA undertook independent validation procedures on the received database and found no material issues. RPA checked 6% of the assays included in the dataset against the assay certificates supplied by the laboratory and identified no significant errors. Twenty-four drill holes were identified as missing downhole surveys, however, only 12 of these intersected the underground mineralisation wireframes and only one was in an area for which underground Mineral Resources were declared. The average deviation of all surveyed drill holes was investigated at the depths of the mineralised intercepts, and no significant impact on the geo-spatial location of the un-surveyed drill holes was identified.

Underground geological wireframes were prepared by Thor geologists at an approximately 2.5 g/t Au cut-off grade. RPA reviewed the data populations, and confirmed that a second grade population starting at 2.5 g/t Au is present which warranted the wireframing cut-off used. RPA reviewed the mineralised intercepts from these wireframes and regenerated underground wireframes using Seequent Leapfrog software. The wireframes were generated using a two-metre minimum mining width.

Assay samples were flagged with the mineralisation wireframes. The mineralised samples within the underground domains were capped at 50 g/t Au, and subsequently composited on one metre lengths. Geostatistical analyses were undertaken on the capped composites to determine the search and kriging parameters. Dynamic anisotropy was employed during interpolation to reflect local variations in the orientation of the mineralisation. The block model was interpolated using ordinary kriging (OK). A density of 2.70 t/m^3 was applied to all domains, based upon the average density of the underground mineralised material.

CIM (2014) definitions were used for Mineral Resource classification. RPA considered a combination of drill hole spacing, distance to nearest sample, mineralisation continuity, minimum number of samples, and the search pass number to inform the Mineral Resource classification. Underground Mineral Resources are reported outside the Stage 3 Design

Reserve open pit shape, south of a demarcation line that limits the open pit Mineral Resources, and inside stope shapes generated at $1,500/oz. Table 14-22 summarises the underground Mineral Resources for the Project.

TABLE 14-22 SUMMARY OF UNDERGROUND MINERAL RESOURCES – 1 DECEMBER 2018
Thor Explorations Ltd – Segilola Gold Project

Category	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Measured	-	-	-
Indicated	93	9.39	28
Total Measured + Indicated	**93**	**9.39**	**28**
Inferred	352	7.9	90

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at a cut-off grade of 2.58 g/t Au and constrained within stope shapes using an average Au price of $1,500/oz.
3. A minimum mining width of 2 m was used along with a lower 2.5 g/t Au wireframing cut-off.
4. Underground bulk density is 2.70 t/m^3.
5. High gold assays were capped to 50 g/t Au.
6. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
7. Numbers may not add due to rounding.

RPA is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

RESOURCE DATABASE

Thor stores the drill hole data in a series of Microsoft Excel workbooks, which are manually updated.

RPA used the validated database that Auralia created to generate the open pit Mineral Resource. Table 14-23 details the Mineral Resource database RPA received for use in the underground model. Metallic screen fire assays were not used in the resource estimation due to the small number available.

TABLE 14-23 UNDERGROUND RESOURCE DATABASE
Thor Explorations Ltd – Segilola Gold Project

Parameter		Value
Drill Hole	Count	194
	Total Length (m)	22,237
Survey Records Count		899
Assays	Count	10,657
	Total Length (m)	10,564
Lithology Records Count		1,947
Number of Density Samples		567

Auralia undertook the following validation steps on the database:

- Checked for missing collar co-ordinates, missing hole depths, missing downhole surveys; miss matched collar, survey or assay depths; and overlapping intervals.
- Checked for missing or overlapping intervals for geology and assay interval data.
- Checked for negative and null assays.

RPA undertook independent validation steps to ensure that the database used was clean and free of errors. RPA identified 33 errors in the drill hole database (Table 14-24). These validation issues were not located within the mineralised domains and did not impact the Mineral Resource estimate.

TABLE 14-24 RESOURCE DATABASE VALIDATION RESULTS
Thor Explorations Ltd – Segilola Gold Project

Dataset	Overlapping Intervals	Intervals beyond hole depth	From > To	From = To
Assay	1	-	0	4
Lithology	17	5	2	1
Density	0	3	-	-

RPA also undertook checks on the assay data to confirm that there were no transcription errors between the assay certificates supplied by the laboratory and the database. In total, RPA checked 632 of the 10,657 assays (6%) in the received dataset, primarily focussing on drill holes that intercepted the underground mineralisation. In total, RPA identified two samples that were different in the assay database to the received certificates, both of which were not located inside the mineralisation wireframes.

RPA identified that 24 drill holes completed prior to Thor's involvement on the Project (SGD197) did not contain downhole surveys. Twelve of these drill holes were used to construct the underground wireframes and only one drill hole was located in an area where underground Mineral Resources were declared. These un-surveyed drill holes were stored in the database with an identical dip and azimuth at the bottom of the drill hole to that at the collar. RPA investigated the average downhole deviation using the drill holes that contained multiple downhole survey measurements. The approximate average deviation downhole (in 3D) is 0.016°/m. RPA reviewed the impact of this deviation on the position of mineralised intercepts in the underground model and found the impact to the un-surveyed drill holes to be negligible at the depths encountered.

RPA recommends that Thor purchase and migrate all Project data into a centrally organised and auditable geological database with stringent cross-validation procedures. Data which fails cross-validation should be held in a buffer table until the issues are resolved. As part of the implementation of the database, the assays should be re-imported from the raw assay certificates.

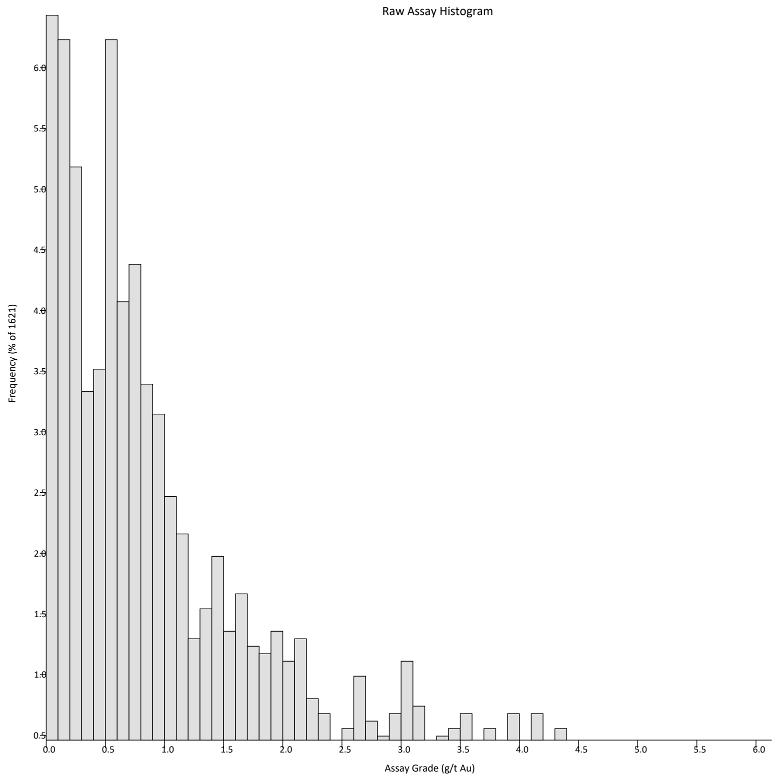
RPA is of the opinion that the drill hole database is valid and suitable to estimate Mineral Resources for the Project.

GEOLOGICAL INTERPRETATION

RPA received draft underground wireframes encompassing high-grade material at a 2.5 g/t Au cut-off grade from Thor geologists. RPA reviewed the data populations, and confirmed that a second grade population starting at 2.5 g/t Au is present (Figure 14-23) which warranted the wireframing cut-off used.

Thor generated these wireframes on drill sections which had been adjusted to the localised drill spacing, which is approximately 20 m to 30 m. RPA reviewed the mineralised intercepts, adjusted where necessary, and generated new wireframes from these intercepts using Seequent Leapfrog geological modelling software. Where required, the minimum grade for wireframing was lowered to ensure geological continuity of the wireframes. The wireframes were constructed with a two-metre minimum mining width, and snapping was turned on. Wireframes were extrapolated approximately half of the average drill spacing past the last mineralised intercept.

FIGURE 14-23 HISTOGRAM OF UNDERGROUND DOMAIN ASSAYS



RPA generated six underground wireframes (Figure 14-24):

- Domain 100: 220 m along strike, 245 m elevation delta, average 4 m width.
- Domain 200: 1,300 m along strike, 240 m elevation delta, average 4 m width.
- Domain 200A: 80 m along strike, 50 m elevation delta, average 3.6 m width.
- Domain 201: 100 m along strike, 60 m elevation delta, 2.8 m averaged width.
- Domain 400: 390 m along strike, 145 m elevation delta, 2 m average width.

In total, the underground mineralisation wireframes strike for approximately 1,300 m at an azimuth of 15° and dip to the southwest at approximately 60°, although there is variation locally within the domains. The underground wireframes extended into the open pit Mineral Resource so that the data support was available, however, additional filters were applied during block model reporting to ensure that the block models did not overlap.

RPA generated mid-point co-ordinates for all mineralised intercepts within the assay dataset. These co-ordinates were then flagged with the underground geological domains.

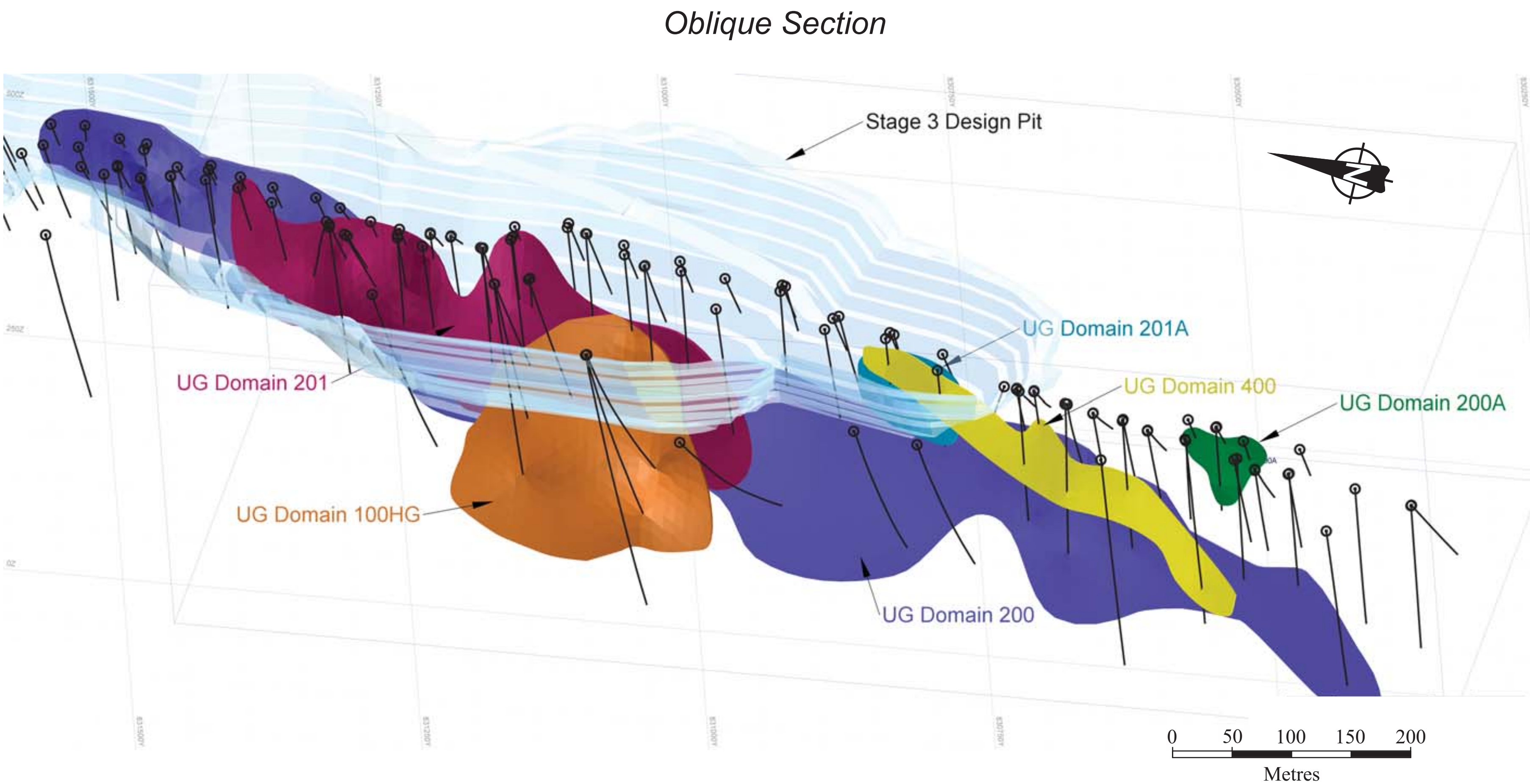


Figure 14-24

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Underground Domains

March 2019

Source: RPA, 2019.

DIGITAL TERRAIN MODEL

A LIDAR survey was undertaken by Southern Mapping in 2018. This survey has a vertical resolution of 5 cm on each 1 m^2 pixel size, based upon the Earth Gravitational Model (EGM) 2008. RPA's underground wireframes did not extend to the topography.

Thor identified that the drill hole collar elevations did not appear in some places to match the elevation in the LIDAR survey. Thor adjusted the collar elevations to match those of the LIDAR survey as they considered the elevations presented by the LIDAR survey to be a better match than those undertaken by the surveyors. The northing and easting locations were not adjusted. RPA is of the opinion that this approach, whilst not best practice, is suitable for the current level of underground study and resource classification as all drill holes have been adjusted to a common surface. RPA recommends that additional collar surveying be undertaken by an independent party to confirm the correct drill hole elevations.

As the underground Mineral Resource was clipped by the Stage 3 Design Reserve pit during reporting, the Mineral Resource does not intercept the weathering surfaces generate by Auralia.

CAPPING AND STATISTICAL ANALYSIS

RPA reviewed the capping levels using a mixture of histograms, decile analysis, log probability plots, disintegration analysis, and other plots.

Due to the limited dataset for the underground mineralisation (462 raw assays), RPA reviewed the capping on all underground domains simultaneously. RPA observed a clear break or change in the histograms, log probability plot, and disintegration analysis (Figure 14-25) at 50 g/t Au. In addition, the decile analysis indicates that at 50 g/t Au the top 1% of the assays are not generating more than 10% of the metal content and that the top 10% of the assays are contributing to approximately 40% of the metal content. Table 14-25 summarises the impact of capping on the raw assays.

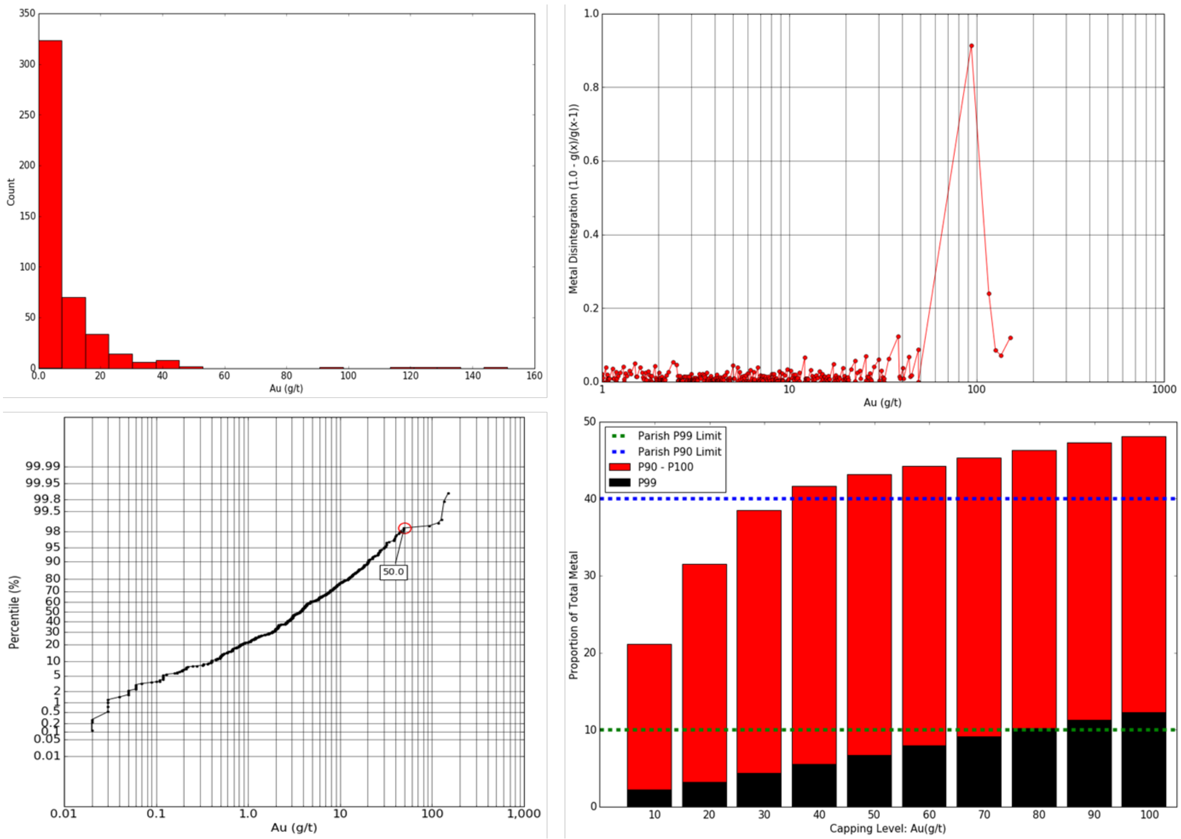
In RPA's opinion, the underground domains constrain the mineralisation well, and the capping helps limit potential over estimation biases from high-grade outliers.

TABLE 14-25 UNDERGROUND CAPPING RESULTS
Thor Explorations Ltd – Segilola Gold Project

Capping	No of Points	Minimum Au (g/t)	Maximum Au (g/t)	Mean Au (g/t)	Variance	Std Dev	Coefficient of Variation (CV)	Number of Caps
Uncapped	462	0.02	151.21	8.21	230	15.16	1.85	5
Capped		0.02	50.00	7.40	98	9.88	1.33	

In RPA's opinion, the underground domains constrain the mineralisation well, and the capping helps limit potential over estimation biases from high-grade outliers.

FIGURE 14-25 UNDERGROUND RAW ASSAY GRADE CAP REVIEW



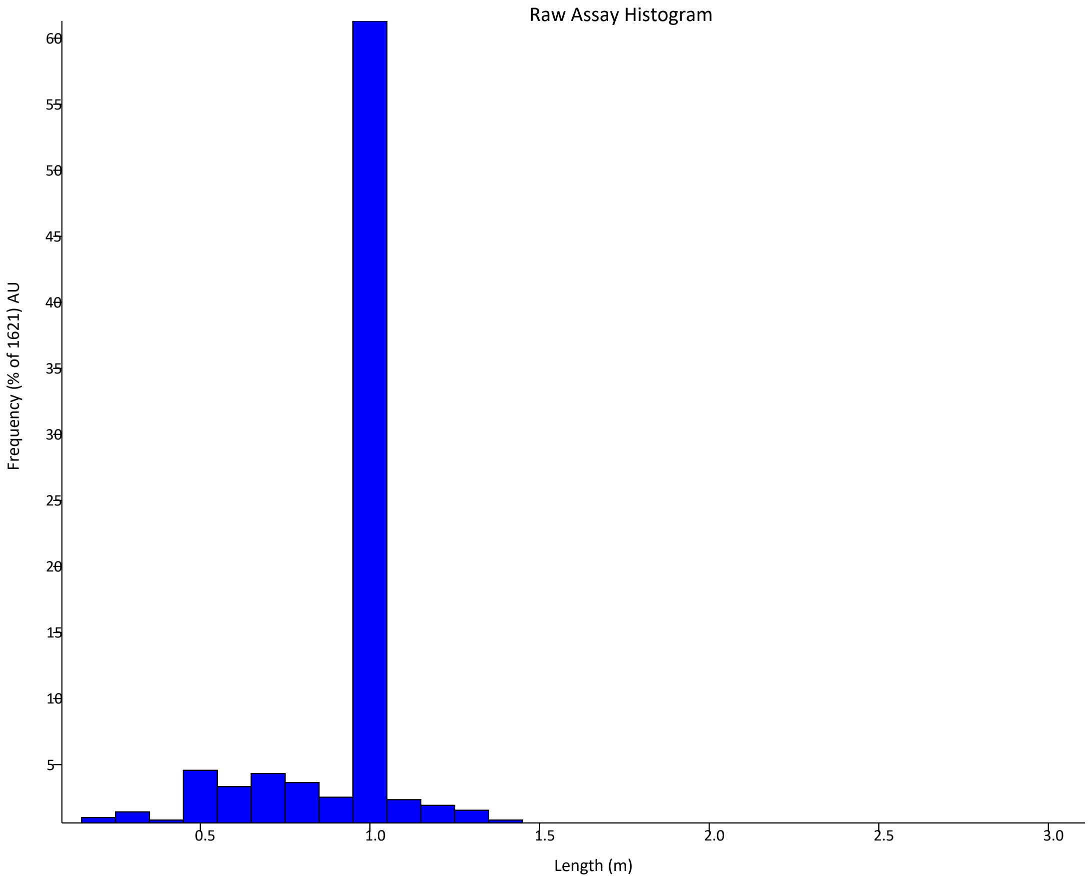
COMPOSITING

To ensure that samples of varying lengths are not given undue bias in the grade estimation, the capped assays were composited within the underground wireframes. RPA identified that over 60% of the raw assays inside the domains were sampled on one-metre intervals (Figure 14-26). Additionally, a significant proportion of the remaining samples were below one metre. Given the minimum mining width of two metres and the majority of samples occurring at one metre, a one-metre composite length was used. A minimum composite length of 0.3 m was applied. Any samples below 0.3 m in length were re-distributed to the previous samples so that they were not ignored during estimation. The impact of the compositing on the capped assays is illustrated in Table 14-26.

TABLE 14-26 UNDERGROUND COMPOSITE RESULTS
Thor Explorations Ltd – Segilola Gold Project

Source	No of Points	Minimum Au (g/t)	Maximum Au (g/t)	Mean Au (g/t)	Variance	Std Dev	Coefficient of Variation (CV)
Capped Assays	462	0.02	50.00	7.40	98	9.88	1.33
Capped Composites	446	0.03	50.00	7.61	92	9.58	1.26

FIGURE 14-26 UNDERGROUND RAW ASSAY LENGTH HISTOGRAM

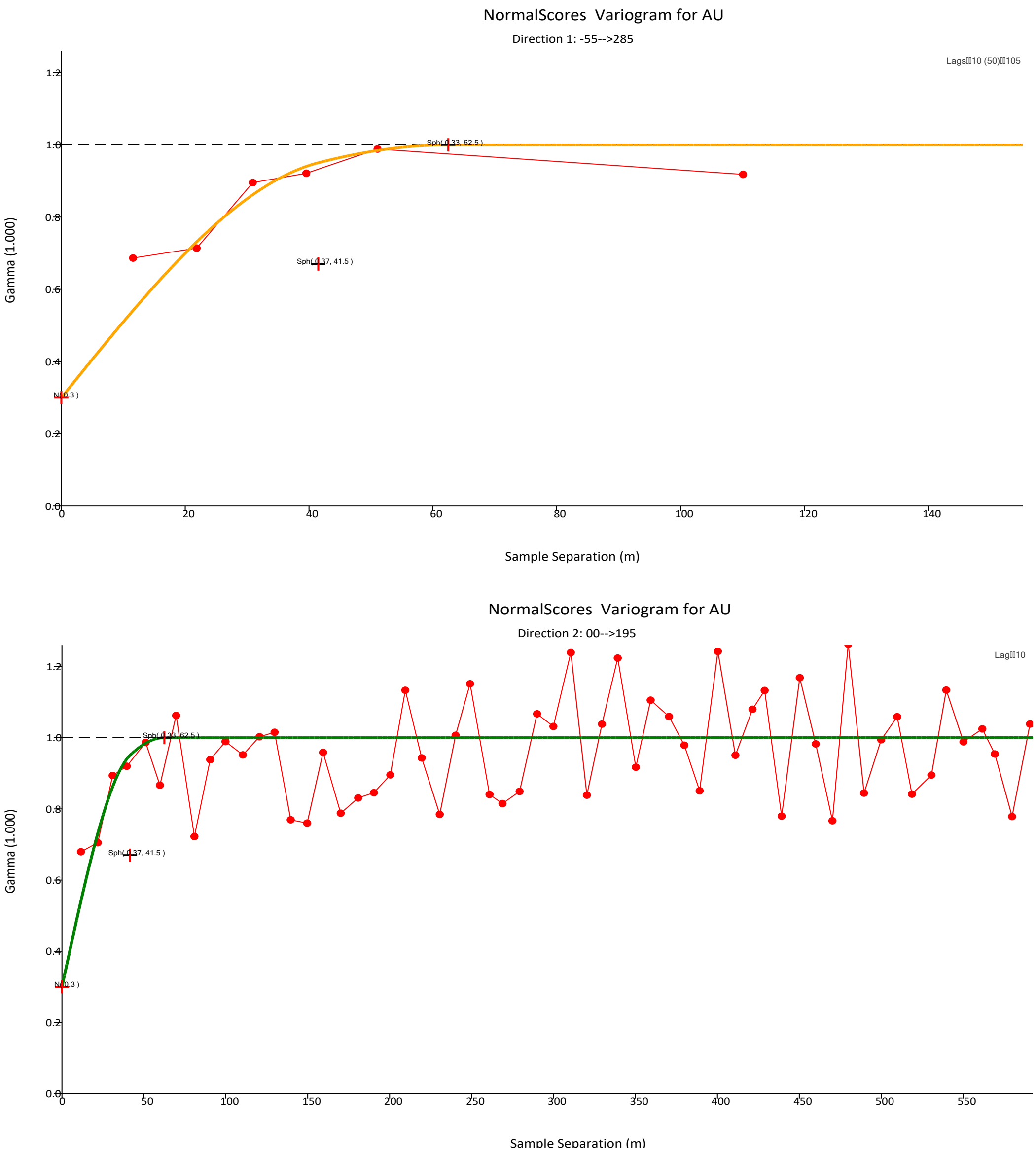


TREND ANALYSIS

The variography of the underground domains was investigated using Snowden Supervisor on the one-metre capped composites. Due to the limited data inside each underground domain and the significant similarities between the underground domains, the data for all underground domains was combined prior to review. The data was transformed with normal score, before the variogram models were constructed, and the resultant variogram model was back-transformed prior to importing into Micromine geological software for the block model interpolation.

Figures 14-27 and 14-28 illustrate, respectively, the variograms and back-transformed model for the Segilola underground Mineral Resource.

FIGURE 14-27 SEGILOLA UNDERGROUND NORMAL SCORE VARIOGRAMS



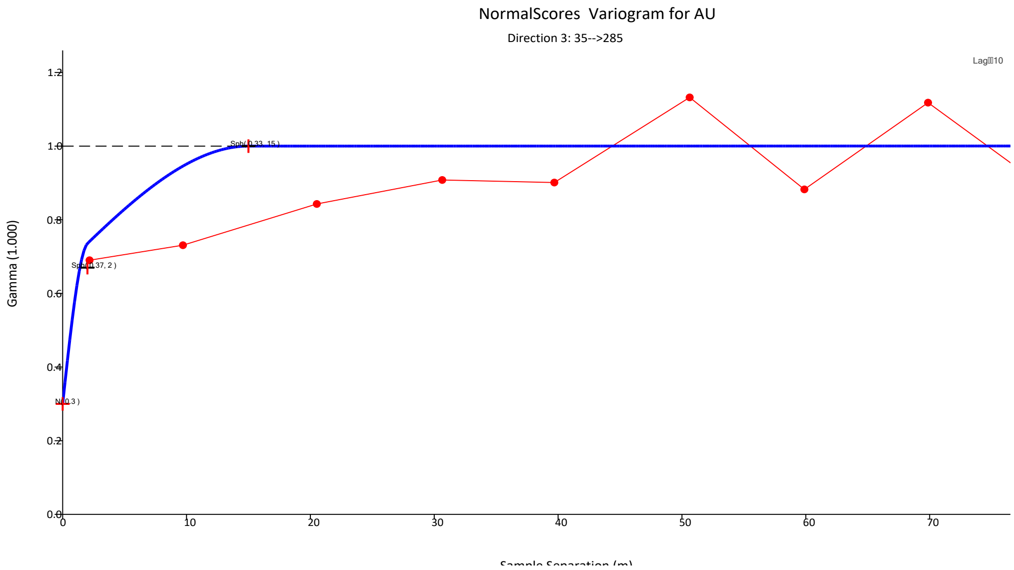


FIGURE 14-28 SEGILOLA UNDERGROUND BACK-TRANSFORMED MODEL

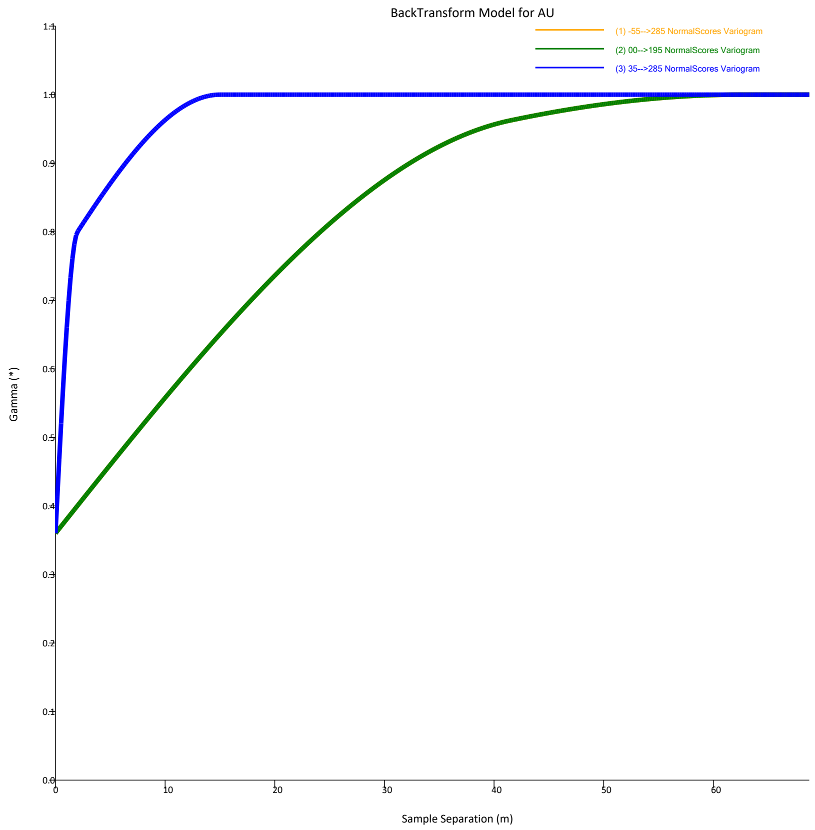


Table 14-27 details the results from the underground variography that were applied to the OK estimate.

TABLE 14-27 UNDERGROUND VARIOGRAPHY
Thor Explorations Ltd – Segilola Gold Project

Direction	Nugget	Sill 1	Sill 2	Direction	Plunge 1	Range 1 (m)	Range 2 (m)
Major	0.36	0.38	0.25	285°	-55°	41.5	62.5
Semi-Major	0.36	0.38	0.25	195°	0°	41.5	62.5
Minor	0.36	0.38	0.25	285°	35°	2	15

BLOCK MODELS

A blank block model was generated to fill the underground wireframes using parent blocks measuring 1 m by 5 m by 5 m (XYZ) in UTM 31N coordinates within Micromine geological modelling software. A minimum sub-block size of 0.5 m by 1.25 m by 1.25 m (XYZ) was employed to ensure that the block model provided a good volumetric representation of the mineralisation wireframes. Parent cell estimation was used, so that all sub-blocks of the same parent cell received the same gold grade. The block model was rotated around the Z axis to 15° from north to match the general mineralisation trend and reduce the amount of sub-blocking required. The underground block model contains 110,686 blocks flagged as being within the mineralised domains.

The parameters used to generate the blank block model are outlined in Table 14-28.

TABLE 14-28 UNDERGROUND BLOCK MODEL PARAMETERS
Thor Explorations Ltd – Segilola Gold Project

Parameter	X	Y	Z
Block Origin	701,080	830,002	-296
Block Size(m)	1	5	5
Number of Parent Blocks	964	598	150
Minimum Sub Block Size (m)	0.5	1.25	1.25
Rotation Around Z Axis		15°	

The block model attributes are summarised in Table 14-29.

TABLE 14-29 UNDERGROUND BLOCK MODEL ATTRIBUTES
Thor Explorations Ltd – Segilola Gold Project

Attribute	Type	Description
X	Numeric	Block Centroid - East
Y	Numeric	Block Centroid - North
Z	Numeric	Block Centroid - RL
_X	Numeric	Block Size - East
_Y	Numeric	Block Size - North
_Z	Numeric	Block Size - RL

Attribute	Type	Description
Class	Character	Classification, 2=Indicated, 3=Inferred, 4=Unclassified
UG_Domain	Character	UG Domain Name
Azimuth	Numeric	DA Block Azimuth
Dip	Numeric	DA Block Dip
Plunge	Numeric	DA Block Plunge (0)
Density	Numeric	Block Density (2.70 t/m^3)
Au_Cap	Numeric	Ordinary Kriging Capped Au
KR_VAR	Numeric	Kriging Variance
KR_STDERR	Numeric	Kriging Standard Error
KR_EFF	Numeric	Kriging Efficiency
SLOPE	Numeric	Kriging Slope of Regression
PERC_NEG	Numeric	Percentage of Negative Weights
Pass	Numeric	Search Pass Number
POINTS	Numeric	Number of Composites Used
Count	Numeric	Number of Drill Holes Used
AVERAGE DISTANCE	Numeric	Average Distance to Composites
CLOSEST DISTANCE	Numeric	Closest Distance to Composite
NN_AU_Cap	Numeric	Nearest Neighbour Capped Au
Resource Stopes at 1500	Character	Stope Constraint Flag
Stage 3 - Jan 8th	Character	Stage 3 Reserve Design Flag

SEARCH STRATEGY AND GRADE INTERPOLATION PARAMETERS

Generally, the underground mineralisation wireframes had a fairly constant azimuth and dip, however, localised variations in the orientations of the wireframes existed. To ensure that the direction of the search ellipsoid honoured these local variations, dynamic anisotropy (DA) was employed to vary the orientation of the search ellipsoid. Mid-point reference surface wireframes were extracted from the Leapfrog project for each domain and the dip and dip direction was extracted from the wireframe surface triangles. These angles were converted to a dip and strike measurement which was flagged to each individual block using a nearest-neighbour (NN) search. Each block's azimuth and dip were used to orientate the search ellipsoid individually for that block.

RPA implemented a three-pass approach to the interpolation, each with a larger search ellipsoid radius and decreasing sample requirements, to ensure that all blocks within the block model were interpolated (Table 14-30). Approximately 40% of the blocks were interpolated in pass one and 40% in pass two, with the remainder being interpolated with the broader pass three. Micromine v2018.1 was used to interpolate the block model.

The initial search pass radius was set at the variogram range, which matched approximately 1.5 times the drill spacing within tighter drilled parts of the deposit.

TABLE 14-30 UNDERGROUND SEARCH PROCEDURE
Thor Explorations Ltd – Segilola Gold Project

Pass Number	Blocks Interpolated in Pass	Search Ellipsoid Radius (m) Direction 1	Direction 2	Direction 3	Max Samples	Min Samples	Minimum Holes	Max Comps Per Hole
1	37%	40	40	10	12	6	2	3
2	44%	60	60	15	12	4	2	3
3	19%	80	80	20	12	2	-	3

A 3 by 3 by 3 discretization was used during interpolation. A hard boundary was used during grade interpolation to ensure that grades were only interpolated using assays from the requisite domain. Additional interpolations using both NN and Inverse Distance Cubed (ID^3) algorithms was undertaken using the same search parameters for resource model validation purposes.

BULK DENSITY

Bulk density measurements were completed on-site on samples drilled by CGA using Archimedean principles using the weight of the core in air and the core suspended in water. The following formula was used:

$$\text{Specific Gravity} = \frac{\text{Weight of Sample (g) in Air}}{(\text{Weight of Sample (g)in Air} - \text{Weight of Sample (g)in Water}}$$

Density measurements on Thor samples have been primarily undertaken by the external MS Analytical laboratory.

RPA reviewed the density data for all drill holes that intersected the underground model. In total, 46 drill holes containing 453 density samples were collected in the underground resource area, and 19 data points were collected from within the underground mineralisation wireframes. During analysis, RPA excluded one measurement in the waste material (SGD130 48.1 m to 48.25 m), as the density was considered an outlier (1.01 t/m^3).

Table 14-31 details the density statistics for the underground resource. Although there is a spread of density samples, the mineralisation and waste both have the same average density (2.70 t/m^3). This could be influenced by the lower grade mineralisation (below 2.50 g/t Au) immediately adjacent to the high-grade mineralisation being flagged as waste for the underground model, yet still having a similar density.

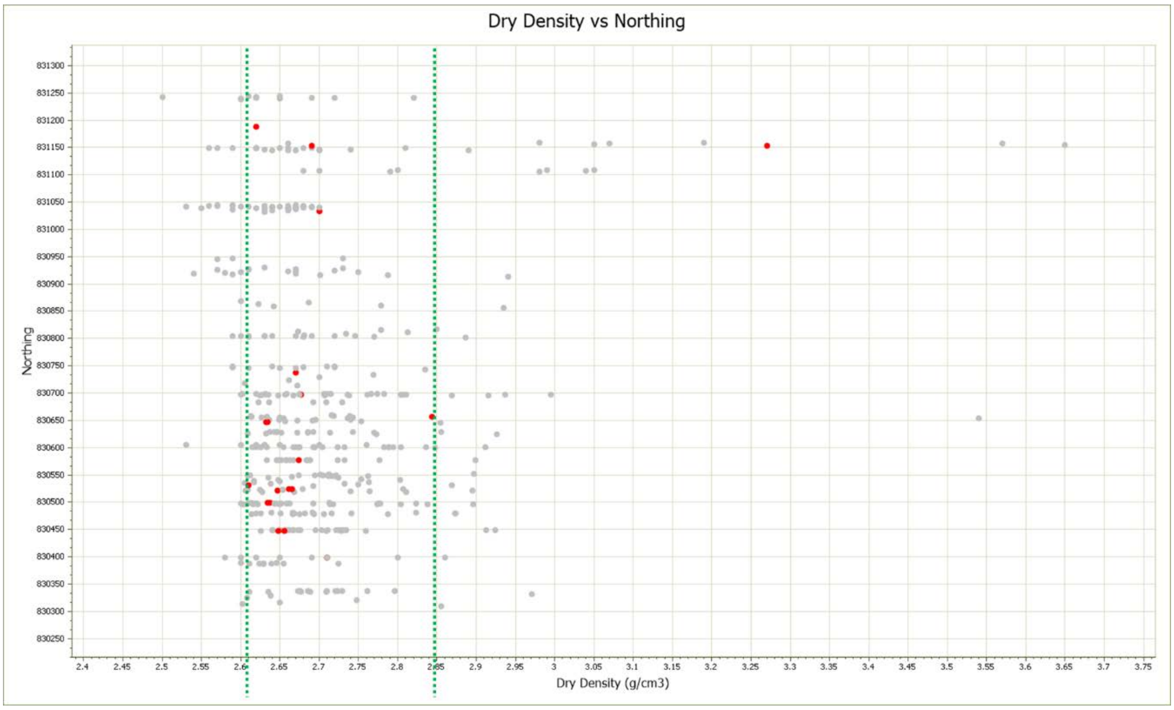
TABLE 14-31 UNDERGROUND DENSITY SAMPLES
Thor Explorations Ltd – Segilola Gold Project

Parameter	Location of Density Samples		
	All	Within Mineralisation	Within Waste
No of Points	453	19	434
Minimum (t/m^3)	2.50	2.61	2.50
Maximum (t/m^3)	3.65	3.27	3.65
Mean (t/m^3)	2.70	2.70	2.70
Variance	0.01	0.02	0.01
Std Dev	0.12	0.15	0.12
Coefficient of Variation	0.04	0.05	0.04

Figure 14-29 illustrates the underground density samples on a scatterplot against the northing of the sample. This plot illustrates that the majority of the mineralised samples (in red) occur within a narrow band. A single density of 2.70 t/m^3 was applied to all underground domains.

RPA recommends that more mineralisation density measurements be taken prior to the next Mineral Resource estimate. RPA recommends that Thor use an on-site scale with a hanging basket/hook to undertake density measurements using Archimedes principles. Calibration samples of known density should be measured every 20th test as part of on the on-site density test work QA/QC procedures.

FIGURE 14-29 UNDERGROUND DENSITY SAMPLES IN MINERALISATION (RED) AND WASTE (BLACK)



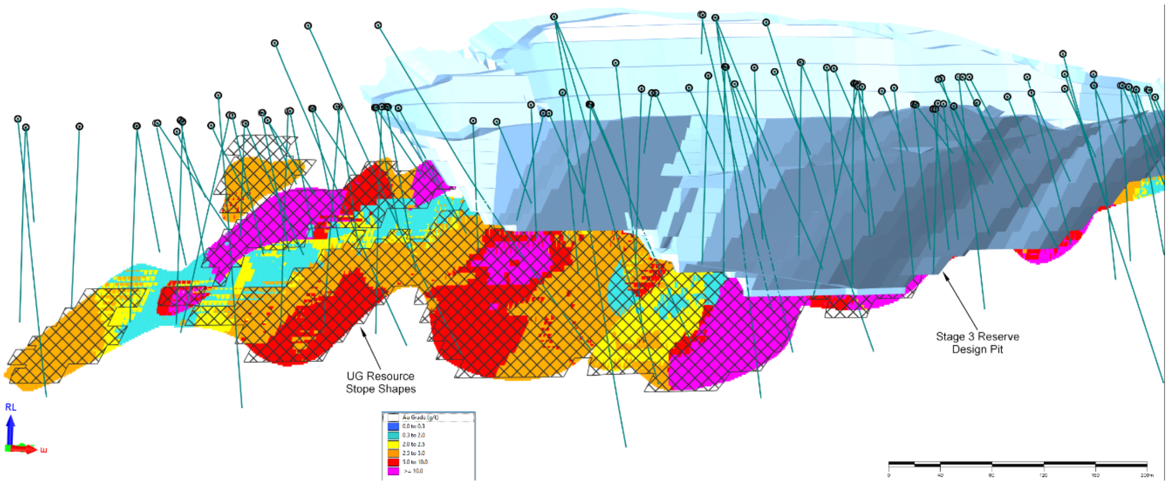
CUT-OFF GRADE AND STOPE CONSTRAINT

RPA reported the underground Mineral Resource at a 2.58 g/t Au cut-off grade based on the parameters as summarised in Section 24.

Metal prices used for reserves are based on consensus, long term forecasts from banks, financial institutions, and other sources. For resources, metal prices used are slightly higher than those for reserves.

In order to determine reasonable prospects for eventual economic extraction, RPA generated stope wireframes to constrain the mineralisation (Figure 14-30). These stope shapes were generated at a $1,500/oz gold price using appropriate mining and geotechnical constraints. A waste halo was generated using the grades of the material that was proximal to the underground wireframes to aid stope design, although this material was not included in the Mineral Resource. The use of stope constraints removed any areas of the block model that were above the cut-off grade yet were not contiguous with the resource or were of insufficient thickness to be considered as economically extractable.

FIGURE 14-30 UNDERGROUND $1,500/OZ STOPE SHAPES (BLACK OUTLINE) USED TO CONSTRAIN RESOURCE (OBLIQUE VIEW)



To ensure that the underground and open pit Mineral Resources did not overlap with each other, the underground Mineral Resources were reported using a filter that only used allowed outside the Stage 3 Mineral Reserve pit design that were south of a demarcation line between the open pit and underground Mineral Resources (Figure 14-31). This demarcation line was generated in a saddle zone where the resource pit shell and the reserve design pit shells were observed to be proximal to each other, thus having minimal impact on the resource pit shell shape. This demarcation line was also generated such that the open pit model contained limited mineralisation south of it and the underground mineralisation contained limited metal north of it (Figure 14-31).

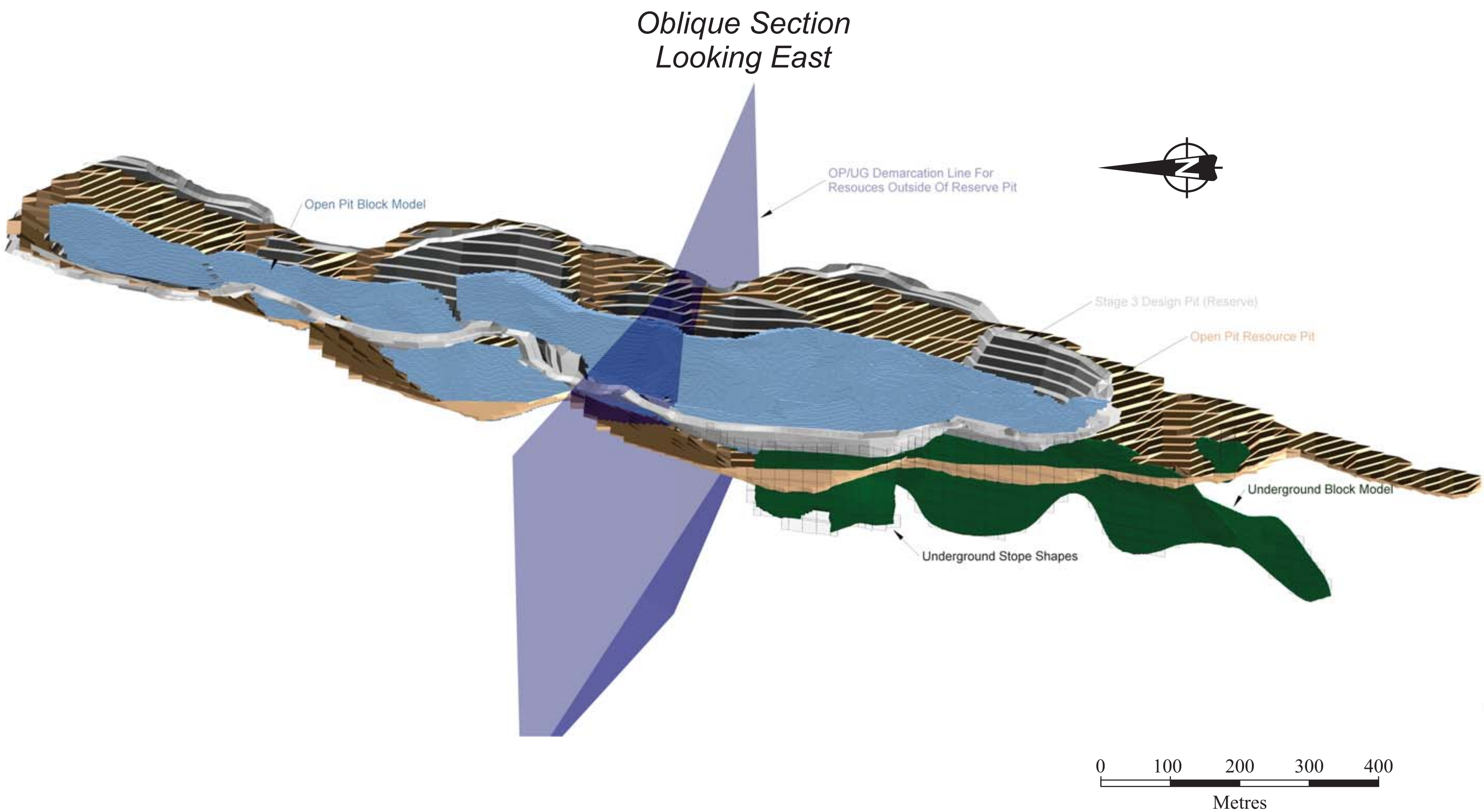


Figure 14-31

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Open Pit and Underground Block Model Constraints

March 2019

Source: RPA, 2019.

CLASSIFICATION

Definitions for resource categories used in this report are consistent with CIM (2014) definitions as incorporated by reference in NI 43-101. In the CIM classification, a Mineral Resource is defined as “a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction”. Mineral Resources are classified into Measured, Indicated, and Inferred categories. A Mineral Reserve is defined as the “economically mineable part of a Measured and/or Indicated Mineral Resource” demonstrated by studies at Pre-Feasibility or Feasibility level as appropriate. Mineral Reserves are classified into Proven and Probable categories.

Categories of Inferred, Indicated, and Measured are recognised in order of increasing geological confidence. However, Mineral Resources are not equivalent to Mineral Reserves and do not have demonstrated economic viability. There can be no assurance that Mineral Resources in a lower category may be converted to a higher category, or that Mineral Resources may be converted to Mineral Reserves. Due to the uncertainty associated with Inferred Mineral Resources, there is no assurance that Inferred Mineral Resources will be upgraded to Indicated or Measured Mineral Resources with sufficient geological continuity.

RPA reviewed each domain separately in longitudinal section to define the classification. RPA considered a combination of drill hole spacing, distance to nearest sample, mineralisation continuity, minimum number of samples, and the search pass number to inform the Mineral Resource classification.

For Indicated, RPA outlined areas of the block model in each domain that were interpolated using a maximum drill spacing of 25 m by 25 m during the first and second pass (minimum of two holes and four samples). For the Inferred material, a 50 m by 50 m maximum drill spacing was employed with a minimum of two samples. Small areas of material that were below these minimum requirements were included to aid the classification continuity where the grade and geological continuity were observed to be consistent. In total, 65% of the Indicated material was interpolated in pass one, and 99% within pass one and two.

These outlines were used to construct wireframes to flag each domain within the block model separately with the resource classification to ensure that the classification was contiguous (Figure 14-32). Figure 14-33 presents a histogram showing the breakdown of classification at Segilola underground based upon the distance to closest sample.

FIGURE 14-32 UNDERGROUND CLASSIFICATION

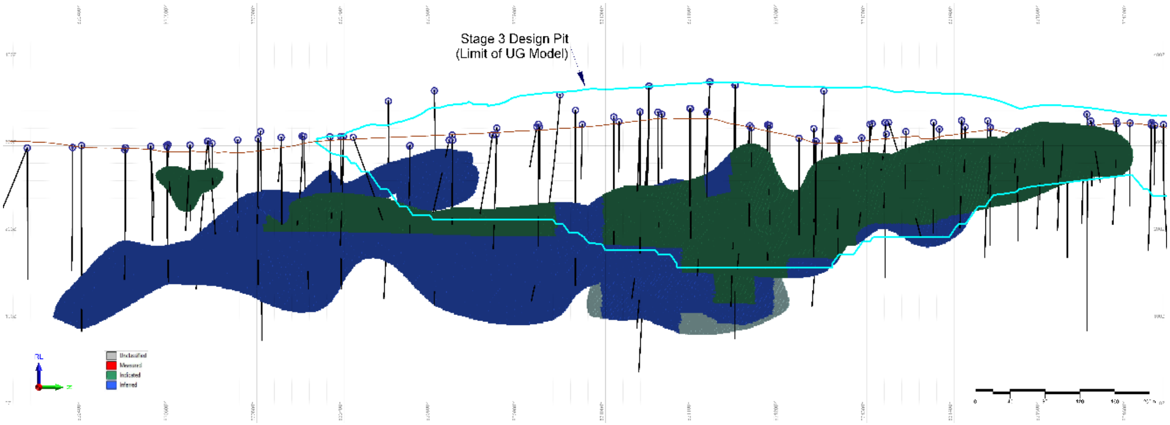
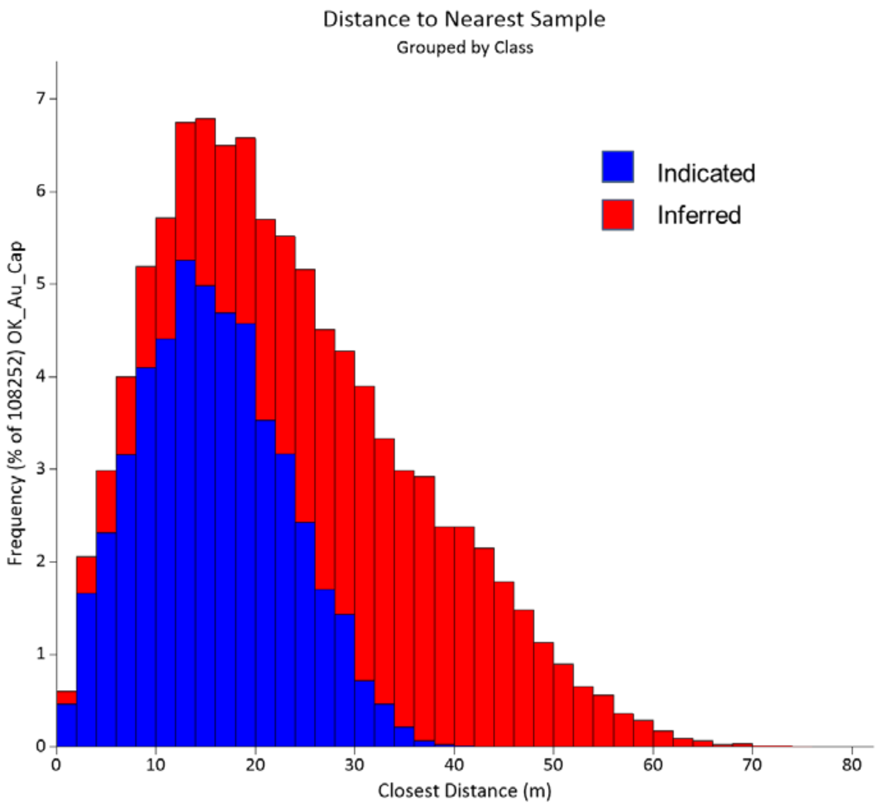


FIGURE 14-33 HISTOGRAM OF CLASSIFIED BLOCKS VERSUS CLOSEST SAMPLE



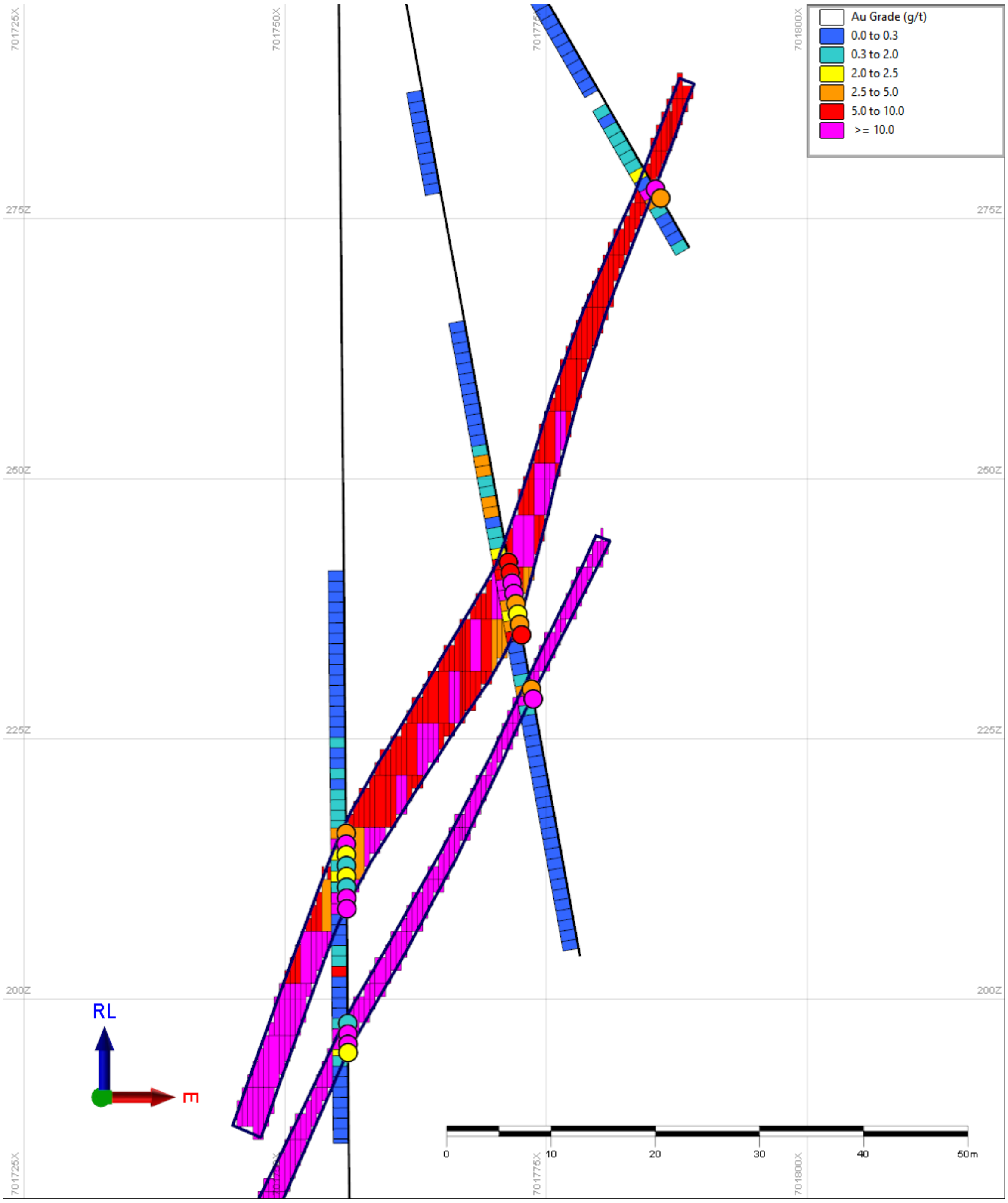
BLOCK MODEL VALIDATION

RPA performed a series of validation exercises on the OK block model to ensure that it was free of errors. These included:

1. Visual inspection of block model grades against composite grades in section and plan.

   The visual comparison of the composite grades against the block model (Figure 14-34) indicates that the block model is providing a good representation of the composite grades in the block model, and that the search strategy and dynamic anisotropy are performing well.

FIGURE 14-34 VISUAL COMPARISON OF COMPOSITE AND BLOCK MODEL GRADES



2. Volumetric comparison between the underground wireframes and the block model. The volumetric comparison check between the wireframe volume and the block model flagged with the wireframes indicates that there is a 0.24% difference between the volumes (Table 14-32). This indicates that the current block model setup is providing a good reflection of the resource wireframes.

TABLE 14-32 WIREFRAME VERSUS BLOCK MODEL VOLUMETRIC COMPARISON
Thor Explorations Ltd – Segilola Gold Project

Domain	Wireframe Volume (m^3)	Block Model Volume (m^3)	Volume Delta (m^3)	Volume Delta (%)
400	20,574	20,591	-16	-0.08%
201A	12,479	12,473	6	0.05%
201	140,197	140,176	22	0.02%
200A	7,399	7,395	4	0.06%
200	290,173	289,219	955	0.33%
100HG	106,200	105,807	393	0.37%
Total	**577,023**	**575,660**	**1,363**	**0.24%**

3. Statistical comparison between the ID^3 and NN check estimates

   A statistical comparison between the OK model, the check models (ID^3 and NN), and the composites (excluding unclassified material) indicates that the block model is honouring the input data (Table 14-33). The lower coefficient of variation and maximum block model grade along with the higher minimum block grade indicate that there is higher smoothing within the OK model (as is customary for OK interpolation), however, visual checks confirm that the level of smoothing is reasonable.

TABLE 14-33 STATISTICAL COMPARISON BETWEEN OK, ID^3, NN, AND COMPOSITES
Thor Explorations Ltd – Segilola Gold Project

Field Name	Composites	OK	OK vs. Composites	ID^3	ID^3 vs. Composites	NN	NN vs. Composites
No of Points	446	110,686	-	110,686	-	110,686	-
Minimum (g/t Au)	0.03	0.44	-	0.07	-	0.03	-
Maximum (g/t Au)	50.00	45.18	-10%	49.99	0%	50.00	0%
Mean (g/t Au)	7.61	8.13	7%	8.15	7%	7.91	4%
Variance	91.80	35.24	-62%	44.78	-51%	81.23	-12%
Std Dev	9.58	5.94	-38%	6.69	-30%	9.01	-6%
Coefficient of Variation	1.26	0.73	-42%	0.82	-35%	1.14	-10%

4. Swath plot and comparison histograms between the OK model and the composite samples.

   RPA generated swath (trend) plots to compare the OK model against the composite grades against the easting (Figure 14-35), northing (Figure 14-36), and elevation (Figure 14-37). These swath plots indicate that the OK block model is reflecting the input composite grade. The variation observed between the NN model and the OK model and composites at the far left of the northing and easting is caused by limited data availability. Visual observations confirm that there is no issue in this area.

FIGURE 14-35 SWATH PLOT BY EASTING

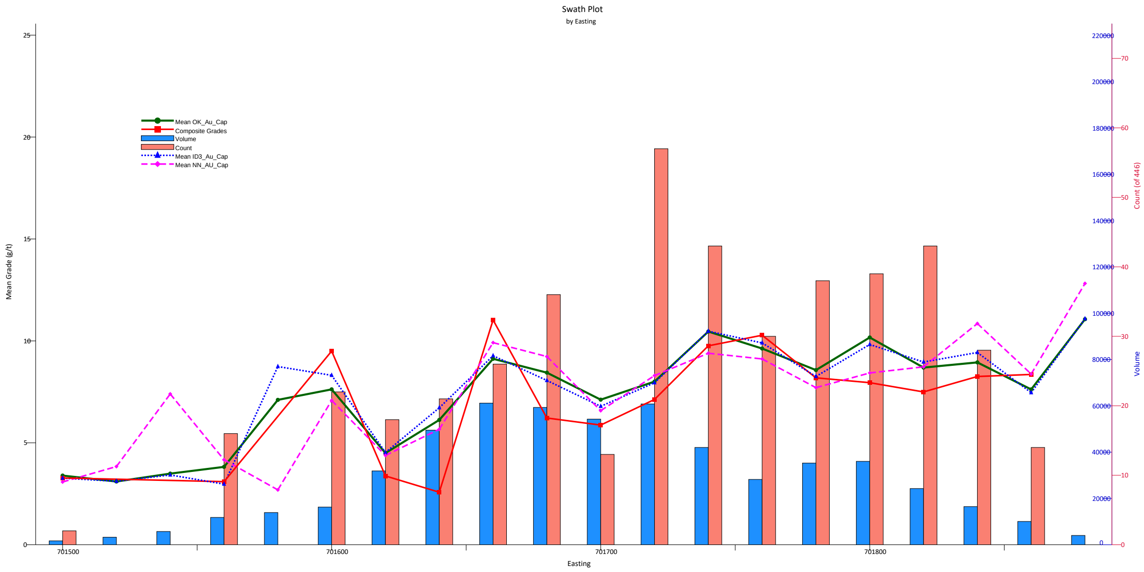


FIGURE 14-36 SWATH PLOT BY NORTHING

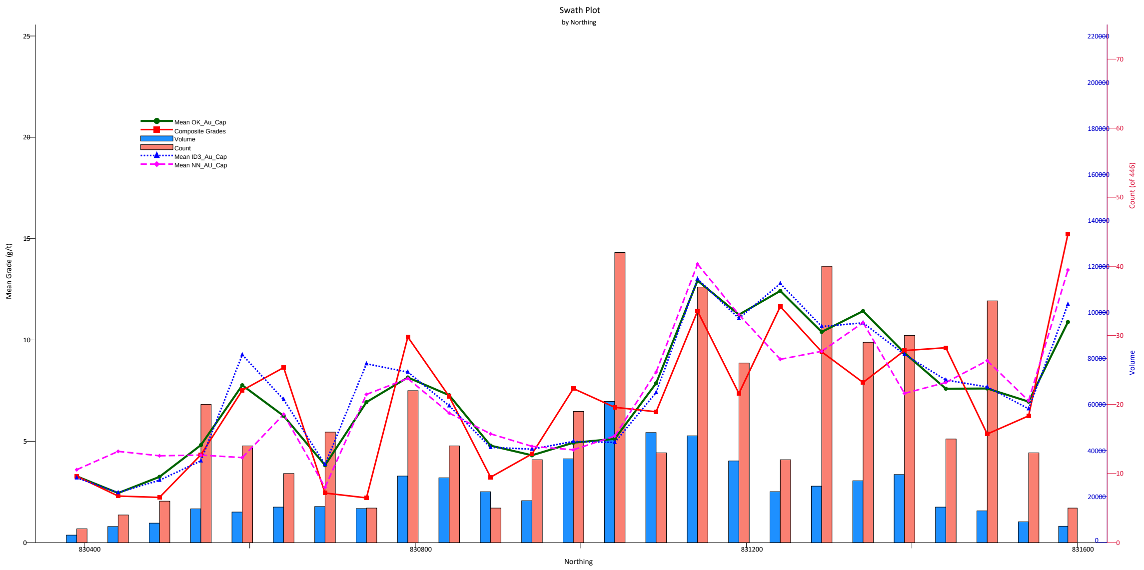
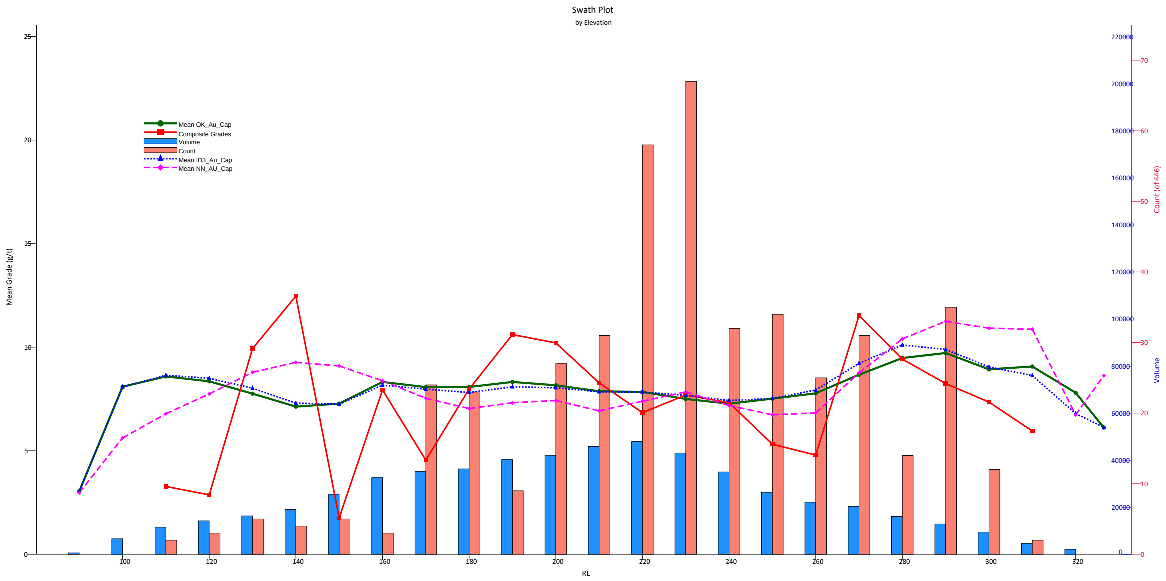


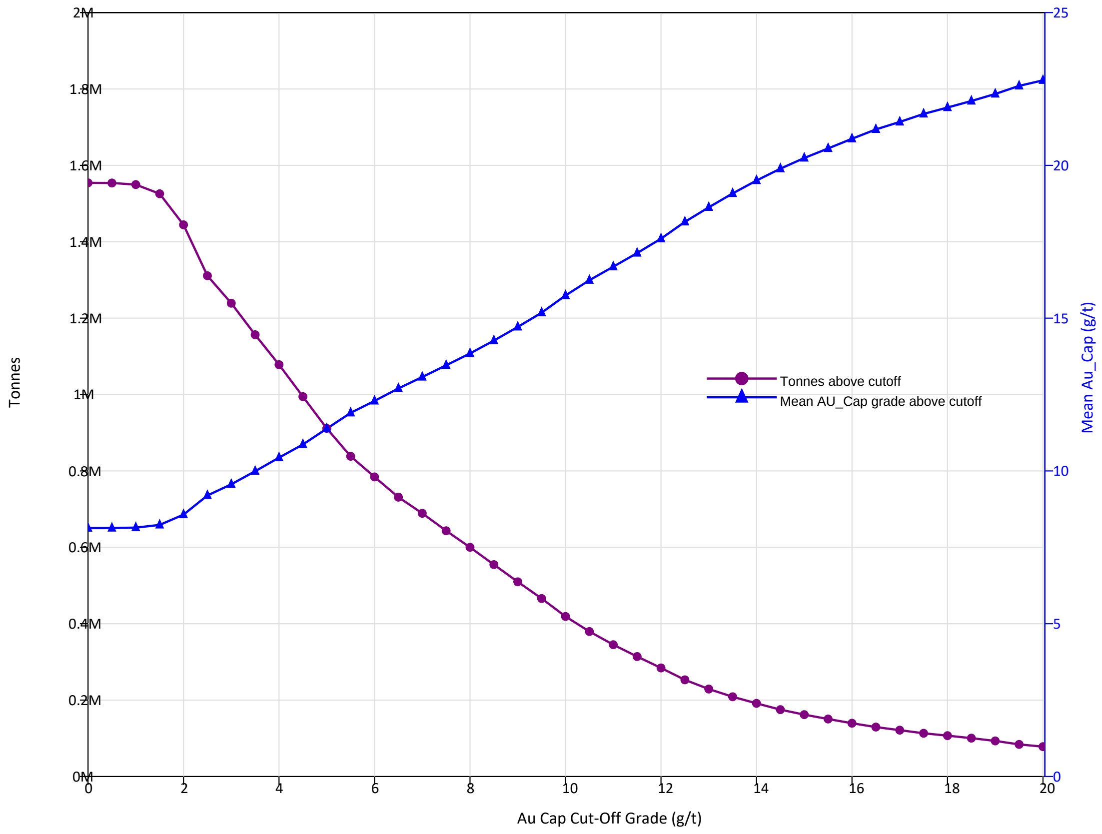
FIGURE 14-37 SWATH PLOT BY ELEVATION



5. Grade tonnage curves.

   The grade tonnage curve (Figure 14-38) indicates that the block model is performing well, with significant flattening of the curves below 2.0 g/t Au to 2.5 g/t Au, which reflects the wireframing cut-off used. There is some variation below the wireframing cut-off value as a result of the minimum mining width being used which has, in some areas, incorporated below cut-off grade material as reflected in the curve.

FIGURE 14-38 UNDERGROUND BLOCK MODEL GRADE TONNAGE CURVE



UNDERGROUND MINERAL RESOURCE REPORTING

In order to demonstrate reasonable prospects for eventual economic extraction, underground Mineral Resources have been reported within $1,500 stope shapes at a cut-off grade of 2.58 g/t Au. A summary of the Mineral Resources effective as of 1 December 2018 is presented in Table 14-34.

TABLE 14-34 SUMMARY OF UNDERGROUND MINERAL RESOURCES – 1 DECEMBER 2018
Thor Explorations Ltd – Segilola Gold Project

Classification	UG Domain	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Indicated	100HG	25	19.77	16
	200	50	6.28	10
	200A	18	3.96	2
	Total	**93**	**9.39**	**28**
Inferred	100HG	60	13.8	27
	200	244	5.7	45
	201	16	6.2	3
	400	33	13.9	15
	Total	**352**	**7.9**	**90**

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Underground Mineral Resources are estimated by RPA at a cut-off grade of 2.58 g/t Au and constrained within stope shapes using an Au price of $1,500/oz.
3. Underground wireframes were defined using a nominal 2.5 g/t Au wireframing cut-off and 2 m minimum mining width.
4. Underground bulk density is 2.70 t/m^3.
5. High gold assays were capped to 50 g/t Au for underground resources.
6. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
7. Numbers may not add due to rounding.

15 MINERAL RESERVE ESTIMATE

SUMMARY

The open pit Mineral Reserve estimate was prepared by Auralia and is summarised in Table 15-1. The Mineral Reserve constitutes the portion of the Mineral Resource which is economically and practically mineable under the specified project parameters. The Mineral Reserve has been estimated in accordance with CIM Definition Standards, and exclude Inferred Mineral Resources.

TABLE 15-1 MINERAL RESERVES – 1 DECEMBER 2018
Thor Explorations Ltd. – Segilola Gold Project

	Category	Tonnage (kt)	Grade (g/t Au)	Contained Metal (000 oz Au)
Open Pit	Probable	3.00	4.2	405
Total		**3.00**	**4.2**	**405**

Notes:
1. CIM (2014) definitions were followed for Mineral Reserves.
2. Mineral Reserves are estimated at a cut-off grade of 0.77 g/t Au.
3. Mineral Reserves are estimated using an average long-term gold price of $1,250 per ounce.
4. A minimum mining width of 10 m was used.
5. Mining dilution of 10% and mining recovery of 95% were applied
6. Numbers may not add due to rounding.

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

RESERVE ESTIMATION PROCESS

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

- The economic cut-off grade was determined based on metal selling price and cost, processing cost and recovery, and general and administration (G&A) costs.
- Appropriate mining dilution and mining recovery factors were selected and applied.
- 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, as detailed in Section 22.

GEOTECHNICAL PARAMETERS

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 was 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 fairly 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.

HYDROGEOLOGICAL IMPACT

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. A number of 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 mineralised 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.

INTACT ROCK STRENGTH

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 Tables 15-2 and 15-3.

TABLE 15-2 SUMMARY OF UCS TESTING RESULTS
Thor Explorations Ltd. – Segilola Gold Project

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

Notes. * 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 15-3 SUMMARY OF DS TESTING RESULTS
Thor Explorations Ltd. – Segilola Gold Project

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

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).

DESIGN FACTORS

'Base case' wall design parameters have been 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 15-4 SUMMARY OF FINAL PIT DESIGN PARAMETERS, SOUTHERN, CENTRAL AND NORTHERN PITS (BASED UPON STAGE 2 DESIGN PROVIDED BY THOR)
Thor Explorations Ltd. – Segilola Gold Project

	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°	

Notes. * This is the inferred pit crest, locally will need to be adjusted for wreathing 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

DILUTION AND RECOVERY

DILUTION

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

Dilution was estimated by determining the width of the orebody on an approximately 10 m x 10 m long-sectional grid. Orebody widths were only calculated for lodes 100 and 200, as Lode 300 contains predominately Inferred material.

The ore is visually discernible from the waste material.

As well as being split by lode, an approximation was made to distinguish between high grade, medium grade, and low grade.

Dilution was calculated as the mining width divided by the in-situ width, where mining width for each intercept was taken as the greater of 2.5 m or in-situ width plus 0.5 m. The minimum mining width of 2.5 m is based on a combination of the drill and blast patterns (burden between 2.5 m and 3.0 m) and expected excavator bucket width (2.2 m to 2.8 m).

The average in-situ width, mining width, and expected dilution for each grade bin and lode is shown in Table 15-5. Although the results show variability in expected dilution depending on lode, the variability between grade bins is less obvious. Despite the variability, a single “unplanned” dilution factor has been applied.

TABLE 15-5 DILUTION ANALYSIS
Thor Explorations Ltd. – Segilola Gold Project

Grade/Lode	Sample Points	Average In-situ Width	Average Mining Width	Calculated Dilution
HG	845	6.4	7.0	8.6%
200	422	9.4	9.9	5.4%
100A	343	3.5	4.1	16.9%
100B	80	3.4	4.0	18.8%
MG	585	6.7	7.3	9.0%
200	319	9.8	10.3	5.4%
100A	90	2.4	3.2	32.9%
100B	176	3.2	3.8	19.6%
LG	703	5.6	6.2	10.6%
200	357	7.7	8.2	6.8%

Grade/Lode	Sample Points	Average In-situ Width	Average Mining Width	Calculated Dilution
100A	98	4.6	5.3	14.7%
100B	248	3.0	3.6	22.4%
Grand Total	**2133**	**6.2**	**6.8**	**9.3%**

Limited internal "planned" dilution is expected from "mineralised waste" associated with economic ore as well as narrow areas of waste that have been modelled either between lodes or in areas of the resource that have been modelled to bifurcate. Both of these areas of internal dilution can be seen in Inset A and B of Figure 15-1. A global value of between 0.5% and 1% has been attributed to internal dilution.

FIGURE 15-1 RESOURCE MODEL SECTION FOR DILUTION ANALYSIS

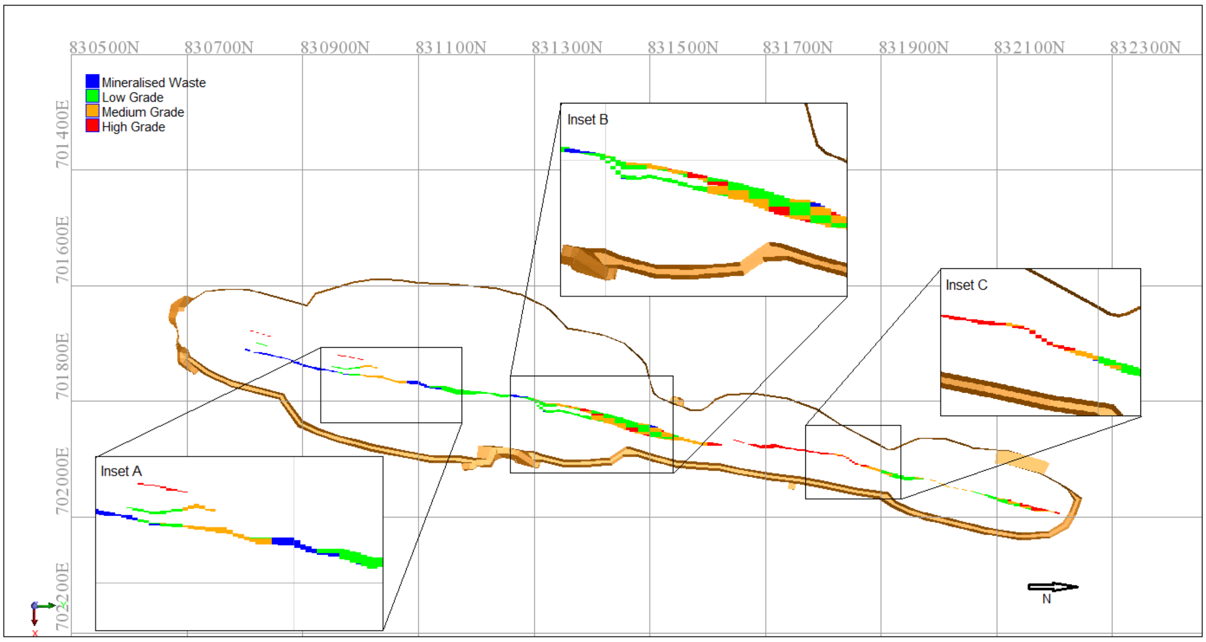


Figure 15-1 shows the Resource model at the 300 mRL inside the pit design. This section is reflective of the entire deposit and shows that in most areas, the full width of the lode is either high grade, medium grade, or low grade. In the wider areas of Lode 200, there is some variability in grade across the lode, however, it is not sufficiently consistent in terms of grade distribution (i.e. a high-grade core surrounded by lower-grade material) to allow a determination that high grade would be diluted by medium grade or medium grade diluted by low grade. As a result, all dilution, regardless of the grade of ore material mined, has been assumed to have zero grade.

A single dilution factor of 10% has been applied to the entire deposit as opposed to separate factors based on location or material type. The dilution includes ‘unplanned dilution’ of approximately 9% and planned dilution of approximately 1%.

As grade control drilling is completed, and mining blocks are planned and marked out as part of the mining operations, a review of dilution factors is recommended.

RECOVERY

A 95% mining recovery factor was applied within Whittle to account for the amount 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.

CUT-OFF GRADE AND ECONOMIC PARAMETERS

The final economic cut-off grade determines what is classified as ore and waste. It varies depending upon the parameters input to the formula. The formula for economic cut-off grade is as follows:

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

Processing cost in this calculation includes the following:

- Fixed/variable direct ore processing cost
- G&A cost
- Grade control cost
- Stockpile/ROM re-handle cost
- Refining cost

The cut-off grade for the parameters used in the formulation of the Mineral Reserve was calculated as 0.77 g/t Au. The costs and revenue used for this determination are given in Table 15-6.

PIT OPTIMISATION

A final pit and two interim stage pits have been designed based on optimisation shells created with the relevant cost, revenue, and physical input parameters.

Optimisation runs were conducted in Whittle using project specific input parameters.

The base case optimisation parameters are summarised in Table 15-6, and detailed in the following sub-sections. All selling prices and costs applied are in US dollars.

TABLE 15-6 BASE CASE WHITTLE OPTIMISATION PARAMETERS
Thor Explorations Ltd. – Segilola Gold Project

Whittle Input Parameter	Value	Unit
Overall Pit Slope	50W/42E	degrees
Surface Mining Cost (Waste)	$2.67	$/t
Mining Dilution	10%	%
Mining Recovery	95%	%
Processing Cost	$19.40	$/t ore
Processing Recovery (Au)	97%	%
G&A Cost	$5.77	$/t ore
Grade Control	$0.34	$/t ore
Re-handle	$0.65	$/t ore
Refining	$0.88	$/t ore
Discount Rate	8%	%
Metal Price Gold	$1,250.00	$/oz
Selling Cost/Royalties*	$14.88	$/oz

Notes. *Royalties paid to previous project owners TML and Ratel (1.5% capped at $4.0M for TNL and $3.5M for Ratel) were not applied in Whittle but were applied in the Economic Analysis.

Optimisations were run based on mining only Indicated Mineral Resources. The extraction of Inferred material was not used as an economic driver for the pit shells.

The deposit is orientated with approximately a north-south strike. Overall wall angles applied in Whittle were

- 42° on the east wall and end walls.
- 50° on the west wall.

Optimisation runs were conducted to determine the likely impact of the location of the ramps on either the east or west wall using the recommended geotechnical design parameters.

The results indicated that placing the ramps predominantly on the west wall should result in a significantly higher (10% to 15%) discounted cash flow due to the lower strip ratio.

The optimisation assumes a process plant throughput of 625,000 tpa of ore.

The commodity price used for the base case optimisation was $1,250/oz for gold.

Royalties payable to the Nigerian government at a rate of 5,400 Naira per recovered ounce of gold were applied in the optimisation. Using an exchange rate of 363 Naira to US$ this equates to $14.88 per ounce.

Two additional royalties were applied in the Economic Analysis/cash flow modelling due to the fact that the royalties have maximum values which is not conducive to accurate application in Whittle. The applicable royalties are as follows:

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

No cut-off grades were forced within the optimisations; Whittle applied the appropriate economic cut-off grade as required per run. The cut-off grade for the parameters used in the base case optimisation was calculated to be 0.77 g/t Au.

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

The maximum cash flow shell resulting from the Whittle optimisation was shell 34 (RF0.96). The final shell selected by Thor was shell 37 (RF1.02), which increases gold production to over 400,000 recovered ounces with a minimal increase in strip ratio. The trade-off in this case was less than 0.2% discounted cash flow (DCF) according to the optimisation outputs for an increase of approximately 15,000 oz. Larger shells were considered, however, the trade-off with DCF and strip ratio did not fit with Thor's desired results.

TABLE 15-7 SELECTED SHELL (37/RF1.02) BASE CASE WHITTLE OPTIMISATION SHELL OUTPUTS
Thor Explorations Ltd. – Segilola Gold Project

Discounted Cash Flow ($M)	Ore Tonnes Input to Processing (Mt)	Grade Input to Mill (g/t Au)	Strip Ratio
$205.2	3.09	4.28	15.3

The pit by pit optimisation results are summarised in Figure 15-2 and Table 15-8, with the maximum cash flow shell (34) highlighted in yellow, and the selected final shell (37) highlighted in green.

FIGURE 15-2 WHITTLE PIT BY PIT VALUE GRAPH

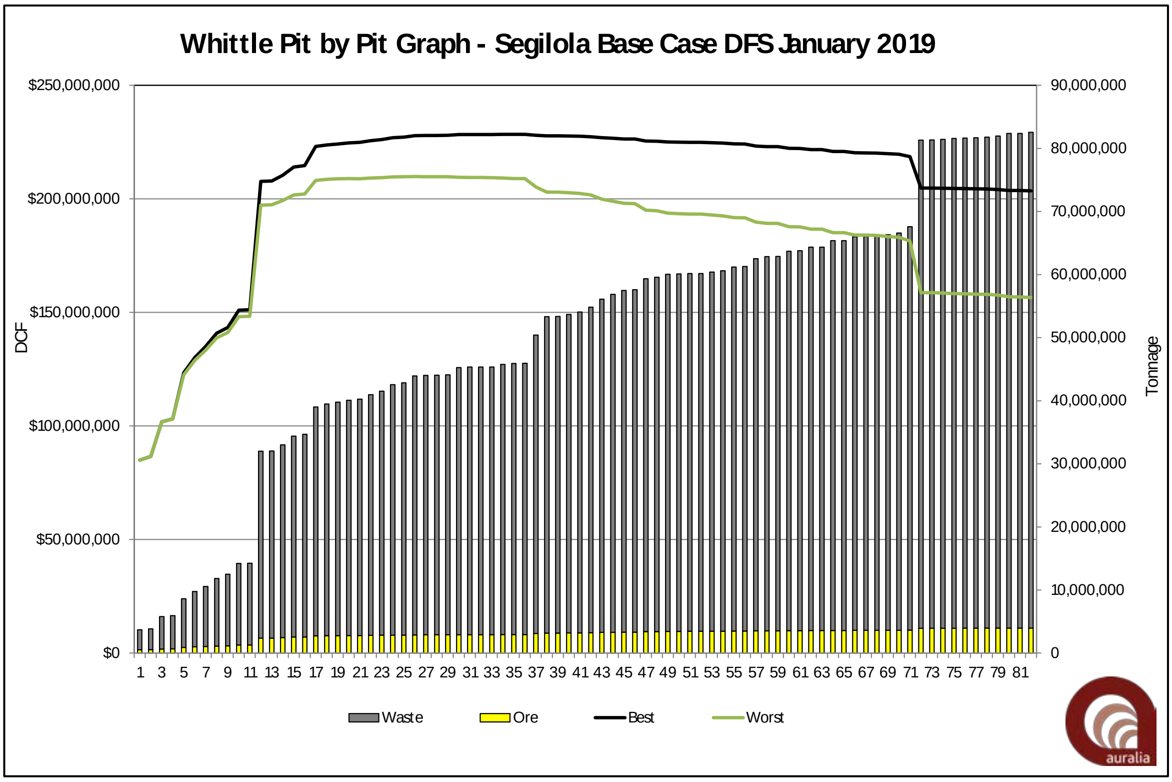
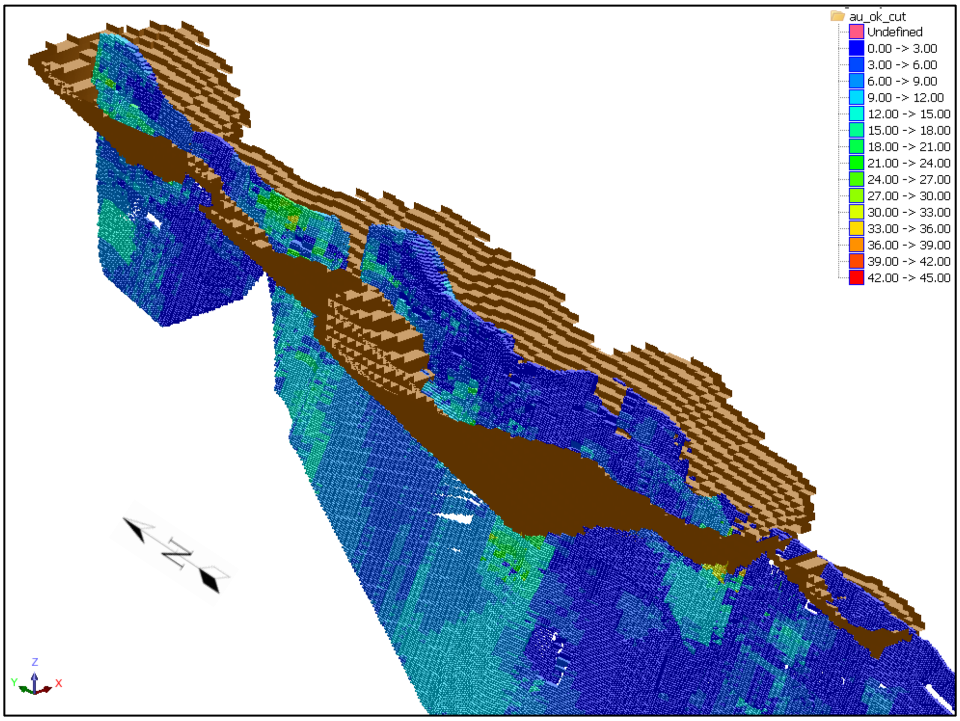


TABLE 15-8 BASE CASE WHITTLE OPTIMISATION SHELL OUTPUTS
Thor Explorations Ltd. – Segilola Gold Project

Final Pit	Revenue Factor	Tonnes input to mill (Mt)	Grade Input to Mill (g/t)	Recovered Ounces (MOz)	Waste tonnes (Mt)	Strip Ratio	Mining Cost ($M)	Processing Cost ($M)	Selling Cost ($M)	Total Cost ($M)	Revenue ($M)	Undiscounted Cash flow ($M)	DCF Best Case ($M)	DCF Worst Case ($M)
1	0.30	0.5	5.86	2.9	3.2	6.3	9.9	13.7	1.4	24.9	115.6	90.6	85.0	85.0
5	0.38	0.9	5.23	4.6	7.7	8.5	23.6	24.6	2.2	50.4	185.2	134.8	123.5	122.6
10	0.48	1.3	4.98	6.1	12.9	10.2	40.1	34.3	2.9	77.3	245.7	168.5	151.0	148.1
15	0.58	2.5	4.35	10.7	31.9	12.6	100.5	68.5	5.1	174.1	429.8	255.7	214.0	201.8
20	0.68	2.8	4.35	11.7	37.3	13.5	118.0	74.9	5.6	198.5	469.6	271.1	224.6	209.0
21	0.70	2.8	4.33	11.7	37.5	13.4	118.5	75.4	5.6	199.5	471.1	271.6	224.9	208.9
22	0.72	2.8	4.33	11.8	38.1	13.6	120.6	76.1	5.6	202.3	475.0	272.7	225.6	209.2
23	0.74	2.8	4.34	11.9	38.7	13.7	122.3	76.5	5.7	204.4	477.9	273.5	226.1	209.4
24	0.76	2.8	4.35	12.0	39.7	13.9	125.5	77.0	5.7	208.3	483.0	274.7	227.0	209.7
25	0.78	2.9	4.36	12.1	40.0	14.0	126.4	77.1	5.8	209.3	484.3	275.0	227.2	209.8
26	0.80	2.9	4.36	12.2	41.1	14.3	129.8	77.9	5.8	213.6	489.8	276.2	227.9	209.9
27	0.82	2.9	4.36	12.2	41.1	14.3	130.1	78.1	5.8	214.0	490.3	276.3	228.0	209.8
28	0.84	2.9	4.35	12.2	41.2	14.3	130.2	78.1	5.8	214.2	490.5	276.3	228.0	209.8
29	0.86	2.9	4.35	12.2	41.2	14.2	130.3	78.2	5.8	214.4	490.7	276.4	228.0	209.7
30	0.88	2.9	4.38	12.3	42.4	14.6	133.9	78.5	5.9	218.3	495.2	276.9	228.4	209.5
31	0.90	2.9	4.37	12.3	42.4	14.6	134.1	78.7	5.9	218.7	495.7	277.0	228.4	209.4
32	0.92	2.9	4.37	12.3	42.4	14.6	134.1	78.7	5.9	218.7	495.7	277.0	228.4	209.4
33	0.94	2.9	4.37	12.3	42.4	14.6	134.2	78.7	5.9	218.8	495.8	277.0	228.4	209.4
34	0.96	2.9	4.37	12.4	42.8	14.7	135.4	79.0	5.9	220.3	497.4	277.1	228.4	209.2
35	0.98	2.9	4.36	12.4	43.0	14.7	135.9	79.3	5.9	221.1	498.2	277.1	228.4	208.9
36	1.00	2.9	4.36	12.4	43.0	14.7	136.0	79.3	5.9	221.2	498.3	277.1	228.4	208.9
37	1.02	3.1	4.28	12.8	47.3	15.3	149.5	83.7	6.1	239.3	516.3	277.0	228.0	205.3
38	1.04	3.2	4.26	13.1	50.1	15.8	158.2	85.8	6.3	250.3	526.9	276.6	227.8	203.0
39	1.06	3.2	4.26	13.1	50.2	15.8	158.3	85.9	6.3	250.4	527.1	276.6	227.8	202.9
40	1.08	3.2	4.26	13.1	50.5	15.9	159.4	86.1	6.3	251.7	528.3	276.6	227.7	202.7
41	1.10	3.2	4.26	13.2	50.9	16.0	160.5	86.2	6.3	253.1	529.5	276.4	227.6	202.4
42	1.12	3.2	4.25	13.2	51.6	16.1	162.9	86.9	6.3	256.2	532.3	276.1	227.4	201.7
43	1.14	3.3	4.21	13.4	52.8	16.1	166.9	88.5	6.4	261.8	537.3	275.5	226.9	199.9
44	1.16	3.3	4.20	13.4	53.6	16.3	169.1	89.1	6.4	264.6	539.7	275.1	226.7	198.9
45	1.18	3.3	4.19	13.5	54.2	16.3	171.0	89.8	6.4	267.3	542.0	274.7	226.4	198.1
46	1.20	3.3	4.19	13.5	54.3	16.3	171.4	89.9	6.5	267.7	542.4	274.6	226.3	197.9
47	1.22	3.4	4.15	13.6	56.0	16.5	176.7	91.8	6.5	275.0	548.3	273.4	225.5	195.1
48	1.24	3.4	4.14	13.7	56.2	16.5	177.4	91.9	6.5	275.9	549.1	273.2	225.4	194.8
49	1.26	3.4	4.12	13.7	56.6	16.5	178.9	92.8	6.6	278.3	551.0	272.7	225.0	193.7
50	1.28	3.4	4.11	13.7	56.7	16.5	179.1	93.0	6.6	278.6	551.3	272.6	225.0	193.5
55	1.38	3.5	4.10	13.8	57.7	16.6	182.6	93.9	6.6	283.1	554.6	271.5	224.2	191.8
60	1.48	3.5	4.06	14.0	60.2	17.0	190.5	95.9	6.7	293.1	561.5	268.4	222.3	187.8
65	1.60	3.6	4.05	14.1	61.8	17.3	195.7	96.8	6.7	299.2	565.4	266.2	220.9	185.1
70	1.70	3.6	4.03	14.1	63.0	17.4	199.6	97.7	6.8	304.1	568.4	264.3	219.7	183.1
75	1.80	3.9	3.91	15.0	77.6	19.7	248.8	106.7	7.2	362.6	601.8	239.3	204.6	158.3
80	1.94	4.0	3.91	15.0	78.4	19.8	251.3	107.0	7.2	365.6	603.4	237.8	203.7	156.9

The selected final Whittle shell is shown in Figure 15-3, with ore displayed at greater than or equal to the 0.77 g/t Au economic cut-off grade.

FIGURE 15-3 WHITTLE RF1 SHELL, AU >=0.77G/T DISPLAYED (3D VIEW)



MINE DESIGN

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

- Ramps 22 m wide (dual lane for 90 t rear dump trucks) at a gradient of 1 in 9
- Minimum mining width (pit floor) of approximately10 m
- Minimum cutback width of approximately 50 m
- Walls designed following the recommended geotechnical parameters

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.

A final pit design has been completed based on the selected final pit shell. The final pit design, shown in Figures 15-4 to 15-6, has the following approximate dimensions:

- Length 1,600 m

- Width 140 m to 430 m
- Depth 55 m to 210 m
- Area 43 ha

FIGURE 15-4 DFS FINAL PIT DESIGN, PLAN VIEW

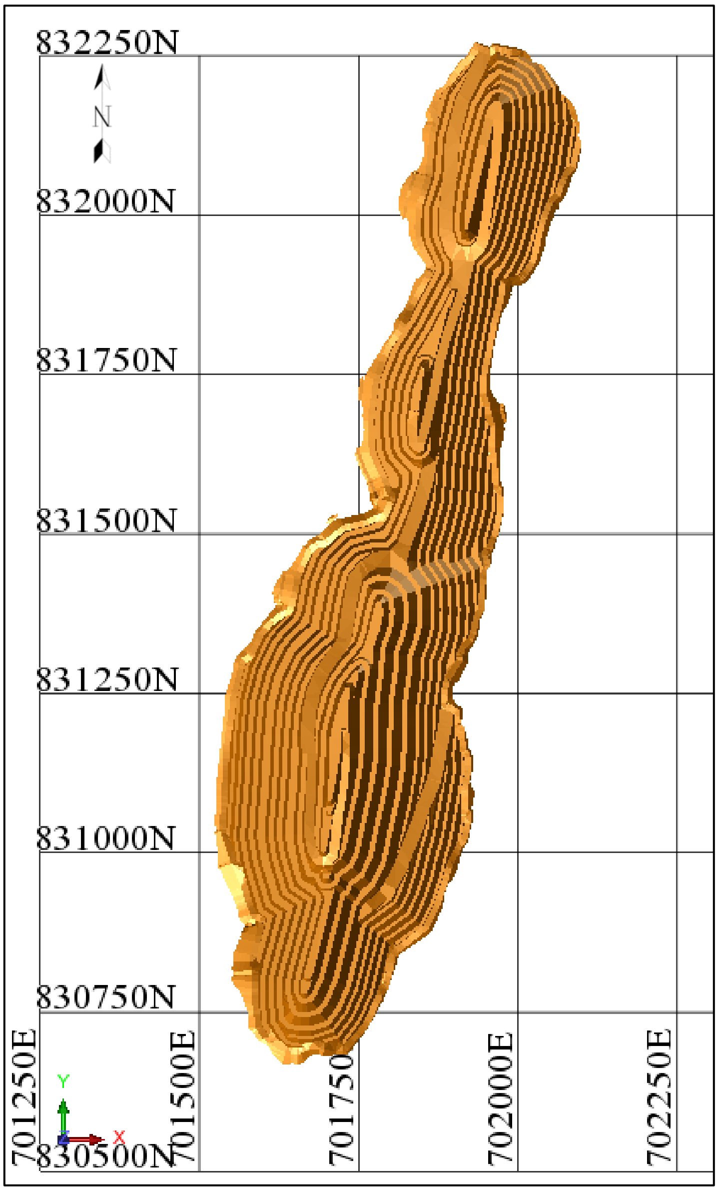


FIGURE 15-5 DFS FINAL PIT DESIGN, SIDE VIEW

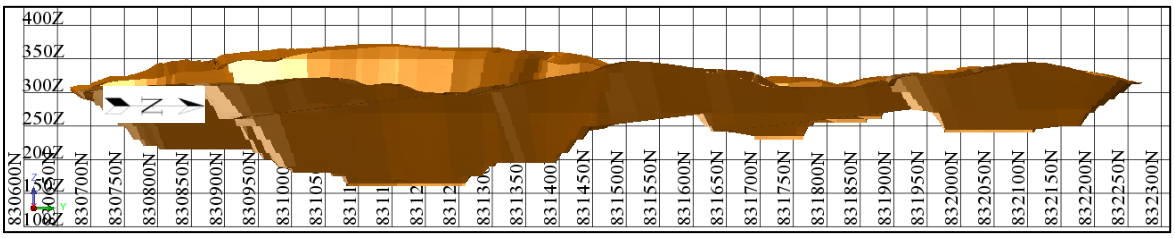
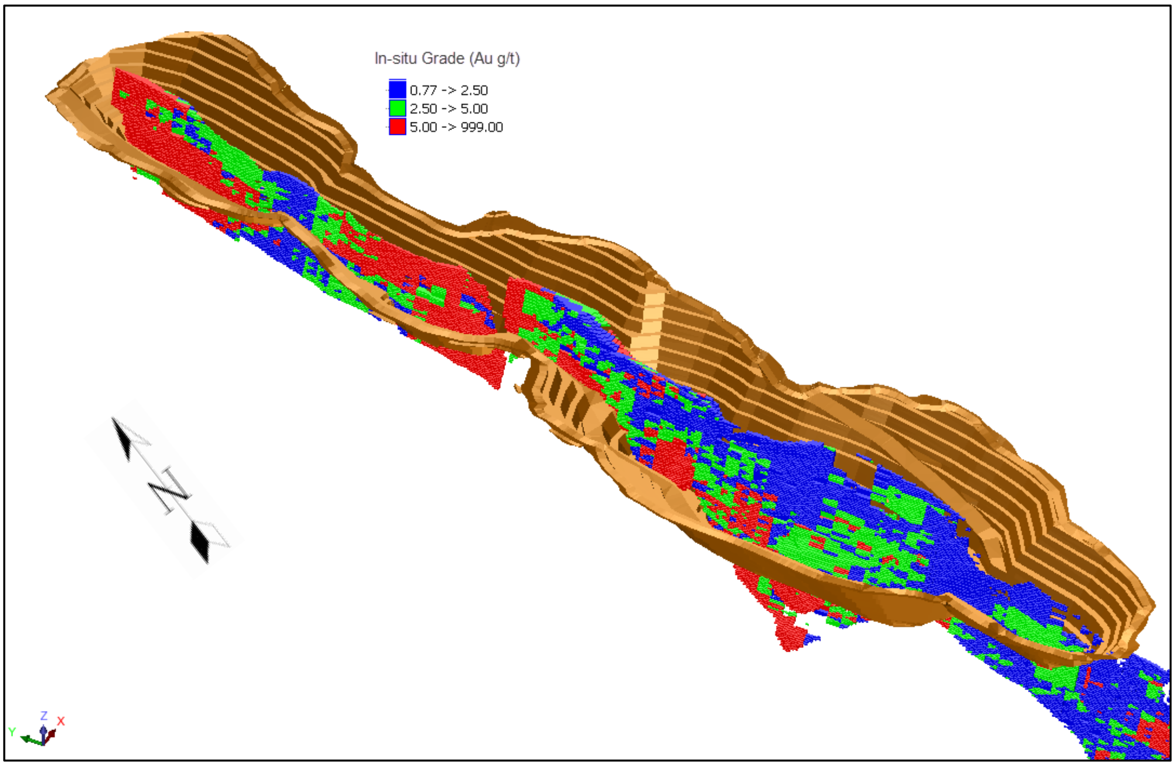


FIGURE 15-6 DFS FINAL PIT DESIGN, 3D VIEW SHOWING INDICATED ORE >=0.77G/T AU



INTERIM STAGE PITS

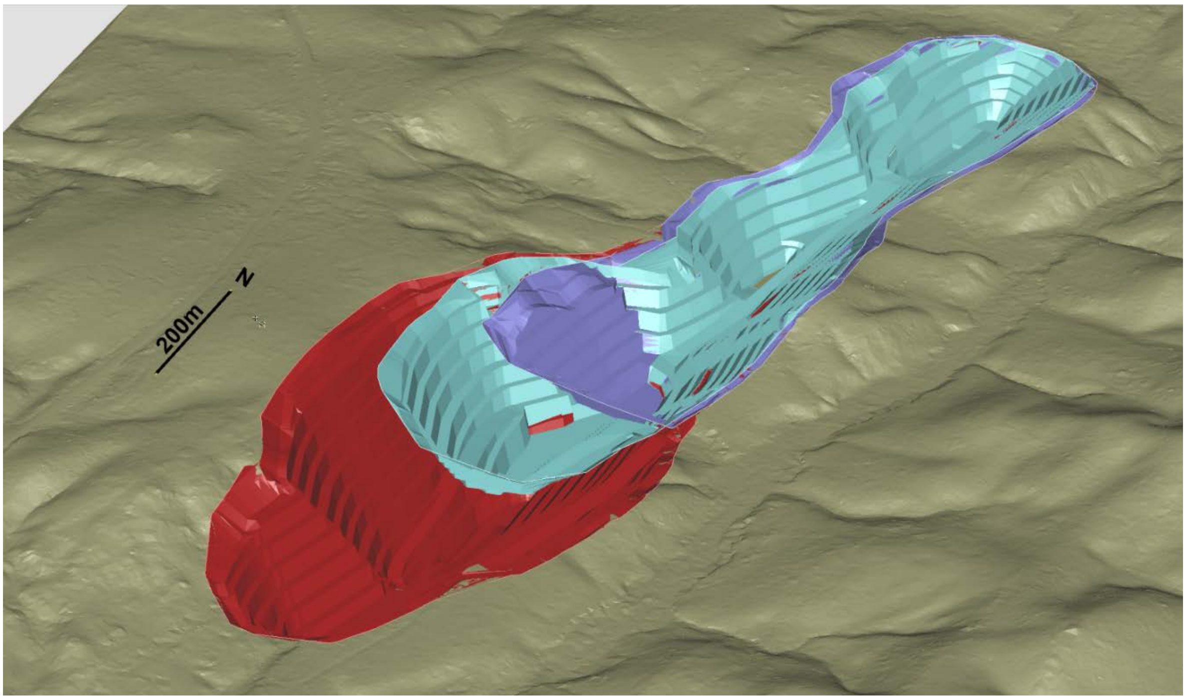
To maximise the DCF of the Project, it is necessary to mine the pit in a series of stages.

The interim stage pits to be developed were selected using the order of development of shells in Whittle as a guide (highest return pit shells targeted first). This resulted in two interim pits (Stage 1 and Stage 2), which will be mined before developing the final pit. The Stage 2 pit is mined in two parts – the main pit is developed with a ramp on the eastern side providing access down to the 250 RL. This ramp is then excavated to uncover additional ore while waste is being mined in the Stage 3 pit. Below the 250 RL access from the Stage 2 pit is possible to the north using the Stage 1 ramp.

The stage and final pit designs are shown in Figure 15-7.

- Purple = Stage 1
- Light Green = Stage 2
- Red = Stage 3/Final Pit.

FIGURE 15-7 INTERIM AND FINAL STAGE PIT DESIGNS



The overall in-situ inventories for each stage pit are summarised in Table 15-9.

TABLE 15-9 IN-SITU STAGE PIT INVENTORIES
Thor Explorations Ltd. – Segilola Gold Project

	Ore Tonnes (Mt)	Grade (g/t Au)	Waste (Mt)	Total Tonnes (Mt)	Strip Ratio
Stage 1	1.0	5.6	15.6	16.6	15.1
Stage 2	0.7	3.7	11.0	11.7	16.5
Stage 3	1.3	4.5	23.6	24.9	17.9

MINERAL RESERVE COMPARISONS

The Mineral Reserve announced as part of the PFS on the Project in 2017 is shown in Table 15-10 along with the current DFS estimate.

TABLE 15-10 MINERAL RESERVE COMPARISON WITH PREVIOUS ESTIMATE
Thor Explorations Ltd. – Segilola Gold Project

Mineral Reserve Estimation	Category	Economic Cut-off (g/t Au)	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
PFS (2017)	Probable	0.64	3,345	4.2	448
DFS (current)	Probable	0.77	3,002	4.2	405

16 MINING METHODS

MINING

The Segilola Gold Project will be operated primarily as a contract mining operation with all mobile equipment to be provided, operated, and maintained by a mining contractor(s) over the life of the Project. Conventional open pit mining methods using excavator and truck fleets will be used. Thor will provide fixed installations including contractor area establishment, workshops, offices, and first aid facilities.

There are four well defined weathering horizons present, allowing the material to be mined according to classification as residual soil, oxide rock, transitional rock, and fresh rock. The vast majority of material is fresh rock.

Drilling rigs operated by the contractor will be Epiroc T45/50 class top-hammer drills with the capability to drill holes from 89 mm to 140 mm in diameter. Ore blasting will be done on five metre benches, while waste blasting will be conducted on 10 m benches where possible. A set of blast patterns ranging in powder factor from 0.43 kg/m^3 to 1.07 kg/m^3 will be applied depending on the material type.

Narrow ore zones will require the implementation of careful blasting techniques to minimise dilution and maximise ore recovery, namely the use of "Blastmaster" bench planning to optimise blast shapes/geometry and sequencing, choke firing where possible to minimise ore movement, blast initiation timing to control movement, the use of blast movement indicators, and a refinement of blast design parameters based on results obtained.

Blasting will be done with pumped emulsion as water is expected in the blast holes.

The mining contractor will use 200 t hydraulic excavators (12.5 m^3 bucket capacity) and 90 t payload rear dump trucks. Excavators will be in backhoe configuration to allow flexibility for mining both ore and waste.

Waste mining will be performed using a double-bench method, where possible, in order to reduce ancillary equipment costs, taking a full 10 m cut with each pass. Ore mining will be on two 2.5 m flitches to reduce the chance of dilution whilst still providing a sufficient bench height for the required production rates.

Grade control drilling and mining of the ore will only take place during dayshifts so that the Thor technical department can guide excavations to minimize ore dilution and loss.

The production equipment will be supported by ancillary equipment such as dozers and graders to achieve level graded floors, smooth out blast heave, and provide face clean-ups. Water carts will be employed for dust suppression, using water sourced from pit dewatering and other storage dams when necessary.

A total of 75% of the Run of Mine (ROM) is planned to be re-handled due to the substantial stockpiling planned. This will be done using a front-end loader (FEL), such as a Caterpillar 988, with 6.4 m^3 bucket capacity. The ROM pad and stockpile area will be located adjacent to the processing plant at the southern end of the west waste dump. All waste rock from the pre-strip period and initial months of mining will be used to create the flat area for the ROM pad and ore stockpiles. The front-end loader may need to be supplemented by a truck from the mining fleet when ROM pad hauls exceed 200 m.

Open pit dewatering requirements have been assessed based on limited groundwater information. The assessment has estimated the inflow into the open pit from groundwater sources could range from 60 m^3/day to 600 m^3/day. The contractor costed the provision of 3 Sykes HH160i pumps, which can each provide 350 m3/h capacity with a head of 100 m).

Sub-horizontal drains will be used to promote passive drainage of the pit walls, typically installed from benches in the pit walls, with a horizontal spacing of 10 m to 30 m between drains. Drilled diameters will be 75 mm to 100 mm, with a 50 mm diameter high-density polyethylene (HDPE) of unplasticised polyvinyl chloride (uPVC) slotted screen installed.

If significant concentration of groundwater seepage and inflow is observed in areas of the open pit, for example in more fractured zones associated with steeply dipping structures, it may be possible to install vertical dewatering wells pumped by submersible pumps. Such wells would likely have a yield of a few litres per second and would have to be located based on geological mapping and seepage observations during mining.

EQUIPMENT

The contractor will provide all the direct mining services, including in pit dewatering and local training.

Mining production equipment operated by the contractor will consist of 200 t hydraulic backhoe excavators with 12.5 m^3 bucket capacity (Liebherr R9200), and a fleet of 90 t payload rear dump trucks (Caterpillar 777G). This class of equipment was selected to meet the waste production requirements whilst still allowing for flexibility and selectivity in mining the ore zones. The use of two separate fleets allows for smoothing of ore delivery as much as possible by pre-stripping waste ahead of ore mining, a requirement due to the high strip ratio.

Table 16-1 shows a full list of plant (except the bulk explosives delivery truck) to be mobilised to site by the mining contractor at the start of the Project, and represents the peak equipment numbers over the mine life:

TABLE 16-1 LIST OF MINING EQUIPMENT, PEAK REQUIREMENT
Thor Explorations Ltd. – Segilola Gold Project

Plant & Equipment	Quantity
Epiroc T45/T50 Drill Rig	5
Liebherr R9200 Excavator	2
CAT 777G Rigid Truck	11
CAT D9R Bulldozer	2
CAT 988H Wheel Loader	2
CAT 16M Grader	1
Komatsu PC300 Excavator/Rock Breaker.	1
CAT 740B ADT (if required)	4
Tipper Truck	2
Water Cart	1
Ancillary Equipment	4
TOTAL	**35**

In order to meet the production requirements in both ore and waste, two excavators and a fleet of 11 trucks (including spares) will operate for the first 27 months of mining, and will be reduced to a single excavator for the remaining 19 months. Both excavators have been allocated a planned average productivity of 1,550 tonnes per hour (tph), at an average of 460 production hours per month.

Additional equipment to be supplied by Thor will include the following:

- Grade control drill rigs (under a separate contractor arrangement).
- Crane for machine assembly.

GRADE CONTROL

Over the life of the open pit mine, a continuous program of reverse circulation drilling on a close-spaced grid together with fire assaying of the mineralised intercepts will be implemented in order to define the boundaries between high grade, low grade and waste material.

The grade control drilling pattern designed by Thor is as follows:

- 8 m burden (along strike)
- 5 m spacing (across strike)
- 20 m vertical depth
- 60° drill hole dip
- direction of drilling normal to the strike of the mineralisation
- sampling every 1 m of hole depth using a three-stage riffle splitter
- assays will be carried out on site

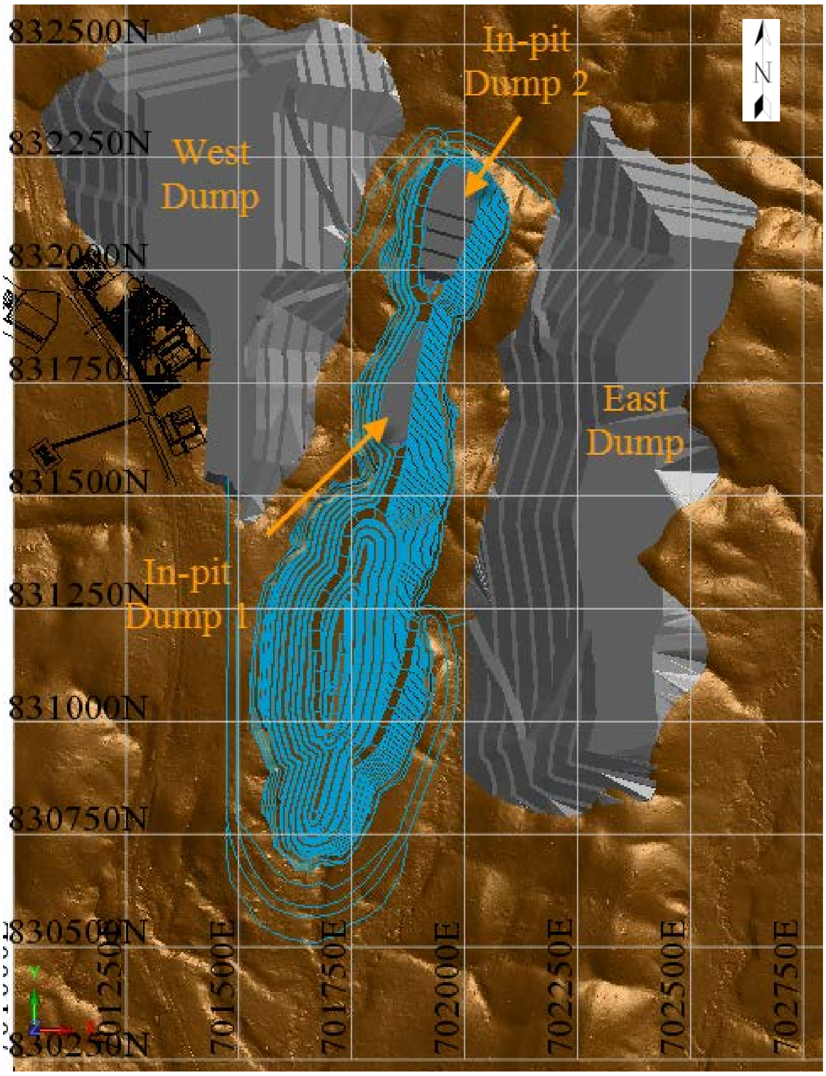
Grade control activities will be supervised and by Thor personnel. Appropriate industry-standard QA/QC procedures will be adopted and data will be captured within a grade control database with the ore mark-outs and reconciliations being managed within modern mining software. Grade control results will be reconciled with the block model to test the performance of the reserve model.

WASTE DUMPS

Waste rock dump designs have been completed, with sufficient capacity to store all of the waste material from the final pit (20.5 Million BCM in-situ). The waste rock dumping strategy consists mostly of out-of-pit dumping, with the two main dumps located immediately to the east and west of the pit. In-pit dumping has also been implemented to reduce haulage costs and minimise the disturbance footprint, where possible, however, the opportunity for this is limited due to the need to maintain haulage access through the pit for most of the mine life. A swell factor of approximately 25% has been applied for the waste rock dump designs.

The locations of the waste dumps are shown in grey in Figure 16-1.

FIGURE 16-1 WASTE DUMPS, PLAN VIEW



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

- Batter angle 18°
- 10 m wide berms every 10 m vertical
- Overall slope approximately 14°

The dumps have been designed in the DFS as final landforms to determine the required disposal capacity. During the construction of the dumps the slope angles will be the angle of repose of the waste rock, which is expected to be approximately 35°. When the designed toe position is reached for the final elevation on each section of the dump, dozers will push the batters down to the final landform design angle in preparation for progressive rehabilitation.

The capacities of each waste dump are as follows:

- Western out of pit dump: 12.5 Mm^3

- Eastern out of pit dump: 12.6 Mm^3
- In-pit dumps: 0.9 Mm^3

The total waste dump capacity is approximately 26 Mm^3, which is sufficient for the estimated LOM requirement of 25.6 Mm^3 of waste rock inclusive of swell. Oxide and transitional waste accounts for approximately 30% of the total waste tonnage. Waste dump construction at the operational level will take into account the physical properties of the material to maximise stability and minimise the potential for erosion.

LIFE OF MINE PLAN

The contractor estimated production rates from first principles, benchmarking against their existing and previous operations.

Relevant assumptions include:

- Average production rate of 1,550 tph per 200 t excavator (Liebherr R9200).
- 7.7 working hours per shift on average for each excavator over the Project life. Weather delays were not specifically accounted for by month in the schedule.
- Two primary excavators will be used for the first 27 months of mining, dropping to a single fleet for the remaining 19 months

Production will commence in the Stage 1 pit towards the northern end of the deposit, targeting high grade ore close to surface with mining progressing generally from north to south. The Stage 2 pit commences after nine months as ore supply from the Stage 1 pit is well established. From Month 24, the eastern access ramp in the Stage 2 pit is mined out to provide access to further ore, with the Stage 2 pit completed by Month 28. Stage 3 (the final pit design) commences in Month 15 upon the completion of Stage 1, and the final pit is completed in Month 46.

The ore was split into the following stockpile grade bins:

- Indicated high grade: greater than 5 g/t Au;
- Indicated medium grade: 2.5 g/t to 5 g/t Au;
- Indicated low grade: 0.77 g/t Au (economic cut-off grade) to 2.5 g/t Au;
- Inferred: All inferred material regardless of grade.

Inferred material represents approximately 5% of the overall ore inventory (approximately 150 kt), and has been excluded from the revenue estimate.

A ramp up period for the processing plant has been incorporated as follows:

- Month 1 (Project Month 16): 7.5% capacity;
- Month 2 (Project Month 17): 52.5% capacity;
- Month 3 (Project Month 18): 100% capacity.

There is a pre-strip period of one month in order to create initial stockpiles for processing plant feed. No further pre-stripping is required in order to meet the schedule objectives.

Prior to the pre-strip, the stage one mining area and stockpile will be cleared, haul roads established and topsoil stripped and stored appropriately for use in rehabilitation.

Key outcomes of the LOM schedule are as follows:

- An average of 625,000 tpa mill feed target is achieved on a monthly basis over the life of the mine.
- Mining operations last for 46 months, with processing continuing for an additional 12 months.
- Maximum overall stockpile sizes are as follows:
    - High grade ore: 100 kt
    - Medium grade ore: 240 kt
    - Low grade ore: 450 kt
    - Inferred material: 150 kt

The schedule outputs are summarised in Table 16-2 to 16-4 and Figures 16-2 to 16-11.

TABLE 16-2 LIFE OF MINE PRODUCTION
Thor Explorations Ltd. – Segilola Gold Project

Calendar Year	Unit	Total	2020	2020	2020	2020	2021	2021	2021	2021	2022	2022	2022	2022	2023	2023	2023	2023
Quarter			1	2	3	4	1	2	3	4	1	2	3	4	1	2	3	4
Total Ore Mined	kt	3,002.4	32.1	121.0	213.0	314.3	218.2	201.5	83.2	181.5	184.8	240.2	26.5	45.3	75.1	151.6	387.5	526.6
Gold Grade, Mined	g/t Au	4.2	3.66	4.8	6.6	5.0	4.4	3.9	1.7	2.1	4.4	3.9	1.8	2.0	3.0	3.2	3.9	4.8
Total Waste Mined	kt	48,991	1,137	4,149	4,222	3,960	4,081	4,132	4,378	4,210	4,093	2,996	2,197	2,186	2,141	2,001	1,851	1,256
Total Material Mined	kt	51,993	1,169	4,270	4,435	4,274	4,299	4,334	4,462	4,392	4,278	3,236	2,223	2,231	2,216	2,153	2,238	1,782
Strip Ratio	W:O	16.3	35.4	34.3	19.8	12.6	18.7	20.5	52.7	23.2	22.1	12.5	82.9	48.3	28.5	13.2	4.8	2.4

TABLE 16-3 MINING PHYSICALS
Thor Explorations Ltd. – Segilola Gold Project

Mine Production Totals		Total/Ave
Material Mined		
Indicated	BCM	1,093,906
	Tonnes	3,002,369
	g/t	4.20
	g	12,615,267
Inferred Resource	BCM	56,346
	Tonnes	151,674
	g/t	5.38
	g	816,416
Waste	BCM	20,485,558
	Tonnes	48,839,038
Total Material Mined	BCM	21,635,810
	Tonnes	51,993,082
Material by Stage		
Stage 1	BCM	7,135,970
	Tonnes	16,643,054
Stage 2	BCM	4,840,290
	Tonnes	11,711,453
Stage 3	BCM	9,659,550
	Tonnes	23,638,575
Material Mined by Grade		
High Grade Ore	t	941,356
	g/t	8.45
Medium Grade Ore	t	947,564
	g/t	3.20
Low Grade Ore	t	1,113,449
	g/t	1.47
Inferred Resource Mined	t	151,674
	g/t	5.38

TABLE 16-4 PRODUCTION BY BENCH AND PUSHBACK
Thor Explorations Ltd. – Segilola Gold Project

Location/Material	Stage 1 Waste		Stage 1 Ore		Stage 2 Waste		Stage 2 Ore		Stage 3 Waste		Stage 3 Ore	
Mining Month	**Bench Toe**	**Tonnes**	**Bench Toe**	**Tonnes**	**Bench Toe**	**Tonnes**	**Bench Toe**	**Tonnes**	**Bench Toe**	**Tonnes**	**Bench Toe**	**Tonnes**
0	325	1,111,534	325	57,724								
1	315	1,272,544	315	43,939								
2	315	1,454,499	315	36,229								
3	295	1,427,427	295	35,613								
4	285	1,439,263	285	62,500								
5	275	1,389,322	275	63,248								
6	275	1,402,819	275	78,081								
7	270	1,386,008	270	123,041								
8	280	1,274,949	280	135,501								
9	290	635,239	290	42,235	335	677,474						
10	270	640,774	270	63,986	325	704,175	325	585				
11	280	633,415	280	62,566	320	690,911	320	5,070				
12	270	677,369	270	71,560	315	743,859	315	5,070				
13	250	607,313	250	87,059	310	688,189	310	6,183				
14	230	259,076	230	68,221	295	740,915	295	16,012	355	421,923		
15					295	704,058	295	15,312	340	719,370		
16					280	733,823	280	17,803	330	751,437	330	189
17					285	707,110	285	20,660	310	726,359	310	1,411
18					275	714,353	275	37,056	320	748,949	320	2,460
19					265	712,348	265	32,768	305	739,214	305	5,902
20					255	663,301	255	64,649	295	727,772	295	178
21					245	655,265	245	67,624	300	720,056	300	2,833
22					220	623,102	220	93,477	305	708,662	305	7,918
23					220	449,565	220	8,521	300	889,136	300	3,011
24					235	319,860	235	54,279	295	1,102,983	295	17,489
25					230	234,492	230	105,344	290	1,012,698	290	6,782
26					205	259,272	205	116,975	290	742,103	290	7,973
27					200	19,795	200	2,198	280	724,524	280	3,516
28									280	738,959	280	14,124
29									275	718,710	275	4,170
30									260	734,603	260	12,807
31									265	734,296	265	16,865
32									265	692,946	265	18,144
33									260	751,048	260	17,907
34									260	708,209	260	21,181
35									255	720,410	255	28,054
36									225	707,345	225	30,548
37									230	650,217	230	34,383
38									235	700,645	235	54,205
39									220	646,657	220	66,863
40									230	669,252	230	87,706
41									220	607,455	220	124,545
42									210	571,652	210	177,432
43									200	565,936	200	184,481
44									185	493,370	185	232,840
45									160	174,545	160	131,219
Totals		**15,611,551**		**1,031,503**		**11,041,867**		**669,586**		**22,321,441**		**1,317,134**

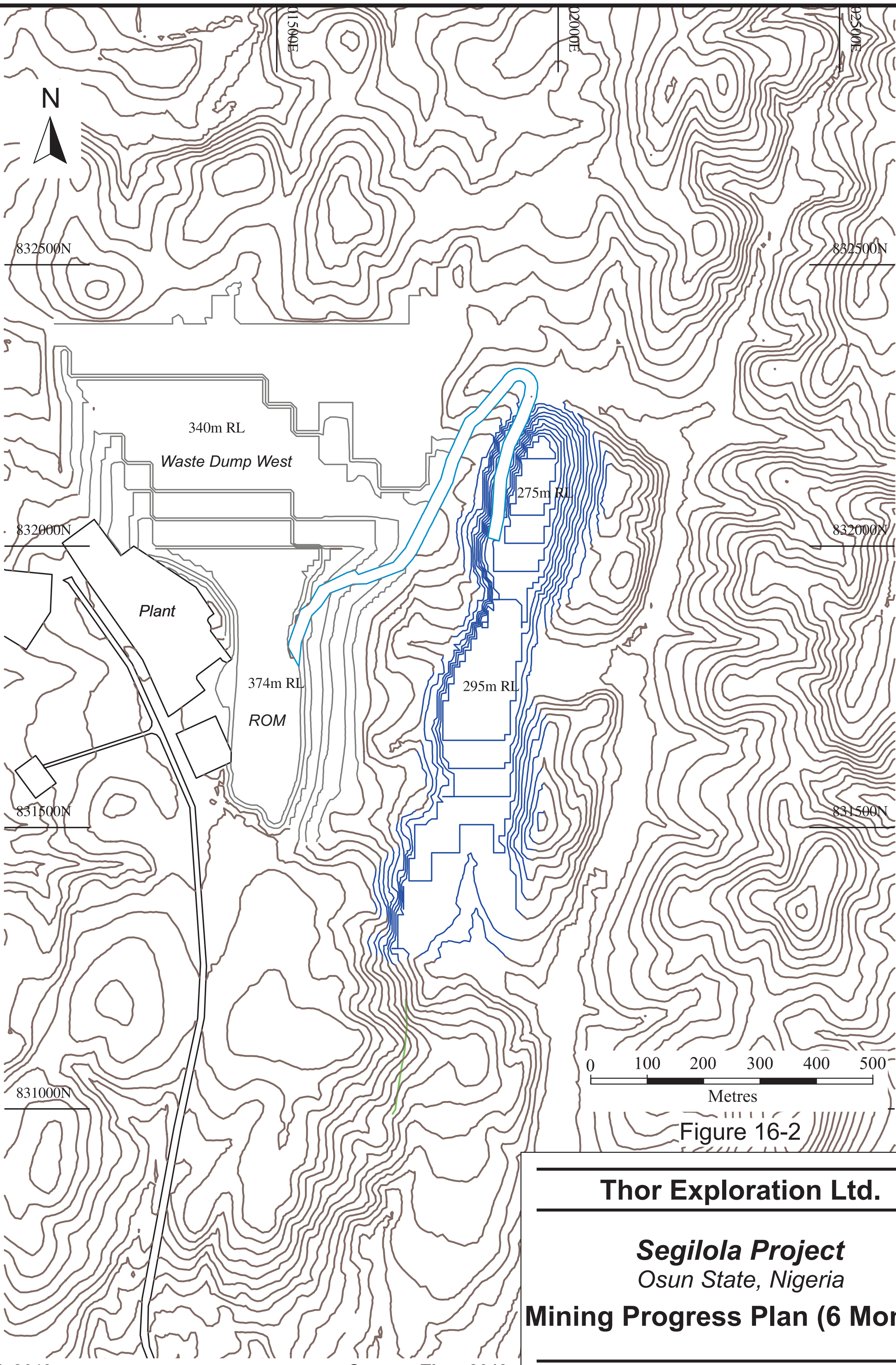


Figure 16-2

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Mining Progress Plan (6 Months)

March 2019

Source: Thor, 2019.

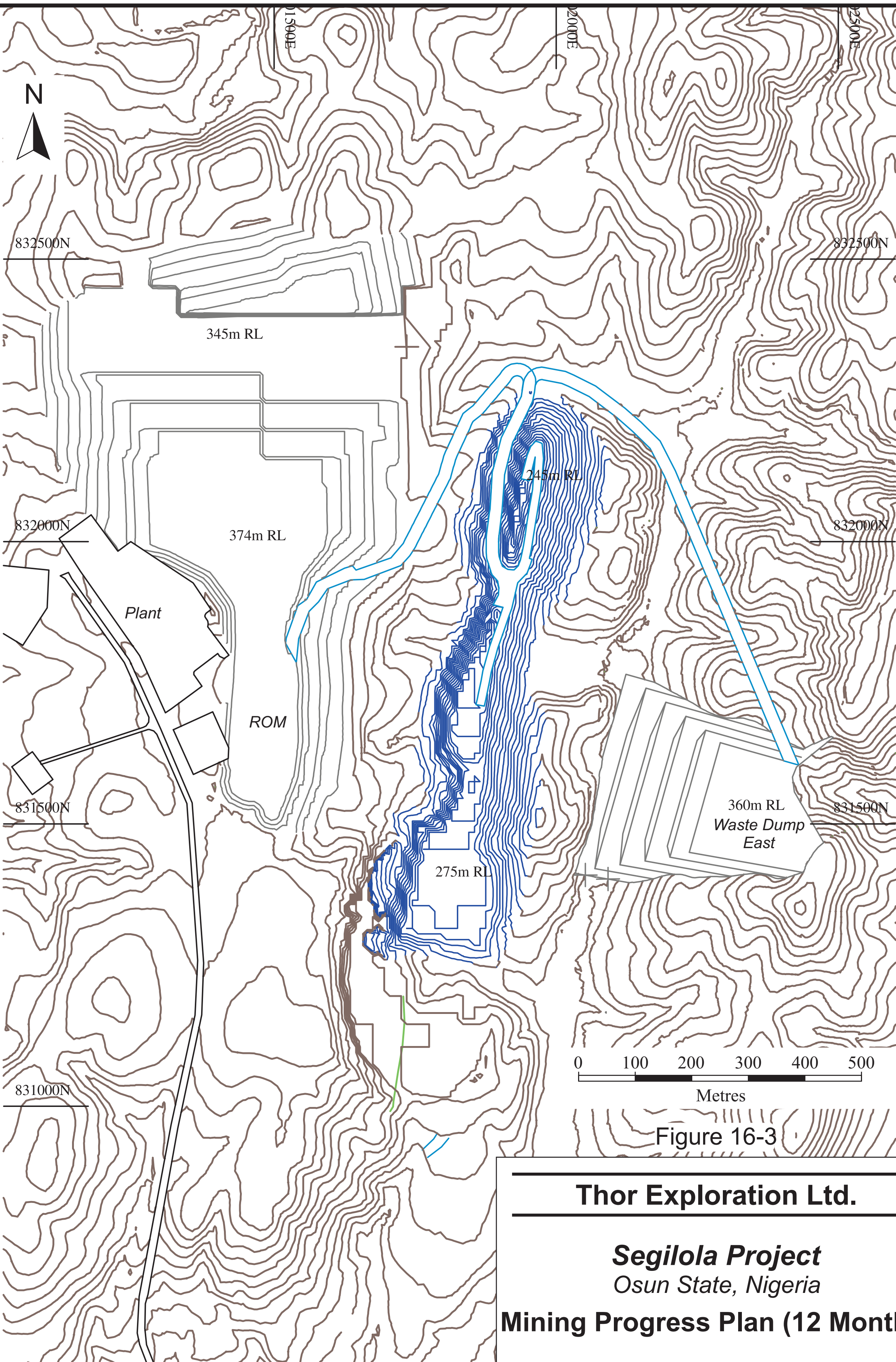


Figure 16-3

Thor Exploration Ltd.

Segilola Project
Osun State, Nigeria

Mining Progress Plan (12 Months)

March 2019

Source: Thor, 2019.

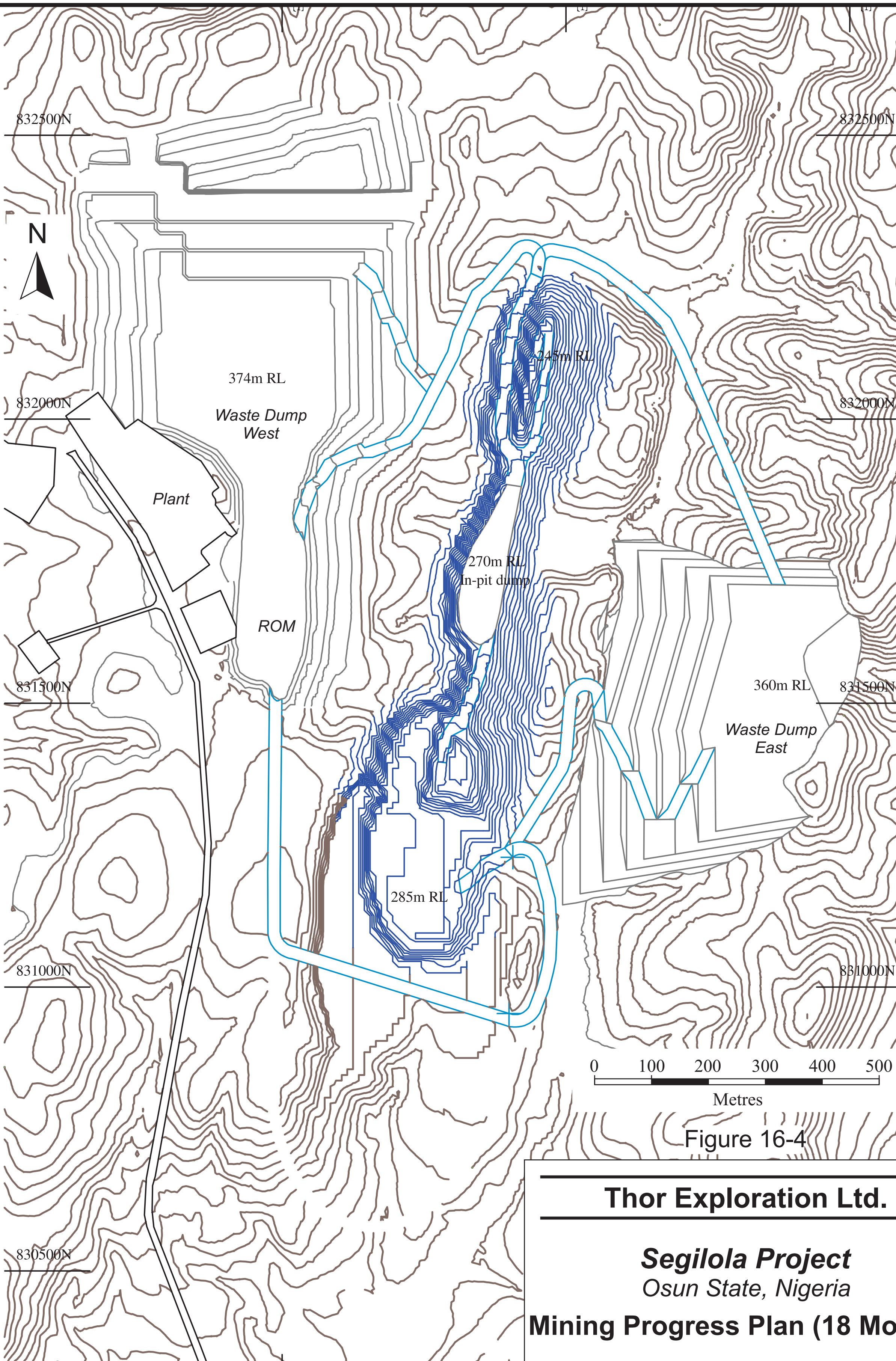


Figure 16-4

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Mining Progress Plan (18 Months)

March 2019

Source: Thor, 2019.

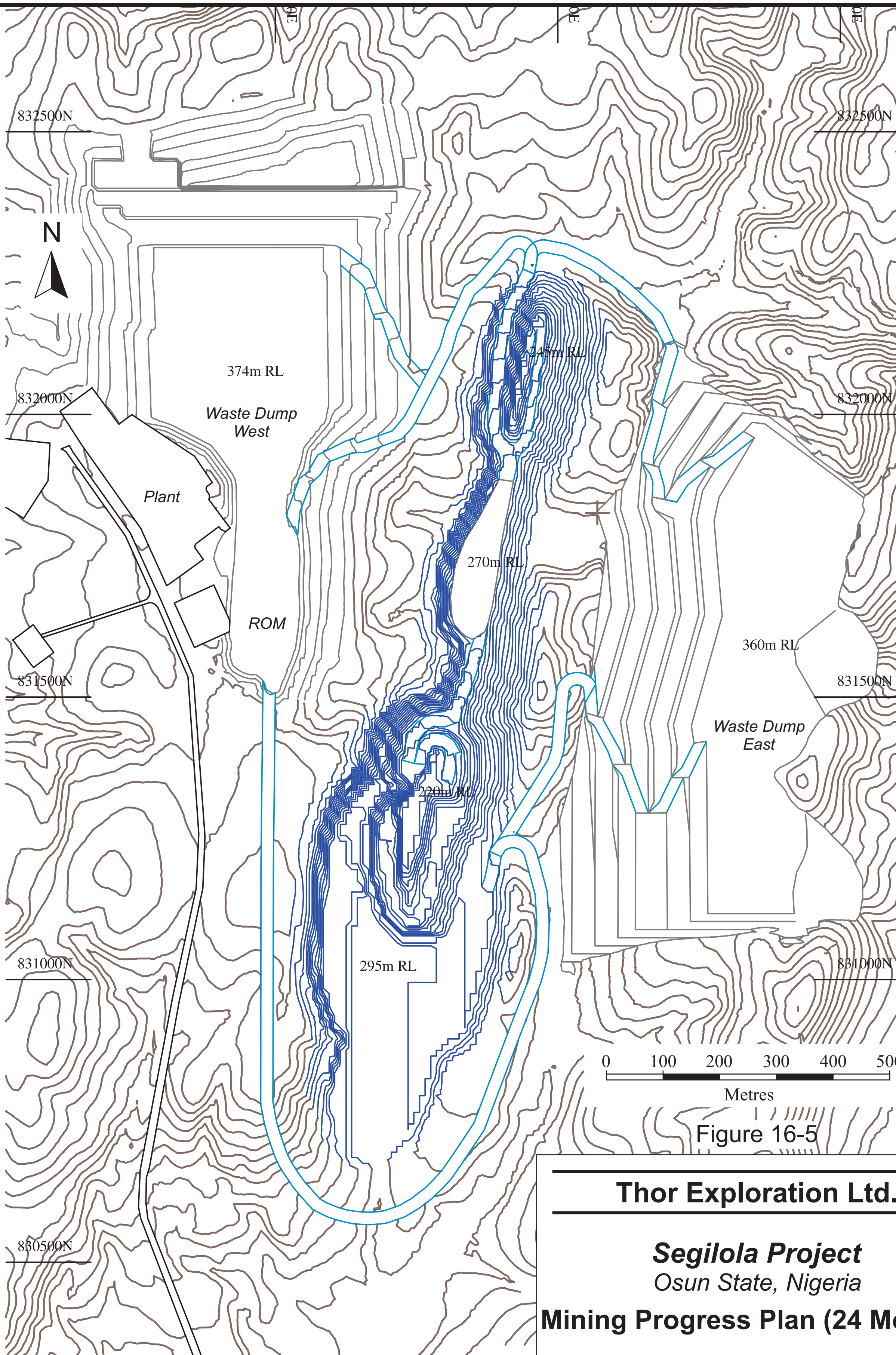


Figure 16-5

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Mining Progress Plan (24 Months)

March 2019

Source: Thor, 2019.

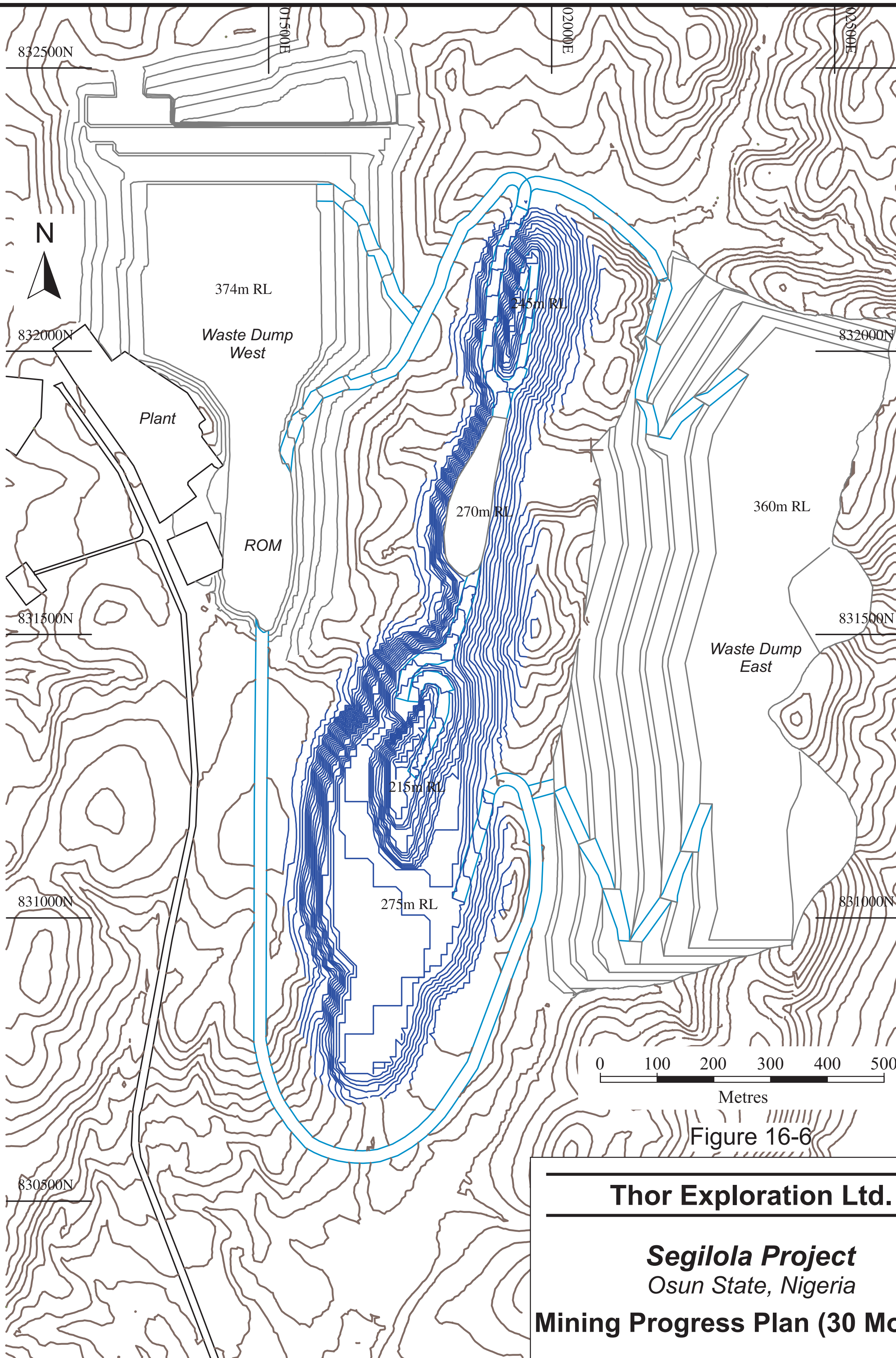


Figure 16-6

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Mining Progress Plan (30 Months)

March 2019

Source: Thor, 2019.

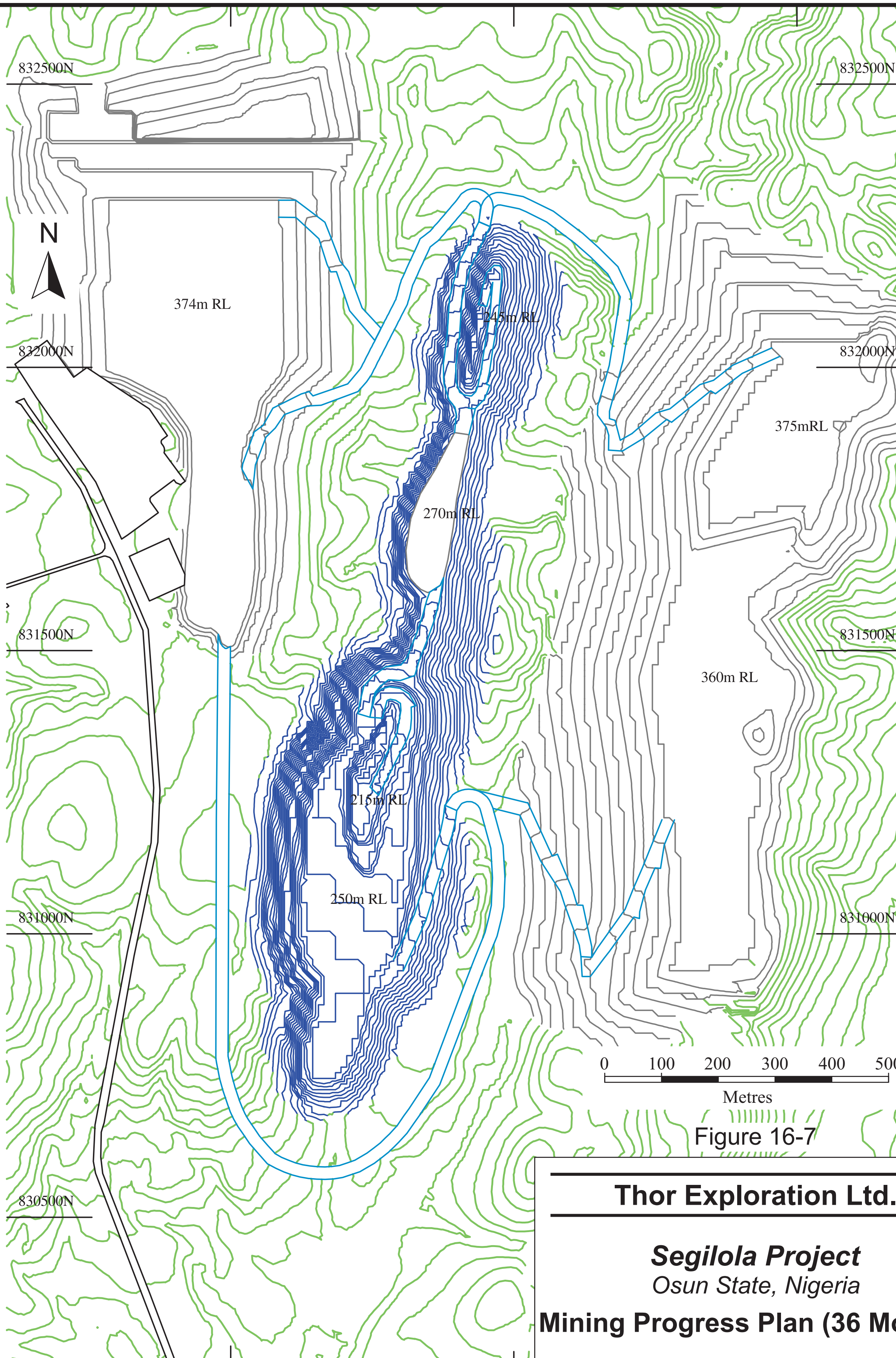


Figure 16-7

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Mining Progress Plan (36 Months)

March 2019

Source: Thor, 2019.

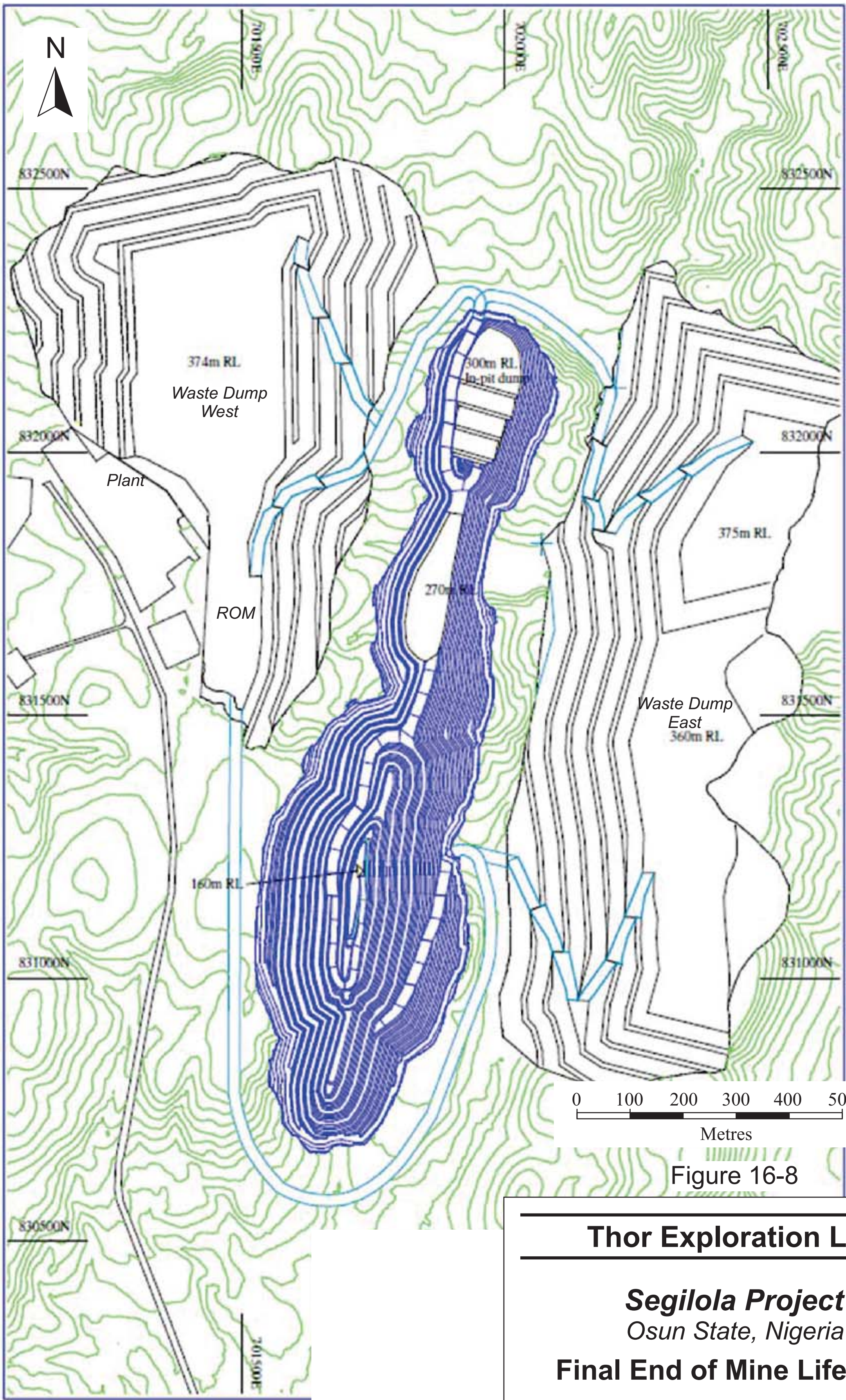


Figure 16-8

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Final End of Mine Life Plan

March 2019

Source: Thor, 2019.

FIGURE 16-9 MATERIAL MINED (BCM)

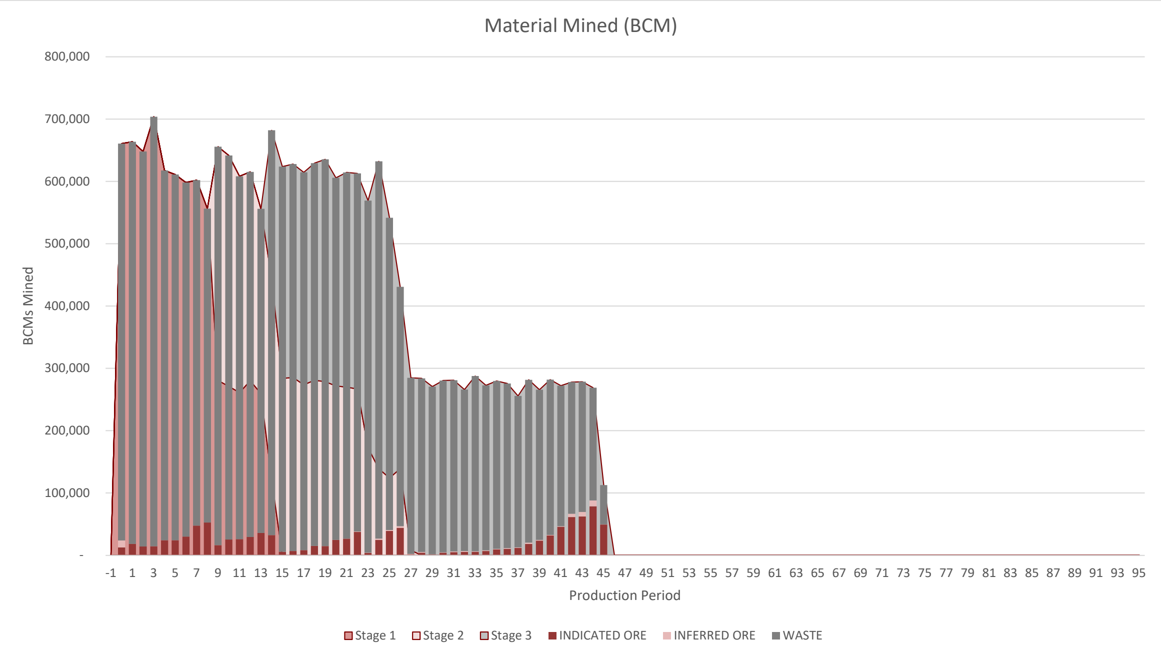


FIGURE 16-10 MATERIAL MINED – TONNES

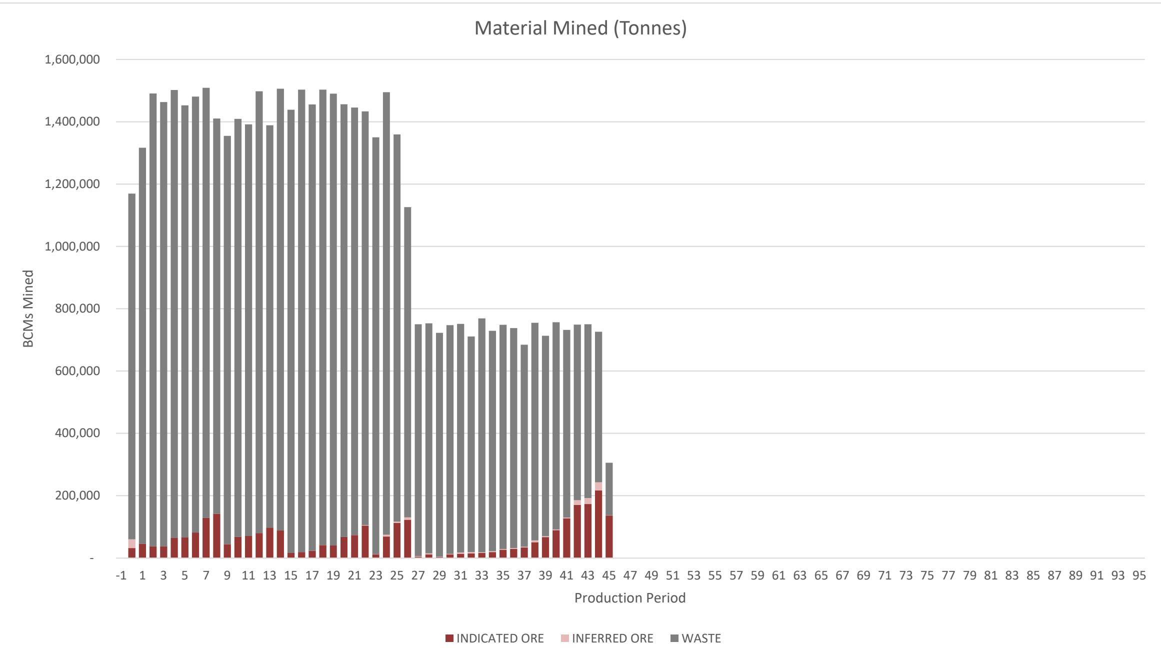
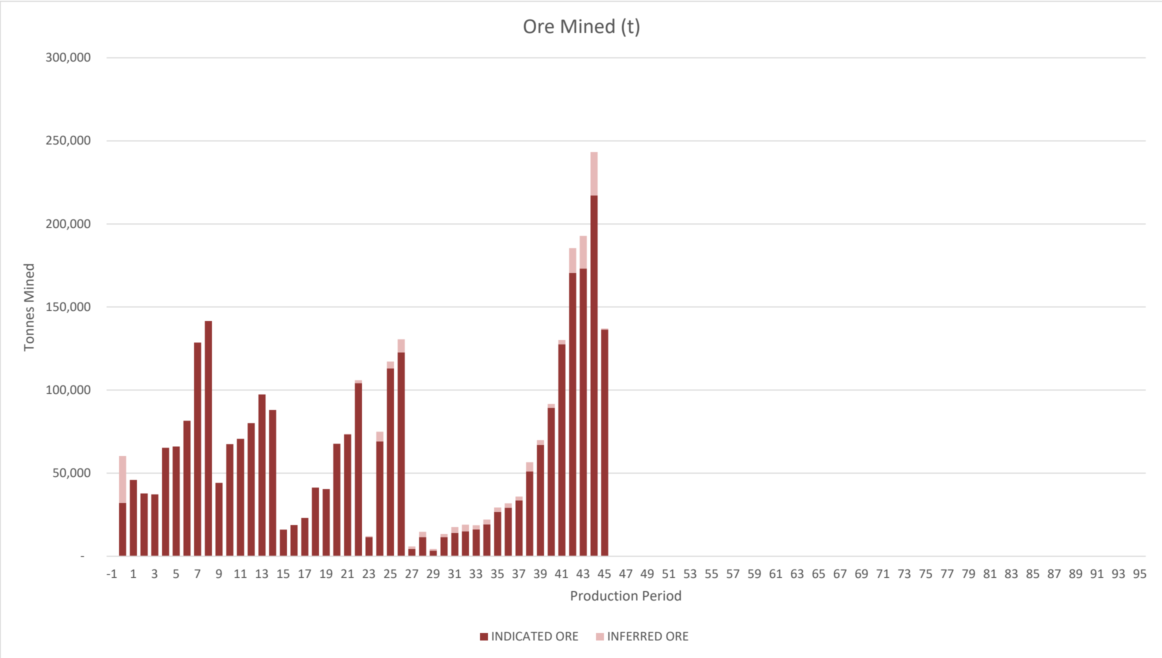


FIGURE 16-11 ORE MINED - TONNES



17 RECOVERY METHODS

PROCESS DESIGN

The process plant design for the Segilola Gold Project is based on a robust metallurgical flow sheet designed for optimum recovery with minimum operating costs. The flow sheet is based upon well proven and conventional technology used in the industry.

The plant is designed to operate 365 day/year, 24 hours/day, with a design utilization of 91.3%, for a nominal ore throughput of 81.3 tph and design ore throughput of 84.5 tph. Production throughput utilises 86.6% of design utilization, resulting in an average LOM annual production of 625,000 tpa.

The process plant was initially designed for a ROM throughput of 500,000 tpa. During the DFS study, to better align with the mine design and optimise project economics, the process plant throughput capacity was increased to 650,000 tpa. This increase in size of the processing facility also allows for operating flexibility and future expansion capacity.

Yantai undertook a detailed cost adjustment exercise for the throughput increase from 500,000 tpa to 650,000 tpa, including vendor quotations for equipment and adjustments to civil work, material and installation labour costs. Incremental capital costs are estimated to be $2,732,000 and are included in the capital estimate for the Project. It is noted that this change was not fully reflected in the final DFS engineering detail as the required modifications are not material to the plan layout and will be incorporated during the front-end engineering design phase.

The major modifications were:

- Crushing section: No equipment modification, utilisation increased from 38.04% to 49.45%
- Grinding section: Size of SAG Mill, Ball Mill and motors increased
- Leaching section: Size of tanks (CIL, CIL and Detox), blowers and pumps increased
- Power section: An additional 1,200 kw generator

The process plant design allows for sufficient automated plant control to minimize the need for continuous operator intervention, and allows for manual override and control when required. The general plant design philosophy will be to provide a low-cost processing facility whilst maintaining high levels of reliability, operability, and maintainability.

The key process design criteria listed in Table 17-1 form the basis of the process plant design. The process flow sheet is in Figure 17-1.

TABLE 17-1 SUMMARY OF KEY PROCESS DESIGN CRITERIA
Thor Explorations Ltd – Segilola Gold Project

	Units	Original Design	Revised Throughput
Plant Capacity	t/y	500,000	650,000
Design Gold Head Grade	g /t Au	4.74	4.74
Gravity Gold Recovery	%	43	[43]
Overall Gold Recovery (Gravity + Leach)	%	97.0	97.0
Crushing Plant Utilization	%	38.04	[49.45]
Crushing Design Throughput	tph	150	150
Plant Utilization	%	91.3	[91.3]
Nominal Throughput	tph	62.5	[81.3]
Design Throughput	tph	65	[84.5]
Crushing Work Index (CWi)	kWh/t	13.8	13.8
Bond Ball Mill Work Index (BWi)	kWh/t	19.3	19.3
SAG Mill A*B		55.3	55.3
Crusher Feed Top Size	mm	600	600
Crusher Product Size	mm	160	160
SAG Mill Feed Size (F80)	mm	112	112
SAG Mill Transfer Size (T80)	mm	6	6
Grind Product Size (P80)	µm	106	106
Leach Circuit Residence Time	h	48	48
Leach Slurry Solids Content	% w/w	40	40
Sodium Cyanide Consumption	kg/t	0.24	0.24
Hydrated Lime Consumption	kg/t	0.19	0.19
Frequency of Elution	strips / week	7	

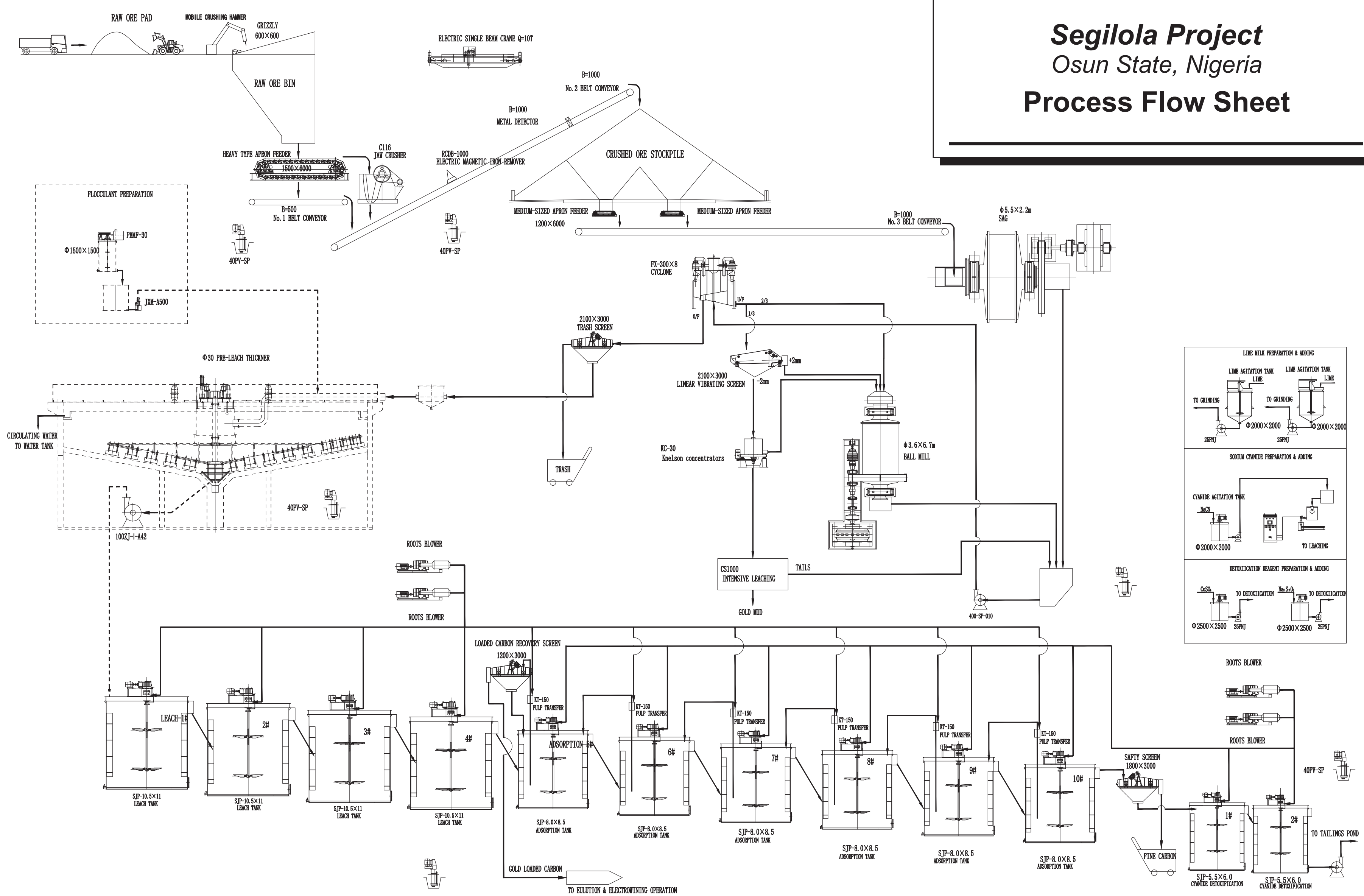
Figure 17-1

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Process Flow Sheet



March 2019

Source: Norinco, 2019.

PROCESS DESCRIPTION

The ROM pad will be located at the south east corner of the plant site. ROM with a nominal top size of 600 mm will be delivered by haul trucks from the open pit mining operation and will either be dumped directly into the ROM bin, or stockpiled on the ROM pad. If oversize rocks bridge the static grizzly, a mobile rock breaker will be used to clear the grizzly.

The crushing circuit consists of a single-stage jaw crusher, and has been sized based on the capacity of the jaw crusher selected for the ROM ore top size. The crushing plant will operate at a crushing rate of 150 tph, treating ore from a maximum lump size of 600 mm to a product top size of 160 mm. The crusher will operate for approximately 13 hours a day.

The crusher circuit product will be transferred to the crushed ore stockpile. From the stockpile, the material will be withdrawn using either of the two apron feeders that will discharge onto the mill feed conveyor, which will transfer the ore to the SAG mill feed hopper. The ore will be mixed with feed water in the mill feed hopper. Hydrated lime, used for pH control, will also be added directly onto the mill feed conveyor

GRINDING AND CLASSIFICATION CIRCUIT

The grinding circuit will consist of a SAG mill, and ball mill in closed circuit with hydro cyclones, having a design throughput of 81.5 dry tph and grinding to a nominal cyclone overflow P_{80} of 106 µm.

The SAG mill dimensions are 5.5 m diameter by 3.0 m long, driven by a 1.2 MW motor. The ball mill dimensions are 4.06 m diameter by 7.5 m long, powered by a 1.75 MW motor.

The SAG mill will be equipped with a variable speed drive (VSD) and synchronous motor, so that variation in ore strength can be addressed by varying the speed of the SAG mill.

The SAG mill discharge will pass through a trommel screen. The screen oversize will be collected in a bunker, while the screen undersize will gravitate to the common mill discharge sump, where it will be diluted before being fed to the cyclone cluster by one of the two cyclone feed pumps.

The cyclone underflow will be collected in the underflow launder, and two-thirds of the underflow will be returned to the feed chute of the ball mill and one-third of the underflow will report to the gravity circuit.

Cyclone overflow, at 40% solids by weight and a grind of P_{80} of 106 µm will gravitate to the vibrating trash screen to remove any wood fragments, organic material, and plastics. Trash material will report to the trash bunker.

The grinding and classification circuit building will be a combined concrete and steel structure, supporting both SAG and ball mills, mill discharge hopper, and the hydrocyclones cluster.

GRAVITY CONCENTRATION AND INTENSIVE CYANIDATION REACTOR

One third of the cyclone underflow will gravitate to the Knelson concentrator scalping screen to remove +2 mm material. Screen oversize and the remaining cyclone underflow will gravitate into the ball mill feed chute and back into the mill for grinding. Screen undersize will report to the Knelson concentrator where free gold will be recovered from the slurry. The Knelson tails will gravitate back into the ball mill feed chute.

The Knelson concentrator will operate in batch mode and the feed will be stopped periodically and the gravity concentrate flushed into a settling cone located beneath the concentrator. The concentrate will be drained into the Acacia reactor every 24 hours for intensive leaching in cyanide. Cyanide, sodium hydroxide, and leach aid will be mixed in the heated intensive cyanidation reactor (ICR) leach solution storage tank before being pumped to the ICR. The leach solution will be used to fluidise the concentrate bed to ensure good contact between the solids and the circulating leach solution. Pregnant solution from the reaction vessel will be transferred to the ICR pregnant solution tank and the gold will be recovered by a dedicated electrowinning circuit. Barren solution will be returned to the CIL tanks, while the leached solids will be washed to remove residual cyanide and then returned to the ball mill feed box.

CIL PLANT

Gold and silver leaching will take place in a conventional CIL type circuit with five leach tanks (no carbon) and five adsorption tanks containing activated carbon, which allows reduced carbon inventory and reduces locked gold in the circuit, whilst target solution tails grades can be maintained if tanks are off line.

Total residence time for the combined leach and adsorption circuit will be 48 hours.

The slurry leaving the last leach tank will report to the adsorption stage, consisting of five tanks having 20 hours total residence time. Dissolved gold and silver cyanide complexes will be adsorbed onto activated carbon contained within the adsorption tanks.

Cyanide solution will be pumped through a ring main, from which it will be dosed into the leach feed splitter box.

Loaded carbon in the CIL slurry will be lifted using air lifters onto the loaded carbon screen. The slurry will return as screen undersize. Spray water will be used to ensure that the clean carbon is discharged as screen oversize product into the column. Barren carbon returning to the adsorption circuit from the carbon reactivation kiln will be screened on the carbon sizing screen to remove fine carbon. The sized and reactivated carbon will report directly to CIL Tank 5. Pulp from the last CIL tank will gravitate to the detoxification section of the plant.

Milk of lime will be added directly to the feed distributor box or leach tanks from a ring main system to ensure that the slurry pH is suitable for cyanidation.

Slurry from the last CIL tank (leach tails) will gravitate to the vibrating carbon safety screen to recover any carbon passing through worn screens or overflowing tanks. Screen underflow will gravitate to the first cyanide detoxification tank. Screen oversize (recovered carbon) will be collected in the fine carbon bin for potential return to the circuit or alternative processing.

The loaded carbon recovery screen will be located above the elution column in the adjacent elution building, as close as possible to CIL tank no. 1.

Stripped carbon will be transferred to the acid wash tank for reactivation. Regenerated carbon will be returned to the last adsorption tank.

ELUTION AND ELECTROWINNING

Slurry containing activated carbon will be pumped from the first adsorption tank and pass over the loaded carbon screen. The screened slurry will be discharged back to the adsorption tank and the recovered carbon will gravitate to the carbon storage tank for the elution circuit.

The loaded carbon in the carbon storage tank is washed with water and then enters the elution column. Elution is carried out at high temperature and pressure, without the use of cyanide. Elution is carried out at 150°C and 0.5 MPa, with a 5% NaOH solution. Elution recycle time will be 6 to 8 hours, and electrowinning will be carried out at a voltage of 2.4 V to 3.0 V, and a current of 250 A. Overall elution cycle time will be 12 to 14 hours.

The stripped barren carbon is sent to a regeneration kiln where it is heated to over 750°C in the absence of oxygen and discharged into a carbon quench tank. The activated carbon is screened for fines, and returned to the circuit at the last adsorption tank to allow counter current movement against the slurry flow for maximum adsorption of gold.

Elution solution (eluate) containing dissolved precious metals will be passed through an electrowinning cell to plate out the contained gold.

After the elution is completed, the gold content in the barren solution will be less than 2.5 g/m3, and the gold content of the stripped carbon will be less than 100 g/t.

The cell will be stripped weekly, where the electroplated gold is taken off the cathodes and collected for smelting as ‘sludge’. Sludge will be filtered in a vacuum filter and dried in an oven prior to smelting to produce bullion.

At the completion of the elution sequence, the stripped carbon will be transferred to the regeneration kiln, if required.

The elution and electrowinning circuit has been designed for a three tonne capacity. Seven or more batches of carbon can be stripped per week, but typically, 21 tons of carbon will be stripped per week.

GOLD ROOM REFINING

Precious metal recovery and refining will be carried out in a secure building.

Gold sludge is thoroughly washed in hot water. Silver and base metals are then removed from the sludge by leaching with nitric acid to form soluble nitrates. After leaching with nitric acid, the dried gold sludge is mixed with a range of fluxes and smelted at 1,100°C to produce bullion. Bullion will be stored in a vault prior to transport and refining off site, although the intention will be to remove bullion immediately off site by helicopter transfer to Lagos International airport for onward transport to the refiner.

TAILINGS DISPOSAL AND RETURN WATER

Tailings slurry will be pumped from the tailings tank to the TSF. Return water from the TSF will be pumped into the reclaim water tank and then back into the process plant.

CIL tailings will be treated to destroy cyanide prior to deposition in the tailings dam. Cyanide destruction (detoxification) will be accomplished using the SO_2 (SMBS) / air process.

WATER BALANCE

Water consumption for the processing plant is summarized in Figure 17-2.

The majority of water added to the process is during grinding, where it is added to the SAG mill and the ball mill feed chutes. Water is also added to the cyclone underflow vibrating screen and Knelson concentrator. Water addition in the grinding circuit is controlled to achieve the target 75% solids in the SAG mill, and 50% solids in the cyclone overflow.

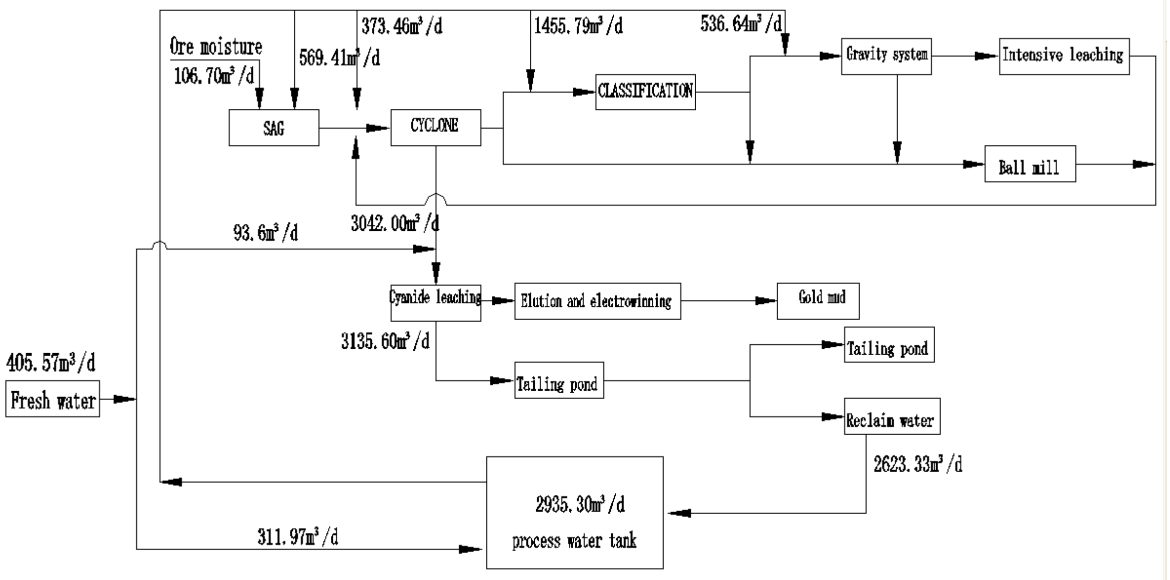
The water demand of 163 m^3/hr (135 m^3/hr x 1.2) is met by reclaim water from the TSF and clean water from the water supply dam, which has a capacity of 0.374 Mm^3. The TSF will supply between 40% and 100% of the monthly water demand, with approximately 78,700 m^3/month water reclaimed on average. To maintain efficient operation the TSF pond will have a minimum volume of 50,000 m^3 and an average of approximately 71,000 m^3.

The water supply dam will make up any deficits from the TSF; supplying up 57,850 m^3/month during the dry season. If commissioning occurs during a dry season the water supply dam will provide up to 100% of the process plant demand during commissioning and up to 60% thereafter.

Raw water from the water dam will be stored in the raw water pond, with a capacity of 1,000 m^3, and the reclaim water from the TSF will be stored in the reclaim water pond, with a capacity of 2,000 m^3. Raw water will provide the majority of the make-up water for the process plant requirements.

Fire water consumption for a two-hour fire duration will be 144 m³ and will be stored in the fire water tank whose capacity is 200 m³, and the make-up water will be pumped into it 30 m³/h.

FIGURE 17-2 WATER BALANCE DIAGRAM



REAGENTS

Cyanide solution will be mixed once per day as a 20% solution by mass in the cyanide mixing tank.

Lime will be added to the lime mixing tank to produce a 5% lime slurry. After total dissolution is achieved, the lime slurry will be pumped by the lime transfer pump to CIL. Lime slurry will be pumped through a ring main by the lime dosing pump, branching to the mill feed hopper, leach feed splitter box, and detoxification tank.

Caustic soda pearls will be added to the caustic soda tank to produce a 20% NaOH solution. The tank will also serve as a caustic soda solution storage tank, with intermittent dosing to the elution/electrowinning, intensive leach, cyanide make-up, and acid wash areas, directly from the tank using the caustic dosing pump.

Copper sulphate solution will be made up once per day. It will be made up as a 20% solution by mass in the copper sulphate mixing tank. The solution will be dosed via one of two variable speed pumps.

Sodium metabisulphite solution will be made up once per day. It will be made up as a 20% solution by mass in the sodium metabisulphite mixing tank.

Activated carbon will be added to the carbon quench tank as required for carbon make-up to the CIL inventory. This addition point will allow attrition of any friable carbon particles with subsequent fines removal on the sizing screen prior to entering the CIL tanks.

Grinding balls will be delivered to site in 200 L steel drums. The balls will be charged to the SAG and ball mill feed chutes using the electric hoist.

Diesel and LNG / CNG will be delivered to site by road tanker.

CIL air will be supplied by duty / standby medium pressure air compressors and air will be reticulated to the CIL tanks and injected into the slurry via the tank agitator shafts.

The compressors (two duty and two standby) will be situated inside the plant area. Four high pressure compressors will be dedicated to supplying plant air and instrument air to the plant.

18 PROJECT INFRASTRUCTURE

There is no existing infrastructure at the Project site. The engineering of the necessary support facilities and infrastructure for the construction and operation of the Project has been considered in the DFS and the construction costs included in the capital cost estimate.

The general layout is shown in Figure 18-1.

SITE ROADS

Access to the site will be from Odo Ijesha – Iperindo Road via a sealed access road which will service both the processing plant and mining compounds.

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

Two major mining roads will be built, 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 road will be unsealed and have a sufficient width to allow two-way traffic flow.

Plant site roads are internal roads between the administration area and plant site facilities.

BUILDINGS

The processing plant will be constructed over three terraced levels, with each terrace containing a number of processing areas. The ROM pad and crusher will be located at the southeast end of the processing plant. Mining waste and other earthworks are required for these terraces.

The main plant will be built on a single level. The gold room will be located adjacent to the mill to facilitate the installation of a gravity circuit at a later stage if required. Reagent mixing will be located to the north of the CIL section. The warehouse, workshop, and reagent storage facility will be located to the southeast of the CIL section.

A plant workshop and warehouse will provide a maintenance area, material storage area, activated carbon storage, and an office for employees.

Most structures will be steel with brick or cladding.

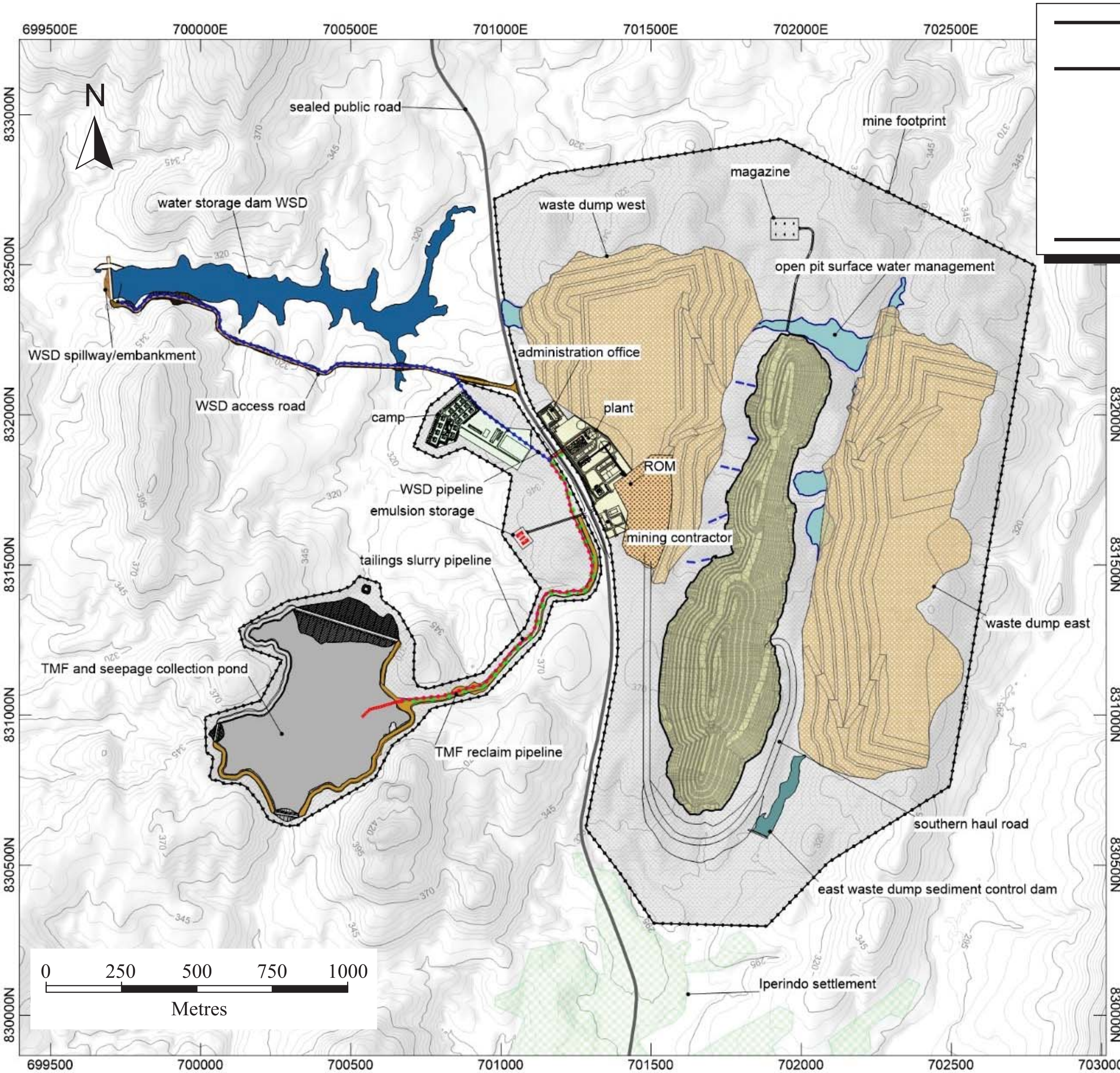
Figure 18-1

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Site Infastructure



March 2019

Source: Norinco, 2019.

A number of administration offices will be built adjacent to the plant. Additional site facilities including combined change room/ablution block, warehouse/stores, and a kitchen will be located adjacent to the administration offices. An assay laboratory will be provided on the south-eastern side of the processing plant.

The mining contractor area is located to the southeast of processing plant. It is separately fenced and consists of a workshop and warehouse, first aid rooms, and dining rooms.

The camp is on the northwest of the processing plant, including two accommodation blocks with 13 single or double rooms with ensuite bathrooms and 13 blocks that can be configured as two, three or four single or double rooms, each room with ensuite bathrooms, dining room, laundry, fitness centre, recreation room, football pitch, swimming pool, and security gate. The construction camp will be located next to the camp and the company may retain some construction camp buildings as provisional overflow accommodation.

The explosive materials will be stored in a magazine located in a remote area to the north of the plant and the pits, well away from the local population and mine activities. The magazine will be secured within a high security fenced compound and surrounded by embankments.

The emulsion storage will be located west of the plant to allow ease of access to the power and water services supply. Two 50 m^3 emulsion tanks are provided with a supply of approximately eight days.

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 MSA and reticulated to the power station, plant day tank, and MSA day tank.

A liquified natural gas (LNG) supply station will be set up to serve the eight 1.2 MW CNG gensets operated on a six duty, two standby basis and one emergency power 0.8 MW diesel genset. A total of five 100 m^3 LNG storage tanks (approximately 10 days' supply), one gasification unit and one 20 m^3 diesel storage tank are proposed.

WATER MANAGEMENT

The total water consumption is estimated to be 3,251 m^3/day, with fresh water consumption at 548 m^3/day, and the recovered water consumption at 2,703 m^3/day. Water in the raw water dam will be replenished by rainfall.

The total fresh water consumption of 548 m^3/day includes water for processing at 406 m^3/day, water for dust suppression and gland seal water at 40 m^3/day, water for greening and spraying road at 12 m^3/day, water for domestic use at 41 m^3/day, and water for unforeseen purposes at 49 m^3/day. The ratio of water reuse processing is 83%.

The fire water consumption for two-hour fire duration is 144 m^3, and the make-up water will be used for this at a pumping rate of 30 m^3/h.

Water supply standards are based on the following parameters:

- The water for processing is as per processing requirement.
- The domestic water is 285 L/person*day.
- The water for unforeseen uses is 10% of the fresh water.

The plant reclaim water will be pumped from the TMF. When required, raw water make up for the plant will be rain water stored in the plant feed water dam west of the processing plant. Process water will be stored in a dedicated 2.000 m^3 reclaim water pond.

Potable water will be produced by treating raw water. Some raw water will be treated as non-potable water for daily use except drinking. The treatment process is physical filtration which includes sand filtration, carbon filtration, and ozone sterilization.

SEWAGE AND SOLID WASTE MANAGEMENT

SEWAGE TREATMENT

Effluent from all water fixtures in the process plant, mine services area and administration areas will drain to gravity sewerage systems. The gravity sewerage system for each area will drain to a sewage treatment plant system located to the north of the process plant.

Treated effluent will be returned to the discharged to an evaporation pond. Treatment plant sludge will be suitable for direct landfill burial.

SOLID WASTES

Wastes will be sorted and reused or recycled as far as the limited access to recycling facilities allows.

Waste lubricating oils will be returned to the supplier or appropriate for recycling.

General non-hazardous solid wastes will be deposited into a suitable mine waste dump and promptly covered to deter vermin and scavengers.

Dangerous or hazardous waste will be collected and stored briefly before being transferred to a suitable permitted facility, either on-site or off-site depending on the specific materials and requirements.

POWER SUPPLY AND DISTRIBUTION

The state grid corporation of Nigeria is unable to provide electricity for the Project, and as a result, generating sets will be used for power supply. A total of eight 1.2 MW generating sets will be installed and operated, all eight of which will be CNG generator sets (10.5 kV/1,200 kW at 0.8 pf). Additionally, there will be one diesel generating unit for emergency power (400 V/800 kW at 0.8 pf). Nominal values for the power distribution system within the mineral processing plant are shown in Table 18-1.

TABLE 18-1 POWER DISTRIBUTION SYSTEM VOLTAGE
Thor Explorations Ltd – Segilola Gold Project

Circuit Type	Voltage
High Voltage System (SAG Mill, Ball Mill, and Transformer)	10 kV, 3-phase
Low Voltage System	220/380 V, 3-phase 4 wire solidly earthed
Uninterruptible Power Supplies	220V single phase, neutral earthed
Contactor Coil	220 VAC
HV Switchgear Trip, Closing	220 VDC
Automatic Control	24 VDC, earthed 0V
Instrumentation	24 VDC, 240VAC (where necessary)

Emergency power from a standby 800 kW/400 V 50Hz diesel powered generator will be provided in the process plant.

The plant site-wide electrical power requirements for mineral processing and infrastructure were calculated on the basis of preliminary equipment sizing. The electric load for the Project is summarized in Table 18-2.

TABLE 18-2 ELECTRICAL LODE
Thor Explorations Ltd – Segilola Gold Project

	Load
Installed Power	8,651 kW
Service Power	7,194 kW
Active Power	5,404 kW
Reactive Power (after LV- Reactive compensation)	3,166 kVar
Apparent Power	6,263 kVA

The SAG mill will be started using a variable frequency drive (VFD) and the ball mill will be started using a liquid resistance starter (LRS). The VFD will limit the SAG mill start-up current to1.6x full load current (frozen load protection) and the LRS will be designed to limit the ball mill start-up current to 1.6x full load current.

The power factor was improved from 0.76 to 0.95 by employing reactive compensation of 1,440 kVar (reactive capacitance compensation) on 400 V side of the step-down transformer.

Table 18-3 lists the power consumption by area.

TABLE 18-3 POWER CONSUMPTION
Thor Explorations Ltd – Segilola Gold Project

Area	Total Installed Power (kW)	Maximum Continuous Power (kW)	K-factor	Maximum Load Utilization Hours (h)	Annual Power Consumption (kW*h)
0.4 kV					
Crushing Section	236	140	0.7	4,368	427,498
Crushed Ore Stockpile	50	31	0.7	5,824	125,484
Milling Section	647	402	0.7	8,000	2,251,917
Leaching and Adsorption	534	346	0.7	8,000	1,938,328
Tailings Handling	289	131	0.7	8,000	735,168
Elution and Electrowinning	315	221	0.7	2,672	412,685
Gold Room	217	152	0.7	1,440	153,327
Reagent Preparation	144	70	0.7	8,000	389,536
Air Services	1,815	641	0.7	8,000	3,586,800
Carbon Regeneration	130	90	0.7	960	60,319
Water Services	349	150	0.7	8,000	840,868

Area	Total Installed Power (kW)	Maximum Continuous Power (kW)	K-factor	Maximum Load Utilization Hours (h)	Annual Power Consumption (kW*h)
WSD & TMF	254	102	0.7	5,000	308,092
Auxiliary facilities	720	517	0.7	3,200	1,156,960
Subtotal	5,701	2,991			12,386,982
Multiply by Coefficient 0.9		2,692			11,148,283
Transformer loss		57	0.7	8,000	317,583
Equivalent to10 kV total	5,701	2,749			11,465,866
10 kV Ball Mill and SAG Mill	2,950	2,655	0.7	8,000	14,868,000
10 kV Total	8,651	5,404			26,333,866

TAILINGS MANAGEMENT FACILITY

The design of the proposed TMF was completed by KP in 2018. KP completed site selection and a subsequent site investigation, which included a ground assessment, sampling, and laboratory testing of geotechnical samples. The TMF has been classified as Significant according to the Canadian Dam Association Dam Safety Guidelines (CDA 2014) representing international best practice.

The TMF will be located 1.3 km southwest of the process plant. The TMF consists of a single valley style rock zoned embankment (north) and two small (less than 5 m) earth filled saddle dams (west and east). To spread construction costs of the TMF over the LOM, the TMF will be constructed to full height over a series of five raises including the starter wall construction (north embankment) using downstream construction methodology. The starter wall will be constructed to a minimum crest elevation of 335 m for a minimum height of 12 m using local borrow materials consisting mainly of silts and clays. The final embankment elevation is specified at 350 m constructed from local borrow material and waste rock from the open pit. The starter wall includes an upstream cut off trench constructed to a minimum depth of 2 m below prepared foundation.

To further aid in reducing seepage potential, an HDPE geomembrane liner will be installed over the upstream slope of the starter wall only. The starter wall (Stage 1) will not be outfitted with an emergency spillway, however, Stage 2 to Stage 5 will be constructed with one located on the west side of the north embankment with any flows routed to the WSD. The TMF has been designed to accommodate the design flood, therefore the emergency spillway is not intended to be used based on the dam classification and operational requirements.

The current LOM is five years with plant commissioning scheduled for January 2020. The TMF has been sized to accommodate 3.3 Mt while providing sufficient capacity for the operating pond and the Inflow Design Flood (IDF). A return period of 1 in 1,000-year 24 hour was adopted for the TMF IDF based on a Significant dam classification. The 1 in 1,000-year 24-hour event equates to 300 mm of rainfall, or 148,700 m^3, to be stored in the TMF.

Stability analysis was completed for the north embankment. Upstream and downstream slopes of 1V:3H and a crest width of 12 m is shown to be stable, meeting all minimum factors of safety according to the CDA 2014 guidelines. A seismic coefficient of 0.1 was selected for this evaluation.

The starter wall crest elevation was chosen to provide approximately one year of tailings storage or 0.35 Mt. Construction of the second raise, Stage 2, will start in mid-2020. The annual storage requirement for years two through six is 0.5 Mt, while the final year is 0.45 Mt. An average tailings in-situ density of 1.35 t/m^3 was applied based on drained and undrained settling testing completed by SGS in 2010.

A 500 m^3 seepage collection pond will be built downstream of the north embankment. Any water passing through the north embankment will be collected through an internal chimney and blanket drain system that discharges into the HDPE lined seepage collection pond. Seepage will be pumped back to the TMF.

A tailings deposition strategy will be implemented to keep the supernatant pond away from the embankments to reduce seepage potential and to push the water towards the decant system located centrally along the eastern side of the TMF. The decant system consists of a floating barge outfitted with two pumps (one operation and one standby). The reclaim barge is capable of supply 100% of the process plant's water demand of 116.5 m^3/hour, however, the average supply rate is 84 m^3/hour or 72% according to the TMF water balance. A floating walkway will be used to access the floating barge, while pump maintenance and removal would require a crane. The reclaim pipe consists of 160 mm diameter HDPE pipe with a pressure rating of 10 bar and is routed along the northern side of the TMF access road with the tailings slurry pipeline.

The tailings slurry pumping and pipeline system feasibility design was completed by Paterson & Cooke. Slurry test work was completed to establish tailings rheology parameters required for the pump and pipeline design. The tailings will be pumped from the detox circuit at the process plant to the TMF along the TMF access road. A series of spigots will be used to control location of tailings deposition into the TMF. The system has been designed for an annual average throughput of 500,000 tpa with varying slurry concentrations, ranging between 35.5% and 40% solids by mass. The identified pumps and pipeline are suitable for the LOM requirements.

Acid mine drainage testing has characterised the tailings as potentially acid generating. Review of the available data indicates that microcline makes up around 5 to 10% of the tailings, which may be able to buffer the acidity generated by the tailings to a pH of approximately 4 to 5. The final pH is, however, dependant on the rate at which the microcline and quartz minerals weather.

The fine grind together with a high moisture content in the tailings should minimise oxygen ingress during operation and restrict sulphide oxidation to a certain degree. As most acid seepage generated can be collected by the reclaim system, a fully lined TMF would not be a significant benefit. However, following drain down, the oxidation front will move through the tailings generating mildly acidic leachate although the very fine nature of the tailings is anticipated to limit sulphide oxidisation. The rate of feldspar dissolution may therefore potentially keep pace with the rate of acid generation.

Carbonate rich, acid neutralising waste rock could also be utilised for acid neutralisation within the TMF and mitigate potential ARD generation in the deposit and minimise metals release post closure, although further work is required to confirm this potential.

19 MARKET STUDIES AND CONTRACTS

MARKETS

The principal commodities at the Project are freely traded, at prices and terms that are widely known, so that prospects for sale of any production are virtually assured.

Metal prices used by Auralia and Thor are in line with the long-term consensus forecasts by independent banks and financial institutions. A gold price of $1,300 per ounce was used for the financial modelling.

CONTRACTS

There are no material contracts or agreements in place as of the effective date of this report.

Refining contracts are typically put in place with well recognized international refineries and sales are made based on spot gold prices. These contracts typically include fees for transportation of the product from the site, insurance, assaying, refining and an allowance for metal losses during refining. The ability to get a contract in place for the sale of doré prior to start of production is not seen as a risk to the Project.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

ENVIRONMENTAL STUDIES

INTRODUCTION AND BACKGROUND

Thor's ML41, in which the Project site is located, consists of a modified environment that has been disturbed by human activities over many decades and are still ongoing (e.g., the area has been logged for over 100 years; hunting for bushmeat still occurs; the site contains the remnants of gold mining activity from the 1950s; and current land use mostly consists of exploiting trees of economic value which provides income for communities in Iperindo, Imogbara and Odo Ijesha, which are to the north and south of the mining lease area) (Figure 20-1).

The area within ML41, which is 17.2 km^2, has not experienced change in terms of development in the past two decades. The existence of exploration licenses and mining leases over a 70-year period has not triggered significant development nor waves of influx. In the 1950s, prospecting and exploration activities for gold had been carried out by other operators in the area; evidence of such activity includes a field campsite and exploratory tunnels.

Communities in the villages of Imogbara (population approximately 1,230), Odo Ijesha including IpoleIjesha (population approximately 4,550) and Iperindo (population approximately 6,150) are agriculture/market-based communities with low incomes. Each community has its own cultural governance system via an Oba, which is an appointed position and is not necessarily a hereditary title. The Atakunmosa East Local Government covering the ML is based in Iperindo.

In terms of environmental studies, a full Environmental Impact Assessment (EIA) was carried out for the Project by independent consultants, in compliance with the EIA Act No. 86 of 1992. An EIA report was submitted to the authorities for review and decision-making (whether the Project could go ahead or not) in 2012. 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 standard conditions.

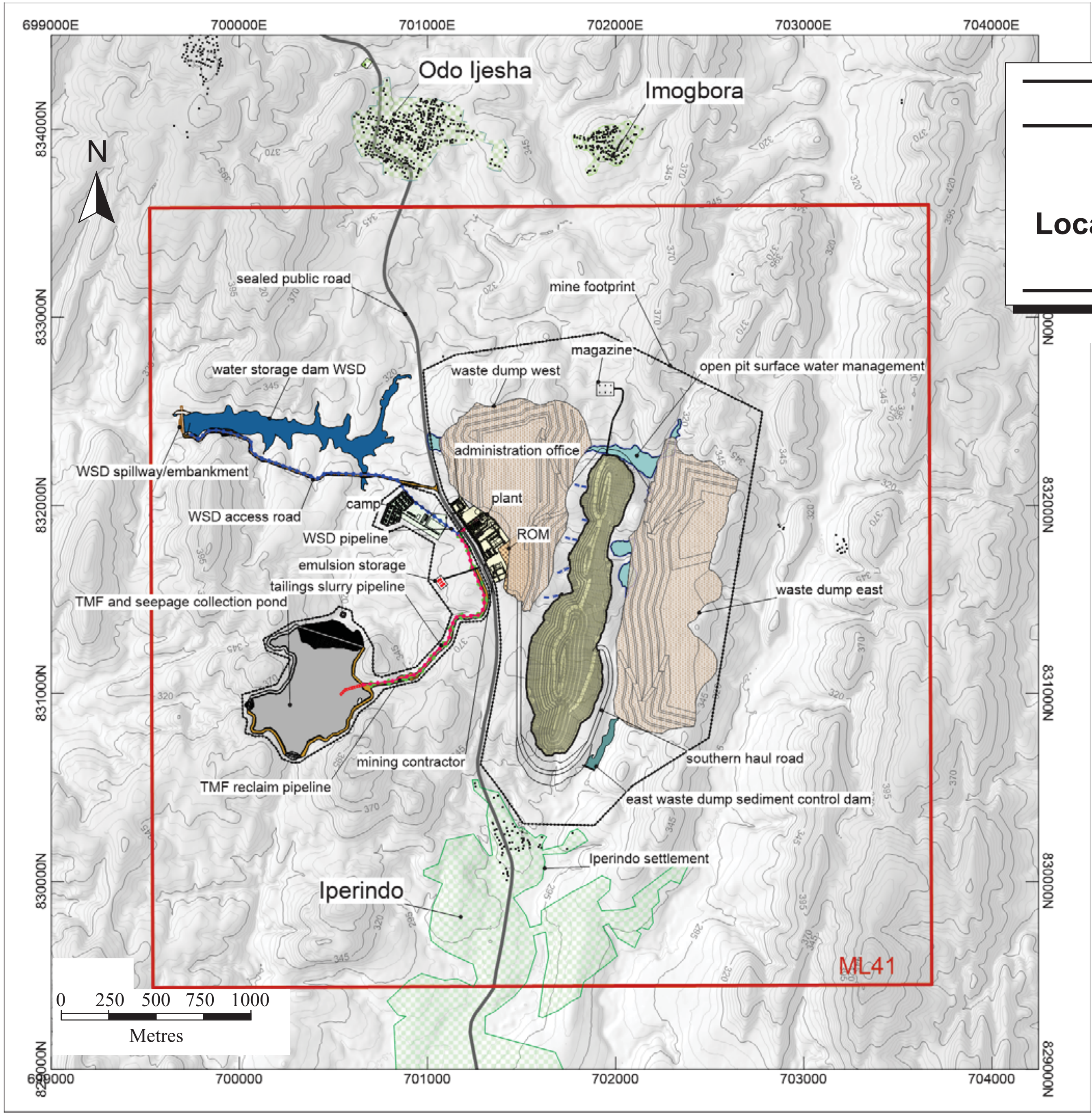
Figure 20-1

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Location of the Mine Site and Surrounding Villages



March 2019

Source: Norinco, 2019.

www.rpacan.com

The EIA approval has been enacted through commencement of exploration activities undertaken by previous owners since 2013, and by SROL since 2017. Work initiated by SROL is in progress to address and fulfil the conditions of the EIA Certificate as the Project moves through the project life cycle. Currently, SROL is preparing a raft of environmental and social management plans and has commissioned additional environmental and social baselines and assessments to update existing information / data. The environmental baselines that have been updated include water quality and ecology (wet season baseline was completed in August 2018 and dry season in Q1 2019), ongoing studies include socio-economic baselines, noise, and air quality. No red flags or showstoppers have been identified through past and present studies.

The suite of studies and management plans have been developed to conform with international standards, particularly the International Finance Corporation's (IFC) Performance Standards (PS) and mining sector-specific Environmental, Health and Safety Guidelines.

The layout of some components described in the EIA have been adjusted and resulted in reducing the footprint from approximately 800 ha to approximately 420 ha; this will reduce the area that is directly impacted by the mine development.

Management plans under the umbrella of an Environment and Social Management System (ESMS) 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. It should be noted that for purposes of aligning with the wording of the EIA certificate, SPROL has entitled the ESMS an "Environmental Management Plan". As living documents, the management plans will incorporate changes to, and updates on, the mine project as outlined in the DFS; potential impacts will be assessed and mitigation measures developed in conformance with international standards, specifically the IFC PS.

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. All three CDAs have been endorsed by the Ministry of Mines and Steel Development (MMSD). Benefits to the communities have commenced with the construction of new boreholes (with handpumps) in the three villages (replacing those which were moribund), and provision of local employment during the exploration stage.

The nearest communities' dwellings and other permanent structures are not directly impacted by the proposed mine, its infrastructure nor buffer zones (i.e., no physical resettlement is required). However, some economic impacts have been identified and compensation for the loss of agricultural crops is under preparation via a Resettlement Action Plan (RAP) which incorporates a Livelihood Restoration Plan, in conformance with IFC PS. SROL has crop compensation sheets as documentary evidence of compensation that has previously been paid to affected parties for crop lost or damaged during exploration phase activities.

Stakeholder consultation occurred during the EIA process (EIA report, 2012) and then intermittently until 2017 when Thor recommenced consultation meetings with communities. A step-up in consultation and public participation has occurred as part of the exploration phase and in the development of the CDAs. CDA committees with representations from each village have been established to aid in managing the CDA process. Thor has developed a stakeholder engagement plan, outlining how consultation has and will occur with identified key project stakeholders for the life of the Project; this includes a grievance management plan. Thor has a team who are managing environmental and social aspects of the Project; the social team, and developing and managing relationships with communities and other stakeholders.

ENVIRONMENT AND SOCIAL BASELINE INFORMATION

The Project environmental baseline was established in 2008/2009 as part of the EIA study; the full suite of studies that are standard for EIAs was undertaken (i.e., biophysical and socio-economic studies). To verify the validity and quality of the baseline, Thor commissioned ecological baseline studies over two seasons; one was conducted in June 2018 wet season and the other was conducted during the dry season of January 2019. Seasonal data collection conforms with requirements of international standards.

Groundwater in southwestern Nigeria is essentially found in semi-confined to unconfined aquifers and is highly influenced by infiltration and percolation of rain. Due to the non-porous nature of the crystalline rock types in this region, permeability is associated with fracturing and weathering processes. Surface water at the site is made up of various shallow streams of less than 1 m depth. The streams serve as reliable source of water for domestic and agricultural uses for local communities.

In line with the monitoring requirements listed in the EIA, the Project commenced water quality monitoring in August 2017. Undertaking water quality testing was strategic as the quality of the water was used as an indicator of the state of the environment. The water quality results recorded since August 2017 have been consistent with EIA baseline values.

The Project area is predominantly secondary rain forest vegetation with high density and composition. The habitat is interspersed with fields where cash crops are grown and trees of economic value provide income for village residents surrounding the mine site. Most of the grasses, shrubs, lianas, and trees in the area are typical to the rain forest zones in Nigeria.

Vegetation in the ML is dense, largely due to the influence of rainfall. Although the impact of rainfall supports the herbaceous flora most effectively, woody flora survives throughout the dry season. The composition and population of the animal communities depends on the vegetation which provide shelter, security, and food for these animals. The latest baseline ecology survey results on flora were consistent with the survey results obtained during the EIA process, in that the natural habitat has been modified by human activity including subsistence agriculture, logging, and hunting.

Air quality samples were taken for analysis and noise levels were measured at five locations within the Project area. Results for air and noise were within the limits as per Nigerian standards.

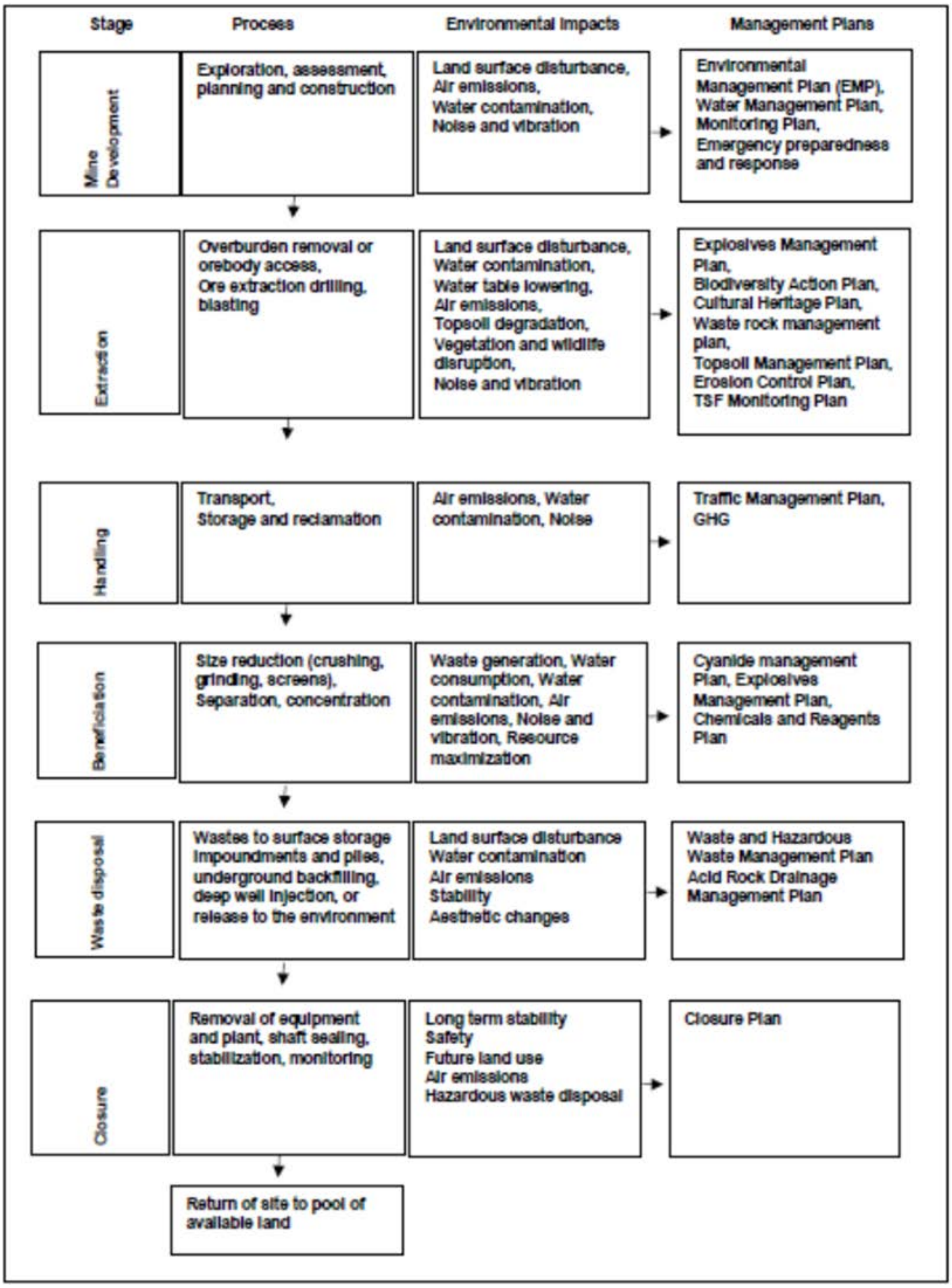
A detailed cultural heritage assessment was undertaken as part of the EIA. It is proposed to update this information as part of the socio-economic surveys planned for Q1 2019.

KEY ENVIRONMENT AND SOCIAL IMPACTS OF THE PROJECT.

The EIA outlined the key environment and social impacts and mitigation factors the essence of which are still considered valid.

The main impacts identified through the EIA (and baseline updates) are typical of those for mine projects and are summarised in in Figure 20-2.

FIGURE 20-2 MINE IMPACTS AND MANAGEMENT THROUGH THE LIFECYCLE



The EIA did not identify any red flags with regard to environmental and social impacts subject to implementation of mitigation measures.

FINANCIAL PROVISION

The main social financial provisions relate to the CDA budgets. SROL manage the budgets for each except for the development fund which is managed by the CDA committee and is subject to criteria for its use.

These budgets all include employment of community selected CLOs for all three communities. Provision of benefits increases during construction and peaks during operation phases. The scheduling of benefits during the project lifecycle is set out in the community development agreements.

Other social financial provisions relate to:

- Provision of SROL CLO and Community Development and Stakeholder Manager for the life time of the mine;
- A budget for corporate social responsibility programmes – based on an annual amount.

SOCIAL OR COMMUNITY REQUIREMENTS

RECRUITMENT AND EMPLOYMENT

Priority will be given to employing people from the local community and immediate surroundings, subject to availability at the time of recruitment and applicants meeting the required level of qualification and professional experience. It is expected that the large majority of unskilled positions will be filled from the local communities. Additionally, training programmes will be provided for some of the semi-skilled positions. These requirements shall be included in contracts where applicable (e.g., for contractors working in the Project).

IN-MIGRATION MANAGEMENT

In-migration (influx) management can become a material issue. The SROL Community Relations team will be fully committed and responsible for managing these issues. The key risks to operation presented by in-migration include:

- Conflict over availability of and selection for jobs.
- Health and social risks that the influx of non-locals might bring with them.
- Pressure on infrastructure.

The Company will:

- Build local capacity.

- Work to prevent employment-linked conflict.
- Develop and implement a communication strategy.
- Provide basic health and safety awareness course for all personnel as well as specific awareness about HIV/AIDS.
- Ensure that its workers are briefed on the socio-cultural norms and sensitivities of the host communities before commencement of work in the area.
- Ensure that all the community leaders are made aware of the project activities and impacts, and encouraged to make inputs in appropriate mitigation measures.
- Support community development efforts as practicable and agreed with community representatives.

MINE CLOSURE AND WORKFORCE DEMOBILISATION

At the end of the commercial life of the Project, SROL's workers likely will have become an established part of the community in the Project region. Due to the life of the Project, no specific plans or forecasts have been currently made concerning workforce demobilisation.

SROL will help its local workforce to transition from an operating Project into the post-closure period, specifically to deal with the impacts of loss of employment.

A workforce transition program will be developed to assist with the loss of employment at mine closure. Regular community and employee consultation will be critical in establishing the best adjustment programs to transition effectively from an operating project to the post closure period, while maintaining local sustainable development.

MINE CLOSURE REQUIREMENTS

The EIA and EPRP outline the parameters for mine closure, as shown in Table 20-1.

TABLE 20-1 MINE CLOSURE COMPONENTS AND OBJECTIVES
Thor Explorations Ltd – Segilola Gold Project

Mine Component	Closure Objectives	Start	Finish
Open Pit	Dewatering will cease on closure of the mine and the open pit will be allowed to flood. The pit slopes will be left in a stable condition (based on geotechnical assessment) and groundwater quality will not be compromised. Reassignment to local community for water storage/use. A protection berm / fence will be formed around the pit to discourage entry.	Mine Closure	5 years post-closure
Waste Rock Dump	Final slopes of the waste rock dump will be stable and any possible re-vegetation carried out.	End of Project	5 years post-closure
Tailings Storage Facility	Profiling of the dam surface and layering with soil to encourage re-vegetation with sustainable crops. Closure of the decant and evaporation of supernatant.	End of Project	5 years post-closure
ROM Pad and Process Plant	Decommissioning of the plant, dismantling of buildings and removal of foundations. Removal of scrap metal and used oils etc. Re-profiling and re-vegetation of the site.	End of Project	5 years post-closure
Offices and Car Park	Dismantling of buildings and removal of foundations. Removal of office waste. Reprofiling and re-vegetation of the site.	End of Project	5 years post-closure
Raw and Process Water Storage tanks	Evaporation of remnant water, removal of contaminated solids to a licensed waste disposal location offsite.	End of Project	5 years post-closure
Raw Water Dam	Reassignment of responsibility to local community, as agreed in closure plan	End of Project	1 year post-closure

The key mine closure parameters from an environment perspective relate to the preparation and implementation of a habitat rehabilitation plan.

Site contouring after closure and removal of structures site investigations and site activities will include:

- Zoning of fauna, flora population and specific soil conditions and types.
- Soil analysis.
- Development of botanical map of the proxy virgin vegetation areas of the mine site using global positioning system to give the historical topography, land form, presence of vegetation compared to the site`s man made modification features pre/post mine site work.
- Inputs into the site recontouring plan (executed through the Mine Plan).
- Development of the overall habitat restoration plan.
- Nursery development and seed production (3 years prior to closure).
- Vegetation re-generation.
- Ongoing maintenance of planted areas.

- Ongoing monitoring (for 5-year period).

Towards the end of the LOM, the operational objective of the TMF will be to reduce the pond volume as much as possible. Following end of LOM, any remaining water on the TMF should be pumped off, treated, and discharged. The proposed closure concept for the TMF is to:

- Use tailings deposition to shape the final surface in a manner that will facilitate the development of a water shedding structure (i.e. produce a suitable surface gradient profile that allows for gravity drainage).
- Facilitate the shedding of water off the tailings surface, by construction of an excavated drainage channel leading to the west side of the north embankment (operating spillway location) will be required once the TMF pond is removed. Rainfall runoff will be routed along the excavated channel and off the TMF, using drainage structures that are designed for long term minimisation of erosion.
- In parallel with construction of drainage engineered features, place a suitable cover profile, at this time anticipated to be a layer of top soil (current consideration of 300 mm) over the entire TMF surface.
- Facilitate the establishment of sustainable vegetation growth on the cover.

The seepage collection pond may require operation for up to five years following end of LOM depending on seepage rates and water quality.

The potential for the oxidation of sulphides in the tailings mass has been identified as a potential long-term risk with respect to water quality. The risk has not been quantified to date and requires more detailed technical assessment.

Further technical work is recommended as follows to increase the level of confidence of the closure design concept:

- Further evaluation of the tailings by field trials, and modelling assessment will be required to establish the materials requirement to maintain a sustainable vegetated cover (based on vegetation type selected).
- Further technical evaluation of the long-term oxygen ingress rate is required to establish the optimal approach to manage long term risks of seepage water quality from the TMF.

21 CAPITAL AND OPERATING COSTS

CAPITAL COSTS

This FS capital cost estimate is compiled as a Class 3 estimate in line with AACE recommended practice 47R-11 Cost Estimate Classification System as Applied in the Mining and Mineral Processing Industries. Based on the level of the Project and engineering definition available, the accuracy is +/-10%.

INITIAL CAPITAL

The initial capital cost estimate for the Project is summarised in Table 21-1.

TABLE 21-1 DEVELOPMENT CAPITAL COST SUMMARY
Thor Explorations Ltd. – Segilola Gold Project

Description	$000s
Mining (ROM pad)	400
Process	19,800
Infrastructure	28,600
Indirects	13,000
Norinco EPC Contract Value	**61,800**
Powerline to TMF/WDS	400
Total EPC Contract	**62,000**
Total TMF	3,700
Total Contract Value	**65,900**
First Fills	1,900
Mine Contractor Mob / Haul Road Est.	800
Owner Cost	11,000
VAT (on local purchases / services)	1,300
Duties (contingency)	1,500
Underground Development	0
Initial Capital	**82,300**
Pre-strip (Month 16)	2,900
Upfront Working Capital	2,400
Total Development Spend	**87,600**

SUSTAINING CAPITAL

No sustaining capital estimates were included in the Project. Estimates for raises to the initial tailings dam are incorporated under operating costs.

EXCLUSIONS

The following items were excluded from the capital cost estimate:

- Project financing and interest charges

- Escalation during construction
- Exchange rate variations

OPERATING COSTS

A summary of the LOM unit operating costs totalling $77.51/t milled is presented in Table 21-2.

TABLE 21-2 LOM UNIT OPERATING COST SUMMARY
Thor Explorations Ltd. – Segilola Gold Project

Description	$/t milled
Contractor Mining (2.85/t moved)	50.97
Process (Thor)	18.58
Fixed G&A	5.78
Infrastructure	2.18
Total	**77.51**

COST ESTIMATE BASIS

The operating costs presented are based upon contractor mining and owner operated production equipment and site facilities, as well as the Owner employing and directing all non-mining operating, maintenance, and support personnel

Operating costs for the Project have been estimated from first principles. Labour costs were estimated using Project specific staffing, salary, wage, and benefit requirements. Unit consumption of materials, supplies, power, water, and delivered supply costs were also estimated.

CONSUMABLE PRICING

The Project's major consumable pricing and basis are as follows:

- Diesel – The Project has a pre-production and LOM diesel price of $0.675/L delivered to site. Diesel is primarily used in the mining cost estimate and the pre-production phase as well as any back up power generation.

- Power – CNG power cost was estimated at $0.11/kWh. Generator set power costs were estimated at $0.168/kWh, assuming a CNG price of $0.45/m^3. Grid power is not available and has not been included in the Project.

MINE OPERATING COSTS

The mining contractor submitted a set of bench by bench mining rates, with a weighted average of $2.80/t for waste and $3.51/t for ore. These rates include fuel, labour, equipment maintenance, consumables, and management. Load and haul as well as drill and blast services are included in these rates, along with dewatering.

Drill and blast costs range from $0.826/t to $1.456/t.

In the total mining cost estimate, a dayworks allowance of $0.09/t and a topsoil stripping and rehabilitation cost of $0.08/t were added. The total mining cost is estimated at $153 million over the mine life, averaging $2.85/t.

According to the Nigerian Minerals and Mining Act, all operators in the mining industry are exempted from payment of custom and import duties in respect of plant, machinery, equipment, and accessories imported specifically and exclusively for mining operations. Equipment duties have therefore not been included in the mining costs, however, this cost was estimated by the contractor at $6.2 million over the mine life.

Total mining costs were determined by writing the bench costs directly into the mine design block model, and reporting the average cost per year as determined in the scheduling software.

PROCESS OPERATING COSTS

The total process operating costs are summarized in Table 21-3, which also includes all labour costs for Thor employees in administration, mining, processing and maintenance.

TABLE 21-3 SUMMARY OF PROCESS OPERATING COSTS
Thor Explorations Ltd. – Segilola Gold Project

Description	$/t milled
Reagents & Consumables	6.47
Maintenance	1.39
Labour[1]	5.26
Power	4.85
Laboratory	0.61
Total	**18.58**

Note: 1) Does not include discretionary annual bonus which is estimated at $0.83/t milled.

REAGENTS/CONSUMABLES

Table 21-4 summarises the reagents used in processing ore.

TABLE 21-4 REAGENT AND CONSUMABLES COSTS
Thor Explorations Ltd. – Segilola Gold Project

Description	Consumption, kg/t	Cost $	$/t
Crusher Wear Parts		128,000	0.21
Mill Liners		500,900	0.83
Mill Balls		1,590,700	2.65
Sodium Cyanide	0.24kg/t	420,500	0.70
Hydrated lime (60% CaO)	0.19kg/t	50,000	0.08
Activated Carbon	0.01kg/t	19,500	0.03
Hydrochloric Acid (32%)	108kg/t carbon	55,300	0.09
Sodium Hydroxide (+99%)	24kg/t carbon	25,800	0.04
Sodium metabisulphite	1.12kg/t	443,800	0.74
Copper sulphate	0.4kg/t	636,000	1.06
Borax	0.8kg/kg AU	3,500	0.006
Silicon dioxide	0.25kg/kg AU	1,300	0.002
Nitric Acid	0.5kg/kg AU	2,100	0.003
Hydrochloric Acid	0.5kg/kg AU	700	0.001
Sodium Nitrate	0.25kg/kg AU	500	0.001
Sodium carbonate	0.6kg/kg AU	3,600	0.006
Total		**3,882,100**	**6.47**

MAINTENANCE

The maintenance cost estimated for the process plant operation totals approximately $832,432,000 per year and covers mechanical spares and wear parts, but excludes crushing and grinding wear components, media, and general consumables, which are covered in the consumables cost estimate above. The maintenance cost excludes all payroll maintenance labour (covered below under Labour).

The maintenance cost for mobile equipment totals approximately $57,000 per year and was estimated based on unit costs for maintenance of medium/heavy equipment and other mobile equipment for the process plant. This excludes maintenance cost of light vehicles which is covered under G&A Operating Costs.

LABOUR

The positions, number of people, and salary estimates for processing are summarized in Table 21-5.

TABLE 21-5 SUMMARY OF LABOUR COSTS FOR THOR OPERATIONS
Thor Explorations Ltd. – Segilola Gold Project

Description	Headcount Number	Annual $000s
Administration	36	469
Mining	18	1,318
Processing	52	1,226
Maintenance	33	640
Total	**139**	**3,652**

POWER

The power consumption estimates are provided in Table 21-6.

TABLE 21-6 SUMMARY OF POWER CONSUMPTION ESTIMATES
Thor Explorations Ltd. – Segilola Gold Project

AREA	Total Installed Power kW	Maximum Continuous Power kW	Maximum Load Utilization Hours h	Annual Power Consumption kWh
0.4kV				
Crushing Section	235	139	4,368	425,786
Crushed Ore Stockpile	50	31	8,000	125,484
Milling Section	647	402	8,000	2,251,917
Leaching and Adsorption	534	346	8,000	1,938,328
Tailings Handling	265	118	8,000	658,448
Elution and Electrowinning	316	221	2,672	413,078
Gold Room	200	140	1,440	141,120
Reagent Preparation	115	66	8,000	366,940
Air Services	1,800	630	8,000	3,528,000
Carbon Regeneration	130	89	960	59,815
Water Services	248	83	8,000	466,200
Auxiliary facilities	680	489	3,200	1,094,240
Subtotal	5,219	2,753		11,469,355
Multiply by Coefficient 0.9	5,219	2,478		10,322,420
Transformer loss		53	8,000	297,374
Equivalent to10kV total	5,219	2,531		10,619,794
10kV Ball Mill and SAG Mill	2,950	2,655	8,000	14,868,000
10kV Total	8,169	5,186		25,487,794
Emergency Diesel Genset	640	640	530	339,200
Total Power Cost, $				2,867,015
Total Power Cost, $/t				4.78

LABORATORY

The operating cost for the laboratory is estimated to be approximately $366,000 per year. The laboratory and assay cost allowance provide for plant control samples and grade control costs. Provision has been made for external metallurgical testing.

INFRASTRUCTURE OPERATING COSTS

The bulk of the infrastructure operating cost is made up of the tailings dam management. Unit operating costs are presented in Table 21-7 which includes post-closure reclamation costs.

TABLE 21-7 SUMMARY OF INFRASTRUCTURE OPERATING COSTS
Thor Explorations Ltd. – Segilola Gold Project

Description	LOM Cost $000s	Annual Cost $000s	$/t milled
TMF	4,317	632	1.44
Tailings Slurry System	1,188	169	0.38
Water Storage Dam	588	84	0.19
Water Mgmt. Misc.	451	64	0.14
Total	**6,544**	**949**	**2.18**

Powerline operating costs incurred to provide the TMF and WDS areas with power are estimated at $56,000 per year.

G&A OPERATING COSTS

Non-labour G&A costs run a nominal $3.6 million per year at $5.8/t milled LOM. Unit operating costs are presented in Table 21-8.

TABLE 21-8 SUMMARY OF G&A OPERATING COSTS
Thor Explorations Ltd. – Segilola Gold Project

Description	Annual Cost $000s	$/t milled
Personnel (Non-Labour)	344	0.56
Site Office	132	0.21
Light Vehicles	258	0.42
Consultants	252	0.41
CDA budget & Fees	424	0.69
Insurances	399	0.65
Security	650	1.05
Camp Management	1,046	1.70
Miscellaneous	84	0.14
Total	**3,589**	**5.82**

OFF-SITE OPERATING COSTS

Doré freight and refining costs are described elsewhere in the report.

22 ECONOMIC ANALYSIS

ECONOMIC CRITERIA

A Cash Flow Projection has been generated from the LOM production schedule and capital and operating cost estimates, and is summarised in Table 22-3. All currency is in US dollars. A summary of the key criteria is provided below.

PRODUCTION

- The pre-production period starts in Q2 2019 with just over 17 months of construction and five weeks of commissioning, with operational trial/commercial production starting in September 2020.
- Mining starts in June 2020 with one month of pre-stripping and continues for 46 months with a maximum rate of 48,000 tpd mined at an average strip ratio of 16.3:1.
- Commissioning starts in July 2020 and processing operates for 58 months at a production rate of 625,000 tpa.
- LOM production is 3.00 Mt at a grade of 4.20 g/t Au, with a constant gold recovery of 97%. The total gold recovered is estimated to be 393,000 oz Au, at an annual LOM average rate of 66,000 oz Au. No Inferred material was included in the gold production estimate.

REVENUE

Payable metal sales total 393,000 oz Au and an annual LOM average 66,000 oz Au per year. The metal price applied was $1,300 per troy ounce Au.

COSTS

Project pre-production initial capital totals $82.3 million, with no sustaining capital estimate included as all maintenance costs are included in operating costs.

The average LOM unit operating costs is $77.51/t milled, as given in Table 21-2. The initial capital currency split is expected to be 90% US$, 10% GBP, while the operating cost split is expected to be 90% US$, 10% NGN.

The following foreign currency exchange rates were used

- US$: GBP – 0.76 per GBP
- US$: NGN – 364.5 NGN per US$
- US$: EUR – 0.88 per EUR

ROYALTIES AND GOVERNMENT PAYMENTS

Royalties are payable to the Nigerian government at a rate of 5,400 Naira per recovered ounce of gold. Using 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.

Royalties and other government payments total $13.4 million, or $4.45/t milled, over the LOM as shown in Table 22-1.

TABLE 22-1 ROYALTIES AND GOVERNMENT PAYMENTS
Thor Explorations Ltd. – Segilola Gold Project

Item	$ 000s
Gov't – $14.89 per payable ounce gold	5,852
Private – 3.0% NSR (capped at $7.5M)	7,500
Total	**13,352**

INCOME TAXES, WORKING CAPITAL, AND CLOSURE COSTS

Income taxes/contributions, upfront working capital, and reclamation/closure costs are shown in Table 22-2. Withholding taxes on corporate dividends and interest payments are not incorporated into the Project economic analysis.

TABLE 22-2 INCOME TAXES, WORKING CAPITAL, AND OTHER
Thor Explorations Ltd. – Segilola Gold Project

Item	$ 000s
Corp. Income Taxes Paid	None
Upfront Working Capital (Credit)	(653)
Mine/Plant Closure/Reclamation	2,172
Infrastructure Reclamation (in opex)	968
Salvage Value	0
Total	**2,487**

CORPORATE INCOME TAX

The Project economic analysis incorporates a five-year statutory income tax holiday from production start-up in July 2020, extended for an additional two years so the Project does not generate tax payable liabilities during LOM.

In accordance with the mining industry pioneer status, Thor expects to receive a five-year exemption from customs duties, levies, and taxes including income tax. However, to estimate the risk of having tax payable liabilities due to the government, a proforma income tax calculation is included in the analysis based on general assumptions. Deductions from income for the purpose of estimating income subject to tax include the following items:

- Operating expenses are deducted 100% in year incurred.
- Depreciation is done with Units of Production (UoP) method.
- Net operating losses are carried forward indefinitely but not be used for prior tax years.
- No stockpile adjustments to operating deductions are included.
- Income tax rate of 30% applied to taxable income.

RPA estimates the LOM income tax payable liability to the Project if the tax holiday shield was not applied would be approximately $55 million with an effective tax rate of 21%.

UPFRONT WORKING CAPITAL

RPA proforma working capital assumptions include five days sales outstanding for accounts receivable, 14 days payable outstanding for labour, and 30 days payable outstanding for supplies, and 3% of cumulative annual balance of property, plant, and equipment (PP&E) for consumable inventories. Plant first fill and critical spares are included in the initial capital estimate.

With relatively small accounts receivable and inventories offset by a much larger accounts payable, an upfront working capital credit of $641,000 is generated in the analysis from start of construction in April 2019 through the first month of commercial production in September 2020. However, all working capital adjustments are recaptured at the end of mine life thus net to zero over the LOM.

RECLAMATION/CLOSURE COSTS

Reclamation/closure costs of the mining and mill areas are estimated at $2.175 million over five years starting in the first year after cessation of processing activities.

An additional $968,000 in post-closure reclamation costs for infrastructure areas was provided for the original 500,000 tpa scenario including TMF ($825,000), water storage dam ($110,000), and water management items ($33,000) are estimated for the Project. The tailings slurry system did not have a reclamation cost estimate. These costs are classified as operating expense as part of the infrastructure cost. It is assumed that the reclamation/closure costs for the 500,000 tpa scenario are valid for the 650,000 tpa base case.

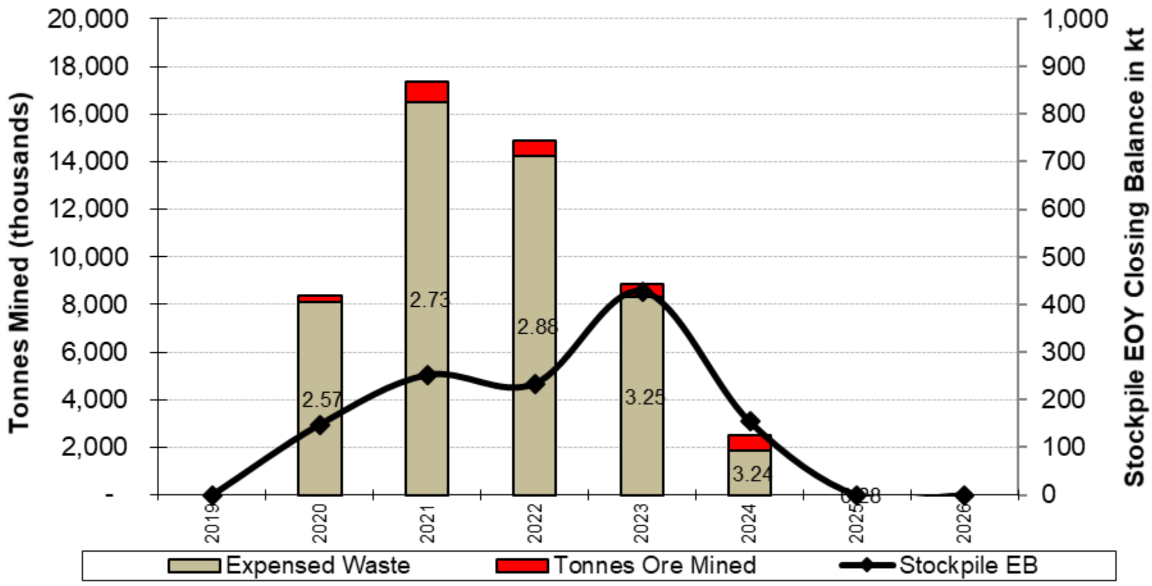
While the quoted closure costs are spread over a five-year period, for simplicity and to avoid starting the final reclamation work during the wet season of May to October, the reclamation/final closure costs are assumed to start in January 2026, approximately eight months after the final processing of ore in June 2025, and complete in December 2026.

No salvage estimates were included in the evaluation.

CASH FLOW ANALYSIS

The LOM plan for the Project results in an average annual ore production of 625,000 tpa and includes significant variations in the ore and waste mining schedule and head grades over its planned five-year life. The base case includes some stockpiling of higher-grade ore in the early stages of mining, which will be processed later in the mine life. These variations are shown in Figures 22-1 and 22-2 and the resulting impact on the pre-tax free cash flow profile is shown in Figure 22-3.

FIGURE 22-1 MINE PRODUCTION PROFILE



Note: Numbers in chart are mining operating costs in $/t moved

FIGURE 22-2 MINED ORE VERSUS MILL PRODUCTION PROFILE

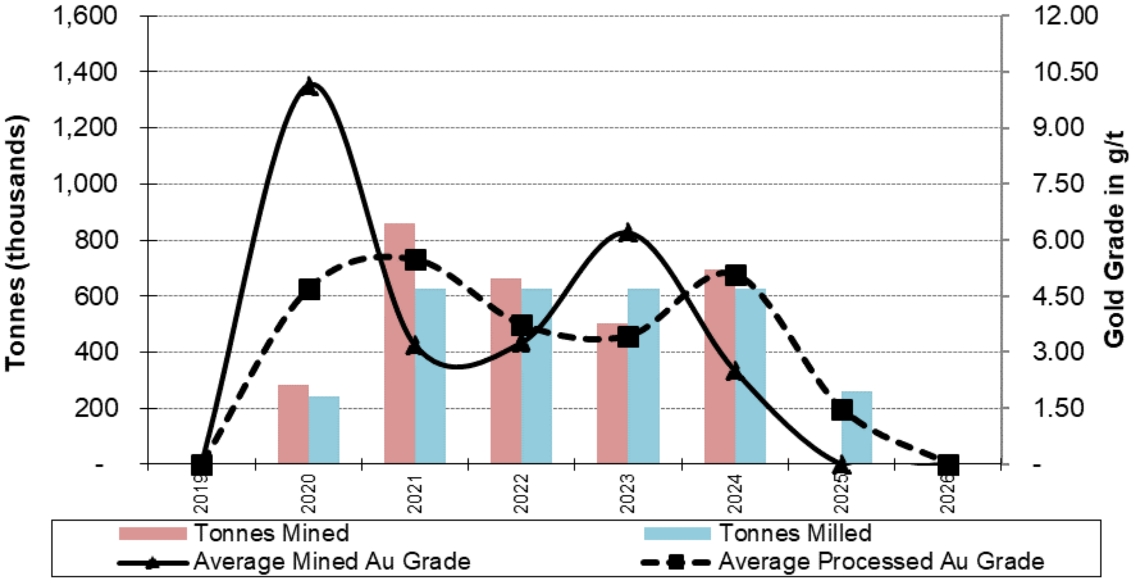


FIGURE 22-3 PROJECT PRE-TAX METRICS SUMMARY

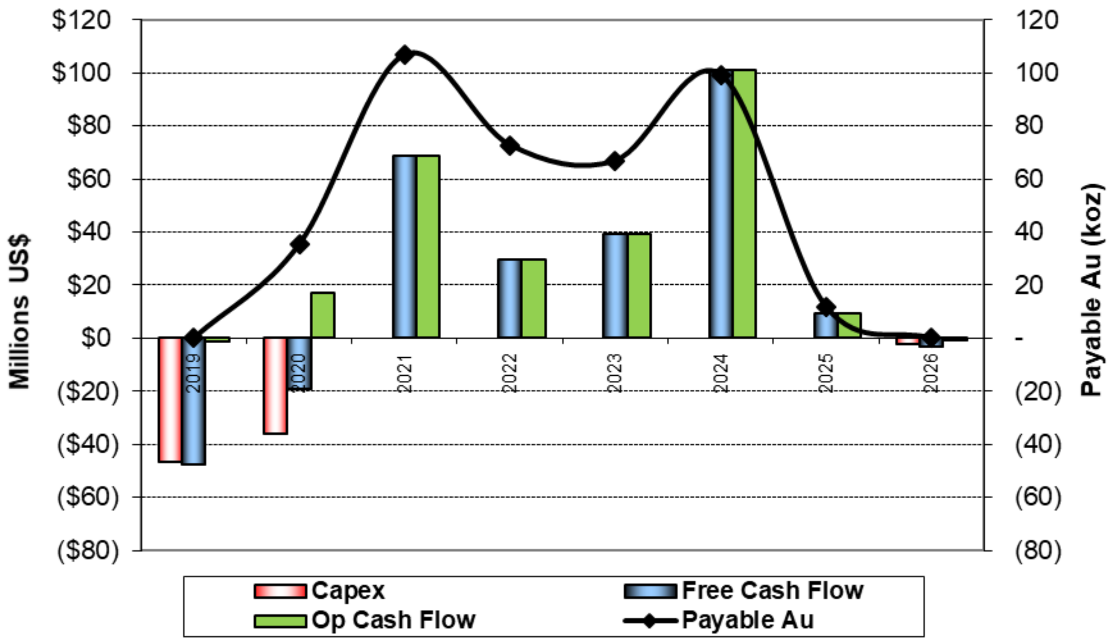


Table 22-3 shows the LOM total metrics for the Project as currently designed. The economic model was constructed on a 100% equity basis with end of period discounting. Whereas the data inputs and schedule are in a monthly format, the results are presented in both quarterly and annual formats with official metrics based on the annual format results.

TABLE 22-3 INDICATIVE PROJECT ECONOMICS
Thor Explorations Ltd. – Segilola Gold Project

Item	$ 000s
Gold Price ($/oz)	$1,300
Payable Gold (oz)	393,000
Total Gross Revenue	**510,900**
Mining Cost	(153,000)
Process Cost	(55,800)
G & A Cost	(17,300)
Infrastructure/Closure Cost	(6,500)
Site Operating Costs	**(232,700)**
Offsite Costs	(2,600)
NSR Royalty	(13,400)
Total Cash Costs	**(248,700)**
Operating Margin (EBITDA)	**262,300**
Operating Margin %	51%
Taxes	0
Working Capital*	(0)
Operating Cash Flow	**262,300**
Initial Capital	(82,300)
Sustaining Capital	0
Closure/Reclamation Capital	(2,200)
Total Capital	**(84,500)**
Pre-tax/Tax Shielded Free Cash Flow	**177,800**
Pre-tax/Tax Shielded NPV @ 5%	**138,000**
Pre-tax/Tax Shielded IRR	**50.5%**
Pre-tax/Tax Shielded Payback Period (Yrs.)	**1.4**

*Note: An upfront working capital credit of $641,000 is generated in period April 2019 through September 2020. All working capital adjustments are recaptured at the end of mine life and so net to zero over LOM.

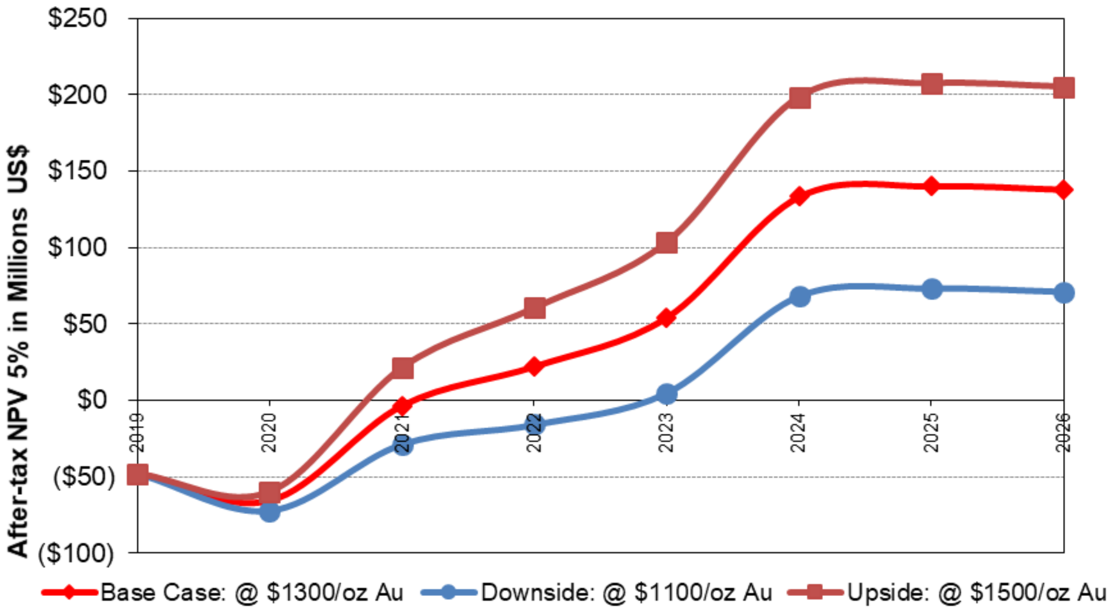
On a pre-tax (or tax holiday shielded) basis, the undiscounted cash flow totals $177.8 million over the mine life with a Net Present Value (NPV) at 5% of $138 million and the Internal Rate of Return (IRR) of 50.5%. The undiscounted payback from start of commercial production is approximately 1.4 years.

RPA also calculated the Project metrics on an after-tax basis using RPA generalized tax assumptions and without the statutory tax holiday shield to test the impact of the tax holiday on overall Project economics. The metrics included undiscounted cash flow of $123 million over the mine life with an NPV at 5% of $93 million and an IRR of 37.9%.

Figure 22-4 shows the cumulative NPV curves for the Project at a 5% discount rate at three different gold price points listed below:

- Base Case: $1,300 per ounce;
- Upside: $1,500 per ounce;
- Downside: $1,100 per ounce.

FIGURE 22-4 PROJECT PRE-TAX CUMULATIVE NPV CURVES



From Figure 22-4, the breakeven pre-tax NPV at 5% gold price is estimated at $887 per ounce. Each curve has a flatter slope in the period 2021 to 2022 that becomes steeper with increasing gold price.

With respect to All-in Sustaining Costs (AISC), Table 22-4 shows an average LOM AISC of $638 per ounce with average annual gold sales during the six years of operation of 66,000 oz per year (393,000 oz over LOM).

TABLE 22-4 ALL-IN SUSTAINING COSTS COMPOSITION
Thor Explorations Ltd. – Segilola Gold Project

Item	$ 000s	$/oz Au
Contractor Mining	153,000	389
Thor Operations	55,800	142
Fixed G&A	17,300	44
Infrastructure/Closure	6,500	17
Subtotal Site Costs	**232,700**	**592**
Offsite Treatment	2,600	7
Total Operating Costs	**235,300**	**599**
Royalty	13,400	34
Total Cash Costs	**248,700**	**633**
Sustaining Capital Cost	0	0
Closure/Reclamation Capital	2,200	6
Corporate G&A	0	0
Off-Mine Exploration	0	0
Total Sustaining Costs	**2,200**	**6**

Item	$ 000s	$/oz Au
Total All-in Sustaining Costs	**250,800**	**638**
LOM Au Sold (oz)		393,000
LOM Average Au Sales per Year (oz)		66,000

SENSITIVITY ANALYSIS

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities:

- Head grade
- Gold recovery
- Gold price
- Exchange rates
- Operating costs
- Capital costs
- Discount rates

Pre-tax NPV and IRR sensitivities over the base case have been calculated for -20% to +20% variations for metal-related categories and exchange rates.

For gold recovery, due to the high overall recovery factor of 97%, the sensitivities have been calculated from -20% to +2.5%. For operating costs and capital costs, the pre-tax sensitivities over the base case have been calculated at -10% to +10% variation.

The sensitivities are shown in Table 22-5 and in Figures 22-5 and 22-6, respectively. Note that in this case, the pre-tax sensitivities are the same with the LOM tax holiday assumption in the economic model.

The analysis shows that the Project is most sensitive to gold grade, price, and (downside) recovery. Capital and operating costs have less effect than the metal-related categories mainly due to the small size and short life of the operation. Exchange rates have very little impact with most costs and all revenue expended in US dollars.

TABLE 22-5 PRE-TAX SENSITIVITY ANALYSIS
Thor Explorations Ltd. – Segilola Gold Project

Factor Change	Head Grade (g/t Au)	NPV at 5% ($ 000s)	IRR (%)
0.80	3.36	$52,500	23.4%
0.90	3.78	$95,200	37.2%
1.00	**4.20**	**$138,000**	**50.5%**
1.10	4.62	$180,800	63.3%
1.20	5.04	$223,600	75.7%

Factor Change	Recovery (% Au)	NPV at 5% ($ 000s)	IRR (%)
0.80	77.60	$52,500	23.4%
0.90	87.30	$95,200	37.2%
1.00	**97.00**	**$138,000**	**50.5%**
1.03	99.43	$148,700	53.7%
1.03	99.43	$148,700	53.7%

Factor Change	Metal Price ($/oz Au)	NPV at 5% ($ 000s)	IRR (%)
0.80	$1,040	$51,100	22.9%
0.90	$1,170	$94,500	37.0%
1.00	**$1,300**	**$138,000**	**50.5%**
1.10	$1,430	$181,500	63.5%
1.20	$1,560	$225,000	76.1%

Factor Change	Exchange Rate (NGN: US$)	NPV at 5% ($ 000s)	IRR (%)
0.80	292	$137,100	50.2%
0.90	328	$137,600	50.3%
1.00	**365**	**$138,000**	**50.5%**
1.10	401	$138,500	50.6%
1.20	437	$138,900	50.7%

Factor Change	Operating Costs ($ 000s)	NPV at 5% ($ 000s)	IRR (%)
0.90	$225,020	$137,100	52.3%
0.95	$229,003	$137,600	51.4%
1.00	**$232,987**	**$138,000**	**50.5%**
1.05	$236,970	$138,500	49.5%
1.10	$240,953	$138,900	48.6%

Factor Change	Capital Costs ($ 000s)	NPV at 5% ($ 000s)	IRR (%)
0.90	$76,275	$146,100	57.1%
0.95	$80,392	$142,100	53.6%
1.00	**$84,508**	**$138,000**	**50.5%**
1.05	$88,625	$134,000	47.5%
1.10	$92,742	$130,000	44.8%

FIGURE 22-5 PRE-TAX NPV 5% SENSITIVITY ANALYSIS

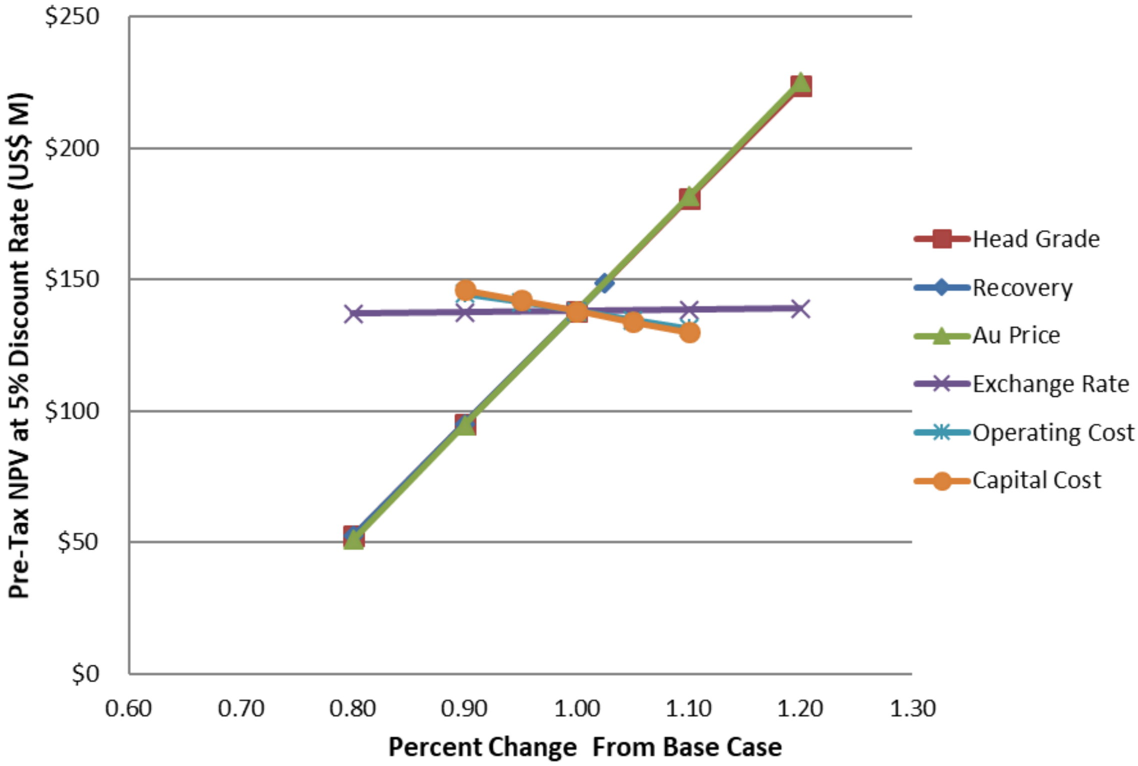
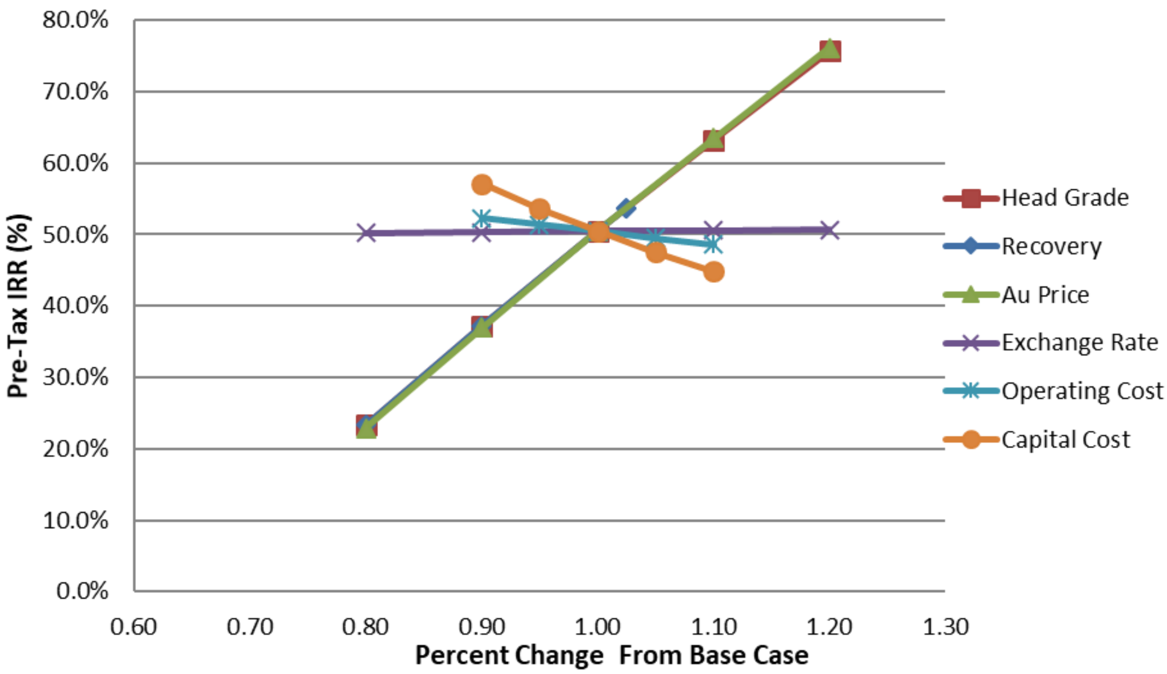
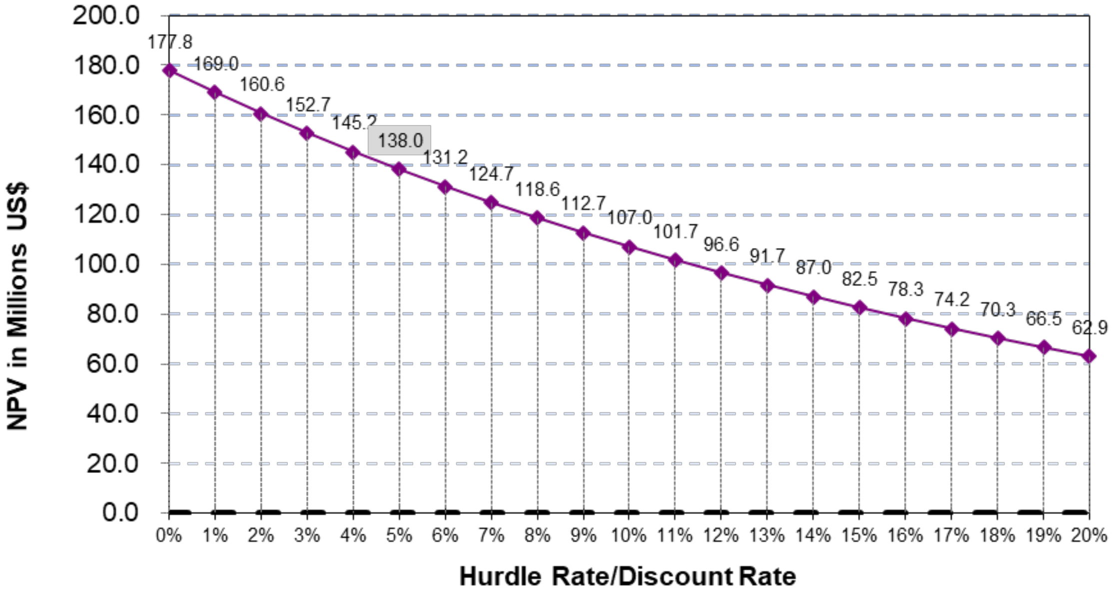


FIGURE 22-6 PRE-TAX IRR SENSITIVITY ANALYSIS



A sensitivity analysis of discount rates presented in Figure 22-7 shows that the Project as currently designed would have positive NPV/IRR even at a 20% discount rate.

FIGURE 22-7 PRE-TAX DISCOUNT RATE SENSITIVITY ANALYSIS



23 ADJACENT PROPERTIES

This section is not applicable.

24 OTHER RELEVANT DATA AND INFORMATION

PRELIMINARY ECONOMIC ANALYSIS OF UNDERGROUND OPERATION

A PEA was completed for the portion of the Segilola deposit that was not included in the open pit design in order to estimate the potential economics of an underground operation. An underground mine design was completed based on Indicated and Inferred Mineral Resources using basic geotechnical and metallurgical data largely extrapolated from the open pit work. The proposed underground mine will run concurrently with the open pit, with the higher-grade underground mineralisation used to increase the volume and grade of the plant feed. This section details this work.

The economic analysis contained in this section of the report is based, in part, on Inferred Mineral Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realised.

MINING

Thor completed a DFS for the Segilola deposit, during which an open pit was designed. Outside of this open pit, resources remained which showed potential value. An underground mine design was completed based only on the resources found outside the open pit. These resources were re-defined for underground mining, where a higher cut-off grade was used to determine the extent of the viable material. The PEA design work was based on Indicated and Inferred material.

No trade-off study was performed between the open pit and the potential underground operation to optimise the operating cost and production schedules of the two operations. Future work should consider this and might see an expansion of the underground mine into the base of the planned open pit, reducing the final open pit strip ratios and operating cost of the open pit mining.

MINE DESIGN APPROACH

The block model and geological wireframes were reviewed for scale and potential mining methods. Following this, basic production rates were estimated using an empirical drop-down rate method and a requirement that the LOM is at least three years. This work indicated that the viable production range was 140,000 tpa to 180,000 tpa.

The Project was benchmarked with operations which were likely to have a similar mining method and production rate. Costs from these operations were used in conjunction with the DFS work to estimate a first pass cut-off grade.

The cut-off grade was used for stope optimisation, in an iterative process. Preliminary results were used to guide mining method decisions, stope dimensions with respect to geotechnical limitations and planned and unplanned dilution. Geotechnical limits and minimum mining widths were applied, and the further results were adjusted for continuity, practicality, and other factors.

These stopes were used to guide the location of level development, primary development, and supporting development.

Primary development was designed to suit the planned level configuration, mining method, and to be cost effective and to allow a rapid ramp-up. Secondary development was included as required to support production.

The elements of the design were then scheduled using basic productivity factors and a mining sequence.

CUT OFF GRADE

An initial cut-off grade of 2.99 g/t Au for a gold price of $1,300/oz was estimated prior to the design work. The inputs for this calculation are given in Table 24-1. Later work saw changes to the royalties and the final mining cost. The same calculation at a gold price of $1,500/oz returned a cut-off grade of 2.58 g/t Au which was applied to the resource.

TABLE 24-1 INITIAL CUT-OFF GRADE ESTIMATE
Thor Explorations Ltd. – Segilola Gold Project

	Units	Value		
Costs				
Mining	$/t	50		
Processing and G&A	$/t	25		
Refining	$/t	1		
Capital Cost and Contractor Cost Premium	$/t	20		
Total	$/t	96		
Gold Price	$/oz	1,300		
Royalties				
TML	%	1.5		
Ratel	%	1.5		
Government	%	1.2		
Effective Gold Price	$/oz	1,245.4		
Cash Breakeven	g/t	2.39		
Process Recovery	%	97		
Head Grade (ROM) Cut-off	g/t	2.46		
Gold Recovery (Mine Call Factor)	%	95		
Dilution Estimate Case		Best	High	With Development
Proportion of Dilution	%	14.2	17.7	19.2
Estimate of Dilution Grade	g/t	0.50	0.50	0.50
In-situ Cut-off Grade	g/t	2.89	2.96	2.99
Optimisation Head Grade	g/t	2.59		
Optimisation In-situ Grade	g/t	2.7		
Applied Design Grade (includes development dilution)	g/t	2.99		
Resource Cut-off Grades				
Price	$/oz	1,500		
In-situ grade	g/t	2.58		

GEOTECHNICAL PARAMETERS

A geotechnical study was completed by POB&A. This work was an extension of the geotechnical work completed for the DFS, but with less data and consequently a lower level of accuracy. Of note is that no oriented drill core was available for the underground geotechnical study and the in-situ stress is not known.

First pass stope spans were estimated using the available data and the Modified Stability Graph method. For the selected interlevel spacing of 15 m, stope strike length of 26 m is indicated, after which rib pillars are required. The planned use of backfill means that rib pillars of a 1:1 width to height ratio should be left to form a barrier for the waste rock backfill, which will also support the hanging wall. In order to reduce the rib pillar requirement and improve ore recovery, cemented rockfill is planned. The sequencing of mining, location, and size of any rib pillars is simplified by the lack of mining of any two lodes adjacent to each other.

A 20 m crown pillar was left and scheduled for extraction at the end of the LOM, with a planned recovery of 50%.

The decline was placed in the better-quality granodiorite in the hanging wall, and is offset from the lodes by approximately 40 m, which is within the recommended 35 m to 50 m range. Development will require bolting with mesh. Cable bolts are likely to be required in large intersection areas.

MINING METHOD

The various mining methods that could be applied to the underground mineralisation were considered. A reasonably applicable method was selected; however, various options remain and should be further investigated. The selected method was sub-level open stoping (SLOS) with rock fill, in a stope by stope sequence. Other methods that were considered included shrinkage stoping and drift and fill.

The average stope thickness was initially estimated at 3.2 m, at a dip of 59°. This made the mineralisation suitable to SLOS.

Geotechnical factors that resulted in practical stopes supported a level spacing that ranged from 25 m to 15 m, with a wider level spacing resulting in shorter stopes on strike. The maximum planned stope width is 6.8 m, however, almost all the potential stopes are under 6 m thick, with an average planned stope thickness at 3.6 m, and at least 60 stopes which contain an orebody of under 2.5 m thick. Given the narrowness of the mineralised zone, dilution will be substantial and the orebody shape and accuracy of drilling will be important factors in dilution control. The orebody shape, in this case the shape of the pay envelope around the mineralised zone, is uncertain due to the low level of geological confidence. Accurate drilling will be required to keep dilution low. As such, a conservative level spacing of 15 m was selected, limiting the drilling span to 13.5 m at 59º. A closer level spacing is possible but the extra development is likely to offset the gain in reduction of dilution. Stopes are designed to be 20 m on strike with a minimum planned width of 2 m, with a further 0.3 m of dilution either side, for a total minimum stope width of 2.6 m. Approximately 1.3% of the total stope tonnes are at the minimum mining width.

Waste rock fill is planned in a chequerboard approach, essentially replacing mined out voids with waste rock. This required waste rock will be transported back to the stopes, which can be done using waste passes and LHDs. Sufficient waste rock is expected to be available from the open pit operation, and trucks hauling ore can return with waste. Cement binder needs to be added to some portion of the waste. The chequer board approach has discrete stopes mined, then filled and cured while mining continues at a different location.

The mine design was completed in Deswik CAD, within which the Stope Optimiser was used to generate the stopes.

OREBODY ACCESS

The mineralised lodes are either immediately below the planned pit or near to the surface. A single ramp was planned that will rapidly reach mineralisation. The starting point of the ramp will be just within the planned footprint of the open pit, which reduces cost and impact and is best suited for the location of the crusher.

Given the probable output of the mine, one ramp is sufficient to handle the traffic and the intake ventilation. A split in the ramp may improve the time taken to open enough working fronts to maintain a constant production profile.

Cross cuts will be developed off the ramp at 15 m vertical intervals, for each mining level, with a re-muck bay where the cross cut intersects the level development. The ramp, cross cut, and remake bay have been sized to accommodate trucks, while the level development has been sized for LHDs only, in order to reduce dilution and development costs.

Level development will be predominantly in mineralisation, except where lodes are being connected or low-grade areas are being mined through. Each level will be developed over the full strike length for stoping to be mined in retreat. Level lengths vary from 90 m to 540 m, with an average length of 300 m.

A sequence of ventilation raises has been planned in two locations, both daylighting into the open pit. Twinned with these raises are waste passes to allow waste rock to be effectively returned to the mine. Trucks can tip waste rock into these passes, after which it will be collected by LHDs on the working levels, for it to be tipped into worked out stopes.

DILUTION AND LOSSES

Dilution was included as planned dilution and an unplanned dilution allowance.

Planned dilution includes any below cut-off grade material that must be mined to retain practical stope and development shapes. Any stopes below cut-off grade after the inclusion of this material were excluded from the design. Any segments of development in mineralisation that is diluted below the marginal grade reports to waste, with the remainder reporting as ore.

Unplanned dilution in the stopes was included as a minimum mining true width and a fixed thickness of dilution in the hanging wall and footwall. Any planned stope had 0.3 m of unplanned dilution added to both the footwall and hanging wall, increasing the overall width by 0.6 m. The minimum mining width is 2.6 m, which is made up of a 2.0 m wide planned stope and 0.3 m of unplanned dilution either side.

As the stopes are defined by a cut-off grade and not a geological contact, this dilution will carry some grade, which was estimated to be 0.48 g/t Au and included in dilution calculations. Approximately 10% unplanned dilution at zero grade was included in ore development.

A 5% ore loss was estimated for all normal stope and development ore tonnes. A 50% loss was estimated for the crown pillar stopes.

MINE DESIGN

The mine design includes 525 kt of diluted mineralised material at a grade of 6.33 g/t Au. Approximately 287 kt of waste is required to be mined. The material included in the design is made up of stope tonnes (75% of the total, at 6.40 g/t Au), ore drive tonnes (16% at 5.81 g/t Au), and crown pillar tonnes (9% at 6.76 g/t Au). The crown pillar has been included in the total mineable material, but at a 50% recovery rate and at the end of the LOM.

The majority of the material included in the design is Inferred mineralisation at 60%, with Indicated mineralisation being 12% of the total. At 25%, mining dilution makes up a substantial portion of the planned mineralised material, increasing the mineable potential to larger than the Mineral Resource used to generate it. A small portion of unclassified material, 3% of the total, was incorrectly included in the design due to the configuration of the design and the block model. This amount is not considered material and was left in the design.

Approximately 10,000 m of horizontal development will be required over the LOM.

Tables 24-2, 24-3, and 24-4 give a basic breakdown of the material included in the design.

TABLE 24-2 MATERIAL CONTAINED IN THE MINE DESIGN
Thor Explorations Ltd. – Segilola Gold Project

	Unit	Total/Ave	% of Total
Total	kt	812	
Ore	kt	525	65
Grade	g/t	6.33	
Waste	kt	287	35

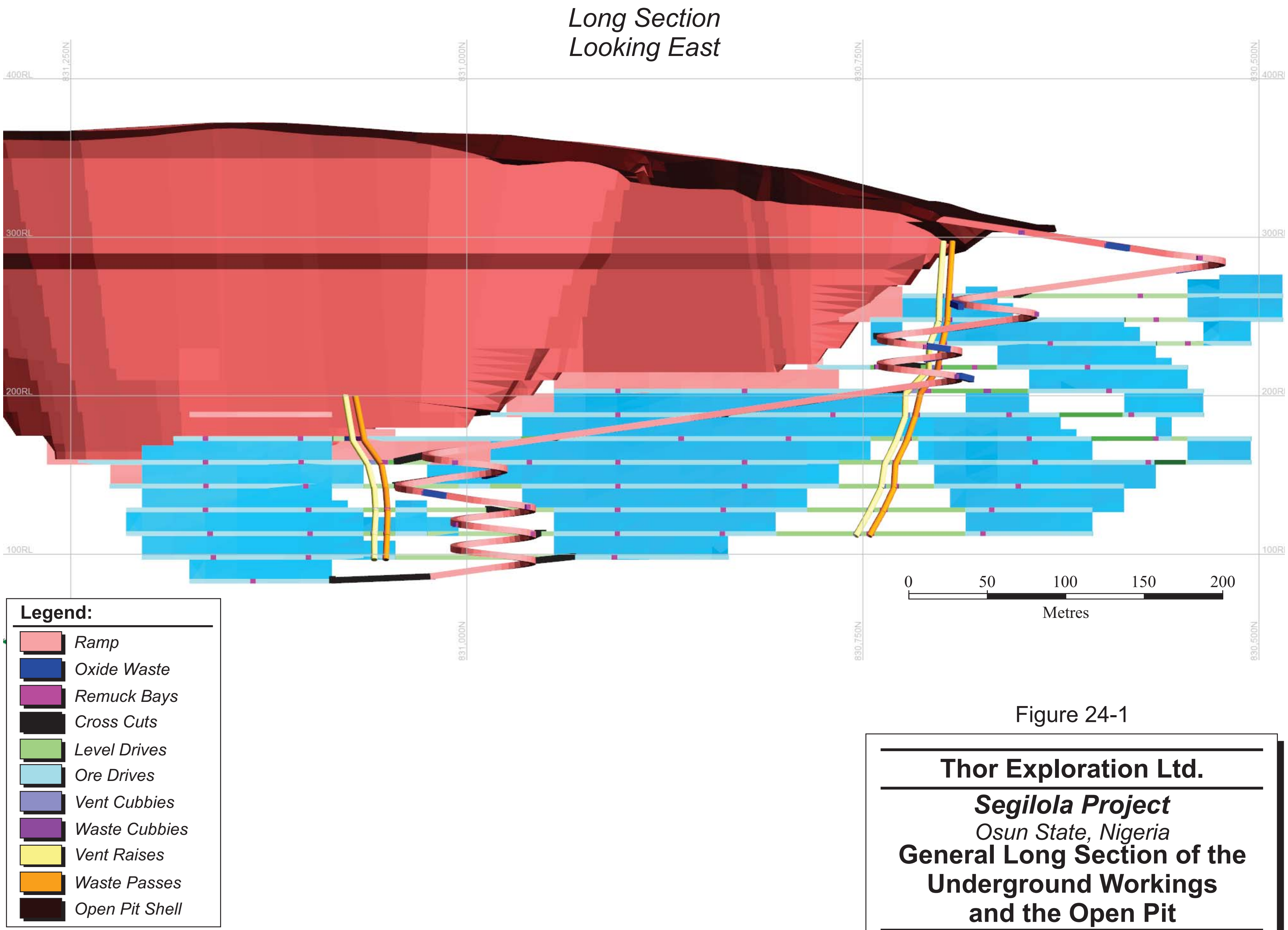
TABLE 24-3 MATERIAL BREAKDOWN BY AREA
Thor Explorations Ltd. – Segilola Gold Project

Area	Kt	g/t	% of Total
Stope	392	6.40	75
Crown Pillar	47	6.76	9
Ore Drives	86	5.81	16

TABLE 24-4 MATERIAL BREAKDOWN BY CLASS
Thor Explorations Ltd. – Segilola Gold Project

Class	kt	g/t	% of Total
Indicated Material	64	9.90	12
Inferred Material	314	7.52	60
Mining Dilution	129	0.45	25
Unclassified Material incorrectly included	17	15.95	3

General views of the mine layout with the planned open pit are shown in Figures 24-1 to 24-4.



March 2019

Source: Auralia, 2019.

Figure 24-1

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

General Long Section of the Underground Workings and the Open Pit

RPA

www.rpacan.com

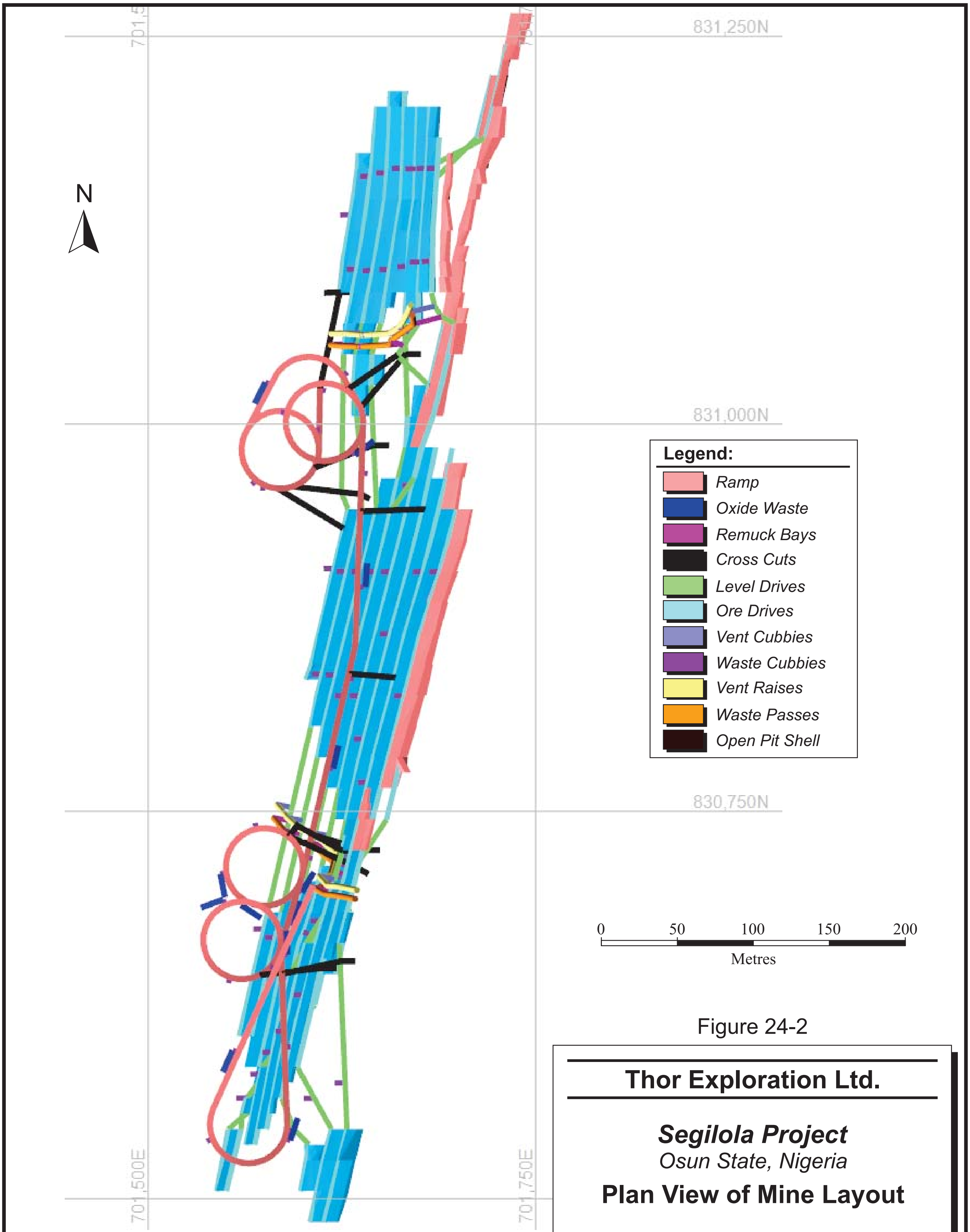


Figure 24-2

Thor Exploration Ltd.

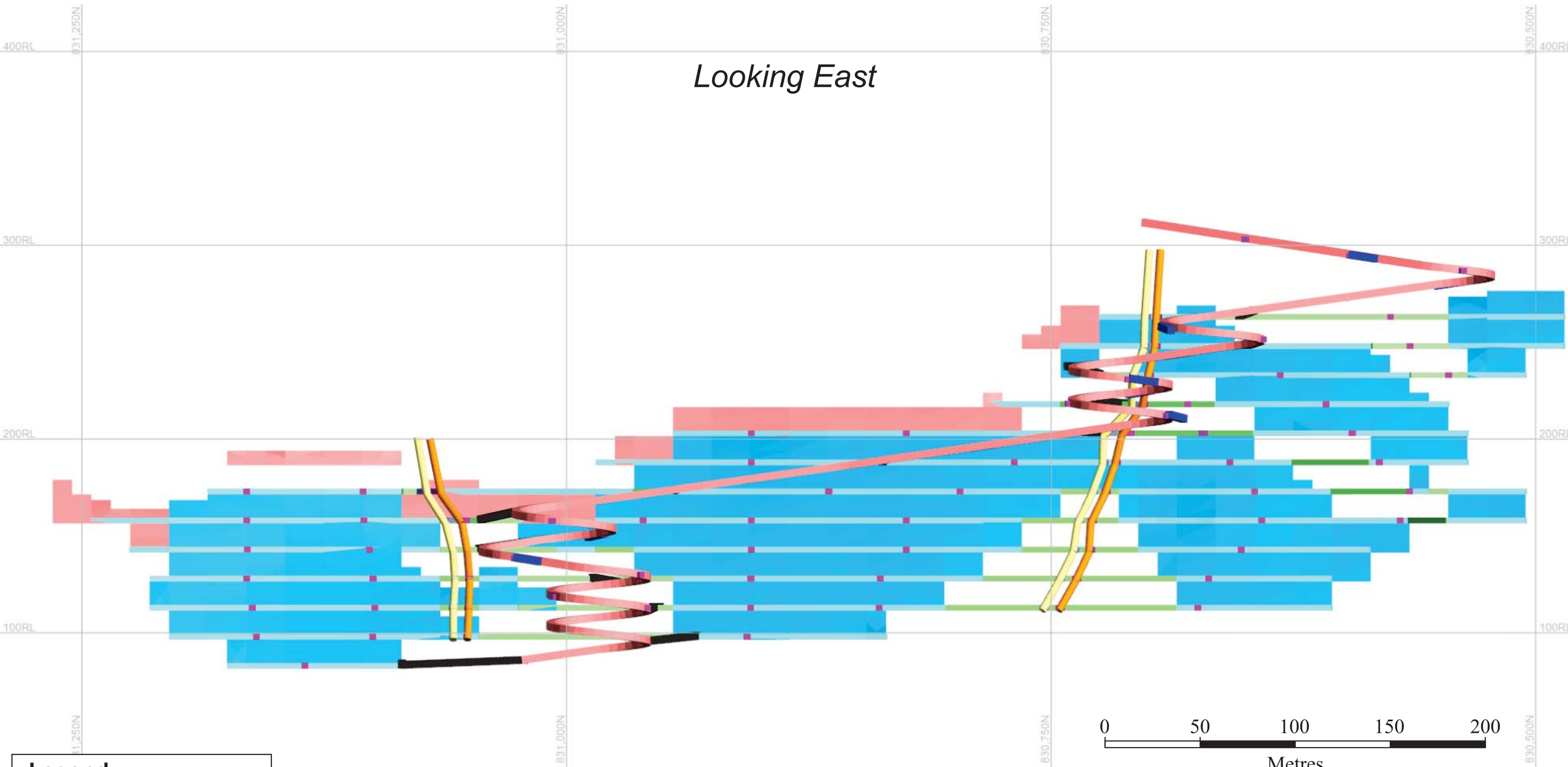
Segilola Project

Osun State, Nigeria

Plan View of Mine Layout

March 2019

Source: Auralia, 2019.



Legend:

- Ramp
- Oxide Waste
- Remuck Bays
- Cross Cuts
- Level Drives
- Ore Drives
- Vent Cubbies
- Waste Cubbies
- Vent Raises
- Waste Passes
- Open Pit Shell

March 2019

Source: Auralia, 2019.

Figure 24-3

Thor Exploration Ltd.

Segilola Project

Osun State, Nigeria

Long Section of Mine Layout

RPA

www.rpacan.com

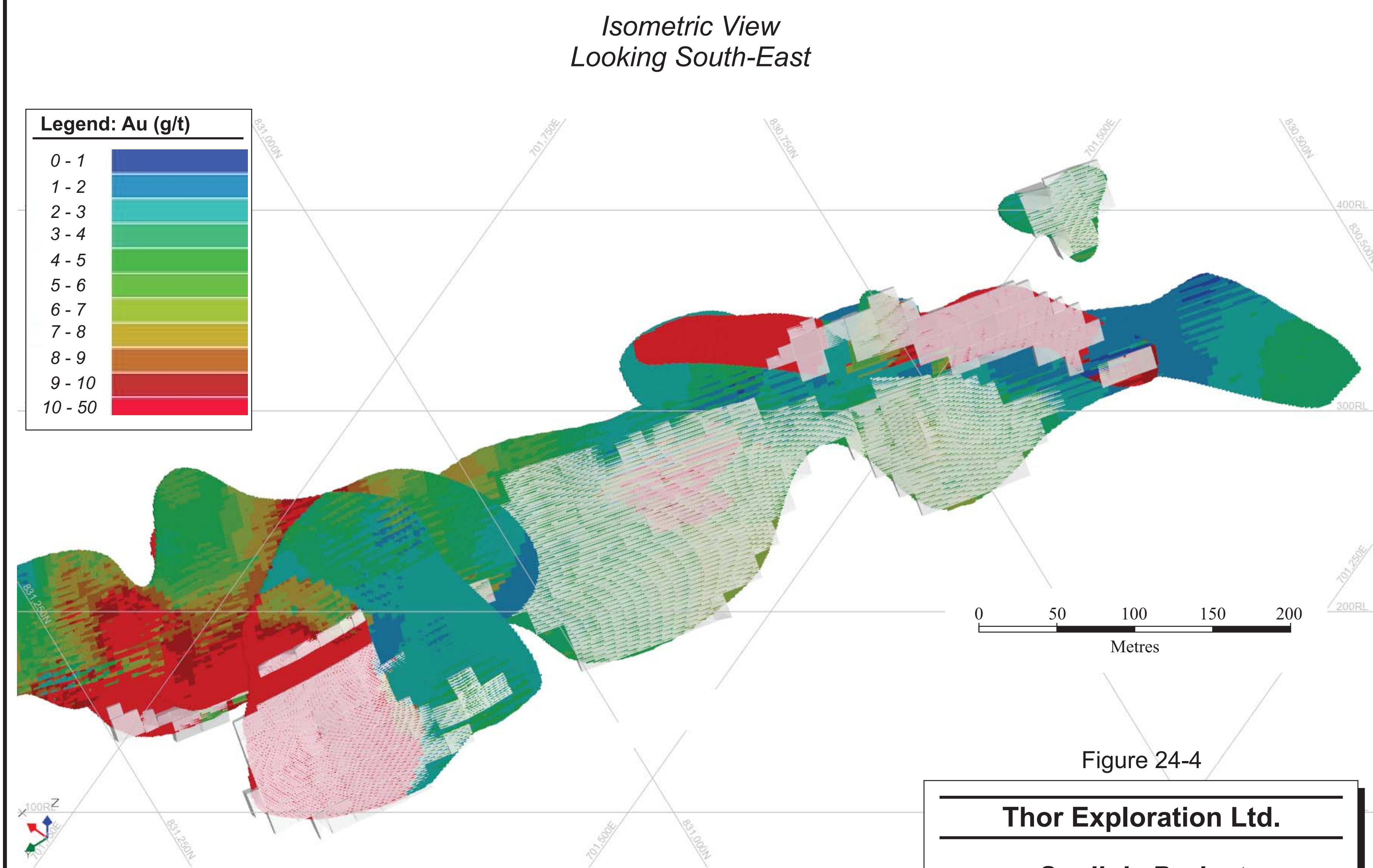


Figure 24-4

Thor Exploration Ltd.

Segilola Project
Osun State, Nigeria

Stope Shapes and Block Model Cells Coloured by Grade (Open Pit is not Shown)

March 2019

Source: RPA, 2019.

EXTRACTION SEQUENCE

Development is scheduled to occur from the ramp, as and when faces become available. Some ends were prioritised to improve the timing of production.

Levels must be mined in retreat, and from the bottom up. In order to allow production before the full development of the mine, levels were paired. The bottom line of stopes of a paired level is mined out first, with the top line of stopes mined out second. The second line of stopes will be drilled off from the surface of the fill in the first line of stopes. In order to allow filling operations to continue below the bottom line of stopes (for the next pair, below), the bottom level drive will need to be kept open while the stopes are filled above it. Some arrangement of mat packs and stulls will be required to retain the rock fill above the level drive.

Levels near the surface of the mine are truncated by the planned open pit, so it will not take long to open these levels in retreat. Later, longer levels should have their development complete by the time the shorter levels are exhausted.

The limits to the stoping sequence are the main limit to the total output from the mine.

MINING

Development has been planned with a slow ramp-up to allow for geotechnical uncertainty and a learning curve. Following that period, an advance rate of 100 m/month was applied to the ramp and 75 m/month per end for other flat ends. A rate of 200 m/month was applied per development team.

Stopes are expected to be fully extracted, filled, and cured over 20 to 28 days, depending on the size of the stope. The tonnage weighted average stope width is 3.6 m, which contains approximately 3,300 t of mineralisation.

The estimated production rate for the operation was between 140,000 tpa and 180,000 tpa, or 11,600 t per month (tpm) to 15,000 tpm. For this tonnage, small production equipment is suitable, such as a 20 t truck and 6.5 t LHD, and a basic twin boom development rig and a top hammer long hole drill rig. Various other machines will be required, such as a charging vehicle, some multi-purpose vehicles (MPVs) and cassettes, and pick-up trucks. Thor plans to contract out the underground mining.

Development will be standard level development with a twin jumbo drill rig drilling the face, blasting, and loading out by an LHD, either into a re-muck bay or directly into a truck.

Once development has reached the end of the half level, stope production can start. Production drilling will be completed, with an initial cut blasted to create a free breaking face, followed by the blasting of rings in sets of 3 to 4. Material will be loaded by LHDs and taken to the re-muck bay where the access cross cut meets the level drive. Material will be re-handled from there into the truck when available, to increase the productivity of the truck.

Trucks will haul the material up the ramp, out of the pit, and to the ROM pad. When returning, trucks may be loaded with sized waste rock for backfill purposes. Substantial waste rock will be available from the open pit mine, and can be sized with the use of a grizzly or small crusher. This rock will be tipped into the top of the relevant waste pass.

After the last ore is loaded out, stopes need to be filled as quickly as possible to allow production to continue on that level. Once a filling barricade is completed, waste will be loaded from the waste pass and tipped into the stope void by dedicated LHDs. A 14-day period has been allowed for curing. The next stope can be drilled while filling and curing take place, and the initial cut can be blasted once the material has cured. Some care will need to be taken to avoid dilution from the rock fill.

PRODUCTION SCHEDULE

The proposed underground operation is best operated concurrently with the open pit, in order to share the operating cost of the processing plant, which has been sized for a throughput substantially higher than can be produced from the underground operation. Concurrent production will also improve the grade being supplied to the plant from the open pit, and level out some of the output in terms of ounces.

In order to best achieve this, November 2020 was selected for the start of development. Should the underground operation be started at a later date, some operating costs will need to be adjusted.

First ore is achieved in the 8th month, after the initial ramp and level development to open up the orebody. From the 9th month, 6,000 tpm will be mined for six months, while development continues and operations are improved. This is followed by production at 13,000 tpm (156,000 tpa) for most of the mine life, until a tail off at the end. Low and full production are sustained over 38 months, with a tail off of 11 months.

The concurrent production from the proposed underground mine and open pit will produce more mineralisation than the processing plant can process. The plant will have 34,000 tpa of spare capacity for underground material, yet 156,000 tpa is planned from the underground mine. The underground material will displace the lowest grade open pit ore. The displacement of open pit ore also means that the immediate unit cost of ore processed from the open pit will increase, and the tail end of the operation in which stockpiled ore is processed but not mined will be extended. The value in the parallel production of more ore than can be processed is in the material increase in grade.

The production schedule is shown in Figures 24-5 and 24-6, and Table 24-5.

FIGURE 24-5 SEGILOLA UNDERGROUND LIFE OF MINE PRODUCTION

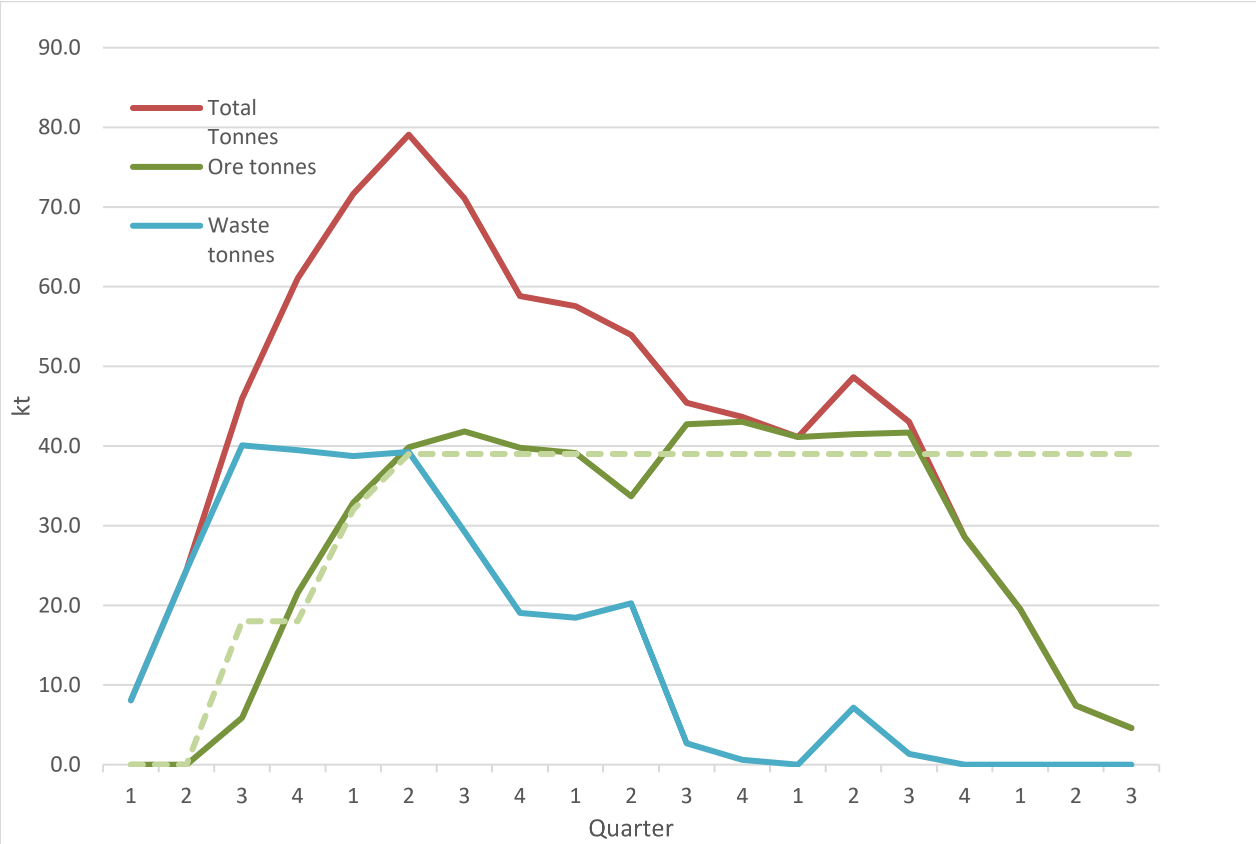


FIGURE 24-6 SEGILOLA UNDERGROUND LIFE OF MINE ORE PRODUCTION

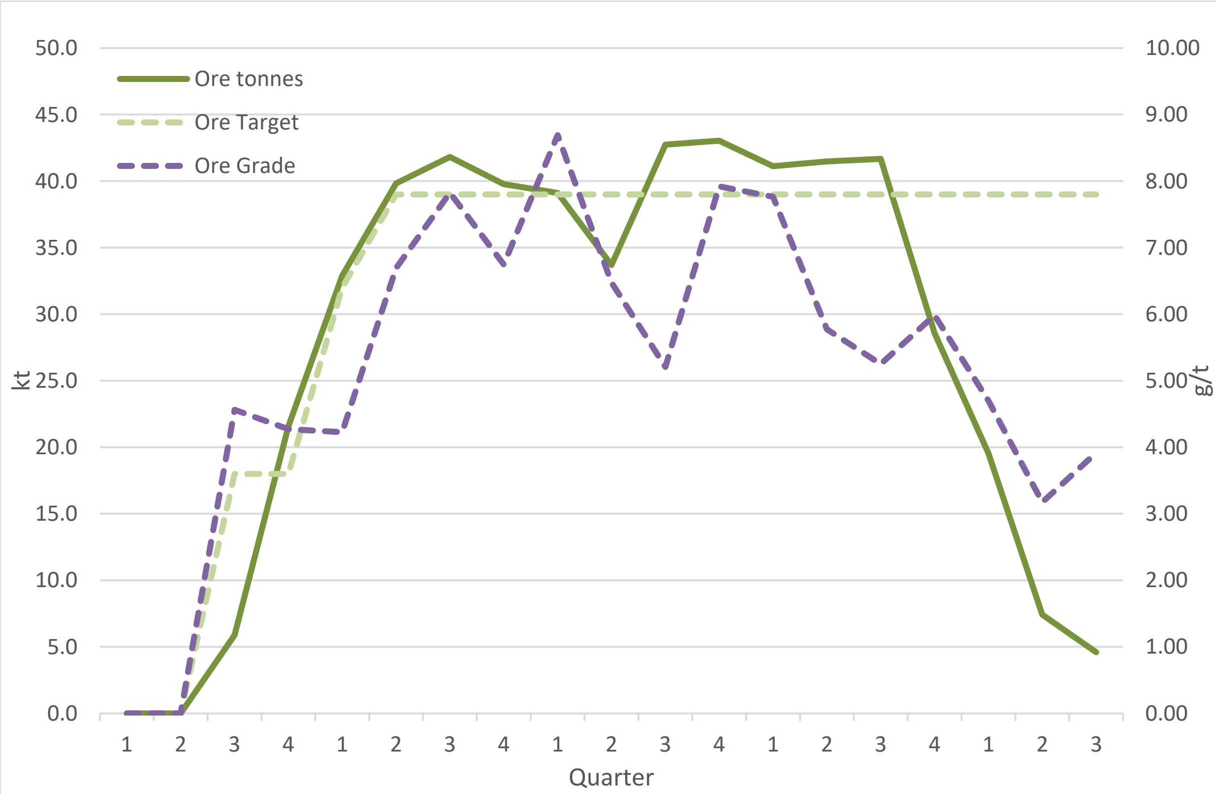


TABLE 24-5 SEGILOLA QUARTERLY UNDERGROUND PRODUCTION SCHEDULE
Thor Explorations Ltd. – Segilola Gold Project

		Total or	2020				2021				2022				2023				2024		
		Average	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3
Total	kt	814	8.1	24.5	46.0	61.0	71.6	79.1	71.1	58.8	57.6	53.9	45.4	43.7	41.1	48.6	43.0	28.6	19.5	7.4	4.6
Ore	**kt**	**525**	**0.0**	**0.0**	**5.9**	**21.6**	**32.9**	**39.8**	**41.8**	**39.8**	**39.1**	**33.7**	**42.7**	**43.0**	**41.1**	**41.5**	**41.7**	**28.6**	**19.5**	**7.4**	**4.6**
Grade	**g/t**	**6.33**	**0.00**	**0.00**	**4.56**	**4.27**	**4.23**	**6.69**	**7.83**	**6.75**	**8.69**	**6.48**	**5.21**	**7.92**	**7.77**	**5.77**	**5.25**	**5.98**	**4.70**	**3.18**	**3.92**
Waste	kt	289	8.1	24.5	40.1	39.5	38.7	39.2	29.3	19.1	18.4	20.2	2.7	0.6	0.0	7.2	1.4	0.0	0.0	0.0	0.0
Target		614	0.0	0.0	18.0	18.0	32.0	39.0	39.0	39.0	39.0	39.0	39.0	39.0	39.0	39.0	39.0	39.0	39.0	39.0	39.0
Ore Production																					
Stope	kt	392	0.0	0.0	0.0	13.4	17.8	24.7	24.4	31.4	34.7	32.0	39.2	42.4	34.8	29.3	23.9	15.9	19.5	7.4	1.0
Grade	g/t	6.40	0.00	0.00	5.84	3.43	4.10	6.25	9.34	6.99	9.38	6.48	5.09	8.00	7.83	5.43	3.60	4.95	4.70	3.18	3.02
Crown Pillar	kt	47	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	6.3	10.4	14.2	12.7	0.0	0.0	3.6
Grade	g/t	6.76	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	7.39	5.82	7.36	7.28	0.00	0.00	4.17
Ore Drives	kt	86	0.0	0.0	5.9	8.2	15.1	15.1	17.4	8.4	4.4	1.7	3.6	0.6	0.0	1.8	3.6	0.0	0.0	0.0	0.0
Grade	g/t	5.81	0.00	0.00	4.56	5.65	4.38	7.41	5.70	5.85	3.34	6.49	6.46	2.42	0.00	11.17	7.85	0.00	0.00	0.00	0.00
Development																					
Total level	m	9,680	162	520	1,067	1,147	1,381	1,534	1,301	757	594	573	200	41	0	252	149	0	0	0	0
Ramp	m	1,919	139	338	322	300	300	280	21	79	139	0	0	0	0	0	0	0	0	0	0
Passing and Re-muck Bays	m	602	23	65	72	103	85	74	58	40	21	13	27	0	0	13	7	0	0	0	0
Cross Cuts	m	697	0	49	123	125	130	76	51	0	70	74	0	0	0	0	0	0	0	0	0
Level Drives	m	1,529	0	40	141	197	67	229	294	100	182	143	0	0	0	136	0	0	0	0	0
Ore Drives	m	4,575	0	28	349	378	762	855	760	518	162	303	173	41	0	102	142	0	0	0	0
Vent Cubbies	m	180	0	0	30	22	19	10	58	10	10	20	0	0	0	0	0	0	0	0	0
Waste Cubbies	m	179	0	0	30	22	18	10	59	10	10	20	0	0	0	0	0	0	0	0	0
Total inclined	m	681	0	0	107	100	35	80	161	90	10	91	8	0	0	0	0	0	0	0	0
Vent Raises	m	340	0	0	64	39	26	30	99	26	5	47	3	0	0	0	0	0	0	0	0
Waste Passes	m	341	0	0	43	60	9	49	62	64	5	43	5	0	0	0	0	0	0	0	0
Total rock	kt	423	8	25	50	51	64	62	52	32	26	29	8	2	0	9	5	0	0	0	0
Level ore (over marginal grade)	kt	136	0	1	10	11	25	23	23	15	5	9	5	1	0	3	4	0	0	0	0
Level ore (below marginal grade)	kt	50	0	1	4	3	10	8	5	7	0	7	2	1	0	1	1	0	0	0	0
Level waste	kt	262	8	24	36	36	37	36	23	13	21	17	2	1	0	6	1	0	0	0	0
Drop raise rock	kt	25	0	0	4	4	1	3	6	3	0	3	0	0	0	0	0	0	0	0	0

MINERAL PROCESSING

Mineralisation will be processed by the plant detailed in the DFS (the DFS plant). The DFS plant has a capacity of 650,000 tpa and the planned throughput of open pit ore is 625,000 tpa, leaving 34,000 tpa of spare capacity. Underground production will be 156,000 tpa when in full production. Currently, Thor plans to add underground mineralisation into the plant until planned capacity is met, which will consume 34,000 tpa of the planned 156,000 tpa of production. The remaining underground material will also be processed by displacing the lowest grade open pit ore, which will be stockpiled instead. This will increase cash flow as the plant will be processing higher grades.

As such, the additional PEA production will impact on the planned open pit production, so the evaluation of the PEA needs to consider the impact of the change in throughput from the open pit.

No metallurgical tests have been performed on the underground mineralisation. However, based on the tests done for the open pit ore, and a geological opinion from Thor that the underground mineralisation is not materially different from the open pit ore, a recovery of 96% (1% lower than for the open pit) was estimated for the PEA.

INFRASTRUCTURE

Both surface and underground infrastructure is required for the underground operation.

The planned underground operations will share parts of the planned open pit infrastructure. Offices, workshop facilities, change rooms, explosives magazines, and other support infrastructure will be modified or extended for underground mining use. Waste dumps and the tailings dam will also be used by the underground operation. The tailings dam wall can be raised to contain the extra tailings from the underground operations, though this means that tailings dam wall raises will occur faster than planned for the open pit alone. There is enough room on the waste dumps for the waste from underground mining.

No cost allowance has been provided for this sharing.

Some underground specific infrastructure is required on surface, such as the portal, fans, and backfill tipping area. Services will be required to some of these points. Except for the supply of services, the cost of these elements has been included.

ENVIRONMENT

The approved EIA for the Project does allow for underground mining. A specific Environment Management Plan will be developed as the planning for the underground operation progresses.

CAPITAL COST ESTIMATE

A detailed capital cost estimate was developed for the open pit operation and reported in the DFS. This capital cost included the processing plant and supporting infrastructure such as roads, power supply, tailings dam, offices, the mining camp, workshops, and magazine. These items have been excluded in the PEA, so only the capital items that directly relate to the underground operation have been included. As the production from the underground operations is materially smaller than the open pit operation, and both the processing plant and tailings dam can handle an increase in production within the currently defined capital cost, all non-underground items are not expected to incur a cost increase and remain unchanged.

A preliminary capital cost was estimated for the items that directly relate to underground mining using 2017 costs for large items and a contingency for small items applied to the results of a basic engineering estimate. Capital costs for ventilation, power, and reticulation infrastructure were based on equipment found in similar operations, or estimated through simple engineering estimates based on an average half level at steady state production. Costs were based on six working half levels and a simple portal into hard rock. While surface fans have been included, the estimate excludes any other surface items such as power lines, pipelines, and extra access roads to the portal.

Thor intends to use a contractor for the mining work, so a mining fleet has not been included in the capital cost. A mining fleet is expected to cost approximately $10.6 million. Assuming this is depreciated by the contractor over the LOM, this will add in the order of $15/t to the operating cost, which was included in the benchmarked operating cost.

A contingency of 30% was included. No Indirects or escalation were included in the operating cost estimate. No closure cost was included as this will be covered by the open pit closure cost. No sustaining capital cost was included given the short life of mine.

Table 24-6 gives the initial capital cost estimate, excluding any capitalised development.

TABLE 24-6 CAPITAL COSTS
Thor Explorations Ltd. – Segilola Gold Project

Item	Cost (k $)
Portal	500
Mobilisation	500
Development	5,887
Ventilation	1,018
Water Reticulation	209
Power	552
Miscellaneous	362
Surface Works	600
Subtotal	9,628
Contingency at 30%	2,888
Total	**12,516**

The capital cost schedule is given in Table 24-7.

TABLE 24-7 CAPITAL SCHEDULE
Thor Explorations Ltd. – Segilola Gold Project

Costs in k $		Month from First Development														
	Total	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**
Portal	250	167	167	167												
Mobilisation	500	500														
Capitalised Operating Cost	3,223				176	336	412	500	670	1,128	1,203	1,461				
Ventilation	1,018			102						407						509
Water Reticulation	209			21						83						104
Power	552			55						221						276
Miscellaneous	362			36						145						181
Surface Works	600		600													
Subtotal	9,628	667	767	381	176	336	412	500	670	1,984	1,203	1,461				1,070
Contingency at 30%	2,888	200	230	114	53	101	124	150	201	595	361	438				321
Total*	**12,516**	**867**	**997**	**495**	**229**	**437**	**536**	**650**	**871**	**2,580**	**1,564**	**1,900**				**1,391**

Note. * Excludes capital for items included in the DFS, such as the processing plant, power supply, tailings dam, roads, offices, mining camp and magazine.

OPERATING COST ESTIMATE

The operating cost estimate was determined by combining costs estimated for the DFS and benchmarks for similar operations in similar conditions.

Detailed costs were estimated during the DFS for the processing, G&A, and surface infrastructure. These costs were applied as a unit cost per tonne of ore mined. Some further G&A costs, treated as a monthly cost, were added in, to include the extra labour, engineering, and geological drilling that can be expected when operating the underground mine.

The application of the DFS costs assumed that the open pit would be operating during the same period as the underground mine, which is what has been planned. It also assumes that the unit costs are not affected by the extra underground material, which is likely to reduce the unit costs to some degree. The accuracy of the DFS derived unit costs is high.

The remaining cost was determined from benchmarks, and this covered the unit mining cost and the unit development cost. Labour, equipment, and consumables were not separately determined.

The benchmark unit costs, DFS derived monthly costs, and the DFS unit costs were combined with the production schedule to create an annual cost profile.

Royalties and refining costs were unchanged from the DFS and included in the overall cost model when completing the economic analysis.

No escalation was included in the operating cost estimate. No escalation was included in the operating cost estimate. Given the use of DFS level estimates and benchmarks to determine the operating cost, no contingency could be determined and has not been included.

While some costs are at DFS level accuracy, the use of benchmarks and the uncertainty in the mine design mean the overall cost estimate accuracy is in line with a PEA at +/-50%.

The main DFS sourced costs are given in Table 24-8. The further G&A costs were estimated with the source rates applied in the DFS. The G&A costs are shown in Table 24-9 and are considered to be reasonably accurate.

TABLE 24-8 DFS BASED OPERATING COST
Thor Explorations Ltd. – Segilola Gold Project

Item	Unit	Value
Operating Consumables	$/t	7.37
Maintenance	$/t	1.46
Labour	$/t	3.07
Power	$/t	4.78
Laboratory	$/t	0.61
General & Administration	$/t	5.77
Subtotal	$/t	23.06

TABLE 24-9 FURTHER G&A COSTS
Thor Explorations Ltd. – Segilola Gold Project

Item	Unit	Value
Labour	k $/month	53.3
Engineering	k $/month	12.1
Infill drilling	k $/month	26.6

Benchmarks were used to determine the unit mining cost which includes drilling, blasting, loading, hauling, and backfilling. The selected cost was an average of some of the benchmarks, adjusted for the relative shallowness of the operation and the cost reduction expected for being an operation with an open pit. The selected cost was $45/t.

A mix of benchmarks was used to determine the cost per metre of three separate development end types, namely the main development (4.0 m x 4.3 m mined), level drives (3.0 m x 3.5 m), and drop raises (2 m diameter).

The operating cost schedule is given in Table 24-10.

TABLE 24-10 OPERATING COST ESTIMATE
Thor Explorations Ltd. – Segilola Gold Project

	Unit	Rate	Total	-1 1 2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
					Quarter from the Start of Development																
Ore Output	kt		525	Capitalised Operating Cost	6	22	33	40	42	40	39	34	43	43	41	41	42	29	20	7	5
Operating Costs																					
	Development																				
Large	$/m	3,994	10,400		2,065	2,106	2,059	1,720	520	476	918	348	108	0	0	54	27				
Small	$/m	3,195	20,425		1,757	1,980	2,766	3,527	3,740	2,039	1,165	1,553	554	130	0	761	454				
Raises	$/m	750	511		81	75	26	60	121	67	8	68	6								
	Production																				
Stopes	$/t ore	45	17,631		1	603	802	1,111	1,100	1,412	1,561	1,438	1,762	1,910	1,566	1,319	1,074	714	879	334	46
	Monthly Costs																				
Further G & A	k $/quarter	160	2,721		160	160	160	160	160	160	160	160	160	160	160	160	160	160	160	160	160
Engineering	k $/quarter	36	619		36	36	36	36	36	36	36	36	36	36	36	36	36	36	36	36	36
Infill drilling	k $/quarter	80	1,360		80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80
Total Direct Underground Cost	**k $**		**53,668**		**4,180**	**5,040**	**5,929**	**6,694**	**5,757**	**4,271**	**3,928**	**3,684**	**2,706**	**2,317**	**1,842**	**2,410**	**1,831**	**991**	**1,156**	**611**	**323**
Total cost per t	**$/t ore**		**6,380**		**1,030**	**703**	**587**	**504**	**414**	**325**	**302**	**331**	**190**	**161**	**134**	**177**	**132**	**105**	**178**	**251**	**854**
Processing and Remaining costs defined in the DFS*	$/t ore	23	12,102		136	497	758	919	964	917	902	777	986	993	948	956	961	659	451	171	106
Total Cost			**65,769**		**4,315**	**5,537**	**6,688**	**7,612**	**6,721**	**5,188**	**4,830**	**4,461**	**3,692**	**3,309**	**2,791**	**3,366**	**2,793**	**1,650**	**1,606**	**782**	**429**
Total cost per t	**$/t ore**		**125**		**733**	**257**	**203**	**191**	**161**	**130**	**123**	**132**	**86**	**77**	**68**	**81**	**67**	**58**	**82**	**105**	**93**

Note. * Operating costs for the processing, infrastructure, other G&A.

ECONOMIC ANALYSIS

Due to the interaction between the planned underground operation and the open pit operation, most notably in the increase in processing throughput, the costs and results of the PEA were combined with the modified costs and results of the DFS in order to understand the correct value of the underground operation. The difference between the DFS results and the combined set of results show the true value of the PEA when added to the current DFS plan. The DFS results are more accurate than the PEA estimates, and caution needs to be exercised when considering the combined value set.

Due to the expected supply of higher-grade material from the underground operation, some open pit ore stockpiling decisions were changed, bringing forward some of the available open pit higher grade ore. This increases the revenue in Year Two before the underground operation has started producing mill feed. Other effects of including the underground production is to change the timing of some costs, such as deferring the mine closure cost. As such the difference, or value add, created by the underground operation is not simply a direct cost or revenue change in line with the PEA cost and revenue schedule.

Details regarding the estimate of revenue, discount rate, taxation, and royalties are unchanged from the DFS. Due to a government agreed tax holiday, pre-tax and post-tax revenues are the same.

The gold price used for the estimate of revenue is $1,300/oz.

The incremental analysis, or difference between the original DFS and the combined results, is shown in Table 24-11. Due to changes in the DFS, some annual values and unit rates determined in the incremental analysis appear unusual, however, the total value is correct.

The net value added by the PEA to the DFS results in an NPV of $34.7M at a 5% discount rate, with a payback period of 1.1 years.

TABLE 24-11 ECONOMIC INCREMENTAL ANALYSIS FOR VALUE ADDED BY THE PEA ALONE
Thor Explorations Ltd. – Segilola Gold Project

Commercial Production Year	Unit	Total/ Ave	2019 -1	2020 1	2021 2	2022 3	2023 4	2024 5	2025 6	2026 7	2027 8
Market Prices											
Gold	$/oz	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Physicals											
OP Mill Feed Mined	kt	0		0	0	0	0	0	0	0	
UG Mill Feed Mined	kt	525		0	47	162	160	138	18	0	
Total Mill Feed Mined	kt	525		0	47	162	160	138	18	0	
Total Waste Mined	kt	287		4	133	112	31	7	0	0	
Total Material Mined	kt	812		4	181	274	191	145	18	0	
Strip Ratio	W:O	1		0	3	1	0	0	0	0	
Total Mill Feed Milled	t	525		4	25	25	25	25	390	30	
Contained Gold, Milled	(000s) oz	107		13	9	25	17	19	22	1	
Average Gold Recovery	%	0.96		0.97	0.96	0.96	0.95	0.96	0.97	0.97	
Payable Gold Sold	(000s) oz	102		12	9	24	16	18	22	1	
Cash Flow											
Gold Gross Revenue	k $	133,246	0	16,117	11,080	31,320	21,092	23,654	28,154	1,829	0
Costs											
Contractor Mining	k $	-62,992	0	0	-13,762	-24,351	-14,253	-9,415	-1,210	0	0
Thor Operations	k $	-9,749	0	-80	-464	-464	-464	-466	-7,244	-566	0
Fixed G&A	k $	-2,094	0	0	0	0	0	0	-1,795	-299	0
Infrastructure/Closure	k $	-424	0	0	0	0	0	0	0	544	-968
Dore Freight/Refining	k $	-679	0	-82	-57	-160	-108	-121	-144	-9	0
Royalty	k $	-1,526	0	-668	-459	457	-242	-271	-322	-21	0
Total Cash Costs	k $	-77,464	0	-830	-14,742	-24,518	-15,066	-10,272	-10,715	-351	-968
Operating Margin	k $	55,782	0	15,287	-3,662	6,802	6,025	13,381	17,438	1,478	-968

Commercial Production Year	Unit	Total/ Ave	2019 -1	2020 1	2021 2	2022 3	2023 4	2024 5	2025 6	2026 7	2027 8
Income Tax	k $	0	0	0	0	0	0	0	0	0	0
Working Capital	k $	0	0	-677	2,957	-1,542	-526	-595	948	-602	102
Operating Cash Flow	k $	55,782	0	14,610	-705	5,260	5,500	12,787	18,386	876	-866
Initial Capital	k $	-12,504	0	-2,563	-9,941	0	0	0	0	0	0
Sustaining Capital	k $	0	0	0	0	0	0	0	0	0	0
Closure Capital	k $	0	0	0	0	0	0	0	0	2,172	-2,172
Total Capital	k $	-12,504	0	-2,563	-9,941	0	0	0	0	2,172	-2,172
Economic Metrics											
Pre-Tax (same as after tax due to tax holiday)											
Free Cash Flow	k $	43,278	0	12,047	-10,646	5,260	5,500	12,787	18,386	3,048	-3,038
Cumulative Free Cash Flow	k $		0	12,047	1,401	6,661	12,160	24,947	43,333	46,381	43,344
NPV @ 5%	k $	34,692	0	11,473	-9,656	4,543	4,525	10,019	13,720	2,166	-2,056
Cumulative NPV	k $		0	11,473	1,817	6,360	10,885	20,904	34,624	36,790	34,734
IRR	%	NA									
Payback Period (from Comm Prod)	Years	1.13									
Mine Life (Processing)	Years	7 years									
Sales Metrics											
LOM Au Sales	000s oz	102	0	12	9	24	16	18	22	1	0
LOM Cash Cost	k $	77,464	0	830	14,742	24,518	15,066	10,272	10,715	351	968
LOM AISC	k $	77,464	0	830	14,742	24,518	15,066	10,272	10,715	-1,821	3,140

25 INTERPRETATION AND CONCLUSIONS

GEOLOGY AND MINERAL RESOURCES

- The Segilola Gold Project is an orogenic-style lode gold deposit which occurs within a regional-scale shear zone. The style of mineralisation is well understood.

- Drilling has delineated three steeply dipping lodes which form an elongate mineralised zone striking 010º and dipping 60º to 70º towards the west within a single robust shear zone, primarily in biotite gneiss. The known mineralised zone is approximately 2,000 m in strike length, between 70 m and 200 m in depth, and between 2 m and 20 m in true thickness.

- The QP considers that the drill hole data has been adequately validated with satisfactory quality control analysis. The quantity and quality of the geological, geotechnical, collar and downhole survey data is sufficient to support Mineral Resource estimation.

- In the opinion of the QP, the analytical data is sufficiently reliable to support Mineral Resource estimation.

- The use of Ordinary Kriging (OK) to estimate the Mineral Resources is considered appropriate based on review of a number of factors, including the quantity and spacing of available data, the interpreted controls on mineralisation, and the style and geometry of mineralisation.

- The estimation was constrained with geological and mineralisation interpretations.

- At a cut-off grade of grade of 0.64 g/t open pit Indicated Mineral Resources are estimated to total 3.0 Mt at an average grade of 4.5 g/t Au for 441 koz of contained gold. The Inferred Mineral Resources estimated for the proposed open pit mine are 0.3 Mt at an average grade of 6.8 g/t Au containing 73 koz of contained gold.

- At a cut-off grade of 2.50 g/t Au the Indicated Mineral Resources estimated for the underground mine are 0.09 Mt at an average grade of 9.4 g/t Au containing 28 koz of contained gold. The Inferred Mineral Resources estimated for the proposed underground mine are 0.3 Mt at an average grade of 7.9 g/t Au containing 90 koz of contained gold.

- The Mineral Resource estimate is consistent with the CIM (2014) definitions as incorporated by reference into NI 43-101.

MINING

- The Segilola deposit is amenable to conventional truck and shovel open pit mining methods and mining is proposed to be undertaken by a suitably qualified mining contractor.

- In the opinion of the QP, the data are sufficiently reliable to support a Mineral Reserve estimate.

- At a cut-off grade of 0.77 g/t Au, the Probable Mineral Reserves proposed to be mined by open pit methods at Segilola are estimated to be approximately 3.0 Mt at an average grade of 4.2 g/t Au, containing 405 koz Au.

- As contemplated in the DFS, the currently estimated Mineral Reserves for Segilola support an open pit mine life of approximately five and a half years at an average mill throughput of 625,000 tpa.

- The open pit operation will mine 49.0 Mt of waste over the life of mine, at an initial rate of approximately 1.4 million tonnes per month (Mtpm) for 25 months and a rate of 0.7 Mtpm for a further 19 months. The average LOM stripping ratio is 16.3.

- Mining ends in Month 46, however milling from a stockpile of ore continues for a further 14 months.

- LOM gold production averages approximately 66,000 oz Au per year.

MINERAL PROCESSING AND METALLURGICAL TESTING

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

- The flowsheet selected is appropriate to the findings of the test work and the process design parameters are in agreement with the test work results.

- The QP considers the modelled recoveries for all ore sources and the process plant and engineering unit costs applied to the Mineral Resource and Ore Reserve process to be acceptable.

- The flowsheet is based on conventional, well-proven gold processing technology and equipment comprising crushing, grinding, gravity recovery and CIL.

- The grind size nominated of 106 microns is considered to be appropriate for optimal gold recovery versus energy input.

- The processing facility has been sized with a nameplate capacity of 650,000 tpa ore. This is adequate for the anticipated rate of mining ore from the open pit and provides additional operating flexibility and future expansion capacity with minimal additional capital expenditure.

INFRASTRUCTURE

- The mine site can be easily accessed by a sealed public road, which is linked to the national road system, however, there is insufficient power and water infrastructure to support the needs of the Project. Site power generation and local water sources will be provided for the Project.

- Planned Project infrastructure includes a processing plant, TMF, power generation station, warehouses, workshops, reagent storage facility, administrative buildings, mining contractor facilities, accommodation camp and natural gas and diesel fuel storage tanks.

- There is adequate space available within the permitted land area for all of the planned Project infrastructure.

- Sufficient natural gas-fuelled generating capacity has been provided to meet the power needs of the Project processing plant and the other on-site infrastructure.

- Adequate provision has been made to meet the water supply needs of the Project through reclaimed water from the TMF and natural rainwater captured by the WSD. The TMF will supply between 40% and 100% of the monthly water demand. The WSD will make up any water deficit from the TMF, particularly during the dry season.

- Raw water from the WSD will be stored in the raw water pond. Raw water provides the majority of the make-up water for the process plant requirements.

- The TMF will be built 1.3 km southwest of the process plant and has sufficient capacity for the current LOM tailings.

ENVIRONMENTAL AND SOCIAL

- Thor has an in-country and local environmental and social team who are managing environmental and social aspects of the Project.

- The location of the Project is in a modified environment which has been logged for over 100 years and has remnants of gold mining from the 1950s.

- The mine area does not directly impact on the permanent dwellings of the nearest communities.

- Compensation for the loss of agricultural crops is being prepared via the RAP in line with Nigerian legislation and international standards. Compensation for damage to crops during exploration activities has been paid and documented.

- Thor has obtained a key environmental permit for the Project, the EIA and the EPRP.

- The EIA has been certified by the Federal Ministry of Environment, subject to standard conditions.

- Management plans have been prepared, or are in the process of preparation, to address environment and/or social impacts during exploration, construction, operation and closure phases.

- Thor has developed a stakeholder engagement plan for consultation with key project stakeholders over the life of the Project.

ECONOMICS

- The Project as envisaged in the DFS, based on an open pit and gold processing operation and a Project operational life of approximately six years, demonstrates strong economics.
- The LOM total gold recovered is 393,000 oz Au, at an annual average rate of 66,000 oz of Au, generating a LOM revenue of $510.9 million using a gold price assumption of $1,300/oz Au.
- Pre-production initial capital is estimated to be $82.3 million
.
- Capital and operating costs have been estimated to a level of accuracy commensurate with a DFS level of study.
- Average LOM AISC are $638 per ounce.
- The Project is expected to benefit from being designated as a Pioneer Industry under Nigerian legislation, resulting in a tax shield for the entire duration of the current LOM.
- Undiscounted payback from the start of commercial production is approximately 1.4 years.
- The Project is most sensitive to gold grade, gold price, and gold recovery. Exchange rates have little impact as most costs and all revenue are in US dollars.

RISKS

A risk assessment was completed for the DFS and the following salient risks were identified. The degree of risk refers to the subjective assessment of how the identified risk could affect the achievement of the Project objectives.

In the QP's opinion, there are no significant risks and uncertainties that could reasonably be expected to affect the reliability or confidence in the exploration information, Mineral Resource or Ore Reserve estimates.

The following definitions were employed by Thor in assigning risk consequence factors to the various aspects and components of the Project:

1. **Low –** Risks that are considered to be average or typical for a deposit of this nature and could have a relatively insignificant impact on the economics. These generally can be mitigated by normal management processes combined with minor cost adjustments or schedule allowances.
2. **Minor –** Risks that have a measurable impact on the quality of the estimate but not sufficient to have a significant impact on the economics. These generally can be

mitigated by normal management processes combined with minor cost adjustments or schedule allowances.

3. **Moderate –** Risks that are considered to be average or typical for a deposit of this nature but could have a more significant impact on the economics. These risks are generally recognizable and, through good planning and technical practices, can be minimised so that the impact on the deposit or its economics is manageable.

4. **Major –** Risks that have a definite, significant, and measurable impact on the economics. This may include basic errors or substandard quality in the basis of estimate studies or project definition. These risks can be mitigated through further study and expenditure that may be significant. Included in this category may be environmental/social non-compliance, particularly in regard to Equator Principles and IFC Performance Standards.

5. **High –** Risks that are largely uncontrollable, unpredictable, unusual, or are considered not to be typical for a deposit of a particular type. Good technical practices and quality planning are no guarantee of successful exploitation. These risks can have a major impact on the economics of the deposit including significant disruption of schedule, significant cost increases, and degradation of physical performance. These risks cannot likely be mitigated through further study or expenditure.

The following definitions were employed by Thor in assigning risk probability factors to the various aspects and components of the Project during the Project life:

1. **Rare –** The risk is very unlikely to occur.
2. **Unlikely –** The risk is more likely not to occur than occur.
3. **Possible –** There is an increased probability that the risk will occur.
4. **Likely –** The risk is likely to occur.
5. **Almost Certain –** The risk is expected to occur.

The predominant Project related risks identified are shown in Table 25-2. A summary of the salient Project related risks identified are shown in Table 25-1.

TABLE 25-1 PROJECT RISK SUMMARY
Thor Explorations Ltd. – Segilola Gold Project

PROBABILITY						
	Almost Certain					
	Likely					
	Possible		2,4	1,3,6,7	2/3a	
	Unlikely			5		
	Rare					
		Low	Minor	Moderate	Major	High
		CONSEQUENCE				

1 – Geology and Mineral Resources: Indicated Resource accuracy limit
2 – Mine Design and Mineral Reserves: Various
3 – Processing: Variations in grade lead to variations in recovery
3/2a – Processing and Mining Equipment Import delays
4 – Environment, Social and Permitting: Various
5 – Capital Cost: Customs duties
6 – Operating Cost: Increase in CNG cost
7 – Other: Major crime incident

TABLE 25-2 SALIENT PROJECT RISKS
Thor Explorations Ltd. – Segilola Gold Project

Project Element	Issue	Risk Consequence	Risk Probability	Mitigation
Geology and Mineral Resource				
	Indicated resource accuracy limits cause a variation of grade and tonnes in the order of +-5%, impacting expected cash flow.	Moderate	Possible	Short term model updates with grade control drilling will allow a better ore grade forecast to be determined in the near term and allow adjustment Potential for higher grades exists too.
	Internal dilution higher than expected, leading to lower grades over the medium term	Moderate	Possible	Short term model updates with grade control drilling will allow a better ore grade forecast to be determined in the near term
Mining				
	Open pit wall stability and potential pit wall failure	Moderate - Major	Rare - Unlikely	Geotechnical work to date has been completed for the wall design, and the hanging wall is known to be competent, reducing this risk. A geotechnical monitoring strategy will be implemented by a Standard Operating Procedure (SOP) Regular reviews of pit wall stability by geotechnical engineers are planned, with implementation of corrective actions where required. Effective wall control blasting and walls scaling practice is planned
	Delays in importing equipment encountered by contractor (lead time as well as import delay at port)	Major	Possible	Correct documents will be obtained from point of export, and a confirmation of the required import documents will be obtained before the equipment leaves. Select logistics provider to give a guarantee of time at the port. If there is an import delay, part of the import tariff may be paid to end the delay, with a view to recovering the tariff at a later date.
Processing				
	Due to variation in ore characteristics and variations in ore grade, periods of variable metallurgical recovery can be expected, including a possible reduction of gravity recovery from 43% to 35%, and gravity and leach recovery below the target 97%.	Moderate	Possible	Variations in ore can be smoothed out by blending off a large enough stockpile on the ROM pad. Finer grind size can be set to prevent loss in recovery (current mill capacity is 650,000 tpa – above the target operating capacity of 625,000 tpa) Gravity change will mean higher grades in CIP circuit

Project Element	Issue	Risk Consequence	Risk Probability	Mitigation
	Delays to equipment delivery for plant during construction period	Major	Possible	Correct documents will be obtained from point of export, and a confirmation of the required import documents will be obtained before the equipment leaves. Select logistics provider to give a guarantee of time at the port. If there is an import delay, part of the import tariff may be paid to end the delay, with a view to recovering the tariff at a later date.
Social				
	Influx – non-locals coming and settling in surrounding villages or along existing public road through mine site: • People coming to site looking for work • Prostitution • Petty trading along existing road • Increase in congestion on road • Increase in pressure on existing services • Squatting • Criminal activity • Adverse impact on project to operate safely • Community unrest due to non- locals taking over villages	Minor	Likely	HR policy – “no jobs at the Gate”, establish employment centre away from site Policing (Govt police) to keep people from moving near site Procurement policy for goods and services – no purchasing at the gate Security Plan and fencing

Project Element	Issue	Risk Consequence	Risk Probability	Mitigation
	Community unrest and project delays caused by: Compensation issues associated with land and crops acquisition – prior to construction: • Difficulty in determining land and crop ownership: • Unrealistic expectation on rates; • Desire for cash not basket of compensation as part of livelihood restoration plan; • Lack of inclusion of gender in compensation process; • Lack of recognition of ecosystems services (wood, bush meat hunting, medicinal plants/herbs) • Ongoing demands for additional compensation as compensation is exhausted by recipients	Delay risk - Moderate General Risk - Minor	Delay risk - Unlikely General risk - Possible	Sequential construction of project reduces delay risk Market rates (not government rates) paid RAP being prepared – contract signed. Process includes market crop rates as compensation (not government rates), methods and witnesses to survey land and assets, and transparency re rates. Team selected who are registered with Govt to undertake and sign off on the surveys. No housing expected to be taken. Gender included in socio-economic survey. Questions on ecosystems services in socio-economic survey and asset survey. Ecosystem components in Biodiversity Management Plan. Livelihood restoration plan as part of RAP. Stakeholder engagement plan documenting ongoing communication with stakeholders – including a grievance mechanism
	Community expectations of benefits from the Project not being fulfilled			Plans and activities to conform with IFC PS
Capital Cost Estimate				
	The rise in the price of raw materials such as steel, cement led to an increase in Capital Cost Estimate, possibly increasing Capital Cost Estimate by 2%.	Minor	Possible	Award the construction contract (turnkey).
Security				
	Major Incident (crime)	Moderate	Possible/ Unlikely	Draw up security plan using expert advice; Implement security plan in phased approach - fencing around site (gold room, camp), cameras, searches, static and mobile guards.
	Ongoing crime (e.g. fuel theft)	Minor	Likely	Draw up security plan using expert advice; Implement security plan in phased approach - fencing around site (gold room, camp), cameras, searches, static and mobile guards.

Project Element	Issue	Risk Consequence	Risk Probability	Mitigation
Operating Cost Estimate				
	Increase in staff salary rate more than inflation	Minor	Likely	For expats, hire and train more local staff
	Increase in CNG cost	Moderate	Possible	Establish stable supply agreement with the local CNG Supplier. Test sensitivities

26 RECOMMENDATIONS

OVERALL

- RPA considers that the DFS demonstrates that the Project generates strong economics and Thor should consider advancing the implementation of the Project as envisaged.

- No significant environmental, social or permitting issues exist that would prevent the development of the Project as presently contemplated.

- Thor's Project implementation approach, based on the appointment of a qualified and experienced EPC contractor offering a fixed price lump sum EPC contract and the use of an experienced mining contractor, is a proven strategy in the global mining industry and elsewhere in Africa. Thor should advance the tender process and evaluation and engage in final negotiations for the EPC and mining contract with the selected contractors.

- The positive results of the PEA for the potential exploitation of the deeper resources by underground methods support the case for further exploration drilling of these resources and the preparation of a PFS.

GEOLOGY

- Generally, positive impacts on grades are observed with bigger sample sizes due to the nuggety nature, therefore, with the sample preparation now being undertaken in country, half-core sampling should be resumed in order to get a higher volume sample.

- Overall, there has been a positive impact on high grades through the use of Metallic Screen Fire Assay, although there are insufficient samples to define a Mineral Resource using them. Thor should undertake Metallic Screen Fire Assays on all samples >10 g/t Au going forward. When there is a sufficient number of analyses, these should be reviewed to determine whether the metallic screen fire assays are suitable to be used for future Mineral Resource estimates.

- The blank insertion rate should be increased to the same frequency as the standards and duplicates for future drilling campaigns. Blanks should also be inserted manually into areas of expected high-grade results to better test sample contamination during preparation.

- Re-instate Inter-laboratory checks (umpire analysis) for all future drilling programmes.

MINERAL RESOURCES

- Although known faults are limited these should be modelled in 3D and incorporated into the geological model to improve the usefulness of the model for future detailed mine planning.

- More bulk density samples should be collected, particularly in the underground resource domains to support the resource estimation.

- Optimised pit shells should be used to guide future drilling programmes to maximise the opportunities to upgrade mineral resource classification.

- A minimum mining width or minimum gram thickness should be applied to exclude any low-grade thin veins in future Mineral Resource estimates.

GEOTECHNICAL STUDIES

- Further geotechnical investigations and analysis will be required as part of detailed mine design, and during open pit mining operations. The data and analysis from the twelve geotechnical drill holes provided a degree of confidence in the recommendations. However, only three holes extended into the east wall, so the analysis should be supplemented with additional triple-tube, oriented drill holes extended into the east wall of the pit and observational methods of design assessment during pit development. This will also provide essential data for the further engineering of the potential underground mine.

- Further hydrogeological studies should be completed to better understand the impact of groundwater on the open pit. This will also be essential input data for the further study of the underground mine.

- Geotechnical studies should be completed for the processing plant and related infrastructure, and for the locations of the waste dumps, if required.

MINING

- As part of the future detailed mine planning, the scheduling of waste mining and the quantity of ore being sent to the stockpiles should be further examined and optimised.

- The positive results of the PEA support further drilling and study work for the preparation of a pre-feasibility study.

PROCESSING

- Additional cyanide destruction test work is recommended to confirm the residence time requirements for effective cyanide destruction.

- A linear vibration screen and belt conveyor can be installed at the end of the SAG mill discharge port to return pebbles to the feed belt of SAG mill. Space for a pebble crusher should be reserved so that it can be installed at a later date if found to be required.

27 REFERENCES

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2014, CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014.

Caby, R. and Boesse, J.M. (2001): Pan-African nappe system in southwest Nigeria: the Ife-Ilesha schist belt. Journal of Africa. Earth Sci. 33 (2): 211-225.

Elueze, A.A. (1986): Petrology and gold mineralisation of the amphibolite belt, Ilesha area, South- Western Nigeria. Geologie en Mijnbouw 65:189-195.

Getsinger, J.S. 1988: Petrological Report, Iperindo, Nigeria. Report for Kanama Industries Nigeria Limited, March 18, 1988. J.S. Getsinger, MPH Consulting Limited, Vancouver, B.C., Canada.

GFDRR, 2018. Osun, Nigeria Hazard Ratings - Oct 2018. Thinkhazard.org: Global Facility for Disaster Reduction and Recovery, World Bank Group, http://thinkhazard.org/en/report/2234-nigeria-osun, Report generated 13-April-2018

Oyinloye, A. O., 2006: Metallogenesis of the Lode Gold Deposit in Ilesha Area of South-Western Nigeria: Inferences from Lead Isotope Systematics Pak. J. Sci. Ind. Res. 2006 49(1) 1-11.

Oyinloye, A.O. and Steed, G.M. (1996). Geology and geochemistry of Iperindo gold deposits, Ilesha schist belt, South-Western Nigeria. Trans. Instn. Min. Metall. (Set. B. Appl. Earth Sci.) 105: B71- B81.

28 DATE AND SIGNATURE PAGE

This report titled "Technical Report on the Segilola Gold Project Feasibility Study, Osun State, Nigeria" and dated 18 March 2019 was prepared and signed by the following authors:

(Signed and Sealed) ***"David JF Smith"***

Dated at London, UK 18 March 2019	David JF Smith, CEng Principal Mining Engineer RPA

(Signed and Sealed) ***"Jack Peter Lunnon"***

Dated at London, UK 18 March 2019	Jack Peter Lunnon, CGeol, EurGeol Senior Geologist RPA

(Signed and Sealed) ***"Grant Malensek"***

Dated at Denver, USA 18 March 2019	Grant Malensek, P.Eng Principal Engineer RPA

(Signed and Sealed) ***"Christopher Speedy"***

Dated at Perth, Australia 18 March 2019	Christopher Speedy, MAIG & MAusIMM Independent Consultant Auralia Mining Consulting Pty Ltd

(Signed and Sealed) ***"Anthony Keers"***

Dated at Perth, Australia 18 March 2019	Anthony Keers, MAusIMM (CP Mining) Principal Mining Engineer Auralia Mining Consulting Pty Ltd

(Signed and Sealed) ***"Qiang Ji"***

Dated at Vancouver, Canada 18 March 2019	Qiang Ji, FAUSIMM Board Chairman CGME Consulting Limited

(Signed and Sealed) ***"Daryl Evans"***

Dated at Perth, Australia 18 March 2019	Daryl Evans, FAUSIMM (224540) Independent Consulting Metallurgist Independent Metallurgical Operations Pty Ltd

(Signed and Sealed) ***"Richard Elmer"***

Dated at London, UK 18 March 2019	Richard Elmer, CEng, MIMMM MCSMI Director and Principal Geotechnical Engineer Knight Piésold Limited

(Signed and Sealed) ***"Marion Ann Thomas"***

Dated at London, UK 18 March 2019	Marion Ann Thomas, Pr.Sci.Nat. Associate: Environmental and Social Consultant Knight Piésold Limited

29 CERTIFICATE OF QUALIFIED PERSONS

DAVID JF SMITH, CENG, FIMMM

I, David JF Smith, C.Eng., as a Qualified Person for this report entitled “Technical Report on the Segilola Gold Project Feasibility Study, Osun State, Nigeria” prepared for Thor Explorations Ltd. and dated 18 March 2019, do hereby certify that:

1. I am Principal Mining Engineer and Managing Director with RPA UK Ltd. of One Fetter Lane, Suite 601, London, EC4A 1BR, UK.

2. I am a graduate of the University of Newcastle upon Tyne, United Kingdom in 1978 with a BSc (Eng) in Mining Engineering.

3. I am registered as a Chartered Engineer (Membership #43860) and a Fellow of Institute of Materials, Minerals and Mining (FIMMM). I have worked as a mining engineer for a total of 40 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Review and report as a mining consultant involved in numerous consulting and engineering assignments including; project technical evaluations, technical report preparation for the purposes of project financing and fund-raisings, regulatory reporting, IPOs, merger and acquisitions, due diligence reviews and engineering studies from scoping to basic engineering
   - Numerous consulting assignments on gold and base metal mine development projects and operating mines in many countries including several countries in Africa
   - Senior positions with a leading international mining and tunnelling contractor, managing international mine and tunnel construction projects as well as developing a successful engineering consulting business
   - Board director for an international mining consulting firm, responsible for leading the UK technical staff, and ensuring the technical quality of the firm’s consulting assignments across the consulting division

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Segilola Gold Project on 19 to 21 September 2018.

6. I am responsible for overall preparation of the report and for related disclosure in Sections 1 to 3, 21, 22, and 25 to 29 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 18th day of March, 2019

(Signed and Sealed) *"David JF Smith"*

David JF Smith, CEng

JACK PETER LUNNON, CGEOL, EURGEOL

I, Jack Peter Lunnon, CGeol, EurGeol, as a Qualified Person for this report entitled "Technical Report on the Segilola Gold Project Feasibility Study, Osun State, Nigeria" prepared for Thor Explorations Ltd. and dated 18th March 2019, do hereby certify that:

1. I am a Senior Geologist of RPA UK Ltd. of One Fetter Lane, Suite 601, London, UK EC4A 1BR.

2. I am a graduate of the University of Southampton, United Kingdom, in 2009 with a Master of Geology (MGeol).

3. I am registered as a Chartered Geologist with the Geological Society of London (Reg. #1022611) and European Geologist with the Federation of European Geologists (Reg. #1456). I have worked as a Geologist for a total of eight years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Mineral Resource estimation and reporting on deposits worldwide for due diligence and regulatory requirements.
   - Experienced user of geological and resource modelling software.
   - Supervision of exploration properties in Africa and the Middle East.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Segilola Gold Project on 9 and 10 February 2019.

6. I am responsible for parts of Sections 1, 14, and 25 to 29 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 18th day of March, 2019

(Signed and Sealed) ***"Jack Peter Lunnon"***

Jack Peter Lunnon CGeol, EurGeol

GRANT A. MALENSEK, P.ENG.

I, Grant A. Malensek, P.Eng., as an author of this report entitled “Technical Report on the Segilola Gold Project Feasibility Study, Osun State, Nigeria” prepared for Thor Explorations Ltd. and dated 18 March, 2019, do hereby certify that:

1. I am a Principal Engineer - Valuations with Roscoe Postle Associates Limited, 143 Union Boulevard, Suite 505, Lakewood, CO, USA 80228.

2. I am a graduate of University of British Columbia in 1987 with a Bachelor’s degree in Geological Sciences. In addition, I have obtained a M.E. in Geological Engineering (Colorado School of Mines, 1997) and a Business Certificate in Foundations of Finance (Daniels College of Business, University of Denver, 2011).

3. I am a Professional Engineer of the Association of Professional Engineers & Geoscientists of British Columbia. I have worked as an Engineer for a total of over 24 years since my graduation from university. My relevant experience includes business experience in financial analysis, project management and business development including, but not limited to, the following relevant projects:
   - Ahafo gold mining district,Ghana (Newmont Mining)
   - Mt. Nimba iron project, Guinea (SMFG JV)
   - Montagne d’Or gold project, French Guiana (Nordgold)
   - Siembra Minera gold project, Venezuela (Gold Reserve)
   - Hope Bay gold project, Canada (Newmont Mining)

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have not visited the Segilola Project property.

6. I am responsible for Section 22 and summary portions of Section 21 and related disclosure in Sections 1, 24, 25, 26, and 27 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 18th day of March, 2019

(Signed and Sealed) ***“Grant A. Malensek”***

Grant A. Malensek, P.Eng.

CHRISTOPHER SPEEDY, MAIG, MAUSIMM

I, Christopher Speedy BSc (Geology), MPAF, MAIG & MAusIMM, as a Qualified Person for this report entitled "Technical Report on the Segilola Gold Project Feasibility Study, Osun State, Nigeria" prepared for Thor Explorations Ltd. and dated 18 March 2019, do hereby certify that:

1. I am an Independent Consultant of Auralia Mining Consulting Pty Ltd, 19-21 Outram Street, West Perth, WA, Australia, 6005
2. I attended the University of Otago, New Zealand where I earned a Bachelor of Science degree in Geology in 2006 and attended Massey University, New Zealand where I earned a Masters in Professional Accounting and Finance in 2018.
3. I am a Member of the Australian Institute of Geoscientists (MAIG) (Reg # 5349) and the Australian Institute of Mining and Metallurgy (MAusIMM) (Reg # 313313). I have worked as a Geologist for a total of 12 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Mineral Resource estimation and reporting on deposits worldwide for due diligence and regulatory requirements.
   - Experienced user of geological and resource modelling software.
   - Supervision of exploration properties in Australia.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Segilola Gold Deposit on February 2, 2019
6. I am responsible for Sections 4 to 12, and parts of Sections 1, 14, and 25 to 29 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I previously contributed to the preparation of the Segilola Preliminary Feasibility Study in October 2017.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains/Section No 5-12 & 14s. in the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 18th day of March, 2019

(Signed and Sealed) ***"Christopher Speedy"***

Christopher Speedy BSc (Geology), MPAF, MAIG & MAusIMM

ANTHONY KEERS, MAUSIMM, CP

I, Anthony Keers, MAusIMM (CP Mining), as an author of this report entitled "Technical Report on the Segilola Gold Project Feasibility Study, Osun State, Nigeria" prepared for Thor Explorations Ltd and dated 18 March 2019, do hereby certify that:

1. I am Principal Mining Engineer and Director of Auralia Mining Consulting Pty Ltd of Suite 1, 19-21 Outram Street, West Perth, WA Australia 6005.

2. I am a graduate of the University of Queensland in 2001 with a Bachelor of Engineering (Mining).

3. I am a member and Chartered Professional (CP Mining) of the Australian Institute of Mining and Metallurgy (AusIMM), registration number 209571. I have worked as a mining engineer for a total of 17 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Review and report as a mining consultant involved in numerous consulting assignments including; project technical evaluations, technical report preparation at PEA, PFS and DFS level, due diligence reviews and production support for projects in Australia, South America, Asia and Africa.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Segilola Gold Project on 4 to 6 May 2017.

6. I am responsible for sections 15, 16 and parts of sections 25 to 29 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have undertaken preliminary technical work (Preliminary Economic Assessment and Pre-Feasibility Study) on the project that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 18th day of March, 2019

(Signed and Sealed) ***"Anthony Keers"***

Anthony Keers, MAusIMM (CP Mining)

QIANG JI, FAUSIMM

I, Qiang Ji, FAUSIMM, as a Qualified Person for this report entitled "Technical Report on the Segilola Gold Project Feasibility Study, Osun State, Nigeria" prepared for Thor Explorations Ltd. and dated 18 March 2019, do hereby certify that:

1. I am the Board Chairman with CGME Consulting Limited Suite 660, 505 Burrard Street, Box 27 Vancouver B.C. Canada V7X 1M4.

2. I am a graduate of Northeast University of China in 1987 with a bachelor's degree in mineral processing.

3. I am a Fellow of the Australasian Institute of Mining and Metallurgy (FAUSIMM) with the Member No: 322442. I have worked as a metallurgical engineer for a total of [32] years since my graduation. My relevant experience includes and is not limited to:
   - Technical Director: Changshanhao (CSH) Gold Mine Expansion Project for Inner Mongolia Pacific Mining Co., Ltd.
   - Technical Director: Phase II Expansion Project of Jiama Copper Gold Polymetallic Mine for the Tibet Huatailong Mining Development Co., Ltd.
   - Technical Director: Wushan Copper Molybdenum Mine of the Inner Mongolia Mining Co. of China Gold Group.)
   - Technical Director: Shangdong Shaling Gold Mine Project for Laizhou Huijin Mining Investment Co., Ltd.
   - Technical Director: KOKA Gold Mine Project in Eritrea.
   - Chief Engineer: Zimudang Gold Mine for Jinxing Gold Mining Co., Ltd in Guizhou Province.
   - Chief Engineer, On-site Instructor: Sanxin Au-Cu Mine Mineral Processing Plant Technical Reform and Expansion Project in Hubei Province.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI43-101.

5. I have not visited the project site.

6. I am responsible for Sections 13 and 17, and parts of Section 18 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 18th day of March, 2019

(Signed and Sealed) ***"Qiang Ji"***

Qiang Ji, FAUSIMM

DARYL EVANS, FAUSIMM

This certificate applies to the technical report prepared for Thor Exploration Ltd. entitled: "Technical Report on the Segilola Gold Project Feasibility Study, Osun State, Nigeria".

1. I am an independent Consulting Metallurgist employed by Independent Metallurgical Operations Pty Ltd (IMO) located at: 88 Thomas St, West Perth Western Australia 6005.

2. I graduated with a degree in extractive metallurgy from the Murdoch University in 1985 and have worked continuously in the Mining Industry as a metallurgist for over 30 years. I have Gold specific consulting and field experience relevant to the project, including all unit operations defined in the testwork reported in Chapter 13 of the Technical Report.

3. I am a Fellow of the Australasian Institute of Mining and Metallurgy, member number 224540.

4. I have read the definition of "qualified person" set out in the National Instrument 43-101 -Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, professional affiliation and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.

5. I was responsible for scoping, implementation and interpretation of the metallurgical testwork performed by IMO and summarised in Chapter 13 of the Technical Report.

6. I have not visited the Segilola Gold Project, Nigeria site.

7. I am not aware of any material fact or material change with respect to the metallurgical testwork performed by IMO in the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

8. I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.

9. I consent to the release of Chapter 13 of the Technical Report and verify that it fairly and accurately reflects the information in the supporting documentation relating to the metallurgical testwork performed by IMO.

Dated 18th day of March, 2019

(Signed and Sealed) ***"Daryl Evans"***

Daryl Evans, FAUSIMM (224540)

RICHARD ELMER, CENG, MIMMM MCSMI

I Richard Elmer, CEng, MIMMM MCSMI, as a Qualified Person for this report entitled "Technical Report on the Segilola Gold Project Feasibility Study, Osun State, Nigeria" prepared for Thor Explorations Ltd. and dated 18th March 2019, do hereby certify that:

1. I am Director and Principal Geotechnical Engineer with Knight Piésold Limited, St Magnus House, 3 Lower Thames Street, LONDON, UK, EC3R 6HD.

2. I am a graduate of the University of Southampton in 1987 with a Bachelor of Science degree in Geology and Master of Science degree in Mining Geology from the Camborne School of Mines in 1988.

3. I am a Chartered Engineer with the UK Engineering Council (No 472325) and Member of the Institute of Materials, Minerals and Mining. I have worked as a geotechnical engineer for a total of 30 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Design and audit of tailings facilities worldwide including the following mines in Africa:
   - North Mara Gold Mine, Tanzania – annual tailings audits
   - Dabola Bauxite Project, Guinea – detailed feasibility study for tailings management facility and water supply dam
   - Inata Gold Mine, Burkina Faso – tailings storage facility raise design for construction
   - Yaoure Gold Project, Cote d'Ivoire – PFS level tailings design
   - Youga Gold Project, Burkina Faso – Tailings expansion design and audits
   - Jilaye Gold Project, Ethiopia – Feasibility level design for combined tailings storage facility and waste rock dump
   - Bisha Gold Mine, Eritrea – Review of annual tailings audits and detailed design of raise

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Segilola Gold Project on 9-10 February 2019.

6. I am responsible for sub-sections in Section 18 of the Technical Report related to the Tailings Storage Facility (TSF), the Water Supply Dam (WSD) and related disclosure in Sections 1, 21, 25, 26, and 27.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 18th day of March 2019

(Signed and Sealed) ***"Richard Elmer"***

Richard Elmer, CEng

MARION THOMAS PR.SCI.NAT

I, Marion Thomas, Pr.Sci.Nat., as a Qualified Person for this report entitled "Technical Report on the Segilola Gold Project Feasibility Study, Osun State, Nigeria" prepared for Thor Explorations Ltd. and dated 18th March 2019, do hereby certify that:

1. I am an Associate: Environmental and Social Consultant with Knight Piésold Limited, St Magnus House, 3 Lower Thames Street, London United Kingdom, EC3R 6HD.

2. I am a graduate of University of Pretoria in 1991 with a MSc degree.

3. I am a professional natural scientist (Pr.Sci.Nat.: number 120813). I have worked as an environmental and social consultant, and engineering geologist for a total of 35 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Environmental and social due diligence of a gold mine in Mali (placed into care and maintenance) on behalf of potential investors
    - Environment and social due diligence for two gold mines in Liberia on behalf of investors
    - Environmental and social audits for a gold mine in Democratic Republic of Congo as part of legal requirements for operating mines
    - Environmental and social due diligence for a proposed gold mine, Finland on behalf of a consortium of banks
    - ESIA for a proposed gold mine in Liberia; Project Director, technical advisor and client liaison

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Segilola Project on 9th February 2019.

6. I am responsible for Section 20 (Environmental studies, permitting, and social or community impact) and related disclosure in Sections 1, 25, 26, and 27 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 18th day of March 2019

(Signed and Sealed) ***"Marion Ann Thomas"***

Marion Ann Thomas, Pr.Sci.Nat.

30 APPENDIX 1

CASH FLOW MODEL BY YEAR

TABLE 30-1 CASH FLOW MODEL BY YEAR
Thor Explorations Ltd. – Segilola Gold Project

Economic Model Annual Summary											
RPA Company: Thor Exploration; Project Name: Segilola; Scenario Name: 625ktpa By Year; Analysis Type: DFS											
Calendar Year			2019	2020	2021	2022	2023	2024	2025	2026	2027
Commercial Production Timeline in Years			-1	1	2	3			6	7	8
Time Until Closure In Years	US$ & Metric Units	LoM Avg / Total	7	6	5	4			1	-1	-2
Market Prices											
Gold	US$/oz	$1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Physicals											
OP Ore Mined	kt	3,002	-	285	857	663	501	697	-	-	-
UG Ore Mined	kt	-	-	-	-	-	-	-	-	-	-
Total Ore Mined	kt	3,002	-	285	857	663	501	697	-	-	-
Total Waste Mined	kt	48,991	-	8,109	16,490	14,222	8,336	1,834	-	-	-
Total Material Mined	kt	51,993	-	8,394	17,347	14,884	8,836	2,531	-	-	-
Strip Ratio	W:O	16.3	-	28.5	19.2	21.5	16.6	2.6	-	-	-
Total Ore Milled	kt	3,002	-	241	625	625	625	627	260	-	-
Gold Grade, Milled	g/t	4.20	-	4.70	5.48	3.74	3.43	5.09	1.48	-	-
Contained Gold, Milled	koz	406	-	36.4	110.2	75.2	69.0	102.6	12.3	-	-
Recoverable Gold, Milled	koz	393	-	35.3	106.9	72.9	66.9	99.5	12.0	-	-
Average Gold Recovery	%	97.0%	--	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	--	--
Payable Gold Sold	koz	393.0	-	35.2	106.7	72.8	66.9	99.4	11.9	-	-
Cash Flow											
Gold Gross Revenue	**$000s**	**510,938**	**-**	**45,792**	**138,767**	**94,696**	**86,946**	**129,203**	**15,535**	**-**	**-**
Contractor Mining Cost	$000s	(153,040)	-	(21,848)	(48,332)	(43,669)	(29,558)	(9,562)	(72)	-	-
Thor Operations	$000s	(55,774)	-	(4,472)	(11,610)	(11,610)	(11,610)	(11,642)	(4,828)	-	-
Fixed G&A Cost	$000s	(17,348)	-	(1,196)	(3,589)	(3,589)	(3,589)	(3,589)	(1,795)	-	-
Total Infrastructure/Closure Cost	$000s	(6,544)	-	(1,040)	(424)	(1,352)	(920)	(479)	(1,361)	(968)	-
Dore Freight/Refining Cost	$000s	(2,605)	-	(234)	(708)	(483)	(443)	(659)	(79)	-	-
Royalty	$000s	(13,352)	-	(1,898)	(5,752)	(3,048)	(996)	(1,480)	(178)	-	-
Total Cash Costs After By-Product Credits	**$000s**	**(248,664)**	**-**	**(30,688)**	**(70,416)**	**(63,751)**	**(47,116)**	**(27,412)**	**(8,313)**	**(968)**	**-**
Operating Margin 51%	**$000s**	**262,274**	**-**	**15,104**	**68,352**	**30,945**	**39,829**	**101,791**	**7,222**	**(968)**	**-**
Income Tax Incl. Tax Holiday	$000s	-	-	-	-	-	-	-	-	-	-
Working Capital	$000s	0	(1,393)	1,869	451	(1,356)	(748)	(707)	1,975	(27)	(66)
Operating Cash Flow	**$000s**	**262,274**	**(1,393)**	**16,973**	**68,803**	**29,589**	**39,082**	**101,084**	**9,197**	**(995)**	**(66)**
Initial Capital	$000s	(82,336)	(46,424)	(35,913)	-	-	-	-	-	-	-
Sustaining Capital	$000s	-	-	-	-	-	-	-	-	-	-
Closure/Reclamation Capital	$000s	(2,172)	-	-	-	-	-	-	-	(2,172)	-
Total Capital	**$000s**	**(84,508)**	**(46,424)**	**(35,913)**	**-**	**-**	**-**	**-**	**-**	**(2,172)**	**-**
Cash Flow Adj./Reimbursements	**$000s**	**-**	-	-	-	-	-	-	-	-	-
LoM Metrics											
Economic Metrics											
Annual Discount Factors	EOP 5%		1.0000	0.9524	0.9070	0.8638	0.8227	0.7835	0.7462	0.7107	0.6768
a) Pre-Tax											
Free Cash Flow	$000s	**177,766**	(47,817)	(18,940)	68,803	29,589	39,082	101,084	9,197	(3,167)	(66)
Cumulative Free Cash Flow	$000s		(47,817)	(66,756)	2,047	31,636	70,717	171,801	180,998	177,831	177,766
NPV @ 5%	$000s	**138,034**	(47,817)	(18,038)	62,406	25,560	32,153	79,202	6,863	(2,250)	(44)
Cumulative NPV	$000s		(47,817)	(65,854)	(3,448)	22,112	54,265	133,466	140,329	138,079	138,034
IRR	%	**50.5**									
Payback Period (from Comm Prod)	Years	**1.4**									
b) After-Tax											
Free Cash Flow	$000s	**177,766**	(47,817)	(18,940)	68,803	29,589	39,082	101,084	9,197	(3,167)	(66)
Cumulative Free Cash Flow	$000s		(47,817)	(66,756)	2,047	31,636	70,717	171,801	180,998	177,831	177,766
NPV @ 5%	$000s	**138,034**	(47,817)	(18,038)	62,406	25,560	32,153	79,202	6,863	(2,250)	(44)
Cumulative NPV	$000s		(47,817)	(65,854)	(3,448)	22,112	54,265	133,466	140,329	138,079	138,034
IRR	%	**50.5**									
Payback Period (from Comm Prod)	Years	**1.4**									
Operating Metrics During Mining Phase											
Mine Life (Processing)	Years	**6**									
Maximum Daily Mining Rate	t/d mined	**47,582**	-	22,832	47,582	40,860	24,241	6,972	-	-	-
Maximum Daily Milling Rate	t/d milled	**1,741**	-	1,338	1,736	1,736	1,736	1,741	722	-	-
Mining Cost	$ / t moved	**$2.85**	-	2.57	2.73	2.88	3.25	3.24	0.28	-	-
Mining Cost	$ / t milled	**$50.97**	-	90.75	77.33	69.87	47.29	15.26	0.28	-	-
Thor Operations	$ / t milled	**$18.58**	-	18.58	18.58	18.58	18.58	18.58	18.58	-	-
Fixed G&A Cost	$ / t milled	**$5.78**	-	4.97	5.74	5.74	5.74	5.73	6.90	-	-
Infrastructure Cost	$ / t milled	**$2.18**	-	4.32	0.68	2.16	1.47	0.76	5.24	-	-
Offsite Costs	$ / t milled	**$0.87**	-	0.97	1.13	0.77	0.71	1.05	0.30	-	-
Royalty/Production Taxes	$ / t milled	**$4.45**	-	7.88	9.20	4.88	1.59	2.36	0.68	-	-
Total Cash Costs	$ / t milled	**$82.82**	-	127.47	112.66	102.00	75.39	43.74	31.99	-	-
Sales Metrics											
LOM Au Sales	koz	**393**	-	35	107	73	67	99	12	-	-
LOM Cash Cost	$000s	**$248,664**	-	30,688	70,416	63,751	47,116	27,412	8,313	968	-
LOM AISC	$000s	**$250,836**	-	30,688	70,416	63,751	47,116	27,412	8,313	3,140	-
LOM Cash Cost / oz Au	$ / oz Au	**$633**	-	871	660	875	704	276	696	-	-
LOM AISC / oz Au	$ / oz Au	**$638**	-	871	660	875	704	276	696	-	-
LOM Avg. Annual Au Sales	koz / yr	**66**									